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23 МЕЖДУНАРОДНАЯ КОНФЕРЕНЦИЯ НАУЧНО-ИССЛЕДОВАТЕЛЬСКИХ ИНСТИТУТОВ ПО ГОРНОЙ БЕЗОПАСНОСТИ

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FOREWORD

The Proceedings for the 23rd International Conference of Safety in Mines Research Institutes contains 100 papers from 20 countries and is organized into 14 sessions. The majority of the papers are published in English (original or translation); where an original text in a foreign language was made available, it follows the English version.

The Organizing Committee wishes to thank the authors and Country Coordinators for their cooperation. Especial thanks goes to Mrs. Ethel T. Wallace and Thomas H. Koslosky of the Bureau of Mines for compiling the Proceedings.

David R. Forshey
Chairman
Organizing Committee

NOTICE

The views and conclusions contained in this document are those of the authors and should not be interpreted as necessarily representing the official policies of the Department of the Interior, Bureau of Mines, or of the U.S. Government.

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Session I

MINE FIRES

REVIEW PAPER: MINE FIRES

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ABSTRACT

The subject of mine fires is a very wide one. To give reasonable coherency, the paper concentrates on the behaviour of fires in the ventilated airways of mines; spontaneous combustions are not considered. The following topics are considered in some detail:

1) Material selection: Performance requirements for mining materials in terms of flammability and smoke/fume production; methods of test and need for relevance to mining conditions.

2) Fire growth: The influence of material properties and method of use on the rate of growth of fire in a mine roadway; the influence of ventilation, slope, material orientation, proximity of walls and roof on fire behaviour.

3) The fire in the ventilation network: The influence of fire on the ventilation distribution; the circulation of fire products in the network.

The influences of these topics in determining fire control strategy are then considered but topics such as fire detection and fire extinction methods are not reviewed in detail.

An extensive Bibliography on papers relating to mine fires is included.

INTRODUCTION

The subject of fires in mines is an important and wide ranging one. In preparing a review paper on this subject for the 23rd meeting of Safety in Mines Research Institutes, I have no need to stress the importance of the subject; most mining countries will have some unfortunate example from their own experience to consider.

Because of the wide ranging nature of the subject, I have restricted my coverage mainly to the topics of fire growth, flammability of materials, ignition sources and fire/ventilation interactions. The important topics of fire detection and extinction are omitted for space reasons. Similarly, I consider only open fires and not spontaneous combustion, on which topic many extensive reviews already exist.

There is an extensive literature on the topic of mine fires, much of it from the previous biennial meetings of Safety in Mines Research Institutes.

In the paper, I have cited a few specific references in order to make specific points; these are listed under 'References' at the end of the paper in the normal way. However, there are so many references that one could cite that, rather than interrupt the flow of the paper, I have compiled a Bibliography which is given separately.

In preparing the Bibliography, I have included papers on mine fires presented at SMRI meetings from the 8th meeting onwards. The choice of this particular meeting, at Pittsburgh in 1959, has a certain symmetry. It represents two complete cycles of meetings, via Washington in 1975 and back to Washington in 1989,

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and it was thirty years ago. (On a purely personal note, it was also the year in which I first became involved in research into mine fires!).

The Bibliography deals only with open fires and not spontaneous combustion. It subdivides references into the following subject areas:

A : general, fire protection, systems approach
B : fire growth, flammability of materials, ignition sources
C : fire/ventilation interactions
D : fire detection
E : fire extinction and control.

The number and comprehensive nature of the papers cited in the Bibliography are themselves an eloquent comment on the value of the biennial SMRI meetings in building up a systematic data base on this and other mine safety topics.

MINE FIRE STATISTICS

For management to control the mine fire problem satisfactorily, reliable statistics are essential. In the UK, any reported fire, no matter how small is the subject of an individual statistical return to the Mines Inspectorate, detailing location, size, materials involved, cause, as far as these are known.

Approximately 50-60 fires are reported each year by this route and this enables a detailed picture of causative factors and materials at risk to be developed. These statistics are reported annually in the Chief Inspector's report and trends or other features of significance are commented upon (e.g. Mines Health and Safety 1985/86).

From time to time, more detailed analyses are published, e.g. fires on conveyor systems were analysed (Cutler and Currie, 1979). Over a period of 21 years, 1205 mine fires were reported of which 405 affected the conveyor in some way. The analysis showed that on 91 occasions, the conveyor belt had been exposed to fire but on no occasion had fire spread along the conveyor belt to an appreciable extent independently of a supporting fire in another material, indicating an acceptable level of performance. Of these 91 fires, 40 had started at the conveyor drive, 34 had started at defective idlers and 14 had been started by friction elsewhere on the conveyor structure. Design features such as the possibility of starting the conveyor drive with the brakes on, accumulation of coal dust around bottom rollers, friction within damaged roller bearings were identified as causative factors. These factors were in turn the subject of research or design improvements on, respectively, interlocking arrangements, elimination of surfaces where dust could accumulate and means of detecting overheated rollers.

Thus, a continuing series of detailed statistics on mine fires is a valuable aid in comparing performance between countries, in comparing performance over a period of time and in identifying areas where improvements, which may require research action, are required.

Apart from such statistical analyses, fires above a certain level of severity need to be studied individually for causative factors and for the development of suitable preventative measures.

However, such an approach is not sufficient of itself; it can only be a basis for reactive policies to fires that have occurred.

In a high risk industry such as coal mining, at a time of great technological change, pro-active policies are required as well, to anticipate problems that might arise and eliminate them before a serious fire occurs.

For instance, the above discussion on conveyor fires relates to statistics gathered in a period when one main type of belt was used and the conveyors were mainly used to trans-

port coal to the pit bottom, where it was wound out of the shaft.

Technological change could lead to the development of new types of conveyor belt; it has led to a much greater use of conveyors to take the coal from the mine via 1 in 4 drifts and to some conveyor installations mounted near to the roof.

In considering the impact of such technological changes, one must ask the question - what if the conveyor catches fire?

What will be the behaviour of the new type of belt?

What will be the effect of slope on fire propagation?

What will be the effect of proximity to the roof on fire propagation?

A crucial role of mine fire research is to provide answers to such questions, in advance of fires occurring, so that mining engineers may make informed decisions on the management of change.

SYSTEMS AT RISK OF FIRE

The mechanical and electrical engineering systems used in mining represent the major potential for ignition, because of their high power inputs, and also a substantial fire load for a developed fire. Mechanical and electrical engineers therefore have an important role to play in the identification of potential ignition sources and their elimination by design improvements.

In the previous section, I discussed some of the ways in which conveyor system design and installation details could affect the fire risks in a mine. This discussion can be extended to other systems where design decisions can affect fire risks.

Hydraulic Power Systems

Hydraulic power systems may contain several hundred litres of combustible hydraulic fluid under high pressure; a loss of containment from such a system could eject the fluid into a mine roadway as a jet or spray, with an attendant high fire risk.

Over the years, much research effort has gone into the development of hydraulic fluids of reduced ignitability that can be used efficiently in mining hydraulic machinery.

The problem is a complex one because of the many characteristics required of hydraulic fluids. The efficient transmission of power, good lubricity, low corrosion potential and reasonable cost are some of these. Except for some specialised applications, such as some designs of fluid coupling which operate using water, and for roof support applications, where a 95% water 5% oil emulsion can be used, the hydraulics fluids that meet the operating requirements are still combustible.

At the moment, the most that can be achieved is to make hydraulic fluids much more difficult to ignite in either a spray or a seepage situation. Test methods have been developed that rank fluids in terms of ignition characteristics. One problem encountered in this development work was that one class of fluids achieved low ignitability by using halogenated compounds; the breakdown products of these fluids on exposure to heat were very unpleasant and the production of smoke and fumes offset their low ignitability advantages.

Prevention of loss of containment therefore remains crucial for fire prevention. Compatibility of seals and fluids is important to prevent leakage. Good design of equipment can prevent vibration and frictional damage to hoses and connectors.

Vehicles

Battery operated vehicles have proved to be ignition sources in the past; tracking between terminals has led to

localised but intense and prolonged fires as the stored energy of the battery discharges and sustains an external fire in battery materials. The incidence of such fires has been greatly reduced by design improvements that reduce the risk of tracking by maintaining clean conditions around the battery terminals and by using improved materials of construction.

Diesel engine powered vehicles represent an appreciable fire risk. The risks associated with the engine itself are addressed by design features that greatly reduce the chances of the engine igniting a flammable atmosphere, if it should encounter one, by the use of flame traps, shut off valves etc and by the requirement to maintain external surfaces of an engine below 150°C, to prevent the ignition of grease, hydraulic fluid, coal dust, textiles etc that might come into contact with engine surfaces.

The fuel load of a vehicle and, for free steered vehicles, the tyres represent substantial fire loads. The fuel load is protected by good mechanical design of the fuel tank and siting it in such a way that it is protected from impact. Nevertheless, a fire on a diesel powered vehicle will, if left unchecked, sooner or later involve the fuel load with very serious consequences. One possibility for reducing this risk is to provide a fire protection coating for the fuel tank; a modern coating could provide 60 minutes of thermal insulation of a tank without undue weight penalty.

Currently, there does not seem any possibility of reducing the fire risk associated with vehicle tyres themselves. In the UK, filling materials are sometimes used in vehicle tyres to improve vehicle stability and usability in the event of tyre damage; progress has been made in a search for filling materials of reduced flammability.

For these reasons, there is considerable interest in providing on board fire fighting systems for FSVs. Such systems may be either manually or automatically operated and direct an extinguishing medium to known points of risk, e.g. brakes, bearings, engine compartment.

Cables and electrical equipment

Faults on power circuits can be a source of ignition for a fire; the continuing discharge of electrical energy before the circuit breakers operate can lead to intense but localised fires in cable insulation which may spread to other materials. Research has been carried out into very rapid and sensitive means of detecting faults on power circuits, where some types of fault can now be detected in 60 microseconds, and on rapid means of breaking circuits. Use of such methods should help to reduce the incidence of fires of electrical origin.

MATERIALS

Coal mines represent a closely controlled environment in which it is possible to exercise rigorous control over the flammability of the materials introduced.

The fire risk represented by a material in a mine is critically affected by the ignition and fire growth characteristics of the material itself, the quantity used, the exposed surface area, the way in which it is used and (for liquids) the integrity of containment.

In this section, I will simply consider some of the major classes of materials encountered.

Roadway Linings

From the fire protection point of view, inorganic roadway linings are obviously to be preferred where these are acceptable on operational grounds. Where organic materials are considered essential, a fire problem obviously arises because of the very large quantities of materials that

will be used.

Wood is the traditional material for this purpose and its fire behaviour has been studied extensively. Surface coating of wood with an inert coating and pressure impregnation with fire retardant salts are techniques that are available for reducing the rate of fire development in wood linings.

Polyurethane foam has been used as a roadway lining, both as a sealant, to prevent air exchange between the roadway and waste areas thereby reducing spontaneous combustion risks, and as a thermal insulation medium, to reduce heat transfer to the ventilating air in deep workings. Several disastrous fires have resulted (e.g. Michael Colliery, UK, 1967, 9 died; Sunshine Mine, USA, 1972, 91 died; Kinross Mine, S Africa, 1986, 177 died). The low density and low thermal conductivity of this form of polyurethane, which make it attractive for these applications, combine to cause a very rapid heating up of the surface when exposed to a small heat source. This behaviour leads to extremely rapid flame spread, orders of magnitude greater than for materials such as wood.

Plastics

The word 'plastics' cover a wide range of substances, some of which burn vigorously and some slowly. The behaviour of plastics in fires may be modified substantially by additives or by changes in formulation. Many plastics emit thick unpleasant smoke when burning.

I have already mentioned conveyor belt, cable insulation and polyurethane foam roadway linings. Other examples of the use of plastics in mining on a potentially large scale are in ventilation ducts, vehicle components, brattice cloth, chilled water pipe insulation and cavity filling.

With such a range of materials and applications it is difficult to generalise beyond saying that materials must be tested in a manner relevant to its proposed conditions of use, taking due account of smoke and toxic fume production.

Liquids and Greases

Lubricating oils and greases, although not used in great quantities in any one installation, represent an important link in the early development of many fires. The leakage of lubricants from mechanical systems, its accumulation in coal dust, other dust or detritus and the possible presence of hot surfaces of brakes, bearings, vehicle exhausts etc give good possibilities for hot surface ignition.

No liquids used in mining should have low flash points but even higher flash point liquids will become involved in fires fairly readily after short periods of heating. The burning of sprays of liquid from pressurised systems or pools of liquid with an appreciable surface area can lead to high rates of burning and serious contamination of the ventilation flow. Bulk usages of flammable liquids in mining include hydraulic fluids, transformer oil and diesel fuel.

METHOD OF USE OF MATERIALS

As stated above, the method of use of a material has an important effect on a material's behaviour in a fire.

A material has certain intrinsic properties - chemical properties such as reactivity, presence of fire retardants, heat of combustion and physical properties such as density and thermal conductivity - that effect its behaviour in fire. However, any assessment of its likely behaviour must also include application factors such as:

surface area of material,
orientation of material (vertical, sloping, horizontal and upwards facing, horizontal and downwards facing),
ventilation velocity and quantity,
roadway cross section and slope,

position in roadway,
continuity of surface (e.g. long continuous runs such as conveyor belts or discrete blocks such as vehicle components).

The interplay between these factors can be summarised in a fire size parameter R, defined (for a one-fuel system) by:

$$R = \frac{K A B}{\rho_A Q} \qquad (1)$$

Where K = mass of air required for the complete combustion of unit mass of volatile products from the burning material (kg/kg).

A = surface area of burning material (m^2).

B = average mass rate of evolution of volatile products from the burning material per unit surface area ($kg/m^2.s$).

ρ_A = density of air (kg/m^3).

Q = ventilation quantity (m^3/s).

The product AB gives the burning rate of the fuel (kg/s) while the product $\rho_A Q$ gives the supply rate of air (kg/s). The introduction of K to the numerator means that when R = 1 fuel and air supplies are in balance and the fumes from the fire would contain virtually no oxygen; for $R < 1$, the fumes would contain some oxygen; for $R > 1$, the fumes would contain excess fuel.

The parameter R describes the threat to life posed by the fire. A fire with a value of R = 0.1 typically has a fume composition of about 19% oxygen, 2% carbon dioxide and 0.1% carbon monoxide and so is close to the boundary of survivability for an unprotected man.

A fire with a value of R = 0.25 typically has a fume composition of about 16% oxygen, 5% carbon dioxide and 0.4% carbon monoxide and so is near the boundary of survivability for a man wearing a CO self rescuer (to a large extent because of oxygen deficiency).

R also defines, to some extent, the mode of propagation of the fire, through its influence on the **average** gas temperature of the fumes leaving the fire zone. R is a fuel/air ratio for the fire and for R = 1, this temperature is at a maximum which, taking account of heat losses, is usually in the range 1100-1300°C; for R = 0.3, the equivalent temperature is about 350°C.

Two modes of fire propagation in tunnels have been described - oxygen rich and fuel rich - for a system with a continuous fuel lining. In the oxygen-rich mode, the value of R is in the range 0-0.3; fire spread is essentially by local processes of heat transfer from flames by radiation and convection to fuel surfaces in the immediate vicinity and the average gas temperature of the products does not significantly affect the process.

In the fuel-rich mode, $R > 1$ and the fire spread process is dominated by heat transfer from the bulk flow of very hot gases past the fuel surfaces, a process that can extend over a considerable distance.

For a value of R = 0.3 or thereabouts, a transitional state develops. Most cellulosic and polymeric materials begin to break down and give off flammable vapours quite rapidly by a temperature of 300°C, so exposure to a gas flow with a temperature of 350°C (i.e. for R = 0.3) begins to have a significant effect on the fire spread process. As heat transfer from the bulk of the gas flow is added to the other preheating processes, the fire front will advance more rapidly, hence R will increase, the importance of heat transfer from the bulk flow will increase further and the fire will rapidly grow in size. For a continuous lining there is no stable value for R between 0.3 and 1.0; the fire will accelerate to the fuel rich

condition.

Experimentally, values of $R \simeq 3$ have been observed in fire trials with wood and with polyurethane foam. Typical fuel compositions in such a state are $O_2 < 1\%$, $CO_2 = 18.20\%$; $CO = 5\text{-}7\%$ and there is a large amount of unburnt fuel which could ignite in the presence of a fresh supply of oxygen, e.g. at a roadway junction. (Roberts and Blackwell, 1969).

Values of B are important in assessing fire behaviour and some typical values are given in Table 1.

In Table 1, the value of B for mineral oil relates to a pool or spillage of oil; the rate of burning of a spray of oil, e.g. from high pressure equipment, would be approximately equal to the discharge rate. Values of B will vary somewhat with circumstances for a particular material but, having regard to other uncertainties, it is probably adequate to take one representative value as characterising the material in the first instance.

The estimation of A for a specific fire scenario is more complex. For relatively compact fuel sources, e.g. a pool of oil, plastic components on a vehicle, filling material in a small roof cavity, it is probably best to equate A to the actual surface area of material exposed; this assumes that flame spread across the surface is relatively rapid compared to penetration of burning into the bulk of the material.

However, for long continuous lengths of material, such as conveyor belts, pipe lagging, roadway linings, this approach would be inappropriate. To take conveyor belts as a simple example, the front edge of the fire will develop in the direction of flow of the ventilation under the influence of local heat transfer processes. The surface area of belt burning will increase until such time as the belt is consumed and burning ceases at the upwind end. In principle, a stable fire system could then move along the conveyor belt with a more or less constant value of A determined by the relative rates of front end advance and rear end burn out.

The burn out time is determined by material density, thickness and the appropriate value of B. The rate of advance of the front end of the fire is determined by a complex set of interactions which are at the heart of the fire growth problem. Despite much research on mine fires and other types of fire there are still many unanswered questions in this area.

One can make certain generalisations:

1) The physical properties of the material affect the rate of flame spread through the 'thermal inertia' of the material, which is defined as the product of the thermal con-

Material	K (kg/kg)	Typical value of B ($kg/m^2.s$)	A for R = 0.1 in a 10 m^3/s air flow (m^2)
Mineral oil	14.0	0.029	3
Wood	4.6	0.007	40
Polyurethane foam (low density)	7.3	0.015	11

Table 1

ductivity, density and specific heat of the material. A low value of thermal inertia means that the surface temperature rises relatively fast when exposed to a heat source. A spreading flame heats up the surface ahead of itself more rapidly and flame spread is therefore more rapid. It is the very low thermal inertia of low density polyurethane foam that is mainly responsible for the rapid flame spread observed across this material.

2) The chemical properties of the material affect the rate of flame spread through such properties as oxygen requirement for complete combustion, head of combustion, thermal stability and char formulation. With a hyrocarbon liquid, flammable vapours are formed by a simple phase change and no char results; flame temperatures are high and burning rates are rapid; these characteristics can be modified by additives such as water emulsified into hydraulic fluids (which alters the thermochemistry to some extent but has its main effect as a diluent and a heat sink in the vaporisation process) or fire retardant chemicals which interfere with the combustion reactions and, in effect, lower the heat of combustion.

For a solid fuel, the solid material is broken down by pyrolysis reactions to give flammable vapours and a solid residue or char. The rate and nature of these pyrolysis reactions is linked to the chemical structure of the material. The char plays an important part in the process. It does not contribute to flaming combustion and its combustion, if it takes place at all, is slower and more localised; in addition, it serves as an insulating layer for the unburnt material beneath and slows down the pyrolysis reactions.

Fire retardant chemicals for solids may act in the vapour phase, as for liquids, or by promoting char formation, at the expense of the formation of flammable vapours.

3) The buoyancy of flames and the orientation of surfaces interact to affect the rate of flame spread; the faster spread of flame up a vertical fuel surface than an equivalent horizontal surface is well known. Rate of flame spread is also affected by edge effects, width of surface, orientation of the burning surface upwards or downwards, slope, burning on both sides. Flame attachment to surfaces is also important. An investigation of a recent serious fire in the UK, along a wooden escalator installed at 30° to the horizontal in a shaft, showed a greatly enhanced rate of flame spread up an upwards facing fuel surface because the flame attached to the surface, through aerodynamic forces, and spread very rapidly. (Moodie 1989).

4) Imposed air velocity has some complicated interactions with rate of fire growth. In the early stages of a fire, a higher air velocity may impede the development of the fire; it may dissipate the heat by rapid mixing and destabilise the flames. For a well established fire, with a reasonable area of fuel in a critically hot state, a higher air velocity may increase the rate of fire growth by increasing the rate of heat transfer back to the fuel surfaces both in the region already burning and in the region towards which the fire is spreading.

Examination of equation (1) shows that the ventilation quantity Q has an important influence on the value of R. If a fire has a value of R = 0.2 say with a value of $Q = Q_1$, then a halving of the ventilation

quantity to $Q = 0.5\ Q_1$, will cause an immediate doubling of R to R = 0.4; this is in the unstable region where a spontaneous acceleration to the fuel rich state is possible.

In other words, a reduction of air quantity to a developed fire can, in some circumstances, make matters worse by raising the fuel/air ratio, hence the exit gas temperature, and causing an uncontrolled runaway of the fire.

This analysis is based on a 'plug flow' model of the airflow, with good mixing across a vertical cross section and only minor vertical temperature gradients. It is possible, in principle, that if one had a fuel source near the roof one could get a channelled flow effect in which the bulk of the air remained cold but a portion of the air near the roof became involved in the fire and created a stable stratification of the flow through buoyancy effects. A fuel rich condition could then arise in the upper level of the flow, independently of the main flow, leading to greatly enhanced rates of fire spread. A surface such as a conveyor, if mounted near the roof, could produce a similar channelling effect.

THE EFFECT OF A FIRE ON THE MINE VENTILATION SYSTEM

There are two main effects to consider here - the fire/ventilation interactions within a single roadway and the interaction in the ventilation system as a whole; the effects of the ventilation on the burning characteristics of the fire have been considered in the previous section.

When considering the interactions in a single roadway, the 3-dimensional nature of the flow must be considered. The buoyancy forces of the fire gases may cause localised flow reversals through creating a layer of smoke and hot gases in the roof of the roadway which flows back against the general direction of the ventilation. This effect will be more pronounced with descensional ventilation but will still be apparent with a modest degree of upwards slope. The main effects of smoke backing are probably to obscure the seat of the fire and to restrict access to it for firefighting purposes.

In a single roadway, with descensional ventilation, the overall buoyancy effects of the fire gases may be sufficient to negate the pressure effects imposed by the mine fans and to cause a complete flow reversal within the roadway; the flow of fire gases, which may contain unburnt fuel, would change direction and take the fire gases back into the fire zone, with an attendant risk of explosion. Such a situation could be inherently unstable and repeated changes in flow direction could occur.

This leads on to the topic of the behaviour of the ventilation network as a whole. The influence of a fire on the stability of a ventilation network has been extensively studied but the problem is broader than that. The circulation of fire fumes by the ventilation system is the main cause of death in mine fires and a knowledge of the route that would be taken by fire fumes and the rate of circulation of fumes from a fire in a specified location are therefore important to pre-planning of strategies for dealing with fire emergencies.

The ventilation network of a mine can be analysed by computer and this is normal practice in many countries, with networks capable of being analysed on micro computers. The extension of these codes to include sub routines that take account of a fire in the network and then analyse the disturbances caused by the fire and the behaviour of the fire gases is in progress.

FIRE TESTING OF MATERIALS

It will be apparent from the preceding discussion that the assessment of the likely behaviour of materials in a mine fire is a complex matter.

For some applications where conditions are reasonably predictable and the amount of material used is sufficient to justify it, standardised tests have been developed. Conveyor belt and hydraulic fluids are examples of this. However, the development of standardised tests is expensive and time consuming and, for most applications, an ad hoc assessment is necessary. In my view, such an ad hoc assessment should be structured on the following lines:

1) The introduction of flammable materials into mines should be rigorously controlled by the operator and subject to audit by the regulatory authority. Any such introduction should have clearly defined advantages not possessed by less flammable alternatives.

2) No easily ignitable materials should be introduced into mines, defined for liquids by reference to flashpoint and for solids by reference to some small scale ignitability test.

3) Where the proposed application is on a relatively small scale (and here, reference to Table 1 will show that even a few square metres of exposed surface can have a significant effect on the mine atmosphere in the event of fire) then small scale flammablility tests may be adequate for screening purposes.

4) Where the proposed application can have a significant effect on the mine atmosphere, small scale screening tests should be supplemented by fire tests on a realistic scale. Such tests should, as far as is practicable, simulate the range of ventilation conditions and geometric arrangements in the proposed application (e.g. fire spread along a bank of four insulated pipes near the roof will be different from that along a single pipe in the middle of the roadway). The igniting source should be of a severity that would be recognised as a serious fire in practice but not of such a magnitude and duration that it would generate lethal fume concentrations in its own right. The assessment should be based on the response of the material to exposure to this igniting source in terms of:

   flame spread beyond the zone of influence of the igniting source (did a self sustaining fire result?)

   additional toxic fume production from the burning material (would it convert a dangerous fire into a lethal one?)

   smoke production (did it produce unusual qualities of smoke that would greatly impede means of escape?)

It will be apparent that I have presented this structure only in qualitative terms; actual quantitative standards are a matter for national authorities!

FIRE PROTECTION MEASURES IN A MINE

In the preceding sections, I have concentrated on matters affecting fire growth in a mine.

The growth of fire, and the associated risks to people in a mine, have to be seen in the context of the mine as a whole and of fire protection measures in force; I have touched on this point to some extent in considering the effect of the fire on the mine ventilation system.

In a recent paper entitled "A systematic strategy for assessing fire protection measures in a mine", (Roberts 1987), I presented an

approach by which the contribution of different fire protection measures to safety could be assessed. I will not repeat the details here but, in essence, fire growth (with the generation of toxic fumes) and the circulation of flames by the mine ventilation were seen as processes adversely affecting workers' chances of survival, in competition with fire detection systems, warning systems, pre planning of evacuation procedures, provision of refuges/safe routes/self rescuers, fire control measures, all of which were intended to improve workers chances of survival.

All these factors can be related to a common time base and the overall objective is obviously to ensure that workers do not encounter toxic fumes from a fire. However, one does need a means of comparing the effectiveness of different courses of action, e.g.:

Is it better to improve detection and warning systems or to provide self rescuers?

Is it better to improve the response time of fire detectors or to install slower response ones at shorter intervals?

The paper does provide an elementary means of relating all these important measures, one to the other, and thereby integrating all the fire protection measures in a mine into a common framework.

CONTINUING RESEARCH NEEDS

Taking advantage of the opportunity provided by a review paper on mine fires, I will offer the following thoughts as continuing research needs on mine fires:

1) A continuing basic research effort on flame spread characteristics of mining materials taking account of orientation of surface, slope air velocity etc.

2) Continuing studies on a realistic scale of the effect of slope and proximity of walls and roof on fire growth characteristics.

3) Codification of the 'ad hoc' assessment of mining materials in fire tests, to give more confidence in comparisons of test data from different countries.

4) The application of (1) and (2) to reduce the 'ad hoc' element of material assessment in fire tests.

5) The development of user friendly computer programmes for micro computers that will enable the effects of hypothetical fires on actual mine networks to be studied in order to assess network stability and fire control and evacuation strategies.

6) The development of the 'system' approach to fire protection measures and the integration of the various elements into coherent strategies.

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B - ignition sources, flammability of materials; fire growth
C - fire/ventilation interactions
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COMPTE-RENDU: INCENDIES DE MINES

A.F. Roberts

Le sujet des incendies de mines est très vaste. Pour tenter de le rendre suffisamment cohérent, notre rapport se concentre sur le comportement des incendies dans les couloirs d'aérage des mines; il ne prend pas en considération les incendies causés par la combustion spontanée des déblets, mais approfondit les sujets suivants :

1) Sélection des matériaux: conditions de performances des matériaux d'exploitation minière en termes d'inflammabilité et de production de fumée/vapeurs; méthodes d'essais et adaptation aux conditions d'exploitation minière.
2) Augmentation des incendies: influence des propriétés des matériaux et des méthodes d'utilisation sur le taux de propagation des incendies dans les couloirs de mines; influence de la ventilation, de l'inclinaison, de l'orientation des matériaux, de la proximité des passages et des toits sur le comportement des incendies.
3) Incendies dans les réseaux de ventilation: effet des incendies sur la distribution de la ventilation; circulation des produits d'incendie dans les réseaux.

Notre rapport examine ensuite l'influence de ces facteurs sur l'élaboration de stratégies de contrôle des incendies. Il n'approfondit toutefois pas les thèmes de la détection des incendies et des méthodes d'extraction des incendies.

GRUBENBRÄNDE

A. F. Roberts

Das Thema Grubenbrände umfaßt einen weiten Bereich. Der Vortrag beschränkt sich auf das Verhalten von Bränden in bewetterten Grubenbauen. Selbstentzündete Brände in Bruchzonen werden nicht berücksichtigt. Die folgenden Themen werden detailliert behandelt:

1) Materialauswahl: Anforderungen an das Material im Hinblick auf Entflammbarkeit und Rauchgaserzeugung; Testmethoden und Auswirkungen bei untertägigen Bedingungen.
2) Brandausbreitung: Einfluß der Materialeigenschaften und der Materialverwendungen auf die Brandausbreitungsgeschwindigkeit in Grubenbauen; Einfluß der Bewetterung, Einfallen der Strecke, der Materialausrichtung und der Streckenhöhe auf das Brandverhalten.
3) Der Brand im Bewetterungssystem: Einfluß von Feuer auf die Wetterverteilung; Verbreitung von Verbrennungsprodukten im Wettersystem.

Eine Diskussion folgt über Feuerbekämpfungsstrategien im Zusammenhang mit den erwähnten Punkten, Themen wie Branderkennung und Löschmethoden werden dabei nicht mit herangezogen.

ПОЖАРЫ В ШАХТАХ

А.Ф. Робертс

Проблема изучения пожаров в шахтах весьма широка. Настоящий доклад ограничивается рассмотрением поведения пожаров в вентиляционных стволах шахт. Пожары, возникающие самопроизвольно в результате возгораний в отработанных породах, не рассматриваются.

Следующие аспекты будут рассмотрены подробно:

1) Выбор материалов: характеристики, требуемые от материалов, применяемых в шахтах, с точки зрения воспламеняемости и дымовыделения; методы испытания материалов для определения их пригодности в условиях шахт.

2) Распространение пожара: влияние характеристик материалов и способов их использования на частоту возникновения пожаров в шахтных стволах; влияние вентиляции, наклона, ориентации пород, размера штольни, высоты кровли и поведения огня.

3) Пожары в вентиляционных стволах: воздействие огня на разводящие вентиляционные стволы, циркуляция продуктов сгорания в вентиляционных стволах.

В докладе рассматривается влияние вышеперечисленных факторов на создание противопожарной стратегии. В то же время такие вопросы, как обнаружение очагов пожара и методы их ликвидации не рассматриваются подробно.

文献调查报告：矿井火灾

A．F．罗勃兹

矿井火灾是个非常大的论题．为了保持一定的条理性，本文献调查将着重考虑矿井通风道中的火灾情况；废料中自发燃烧引起的火灾将不予于论述．下面是一些本文要详细讨论的论题：

1）材料的选择：从材料的易燃性和烟／硝的产生为出发点的采矿用材料性能的要求；试验方法和适用于矿业情况的需要．

2）火灾的扩散：材料的性质和使用方式对火灾在矿井巷道中扩展速度的影响；通风，坡度，材料放置方向，离人行道的距离及顶板对火灾情形的影响．

3）通风网路中的火灾：火灾对于通风分配的影响；火灾所产生气体在通风网路的循环．

而后，本文将讨论这些论题对灭火对策的决定的影响，但是本文将不详细论述火灾探测和扑灭的具体方法．

STRATEGY FOR FIRE SAFETY

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SUMMARY

Fire is one of the major risks of collective accidents in mines. Spontaneous heating of coal and explosions are sources of risk that have long been studied. More recently, the ever-increasing introduction of polymeric, chemical materials has become important. Although this allows an improvement in working conditions and increases the profitability of the mines, it means that there must be a new approach to fire hazard control, special attention being paid to the thermal and toxic consequences of disasters.

The present communication illustrates an approach to a strategy for fire safety in mines:

- analysis of the risk: full-scale experiments on the behaviour, in fire, of a polymeric gunite mass (100 m^2 applied to walls in the CERCHAR gallery),
- evaluation, in a fire situation, of different types of refuge bays and of their equipment : collective protection,
- evaluation of a self-contained breathing appliance working on chemical oxygen (mini-Apeva): individual protection,
- introduction of a strategy for fire safety.

INTRODUCTION

In underground mines, the risks of fire have long been known. These risks were related to the combustible materials naturally present, which may cause autoxidation phenomena, as is the case with coal and sulphur, and are also related to products and objects introduced for operational requirements, such as wood, electric wiring, conveyor belts, oils, greases etc.

At present, increasing quantities of chemical and of polymeric products are being introduced, the operational characteristics of which present important advantages compared with the traditional materials.

As against this, these materials are combustible and may be more or less flammable and, when they burn, they can emit very toxic gases. In a mine, therefore, polymeric materials modify the fire risks, particularly the risk of fire propagation and the toxicity of the gases.

We thus deemed it necessary to evaluate the risks presented by these materials, in order to avoid the use of the most dangerous and to set up a strategy for the protection of the personnel in the case of fire.

The main elements of this strategy are:

- Prevention: selection of materials by an approval procedure (this is the case with conveyor belts) or in accordance with full-scale tests when this appears necessary, as in the case of products used over a large area for lining the walls of galleries.

- Detection: choice of detectors, determination of their positions.

The aim is to get rapid and reliable detection.

- Rapid warning: to all persons concerned by the hazard detected.

- Immediate protection of the personnel by breathing appliances worn on the belt or, in certain cases, placed nearby. These appliances provide the protection necessary to enable the person to reach a "pressurisable recess" even if the atmosphere has become unbreathable because of the presence of toxic gases or the absence of oxygen.

- Preparation for evacuation: The "pressurisable recess" is a place supplied with unpolluted air by the compressed air system. The personnel from the nearest workplaces can, in these "pressurisable recesses", prepare with the deputies, for the next stages in the evacuation.

The means available in these recesses are:

- long-life replacement cartridges for the self rescuers,

- a telephone for obtaining information on the situation and or choice of the best road for the evacuation, also any other useful information concerning rescue,

- a plan of the district for marking out the evacuation route.

In the following we describe our latest work and results concerning polymeric materials (materials already in place and basic products ready for use, the pressurisable recesses and the breathing apparatuses.

BEHAVIOUR OF POLYMERIC MATERIALS IN FIRE

In order to be able to study, in full-scale, the problems raised, with regard to fire, by the introduction of new polymeric objects and materials in underground workings, CERCHAR, in 1983, put a fire gallery into service.

Description of the fire gallery

This gallery, shown in fig. 1, was fully described in a paper presented at the SYDNEY conference (R. CHAUVIN, 1985).

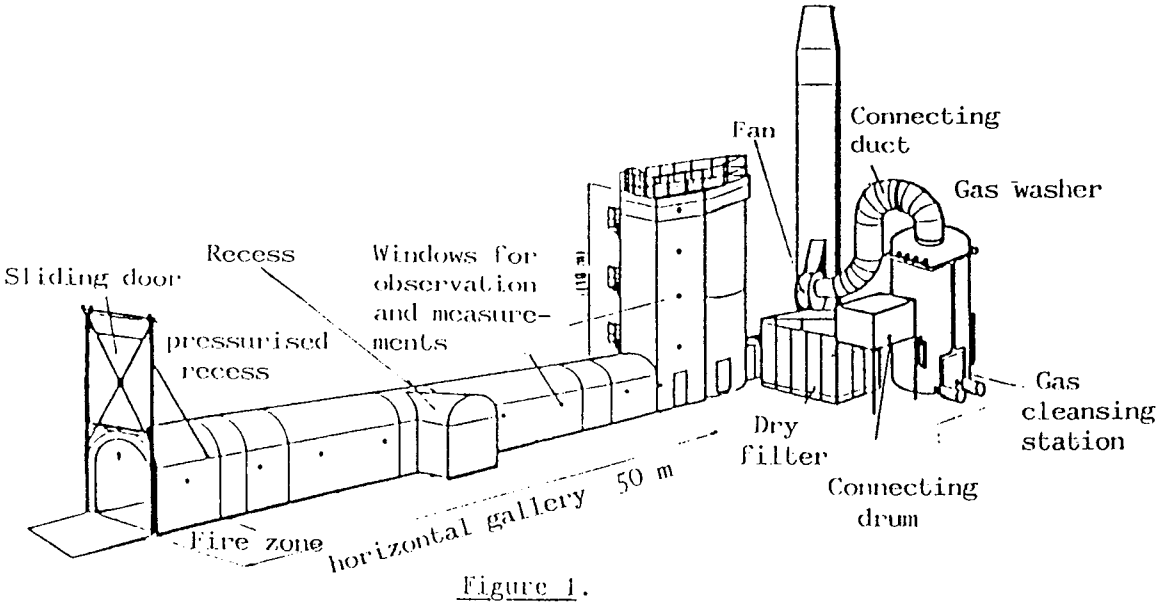


Figure 1.

Let us mention only the following points:

- for the tests, the gallery has a horizontal part, cross-section 10 m^2, length 50 m and a vertical part, cross-section 6 m^2 and height 10 m; halfway along the horizontal part there is a recess, which was used for the study of pressurisable recesses;

- the gallery also contains a system for cleansing the fumes;

- it can be ventilated with a maximum air speed of 1.5 m/s.

Polymeric materials already examined at CERCHAR

The most frequent cause of underground mine fires is the heating and ignition of conveyor belts. Recent examples have reminded us that these fires can have serious consequences both in loss of human life and from the economic point of view.

Among the other polymeric materials used in large quantities in underground workings, expanding foams are becoming increasingly important and are injected for consolidation of strata, waterproofing, construction of rapid barriers, lagging of roadways.

A preliminary programme of research on the behaviour, in fire, of PVC- and chloroprene-based conveyor belts, and of urea-formalin-based expanding foam was subsidised by the European Coal and Steel Community (ECSC) (J. DANGREAUX, 1986).

Tests on three conveyor belts showed that, in our ignition conditions, with a 300 kg pile of poplar wood, the chloroprene Type 500 belt burnt over its entire length of 15 metres, that of chloroprene Type 800 over a length of 6.4 m, while a belt of PVC Type 1250 did not burn beyond the 2.3 m sandwiched within the pile. We were able to demonstrate a propagation speed limit of 15 to 18 cm/min. for the two chloroprene belts below which the fire died out.

As regards the toxicity of products from the thermal degradation of the belts examined, the CO concentration may rise to a high value, like the concentration of HCl, although a large part of this gas is condensed in the gallery. Their conjugate action could, with the concentrations observed, cause serious physiological disturbances. The presence of polyaromatic hydrocarbons fixed on the soots is likewise observed. Finally one must draw attention to the very opaque fumes released by a fire in chloroprene and in PVC belts.

In the case of the urea-formalin foam, the lagging of the gallery walls obtained by the application of this foam was subjected to the action of a 300 kg wood fire. After a pyrolysis phase with a lot of gas emission near the surface of the foam, there was a general conflagration caused by the sudden ignition of the flammable gases. The lining was completely destroyed. The thermal degradation of the foam causes the emission of large quantities of hydrocyanic acid and nitrogen oxides. This foam is no longer used for lagging roadways but it has been replaced by a foam based on phenolic resin.

Behaviour of a formophenolic foam in fire

Test protocol

The case considered is that of formophenolic foam lagging on walls.

The foam was applied over an area of 100 m^2 to an average thickness of 5 cm; it lined the walls and arch of the gallery over a length of slightly more than 10 m.

300 kg of poplar wood was arranged so as to be entirely within the foam-lined part of the gallery. The wood was ignited by 2 litres of fuel-oil and 0.5 litres of methylated spirits.

A large number of transducers were installed to observe the development of the fire:

- thermocouples at the surface of the foam and inside the foam,

- thermocouples downwind in the gallery, at a horizontal distance of 38 m from the seat of the fire, in the vertical part of the gallery and at the exit,

- flowmeters in the ambient atmosphere around the seat of the fire,

- densitometers downwind of the fire,

- various sampling tubes for the fumes, providing, in particular, means for the continuous measurement of CO, CO_2, O_2, bubblers in series for the dosage of quantities of freon, phenol and formalin and recovery of soot.

Progress of test-observations:

The test was carried out with an air flow rate, in the gallery, of 27000 m^3/h, that is, an air speed of about 0.75 m/s.

The test lasted 75 mins., counting from the ignition of the pile of wood.

The foam lining first of all turned pink and then became brown on contact with the flames. Then the foam became covered with small cracks, became incandescent at various places, with, here and there, flakes of fire (at the times 13 min. 30s, 15 min.).

After the burning wood collapsed (16 min.), the temperature in the gallery decreased considerably and, after 36 min., only one part of the foam was still incandescent. The following 39 min. did not bring about any resumption of the fire.

Test on a fire involving only a pile of wood

In order to be able to determine the part played by the foam in the developments and consequences of the fire, a comparative test was carried out using only a pile of wood in a bare, unlagged gallery.

The various transducers and means analysed were, of course, placed in the same positions as in the test on the foam.

Results obtained - Thermal and toxic effects

With the effect of this fire, we were able to observe the satisfactory behaviour of the foam since, even closest to the fire, it remained in place and was charred only on the surface.

The foam did not propagate the fire; the charring lasted only for the period during which the wood burned.

The temperatures recorded at the top of the arch of the gallery in the two types of fire: wood alone and wood + foam, are shown in fig. 2.

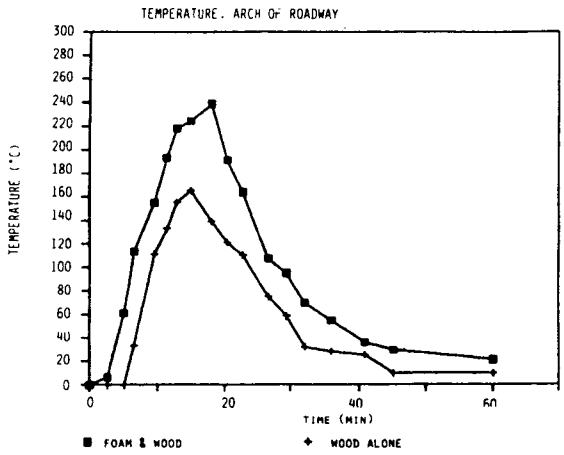


Figure 2.

Without propagating the fire, the foam, in spite of everything, plays a considerable part in heating the ventilation air.

As regards the toxic effect of the gases produced, one may note that it is characterised, essentially, by a release of CO (v. fig. 3). The content of CO attributable to the decomposition of the foam rises to some 3500 ppm and this over some minutes with our fire arrangements. Furthermore, a specific analysis of the products, freon, phenol and formalin was carried out, as a whole, throughout the test.

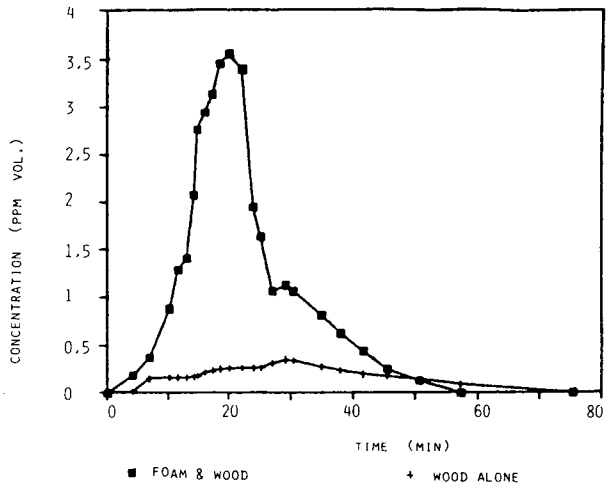


Figure 3.

The average contents measured in our operational conditions were:

Freon : 18 ppm

Phenol : 3 ppm

Formol : 2.7 ppm

In conclusion, it will not be forgotten that arrangements for the safety of persons in the environment of such a fire must also take into account the oxygen content of the atmosphere, a content which, in our conditions, dropped to 15%.

Another test was carried out on three metal drums, each containing 30 kg basic resin and placed in the immediate vicinity of a 300 kg pile of wood.

The pipes, normally closed, were heated, leading successively to deformation in the container and first of all slight leakages of fumes at the plug, and then breakage of the drums.

The resin released and poured out of the drums was not ignited. This may be explained by its complex formula (aqueous solution, mineral and freon content).

CERCHAR will continue to evaluate the toxicity of the fumes released during fire in other moulded polymeric materials (polyurethane, acrylic resins) and of their basic constituents, that are stored in various conditions.

PRESSURISABLE RECESSES

In the case of a fire in polymeric products in underground workings, the plans for evacuation provide for the assembling of personnel at some preferred points as regards the ventilation and then for evacuation, via preselected routes, to the surface.

In the case where the evacuation routes cannot be used because of toxic gases in the galleries or because of lack of oxygen, the procedure is to reach pressurisable recesses.

Such a recess is an enclosed niche sufficiently insulated from the external atmosphere (which may be at a high temperature and/or polluted by fumes). The recess is kept at a slight overpressure by a controlled admission of air supplied from the compressed air network.

Pressurisable recesses could be, a priori, considered as:

- safe places for gathering the personnel while awaiting information from the surface, in order to organise evacuation in the best safety conditions.

- relay points during an evacuation in progress after a fire warning; the workers, equipped with their APEVA (Appareil d'Evacuation Autonome - Self-contained evacuation

apparatus) reach the recesses and can then exchange these appliances (the useful life of which is generally about twenty minutes) in good safety conditions, after which the evacuation can go on;

- places of protection from effect of the fumes; the miners then wait until the danger is past or until the arrival of rescue teams equipped with much more powerful means of protection.

How pressurisable recesses work

Test protocol

The research, undertaken at CERCHAR and subsidised by ECSC, was for the purpose of defining the limits within which the miners could be protected, not only against a toxic and asphyxiating atmosphere but also against an excessive fumes temperature. The project likewise enabled us to investigate flexible materials for making curtains whose nature, structure and mode of operation enable us to ensure fire-break action, the heat insulation as well as making the place gas-proof, providing effective and durable protection against the effects of the fire.

Two different designs for the recesses were studied within the research contract :

- a recess consisting of tubing elements and positioned right on the floor in the centre of the gallery;

- a recess hollowed out in the side of the gallery and closed by a curtain, which ensures protection against the flow of heat and the toxic gases.

The tests consisted in subjecting each type of recess, one after the other, to the hot fumes from a fire in which we were able to control the calorific power as we wished.

A gas burner was developed, capable of continuously maintaining the desired temperatures with respect to the fan ventilating the gallery. This burner, the power of which could be adjusted between 1000 and 10,000 kW was placed at the entrance to the gallery.

The following important factors were taken into account:

- the natural ventilation tending to establish itself between the gallery and the selected recess (problems of pressurisation);

- the heat exchanges between the hot fumes in the gallery and the atmosphere inside the shelter;

- the heat resistance of the flexible components closing the recess (curtains).

After evaluation of the behaviour and characteristics of these recesses in their current design, we tried to optimise their conditions of use, by attempting to:

- cool the hot fumes in the gallery upwind of the recess, by means of sprinklers,

- cool the tubing elements by spraying with water,

- cool the curtains by spraying,

- cool the air injected into the recess,

- heat-insulate the tubing type of recess.

Results obtained

A bibliographical study enabled us to show that the average temperature level which one could obtain at a distance of some hundreds of metres from the seat of a fire was of the order of 100 to 200°C. These values were reproduced, experimentally, in our installations.

Behaviour of the "tubing-type" recess

To begin with, we tested the gas-tightness and defined the quantity of air to be injected into the recess in order to purge it; a diagram of the ventilation as a function of the volume to be purged and of the degree of purity sought was devised. The curves make it possible to establish the quantity of air to be injected into a polluted enclosure, before entering it; this volume of air was estimated to be at least 5 times the volume of the enclosure. An air volume of 300 Nm^3/h makes it possible to keep the enclosure at a slight overpressure (15 to 25 pascals) when the temperature of the gases in the gallery is 200°C.

Secondly, we determined how the temperatures in the recess developed, with and without a cooling device, and as a function of the temperature in the gallery.

The following are the main results obtained:

For a fixed temperature level of 100°C in the gallery, the temperature measured in the recess rapidly reaches a value of 80-85°C.

Cooling the walls of the recess by sprinkling water enables us to reduce the temperature to 60°C for 2.5 m^3/h water and 40°C for 6 m^3/h water.

When the temperature level in the gallery is raised to 160°C, the temperature inside the water-cooled recess exceeds the limit of human tolerance and reaches 65°C.

When the temperature level in the gallery is raised to 200°C, the temperature inside the water-cooled recess becomes stable at around 100-110°C.

Behaviour of the "lateral-type" recess

The supply of compressed air (300 Nm^3/h) injected into this recess enabled us to keep it at a slight overpressure (5 to 10 pascals). The gas analyses carried out showed that, in these conditions, the volume in the recess remains purged and this no matter what the level of temperature is in the gallery.

Three different flexible curtains, based on glass fabric, were tested; they are coated either with hypalon or with silicon resin. The tests showed that these materials have an excellent fire-resistance.

Temperature measurements in the recess gave the following results:

- for a stabilized temperature of 200°C on the external surface of the sprayed curtain, the temperature measured in the recess did not exceed 25 to 35°C after several tens of test minutes,

- the average rate of temperature increase in the recess is 0.15°C/min when the curtain is cooled; it rises to 4°C/min. when the spraying of the curtain is discontinued.

Whatever the temperature level at right angles to the curtain in the gallery (value between 70°C and 220°C), the average temperature level in the recess never exceeds the limit of human tolerance, fixed at 60°C.

Although the lateral recesses behave better than the shelters in the gallery, we abide, for obvious safety reasons, by the following principle: pressurisable recesses are not places for a prolonged stay in the case of fire; they are only halts, making it possible to prepare for the further evacuation in the best conditions.

BREATHING APPLIANCES WORN ON THE BELT

When there is a lack of oxygen or presence of toxic gases, it is necessary to have respiratory protection very quickly.

To answer this need we use, in French mines, self-contained

respirators for evacuation; these are placed close to the workplace or worn permanently on the belt.

These appliances are autonomous, containing a KO_2 cartridge which provides the necessary oxygen; they thus afford protection whatever the composition of the atmosphere.

The appliances worn on the belt are becoming more and more widely used.

For a weight of approximately 2 kg, they can be used, for evacuation, over a period of 20 to 30 mins. The original cartridge can be exchanged for a replacement cartridge giving a further period of use of 70 to 90 mins.

Durations observed in tests carried out on a respirator:

Output	Original cartridge	Replacement cartridge
35 l/min	26 min	72 min
30 l/min	31 min	88 min
10 l/min	110 min	6 h 35 min

The exchange cartridges are kept in the pressurisable recesses and the placement has been determined so that the autonomy provided by the original cartridge in the self rescuer allows the person to leave from any workplace and to reach the nearest refuge.

The autonomy was observed during tests in actual workings, the travelling conditions in which correspond to the main types of workings in our mines. The results were noted in the form of functioning times and distance covered.

Type of working	Autonomy of original cartridge: Time (in min).	Distance (in m)
Strike face	33	1 870
Strongly inclined	18	1 010
Steep	38	1 470

The pressurisable recesses are sited in accordance with these results, allowing for a safety margin.

CONCLUSION

Investigations and tests are in progress in order to make detection and rapid alarm signals more effective but, already, the strategy for fire safety, of which we have just described the essential elements, constitutes a significant improvement in safety.

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STRATEGIE DE SECURITE INCENDIE

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RESUME

L'incendie est un des risques majeurs d'accident collectif en mine. L'auto-échauffement du charbon, les explosions sont des sources de risque étudiées depuis longtemps. Plus récemment, l'introduction de plus en plus importante de matériaux chimiques, polymériques s'est affirmée. Si elle permet une amélioration des conditions de travail, une meilleure rentabilité des exploitations, elle implique par contre une nouvelle gestion du risque incendie en prenant plus particulièrement en compte les impacts thermique et toxique des sinistres.

La présente communication illustre une approche de stratégie de sécurité incendie en mine :

- analyse du risque : expérimentation en grandeur du comportement au feu d'un gunitage de masse polymérique (100 m^2 projetés sur les parois de la galerie du CERCHAR),

- évalutation en situation d'incendie de différents types de relais, refuges (niches pressurisables) et de leurs équipements : protection collective,

- évaluation d'un appareil respiratoire autonome à oxygène chimique (mini-Apeva) : protection individuelle,

- mise en place d'une stratégie de sécurité incendie.

INTRODUCTION

Dans les mines souterraines les risques d'incendie sont connus depuis longtemps. Ils étaient liés aux matériaux combustibles présents naturellement qui peuvent donner lieu à des phénomènes d'auto-oxydation, c'est le cas du charbon et des sulfures, ainsi qu'aux produits et objets introduits pour les besoins de l'exploitation comme le bois, les canalisations électriques, les bandes de convoyeur, les huiles, les graisses, etc.

On introduit actuellement des quantités croissantes de produits chimiques et de produits polymériques dont les caractéristiques d'emploi présentent des avantages importants par rapport aux matériaux traditionnels.

En contrepartie ces matériaux sont combustibles et peuvent être plus ou moins inflammables et leur combustion peut dégager des gaz hautement toxiques. Les matériaux polymériques en mine modifient donc les risques d'incendie, en particulier le risque de propagation et la toxicité des gaz.

Nous avons donc estimé nécessaire d'évaluer les risques présentés par ces matériaux, afin d'éviter l'emploi des plus dangereux, et de mettre sur pied une stratégie pour la protection du personnel en cas d'incendie.

Les éléments principaux de cette stratégie sont :

- Prévention : sélection des matériaux par une procédure d'homologation (c'est le cas des bandes de convoyeur) ou d'après des essais en vraie grandeur lorsque cela paraît nécessaire, comme dans le cas des produits utilisés en grande surface pour le revêtement des parois des galeries.

- Détection : choix des détecteurs, détermination de leurs emplacements. L'objectif est rapidité et fiabilité de la détection.

- Alerte rapide de toutes les personnes concernées par le risque détecté.

- Protection immédiate du personnel par des appareils respiratoires portés à la ceinture ou dans certains cas déposés à proximité. Ces appareils fournissent la protection nécessaire pour rejoindre une "niche pressurisable" même si l'atmosphère est irrespirable par présence de gaz toxiques ou absence d'oxygène.

- Préparation de l'évacuation : La "niche pressurisable" est un lieu alimenté, par le réseau d'air comprimé, en air non pollué. Le personnel venu des lieux de travail les plus proches peut dans ces niches pressurisables préparer, avec les agents de maîtrise du secteur, la suite de l'évacuation.

Les moyens disponibles dans ces niches sont :

- des cartouches de rechange longue durée pour les appareils respiratoires,

- un téléphone permettant d'obtenir des informations sur la situation et le choix du meilleur chemin pour l'évacuation, ainsi que toute autre information utile concernant le sauvetage,

- un plan du quartier pour repérer le chemin d'évacuation.

Nous présentons ci-après nos derniers travaux et résultats concernant les produits polymériques (matériaux en place et produits de base prêts à l'emploi), les niches pressurisables et les appareils respiratoires.

COMPORTEMENT AU FEU DE MATERIAUX POLYMERIQUES

Pour être en mesure d'étudier en grandeur et en situation les problèmes que posent, vis-à-vis de la sécurité incendie, l'introduction dans les travaux souterrains des matières et matériaux polymériques nouveaux, le CERCHAR a mis en service en 1983 une galerie incendie.

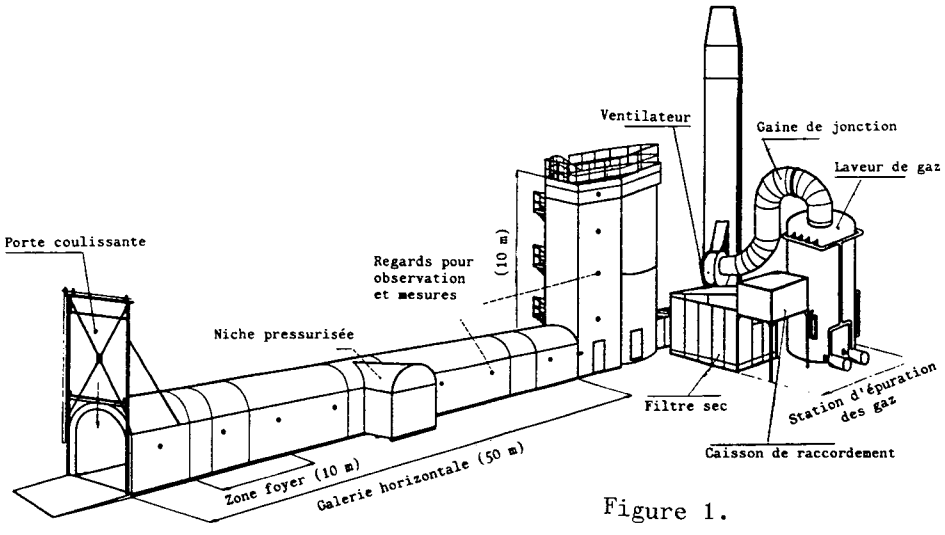


Figure 1.

Description de la galerie incendie.

Cette galerie représentée sur la figure 1 a été complètement décrite dans une communication à la conférence de SYDNEY (R. CHAUVIN, 1985).

Rappelons seulement les points suivants :

- pour les essais, la galerie comprend une partie horizontale de section 10 m^2 et longueur 50 m et une partie verticale de section 6 m^2 et hauteur 10 m ; à mi-longueur de la partie horizontale est aménagée une niche qui a été utilisée pour l'étude des niches pressurisables ;

- la galerie est pourvue en outre d'un système de dépollution des fumées ;

- elle peut être ventilée avec une vitesse maximale d'aérage de 1,5 m/s.

Matériaux polymériques déjà examinés au CERCHAR.

La cause la plus fréquente d'incendie au fond de la mine est l'échauffement et l'inflammation des bandes transporteuses. Des exemples récents sont venus rappeler que ces incendies peuvent avoir des conséquences graves tant en pertes de vies humaines que sur le plan économique.

Parmi les autres matériaux polymériques qui sont utilisés en quantité importante dans les travaux souterrains, les mousses expansibles occupent une place grandissante et sont utilisées en injection pour la consolidation des terrains, l'étanchement, la réalisation de barrages rapides, le garnissage des voies.

Un premier programme de recherche sur le comportement dans un incendie au fond de la mine de bandes transporteuses à base de PVC et de chloroprène et d'une mousse expansible à base de résines urée-formol a été mené avec l'aide financière de la Communauté Européenne du Charbon et de l'Acier (J. DANGREAUX, 1986).

Les essais sur trois bandes transporteuses ont montré que dans nos conditions d'allumage avec un bûcher de bois de peuplier de 300 kg, la bande en chloroprène type 500 a brûlé sur toute sa longueur de 15 mètres, celle en chloroprène type 800 sur 6,4 m alors que celle en PVC type 1 250 n'a pas brûlé plus loin que les 2,3 m pris en sandwich dans le bûcher. On a pu mettre en évidence une vitesse limite de propagation de l'ordre de 15 à 18 cm/min. pour les deux bandes en chloroprène au-dessous de laquelle le feu s'éteint.

En ce qui concerne la toxicité des produits de dégradation thermique des bandes examinées, la teneur en CO peut atteindre une valeur élevée ainsi que celle en HCl bien qu'une partie importante de ce gaz se condense dans la galerie. Leur action conjuguée pourrait, aux teneurs constatées, provoquer des troubles physiologiques sérieux. On constate également la présence d'hydrocarbures polyaromatiques fixés sur les suies. Enfin, il faut insister sur les fumées très opaques que dégage l'incendie des bandes en chloroprène et en PVC.

Dans le cas de la mousse urée-formol, le garnissage des parois de la galerie obtenu par projection de cette mousse a été soumis à l'action d'un bûcher de 300 kg de bois. Après une phase de pyrolyse avec dégagement gazeux important au voisinage de la surface de la mousse, il s'est produit un embrasement généralisé par inflammation brusque des gaz inflammables. Le revêtement a été complètement détruit. La dégradation thermique de la mousse provoque l'émission de quantités élevées d'acide cyanhydrique et d'oxydes d'azote. Cette mousse n'est plus utilisée pour le garnissage des voies mais elle a été remplacée par une mousse à base de résine phénolique.

Comportement dans un incendie d'une mousse formophénolique.

Protocole d'essais :

Le cas considéré est celui d'un garnissage de parois à l'aide d'une mousse formophénolique.

La mousse a été projetée sur une surface de 100 m^2 pour une épaisseur moyenne de 5 cm : elle a revêtu murs et voûte de la galerie sur une longueur d'un peu plus de 10 m.

Un bûcher de bois de peuplier de 300 kg a été disposé de manière à être engagé entièrement dans la partie garnie de mousse de la galerie. La mise a feu a été effectuée à l'aide de 2 litres de fuel-oil et de 0,5 litre d'alcool à brûler.

De nombreux capteurs ont été installés pour suivre le déroulement de l'incendie :

- thermocouples en surface de la mousse et au sein de la mousse,

- thermocouples en aval de la galerie, à 38 m du foyer en zone horizontale, dans la partie verticale de la galerie et en sortie,

- fluxmètres dans l'environnement du foyer,

- opacimètres en aval du feu,

- diverses cannes de prélèvement des fumées avec plus particulièrement des possibilités de mesure en continu de CO, CO_2, O_2, des barboteurs en série pour dosage des quantités de fréon, phénol et formol et récupération de suies.

Déroulement de l'essai - observations :

L'essai a été effectué avec un débit d'air dans la galerie de 27 000 m^3/h soit une vitesse d'aérage d'environ 0,75 m/s.

L'essai a duré 75 min à partir de l'inflammation du bûcher.

Le revêtement de mousse a d'abord rosi puis bruni au contact des flammes. Puis la mousse s'est craquelée, est devenue en divers endroits incandescente avec de part en part apparition de flamèches (aux temps 13 min 30 s, 15 min).

Après l'effondrement du bûcher (16 min) la température a fortement chutée en galerie et au bout de 36 min seule une partie de la mousse était encore incandescente. Les 39 min suivantes n'ont donné lieu à aucun regain de l'incendie.

Essai au bûcher de bois seul :

Afin de pouvoir déterminer la part de la mousse aux développements et conséquences de l'incendie un essai de référence a été fait avec un bûcher de bois seul dans la galerie nue.

Les divers capteurs et moyens analysés ont été bien sûr disposés aux mêmes endroits que lors de l'essai réalisé sur la mousse.

Résultats obtenus - Impacts thermique et toxique :

Sous l'effet de cet incendie on a pu observer le bon comportement de la mousse, puisque même au plus près du bûcher, elle est restée en place et n'a été calcinée qu'en surface.

La mousse n'a pas propagé l'incendie ; sa calcination n'a duré que le temps de la combustion du bûcher.

Les températures enregistrées en haut de la tour de la galerie dans les deux configurations de feu : bûcher seul et bûcher + mousse sont données sur la figure 2.

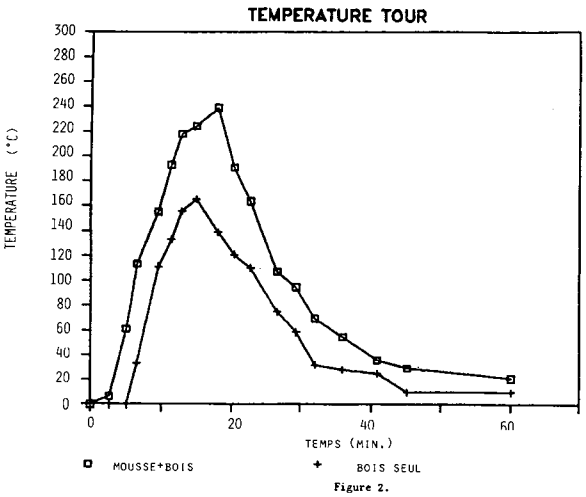


Figure 2.

Sans propager l'incendie, la mousse participe malgré tout de façon notable à l'échauffement de l'air d'aérage.

En ce qui concerne l'impact toxique des gaz produits, on peut remarquer qu'il est caractérisé essentiellement par un dégagement de CO (voir figure 3). La teneur en CO attribuable à la décomposition de la mousse, atteint quelque 3 500 ppm et ce pendant quelques minutes dans notre scénario d'incendie. Par ailleurs, une analyse spécifique des produits, fréon, phénol et formol a été effectuée globalement sur l'essai.

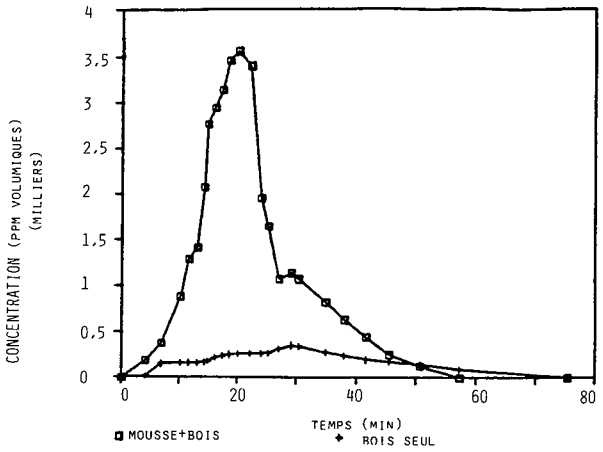


Figure 3.

Les teneurs moyennes mesurées dans nos conditions opératoires ont été :

Fréon : 18 ppm.

Phénol : 3 ppm.

Formol : 2,7 ppm.

Enfin, on n'oubliera pas que la gestion de sécurité des personnes dans l'environnement d'un tel incendie doit aussi tenir compte de la teneur en oxygène de l'atmosphère, teneur qui est descendue à 15 % dans notre scénario.

Un autre essai a été effectué sur trois bidons métalliques contenant chacun 30 kg de résine de base disposés à proximité immédiate d'un bûcher de bois de 300 kg.

Les fûts normalements fermés ont subi des échauffements qui ont conduit successivement à des déformations du conteneur et fuites légères de fumées au bouchon d'abord et à une rupture des bidons ensuite.

La résine qui s'est dégagée et s'est déversée des bidons ne s'est pas enflammée. Sa formulation complexe (solution aqueuse, charge minérale et fréon) peut expliquer ceci.

L'évaluation de la toxicité des fumées dégagées lors d'incendie d'autres matériaux polymériques formés (résines polyuréthanne, acrylique) et de leurs constituants de base qui font l'objet de stockages dans des conditions variées, sera poursuivie au CERCHAR.

NICHES PRESSURISABLES

En cas d'incendie de produits polymériques dans les chantiers souterrains, les plans d'évacuation prévoient de rassembler le personnel en quelques points privilégiés par rapport à l'aérage, puis de l'évacuer par des itinéraires choisis d'avance vers la surface.

Dans le cas où les itinéraires d'évacuation ne pourraient être empruntés par suite de l'envahissement des galeries par des gaz toxiques, ou par la raréfaction de l'oxygène, la procédure est de gagner des niches pressurisables.

Une niche pressurisable est une enceinte close suffisamment isolée de l'atmosphère extérieure, qui risque d'être à température élevée et/ou polluée par les fumées. La niche est maintenue en légère surpression par une arrivée contrôlée d'air fourni par le réseau d'air comprimé.

Les niches pressurisables pouvaient être considérées a priori comme :

- des lieux de regroupement sûr pour le personnel dans l'attente d'informations en provenance du jour afin d'organiser son évacuation dans les meilleures conditions de sécurité,

- des relais en cas d'évacuation en cours après l'alerte incendie ; les ouvriers équipés de leur APEVA (Appareil d'Evacuation Automone) gagnent les niches pour y réaliser l'échange de ces appareils (dont la durée d'utilisation est généralement d'une vingtaine de minutes) dans de bonnes conditions de sécurité et poursuivent ensuite l'évacuation,

- des lieux de séjour protégés contre les effets des fumées ; les mineurs attendent alors que le danger soit écarté ou qu'interviennent des équipes de secours équipés de moyens de protection beaucoup plus puissants.

Comportement des niches pressurisables.

Protocole d'essai :

La recherche entreprise au CERCHAR avec l'aide financière de la CECA avait pour objet de fixer les limites dans lesquelles les mineurs pourront être protégés, non seulement contre une atmosphère toxique et asphyxiante, mais aussi contre une température excessive des fumées. Elle a également permis de rechercher des matériaux souples pour la confection de rideaux dont la nature, la structure et la mise en oeuvre permettent d'assurer l'action coupe-feu et l'isolation thermique ainsi que l'étanchéité aux gaz pour une protection efficace et durable contre les effets de l'incendie.

Deux conceptions différentes de niches ont été étudiées dans le cadre de cette convention :

- une niche composée d'éléments de tubing et disposée à même le sol au centre de la galerie,

- une niche creusée sur le côté de la galerie et fermée par un rideau qui assure la protection contre le flux thermique et les gaz toxiques.

Les essais ont consisté à soumettre chaque type de niches successivement aux fumées chaudes d'un foyer d'incendie dont on a pu régler à notre guise la puissance calorifique.

Un brûleur à gaz capable d'assurer en continu les températures voulues au débit du ventilateur assurant l'aérage de la galerie, a été mis au point. Ce brûleur de puissance réglable entre 1 000 et 10 000 kW a été disposé à l'entrée de la galerie.

Les facteurs importants suivants ont été pris en compte :

- la ventilation naturelle qui tend à s'établir entre la galerie et la niche choisie (problèmes de pressurisation),

- les échanges thermiques entre les fumées chaudes de la galerie et l'atmosphère à l'intérieur de l'abri,

- la résistance à la chaleur des éléments souples de fermeture (rideaux).

Après évaluation des comportements et caractéristiques de ces niches dans leur conception actuelle nous avons cherché à optimiser leurs conditions d'utilisation en cherchant à :

- refroidir les fumées chaudes dans la galerie en amont de la niche par des sprinklers,

- refroidir les éléments de tubing par arrosage,

- refroidir les rideaux par arrosage,

- refroidir l'air injecté dans la niche,

- isoler thermiquement la niche type tubing.

Résultats obtenus :

Une recherche bibliographique nous a permis de montrer que le niveau moyen de température que l'on pouvait atteindre à quelques centaines de mètres d'un foyer d'incendie était de l'ordre de 100 à 200°C. Ces valeurs ont été reproduites expérimentalement dans nos installations.

Comportement de la niche "type tubing".

En premier lieu nous avons expérimenté l'étanchéité aux gaz et défini la quantité d'air à injecter dans la niche pour en assurer la purge ; un diagramme de ventilation en fonction du volume à purger et du degré de pureté recherchée a été mis au point. Les courbes permettent d'établir la quantité d'air à injecter dans une niche polluée avant d'y pénétrer : ce volume d'air a été estimé à au moins 5 fois le volume de la niche. Un débit d'air de 300 Nm^3/h permet de maintenir la niche en légère surpression (15 à 25 pascals) lorsque la température des gaz dans la galerie est de 200°C.

En second lieu nous avons déterminé l'évolution des températures dans la niche avec et sans dispositif de refroidissement en fonction de celle dans la galerie.

Les principaux résultats suivants ont été obtenus :

Pour une température de 100°C fixée en palier dans la galerie la température mesurée dans la niche atteint rapidement une valeur de 80-85°C.

Le refroidissement des parois de la niche par ruissellement d'eau permet de ramener la température à 60°C pour un débit d'eau de 2,5 m^3/h et 40°C pour un débit d'eau de 6 m^3/h.

Lorsqu'on porte le niveau de température dans la galerie à 160°C, la température à l'intérieur de la niche arrosée dépasse la limite de tolérance humaine et atteint 65°C.

Lorsqu'on porte le niveau de température dans la galerie à 200°C, la température à l'intérieur de la niche arrosée se stabilise autour de 100-110°C.

Comportement de la niche "type latérale".

Le débit de l'air comprimé (300 Nm^3/h) injecté dans cette niche a permis de la maintenir en très légère surpression (5 à 10 pascals). Les analyses des gaz effectuées ont montré que dans ces conditions le volume de la niche reste purgé et ce quel que soit le niveau de température dans la galerie.

Trois rideaux souples différents à base de tissus de verre ont été expérimentés ; ils sont enduits soit d'hypalon, soit de résine silicone. Les essais ont montré que cinq matériaux sont dotés d'une excellente résistance au feu.

Les mesures de température dans la niche ont donné les résultats suivants :

- pour une température de 200°C fixée en palier au niveau de la face extérieure du rideau arrosé, la température mesurée dans la niche n'a pas dépassé 25 à 35°C après plusieurs dizaines de minutes d'essai,

- la vitesse moyenne de montée de température dans la niche est de 0,15°C/min lorsque le rideau est refroidi ; elle passe à 4°C/min lorsqu'on arrête l'arrosage du rideau.

Quel que soit le niveau de température au droit du rideau dans la galerie (valeur comprise entre 70°C et 220°C) le niveau moyen de température dans la niche ne dépasse jamais la limite de tolérance humaine fixée à 60°C.

Bien que les niches latérales aient un comportement meilleur que celui des abris en galerie, nous retenons, pour des raisons évidentes de sécurité, le principe suivant : les niches pressurisables ne sont pas des lieux de séjour prolongé en cas d'incendie, elles sont seulement des lieux de halte permettant de préparer dans les meilleures conditions la suite de l'évacuation.

APPAREILS RESPIRATOIRES PORTES A LA CEINTURE

En cas d'arrivée de gaz toxiques ou irrespirables par défaut d'oxygène il faut disposer à bref délai d'une protection respiratoire.

Pour répondre à ce besoin nous utilisons dans les mines françaises des appareils respiratoires autonomes d'évacuation qui sont soit disposés à proximité du poste de travail soit portés en permanence à la ceinture.

Ces appareils sont autonomes, comportant une cartouche de KO_2 qui fournit l'oxygène nécessaire ; ils protègent donc quelle que soit la composition de l'atmosphère.

Les appareils portés à la ceinture sont de plus en plus répandus.

Pour un poids de 2 kg environ ils sont utilisables pour l'évacuation pendant une durée de 20 à 30 min. La cartouche initiale peut être remplacée par une cartouche de rechange procurant une durée d'utilisation supplémentaire de 70 à 90 min.

Durées observées lors des tests effectués sur une machine a respirer.

Débit	Cartouche initiale	Cartouche de rechange
35 l/min	26 min	72 min
30 l/min	31 min	88 min
10 l/min	110 min	6 h 35 min

Les cartouches de rechange sont déposées dans les niches pressurisables dont l'implantation a été déterminée de manière à ce que l'autonomie procurée par la cartouche initiale de l'appareil respiratoire permette à partir de n'importe quel poste de travail, de rejoindre la niche la plus proche.

L'autonomie a été observée au cours d'essais dans des chantiers réels dont les conditions de circulation correspondent à nos principaux types de chantiers. Les résultats ont été notés sous forme de temps de fonctionnement et de distance parcourue.

Type de chantier	Autonomie de la centrale initiale	
	Temps (en min)	Distance (en m)
Taille au pendage	33	1 870
Demi-dressants	18	1 010
Dressants	38	1 470

Les niches pressurisables sont implantées d'après ces résultats en prenant une marge de sécurité.

CONCLUSION

Des recherches et essais sont en cours pour rendre plus efficaces la détection et l'alerte rapide mais dès maintenant la stratégie de sécurité incendie dont nous venons de décrire les éléments essentiels constitue une amélioration importante de la sécurité.

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STRATEGIE ZUR AUFRECHTERHALTUNG DER FEUERSICHERHEIT.

C. Cwiklinski et al.

Feuer ist eine der Hauptgefahren in Gruben. Selbstentzündungen von Kohle und Explosionen sind wohl-bekannte Risiken, wobei seit kurzem die Einführung wachsender Mengen von Chemikalien und Polymeren neue Risiken mit sich bringt.

Diese Produkte verbessern die Arbeitsbedingungen und die technischen Ergebnisse, erfordern aber neue Verhaltensweisen im Hinblick auf Feuer-Sicherheit, welche der thermischen Auswirkung und der Giftigkeit der Rauchgase Rechnung tragen.

Der Vortrag schlägt eine neue Strategie zur Feuersicherheit in Gruben vor, die zum Beispiel das folgende einschließt:

Risikoanalyse in Experimentaleinrichtungen, um das Verhalten von Polymer-Beschichtungen unter dem Einfluß von Feuer zu beobachten (100m² Beschichtung in der Teststrecke in Cerchar).

Betriebsanalyse von verschiedenen Arten von Schutzräumen, Fluchtkammern und ihren Ausrüstungen für einen kollektiven Personenschutz unter dem Einfluß von Feuer

Betriebsanalyse von am Gürtel getragenen Sauerstoff - Selbstrettern zum individuellen Schutz

Einführung eines strategischen Feuersicherheitskonzeptes.

СТРАТЕГИЯ ПОЖАРНОЙ БЕЗОПАСНОСТИ

К. Цвиклинский и др.

Пожар - главная угроза безопасности в шахтах. Самовозгорание угля и взрывы являются хорошо известными проблемами. Возросшее в последние годы применение в шахтах химикатов и полимерных материалов принесло новую угрозу безопасности.

Эти материалы улучшают условия работы и помогают достигать более высоких технических результатов. Однако, они требуют нового подхода к проблеме угрозы пожаров, при котором будет обращаться специальное внимание на воздействие температуры и токсичность отходящих газов.

Настоящий доклад предлагает новую стратегию пожарной безопасности в шахтах, которая включает следующие компоненты:

- анализ риска, использующий лаборатории для изучения поведения полимерного покрытия при пожаре (100 кв.м покрытия в экспериментальном контейнере Серчара);
- анализ различных типов убежищ, спасательных камер и их оборудования для защиты групп людей во время пожара;
- испытание автономного переносного, прикрепленного к поясу кислородного аппарата для индивидуального спасения;
- представление стратегической концепции пожарной безопасности.

防火安全的策略

C．隋克灵士基 等

火灾是煤矿中主要的危险．煤的自燃和爆炸是已经很熟悉的危险．近来矿里增加了很多化学药品和高分子成品，产生了不少新的危险．

这些成品可以改进操作情形，得到更好的技术成果，但亦需要对热冲击和气烟的毒性有新的防火办法．

本文提出一种防火安全的新策略，包括：

. 危险分析，用试验设备研究高分子涂料在火灾中的变化，（在塞尔夏试验煤矿里１００ 平方米的涂料）

. 在火灾中各种不同的掩蔽处，躲避室和它们的集体保护设备操作的分析

. 个别保护用的系带氧气自救器的操作分析

. 防火安全策略的观念．

AN INVESTIGATION OF THE CONNECTIONS BETWEEN LARGE FIRE-AREAS BY SF_6 TRACER GAS TECHNIQUES

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ABSTRACT

This paper presents briefly the results of the investigations on the air leakage paths between large fire areas by the sulphur hexafluoride tracer gas techniques.

The field tests were carried out in Baidong coal mine. In the tests, large amount of sulphur hexafluoride was injected from the surface rock subsidence fissures. It was found out that the four fire areas, which have a total area of 1.5 square kilometers within bounds of the mine , were all connected to the surface through air leakage paths. Three of the fire areas were mutually connected, the other was isolated. These information provided a reliable basis for the working out of the comprehensive measures for fire prevention and fire fighting.

INTRODUCTION

The Baidong coal mine of Datong Mining Administration in Shanxi Province of China has an annual production of 1500 thousand tonnes, an area of 9.5 square kilometers . There are four minable coal seams which were numbered from top to bottom as No.9, No.11, No.12 and No.14. The mean seam thickness is 1.0 m, 5.5 m, 1.5 m, 3.2 m respectively . The mean distance between seams is 33m, 22m, 17m respectively. The dip angle of seams is about 5 degree. The No. 9 and No. 11 seams have been mined-out by the shortwall and pillar method. Since the 70's, the No.12 and No. 14 seams have been mined by the longwall method sequentially. Later , the No. 14 seam was mined with comprehensive mechanization. The roof caved in large area, this caused extensive subsidence fissures , which lead up to the hill surface . The mine is ventilated by the exhaust method. Large amount air leaks into the mine through the surface subsidence fissures, which caused spontaneous combustion at several places in the large aera of gob in seam No. 11. After being suppressed by grouting through drillings, there are still several inextinguished fire source in the four panels numbered 301, 509, 402 and 303. The total area of the four sealed fire areas exceedes 1.5 square kilometers, the extents of the fire areas are shown in figure 1. The working faces beneath the fire areas in seam 12 and seam 14 are endangered by the intrusion of fire gases. The pressure balance method was suggested by the Fushun Coal Research Institute to speed up the extinction of the fire, and to prevent the fire gases to intrude into the working panels. Huainan Mining Institute was invited to investigate the leakage connections between the fire areas and the surface and between the fire areas themselves by the sulphur hexafuoride(SF_6) tracer gas techniques.

THE FIELD TESTS

On the hills above the Baidong coal mine, a lot of open subsidence fissures were found. The width of the fissures vary from several cen-

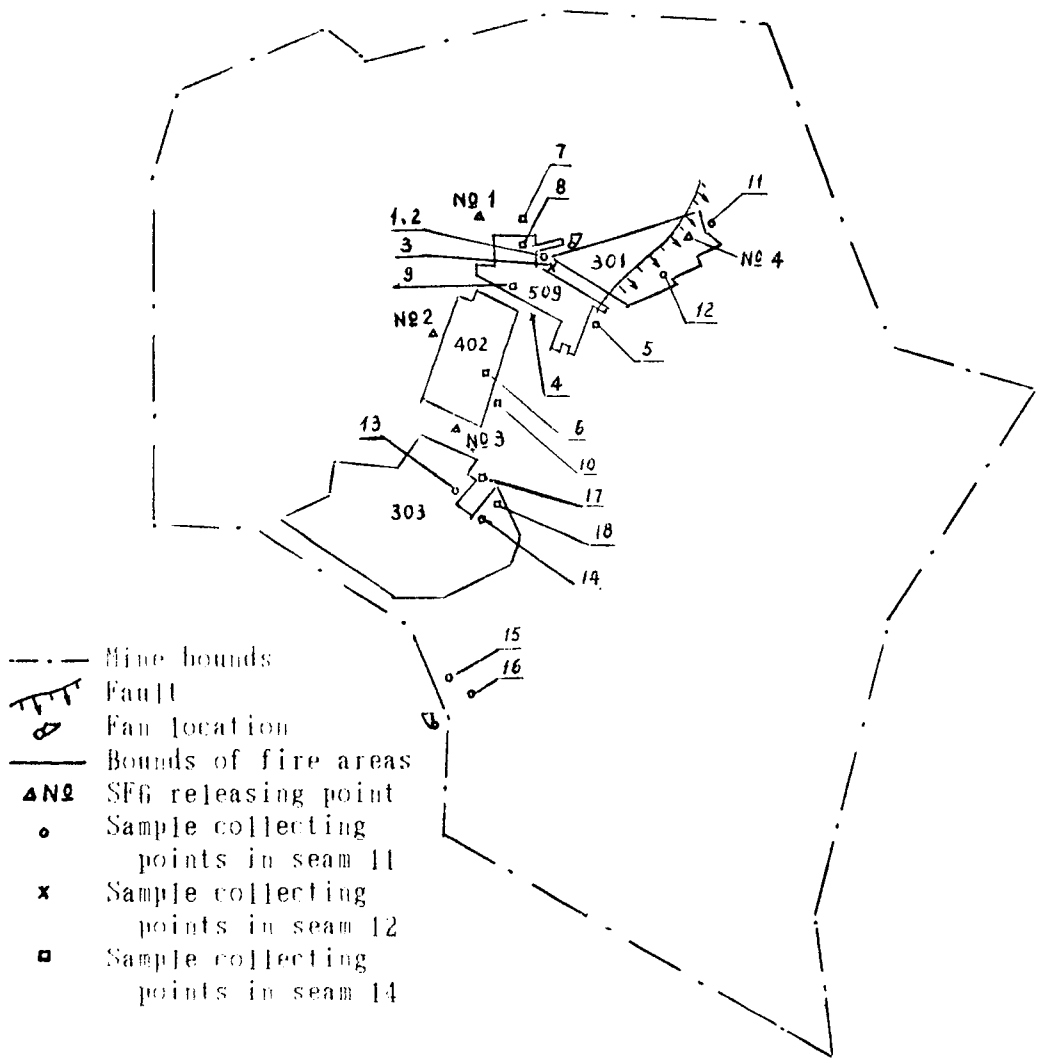


Figure 1

timetres to several decimetres, the length vary from several decametres to several hectometres. Based on the information of mine ventilation resistance survey, the air pressure distribution in the airways connected to the fire areas by air leakage paths were studied, the number and positions of SF6 releasing points on surface and the sample collecting points underground were determined.

The First Test

The purpose of this test is to investigate the connections of the two fire areas 509 and 402 with the surface above, and with the gobs in coal seams 12 and 14 underneath, and also the connections between the two fire areas themselves.

The SF6 releasing points are indicated with symbol △ in figure 1. The coordinates of the SF6 releasing points and the amount of SF6 released are shown in table 1. The SF6 releasing process at all the three releasing points were finished within an hour.

Table 1. SF6 Releasing Information

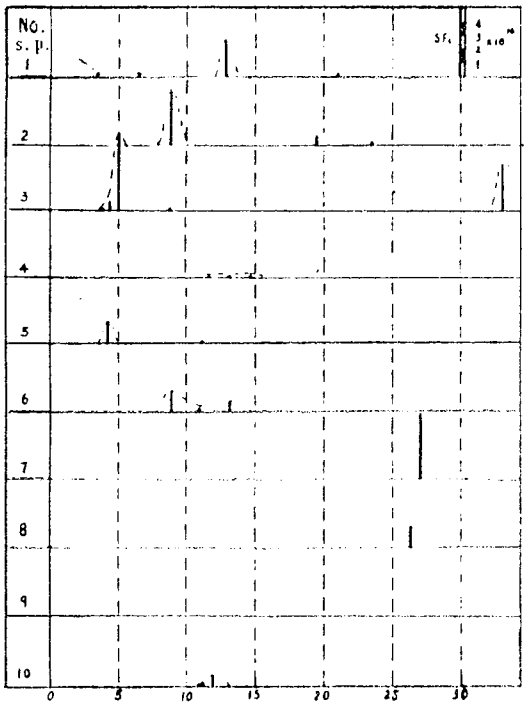
No.	Coordinates X	Y	Z	Amount (l)	Location
No. 1	46170	30670	1375	13.3	Surface
No. 2	45950	30230	1448	90.0	fissure
No. 3	46020	29820	1445	13.0	openings

The locations of the sample collecting points in the roadways in different coal seams are shown in figure 1. The distances between the releasing points and the collecting points, the relative pressure differences between the collecting points and the surface are shown in table 2. The time intervals of sampling vary from 15 minutes to 1 hour. The results of this test are shown in figure 2.

Table 2. Information of Sample Connecting Points

No. of Samp. Point	Location	Distances from Rel. to Col. Points (m)									Dif. of R. Pres. (Pa)
		From No. 1			From No. 2			From No. 3			
		Ver	Hor	Sum	Ver	Hor	Sum	Ver	Hor	sum	
1	At the outlet of the leakage hole connected to the 509 fire area in seam 11 at the bottom of the Zhengjiagou air shaft.	199	280	479	250	455	705	240	690	930	-1300
2	At the No. 304 seal connected to the 509 fire area in seam 11.	199	285	484	250	450	700	240	685	925	-1300
3	At the No. 400 seal connected to the intake of the former 8407 working-face in seam 12 (Beneath the 509 fire area)	207	310	517	272	500	772	264	730	994	-300
4	At the No. 422 seal connected to the return of the former 8418 working-face in seam 12 (Beneath the 509 fire area)	207	340	547	272	380	652	264	540	804	-300
5	At the No. 432 seal connected to the intake of the former 8424 working-face in seam 14 (Beneath the 509 fire area).	228	650	878	291	320	611	282	630	912	-700
6	At the No. 380 seal connected to the intake of the former 8401 working-face in seam 14 (Beneath the 402 fire area)	228	600	828	291	270	561	282	282	562	-300
7	At the No. 410 seal connected to the return of the former 8409 working-face in seam 14 (Beneath the 509 fire area)	228	240	468	291	600	891	282	880	1162	-380
8	At the No. 411 seal connected to the intake of the former 8409 working-face in seam 14 (Beneath the 509 fire area)	228	250	478	291	450	741	282	720	1002	-380
9	At the No. 392 seal connected to the return of the former 8407 working-face in seam 14 (Beneath the 509 fire area)	228	370	598	291	360	651	282	560	842	-300
10	In the return of the 8401 working face in seam 14. (mined out, not sealed)	228	610	838	291	250	541	282	270	552	-580

※ The sum of the vertical and horizontal distances can be considered as the maximum distances(Lm) that the tracer gas can flow. Supposing the time of the first SF6 detecting is known as t, the gas flow velocity (v) can be approximated by; $v = Lm / t$.



Time of sample collecting, h
(elapsed from releasing)

Figure 2

In this test, SF6 was detected at all the sample collecting points except one. Samples were collected at the outlet of the main fan evase of the Zhengjiagou air shaft, and SF6 were detected for several consecutive samples within 45 minutes since its releasing at point No. 1. The peak concentration reached 2 x 10 . This indicates that the releasing point No. 1 is extensively connected to the 509 fire area in seam 11. The No.1 collecting point is in a natural hole exposed to the wall of an airway at the bottom of the return air shaft. The air leakage from this hole is about 1.7 m/min, it is obviously the main leakage path from the surface to the return air shaft through fissures and the 509 fire area. It is reasonable that SF6 arrives here earier than the outlet of the main fan evase since the air leakage velocity here is as high as 10 m/min. The SF6 detected at this point 35 hours after its releasing should be in the "tailing" behind the peak concentration.

It is evident from the SF6 capturing information at each collecting point that the surface fissures connected with not only the 509 fire area in seam 11, but also the gobs in seams 12 and 14 beneath the fire area. The velocities of air leakage vary from 0.3 m/min to 2.4 m/min.

The information from the No. 6 collecting point shows that the 402 fire area connected with the surface and the gob of the former 8401 working face in seam 14. The No. 6 collecting point was near the No. 2 and No. 3 releasing points. The velocity of air leakage here is about 1 m/min.

SF6 were detected five times altogether at No.3 collecting point. The SF6 concentration at the first three times shows a continuous curve, the SF6 came obviously from the No.1 releasing point. The air leakage velocity is about 2 m/min. There is a 24.15 hours interruption between the fourth time and the fifth time, and a peak concentration appears at the fifth time. It is believed that SF6 was from the No.2 or No.3 releasing points. The air leakage velocity is about 0.4 - 0.5 m/min. This indicates that the 402 fire area was connected with the 509 fire area. According to the information from the mine, fumes from the 402 fire area appeared in the 509 panel in seam 11 while being mined, the fume temperature reached 50℃. In the meantime the temperature of water from the hot point in the 402 fire area exceeded 80℃. In the process of fire areas grouting, the temperature in both areas rose and fell synchronously. These also indicate that the 509 fire area was connected with the 402 fire area. This is due to the damage of the panel barrier pillar by the advancing of the 8407 comprehensively mechanized working face in seam 14 from the underneath of the 402 fire area to the underneath of the 509 fire area.

The No. 7 and No. 8 collecting points were added subsequently. High concentration of SF6 was captured at the first time, which shows that they were extensively connected with the

releasing points. No SF6 was captured at the No.9 collecting point, due to the low difference of relative pressure across the several permanent stoppings out of the mined-out working face.

The Second Test

The purpose of this test is to investigate whether the fault pillar in the centre of the 301 fire area served the function of isolation, the connections between the surface and the gobs in seams 11, 12 and 14 in this area, and whether the 301 fire area was connected with the 509 fire area.

In the test SF6 was injected on surface above the 301 fire area from the No. 6 grouting drilling located at the upper panel of the fault. The drilling led to the gob in seam 11, the related information are shown in table 3 and table 4.

Table 3. SF6 Releasing Information

No.	Coordinates X	Y	Z	Amount (l)	Location
No.4	46950	50500	1300	99.7	No. 6 grouting drilling

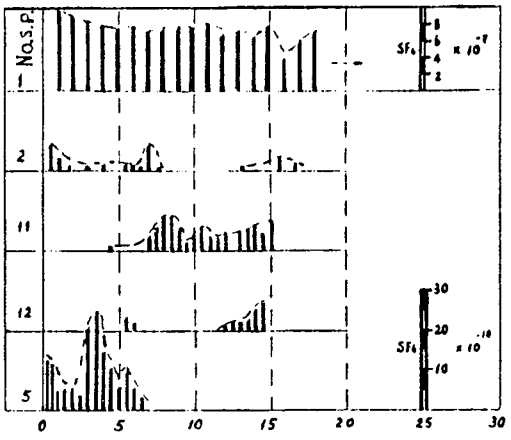
Table 4. Information of Sample Connecting Points

No. of Samp. Points	Location	Distance between No.4 Releasing Point and Collecting Points (m) Vertical	Horizon.	Max. Flow Dis.※	Dif. of R. Pres. (Pa)
1	At the outlet of the leakage hole connected to the 509 fire area in seam 11 at the bottom of the Zhengjiagou air shaft.	147	600	600	-1300
2	At the No.304 seal connected to the 509 fire area in seam 11.	147	600	600	-1300
11	At the No.217 seal connected to the 301 fire area in seam 11. (At the upper panel of the fault)	147	110	110	-100
12	At the No. 47 seal connected to the 301 fire area in seam 11. (At the upper panel of the fault)	147	200	200	-300
5	At the No.432 seal connected to the intake of the former 8424 working face in seam 14 (Beneath the 509 fire area).	184	510	547	-700

※Since SF6 was injected from the drilling, the maximum flow distance in seam 11 is approximated by the horizontal distance, while in seam 14 it is calculated from the horizontal distance plus the vertical distance between seam 11 and seam 14.

The analyzed results of samples collected are shown in figure 3.

High concentration of SF6 were captured continuously for more than 64 hours at No. 1 collecting point. Continuous SF6 concentration waves also appeared at other collecting points. These curves are obviously different from those of the first test. This shows the differences



Time of sample collecting, h
(elapsed from releasing)

Figure 3

between passing and not passing the fissures in the gas flow processes.

Information reveals that, the fault pillar in the 301 fire area was damaged by the mining process of the coal seams underneath, the fire areas in the upper and lower panels of the fault were connected with each other, this fire area was also connected to the 509 fire area extensively with the air leakage velocity exceeded 10 m/min. The quickly capturing of SF6 at the No.5 collecting point indicates that the gobs in seam 11 and seam 14 were also connected with each other extensively. The delay of SF6 capturing at No.11 and No.12 collecting points is due to the long time elapse since the roof caving in this area, which resulted in the low pressure difference relative to the surface. The air leakage velocity is as low as 0.4 to 0.6 m/min.

The Third Test

The purpose of this test is to investigate the connections of the 303 fire area with the surface and other fire areas. The 303 fire area and the working sections underneath were under the influence of the Yangjiangou air shaft ventilation system. The information related to the SF6 releasing points and the collecting points are listed in table 5 and 6.

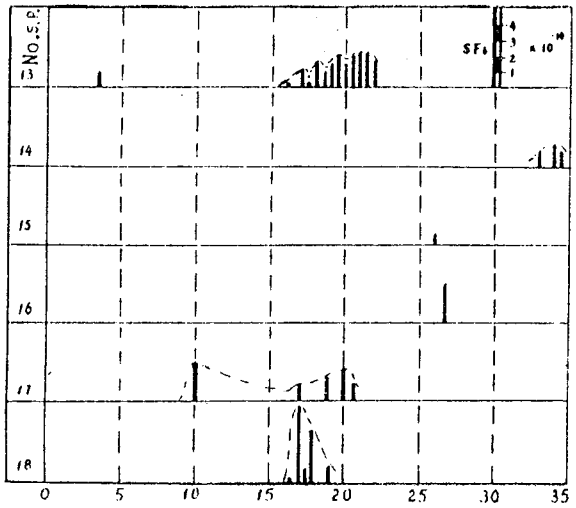
Table 5. SF6 Releasing Information

No.	Coordinates X	Y	Z	Amount (l)	Location
No.5	45900	49350	1400	130.4	Surface fissure openings

Table 6. Information of Sample Connecting Points

No. of Samp. Points	Location	Distance between No.5 Releasing Point and Collecting Points (m) Vertical	Horizon.	Sum	Dif. of R. Pres. (Pa)
13	In the air leakage hole led to the winder chamber.	191	220	411	-250
14	At the No.900 seal connected to the gob in seam 11.	191	180	371	-1000
15	At the No. 48 seal connected to the main return led to the Yangjiangou air shaft.	191	540	731	-3400
16	At the No.729 seal connected to the main return in seam 11.	191	540	731	-3400
17	At the No.216 seal connected to the gob in seam 14.	223	260	483	-1500
18	At the No. 63 seal connected to the gob in seam 14.	223	283	506	-1500

The analyzed results of samples collected are shown in figure 4.



Time of sample collecting, h
(elapsed from releasing)

Figure 4

The information from sample analysis shows that no SF6 was captured at any of the collecting points until 3.4 hours after the releasing of SF6 at the No.5 releasing point. This means that no SF6 from the preceding two tests reached this area. In the first test, samples were collected at the No.13 collecting point over a long period of time, also no SF6 was captured. Thus it can be concluded that the 303 fire area was isolated from other areas. This is due to the barrier effects of the panel pillar between panel 402 and panel 303. This pillar was about 40 meters in width, and was kept intactly during the mining of coal seam 12 and seam 14.

In this test SF6 was captured at all the collecting points, but the time of capturing were late except for the No.13 collecting point, which was connected to the fire area directly through a hole. The air leakage velocities are low, vary from 0.2 m/min to 0.8 m/min. These mean that the surface fissures were connected with the gobs in coal seams 11, 12 and 14, but the connections were not very extensive. This is due to that the coal in this area was mined out long time before by mainly the shortwall and pillar method.

CONCLUSIONS

The 301, 509 and 402 three fire areas in coal seam 11 were connected upper with the surface and lower with the gobs in seam 12 and seam 14 through the rock subsidence fissures, meanwhile, they were also connected with each other. Because large amount of air were inhaled from surface, the fire areas could not be extinguished over a long period of time, this endangered greatly the mining of coal seams underneath. During the treatment of the fire areas to prevent the fumes from spreading to the working panels by the pressure balance techniques, the three fire areas should be considered as an integral. The 303 fire area in seam 11 is an isolated fire area, which is not extensively connected with the surface fissures. The air leakage in the hole connecting the No. 13 sample collecting point with the fire area is obvious, it should be filled up as soon as possible so as to prevent the occurence of spontaneous combustion in the coal pillar near the winder chamber.

REFERENCE

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用六氟化硫示踪技术探查煤矿大面积火区之间的连通关系

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论文提要：本文扼要介绍应用六氟化硫示踪技术探查火区间漏风通道的结果。

在白洞矿进行了从地面岩层沉降裂隙注入大量六氟化硫的试验，查明了井田范围内总面积约1.5平方公里的四个火区与地面均有漏风联系，而其中三个火区是相互连通的，一个火区是孤立的，为制订全矿综合防灭火措施提供了可靠的依据。

引　言

中国山西省大同矿务局白洞煤矿年产150万吨，井田面积约为9.5平方公里。从上而下有9号、11号、12号、14号四个可采煤层，平均层厚分别为1m、5.5m、1.5m、3.2m。各煤层间距分别为33m、22m、17m，各煤层顶板岩层多为砂岩，煤层倾角3～5度。9号、11号煤层大部已用刀柱法采完，七十年代起，12号、14号煤层先后用长壁冒落法开采，特别是14号煤层采用综合机械化开采后，顶板大面积冒落，岩层沉降裂隙发达，直通山头地表，矿井采用抽出式通风，从地面经裂隙大量向矿内漏风，在11号煤层大面积采空区中多处发生煤炭自燃，经打钻灌浆处理，还有原301、509、402、303四个盘区的采空区内的火源没有熄灭。这四个封闭的火区面积达1.5平方公里，火区范围如图1所示。在火区下开采12号、14号煤层，回采工作面受到火烟侵入的威胁。抚顺煤炭研究所拟采用均压调节技术加速火区的熄灭，并防止烟气向生产盘区漏出，邀请淮南矿业学院合作，应用SF_6示踪技术查清火区与地面以及火区之间的连通情况。

白洞煤矿火区漏风通道的探查

在白洞矿地表山头上出露有大量塌陷裂隙，宽度从几厘米到几十厘米，长度从几十米到几百米。根据全矿井巷通风阻力测定资料，分析与火区有漏风联系的井巷风压分布，确定地面的SF_6释放点和井下取样点的数量和位置。

第一次实验　目的是探查井田中部509、402两火区上与地表、下与12号、14号层采空区的连通情况，以及两火区相互连通的情况。

释放点用符号A标注在图1中，其位置与SF_6释放量见表1。三释放点于1小时内释放完毕。

在不同煤层的巷道中设置的取样点位置见图1。取样点与释放点间距，取样点与地表的相对压能差参见表2。取样间隔时间从15分钟到1

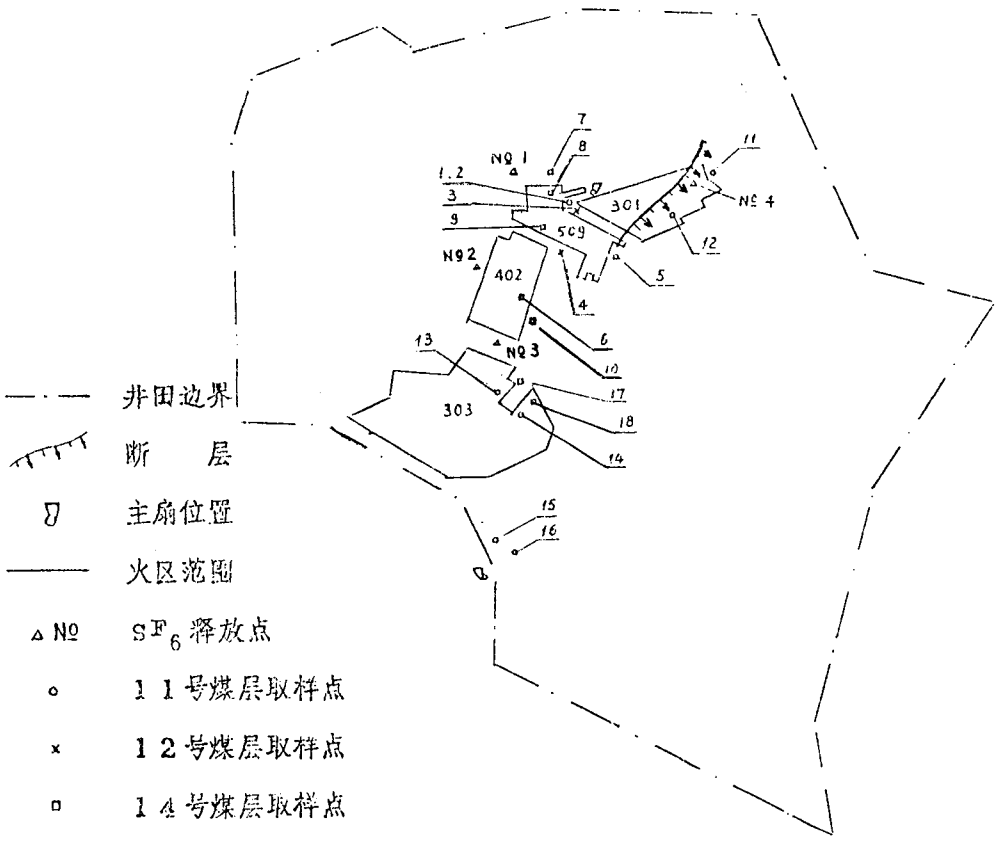


图1　白洞煤矿火区分布情况

小时不等，测定结果见图2。

本次实验中只有1个井下取样点采捕得SF_6，在郑家沟风井主扇扩散器口取样，于№1点释放45分钟起即连续几次捕得SF_6，最高浓度为2×10^{-10}，说明№1释放点与11号煤层的509火区连通程度很高。1号取样点是风井底垂直于巷道壁的一个天然孔洞，漏出风量约1.7 m³/min，显然是从地面经裂隙、509火区到风井的主要漏风通道，该处出现SF_6的时间应早于主扇扩散器口，漏风风速高达10 m/min，该取样点在3.5小时取样捕获的SF_6应是高浓度波后的"拖尾"。

从各取样点捕获SF_6的资料说明，地表裂隙不仅与11号层509火区相通，而且与火区下部12号、14号煤层的采空区相通，漏风风速从0.3 m/min至2.4 m/min不等。

表1

释放点号	释放点座标			SF_6释放量(l)	位置
	X	Y	Z		
№1	46170	30670	1375	13.3	地表裂隙口
№2	45950	30230	1448	90.0	
№3	46020	29820	1445	13.0	

6号取样点的资料表明，402火区上通地表，下通14号煤层原8401工作面采空区。该点与№2、№3释放点较近，其漏风风速约1 m/min左右。

3号取样点先后五次捕获SF_6。前三次呈连续波形，SF_6显然来自№1释放点。漏风风

表2

取样点号	位置	取样点与排放点间距 (m) 离 №1点 垂距	离 №1点 平距	离 №1点 垂、平距之和*	离№2点 垂距	离№2点 平距	离№2点 垂、平距之和	离 №3点 垂距	离 №3点 平距	离 №3点 垂、平距之和	相对压能差 (Pa)
1	郑家沟风井底，通11号层509火区的漏风孔洞中	199	280	479	250	455	705	240	690	930	-1300
2	通11号煤层509火区的304号密闭	199	285	484	250	450	700	240	685	925	-1300
3	通12号煤层原8407工作面进风道的400号密闭（509火区下部）	207	310	517	272	500	772	264	730	994	-300
4	通12号煤层原8418工作面回风道的422号密闭（在509火区下部）	207	340	547	272	380	652	264	540	804	-300
5	通14号煤层原8424工作面进风道的432密闭（在509火区下部）	228	650	878	291	320	611	282	630	912	-700
6	通14号煤层原8401工作面进风道的380密闭（在402火区下部）	228	600	828	291	270	561	282	282	562	-300
7	通14号煤层原8409工作面回风道的410密闭（在509火区下部）	228	240	468	291	600	891	282	880	1162	-380
8	通14号煤层原8409工作面进风道的411密闭（在509火区下部）	228	250	478	291	450	741	282	720	1002	-380
9	通14号煤层原8407工作面回风道的392密闭（在509火区下部）	228	370	598	291	360	651	282	560	842	-300
10	14号煤层8401工作面回风道（已采完，未封闭）	228	610	838	291	250	541	282	270	552	-580

* 垂、平距之和可认作示踪气体流动的可能最大距离 Lm，已知初次检得 SF_6 的时间 t，可估算其流速 V，$V = Lm / t$。

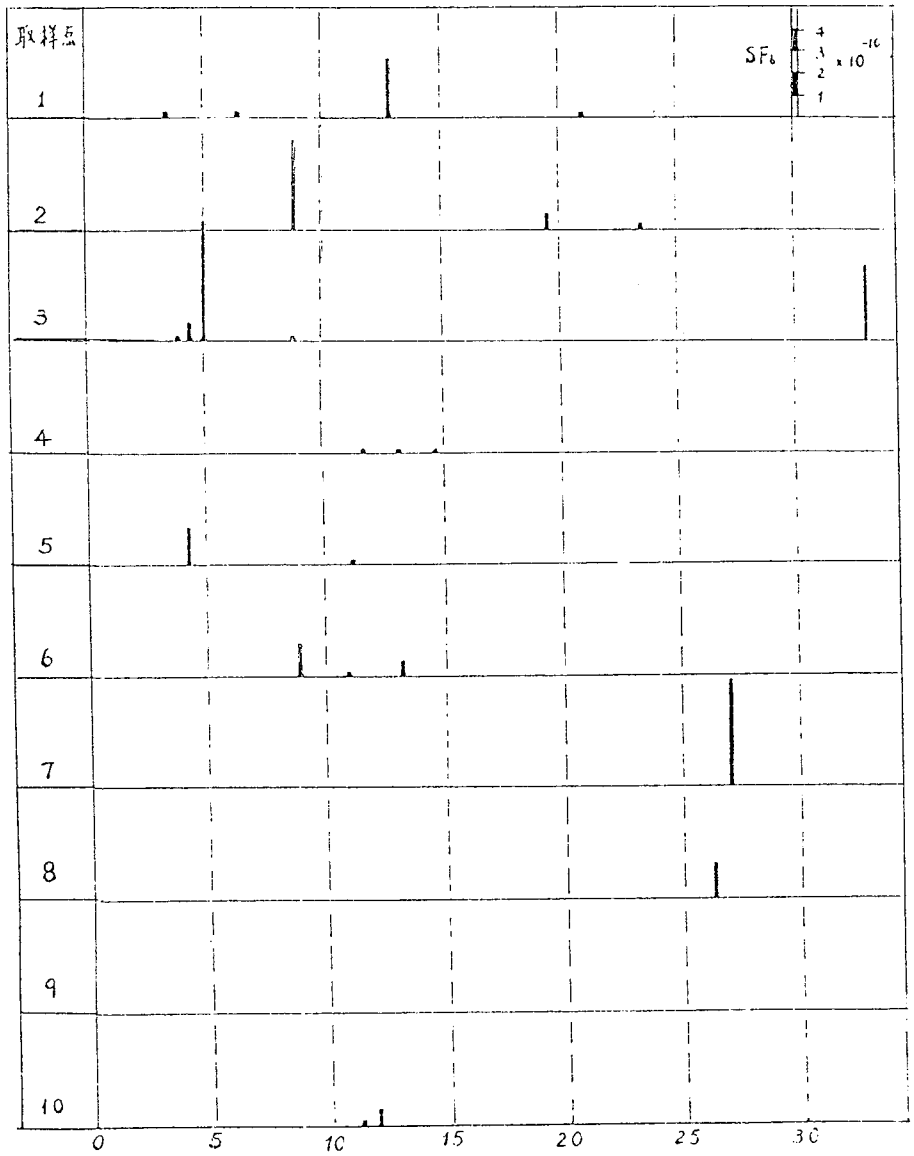


图 2

速约2 m／min。第五次与第四次间隔24.15小时。出现高浓度峰，可认为SF_6来自№2或№3释放点，漏风风速为0．4～0．5m/min，说明402火区与509火区是连通的。据矿方资料，在11号煤层509盘区开采时曾出现来自402火区的烟气，温度达50℃，当时402火区高温点出水温度在80℃以上。在火区灌浆过程中，两处的温度同步升降，也说明509火与402火区是连通的，原因是14号煤层内的8407综采工作面从402火区下部采到509火区下部，盘区间煤柱已遭破坏。

第7、8两取样点是后来追加的，第一次取样已捕获较高浓度的SF_6，说明该处连通程度较高。第9取样点未捕得SF_6是因为此工作面早已采完，且有多道永久密闭封闭，相对压能差也较小。

第二次实验　目的是考察301火区中部的断层煤柱是否起到隔断作用，在该区范围内从

地面到１１号、１２号、１４号煤层采空区的连通情况，３０１火区与５０９火区是否连通。

这次实验将SF_6从３０１火区地面位于断层上盘的6号灌浆孔注入，钻孔打到１１号煤层采空区，有关情况见表３、４。取样分析资料见图3.

表3

释放点号	座标			释放量(t)	位置
	X	Y	Z		
№４	46950	50500	1300	99.7	6号灌浆孔口

１号取样点连续捕得高浓度SF_6达６４小时以上，其它各取样点也普遍出现连续的SF6浓度波，与第一次实验的波形有明显不同，反映了流动过程中是否通过裂隙通道的差异。

资料表明：３０１火区断层煤柱已受下部煤层开采破坏，不仅其上下盘火区连成一片，而且与５０９火区相连通，连通程度很高，漏风风速大于１０m／min；５号取样点很快捕得SF_6。说明１１号层到１４号层采空区的连通程度也很

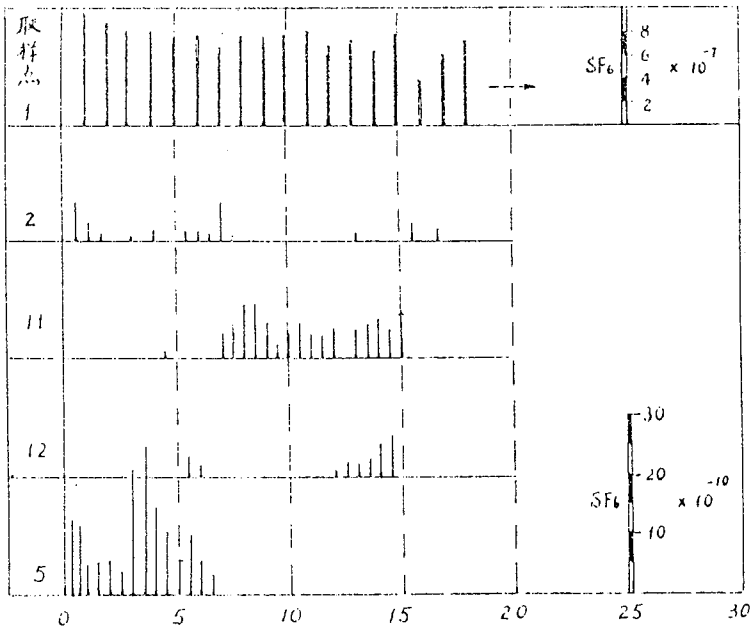
表4

取样点号	位置	取样点与№４释放点间距（m）			相对压能差（Pa）
		垂距	平距	最大流动距离*	
1	郑家沟风井底通１１号煤层５０９火区的漏风孔洞	１４７	６００	６００	－１３００
2	通１１号煤层５０９火区的３０４密闭	１４７	６００	６００	－１３００
11	通１１号煤层３０１火区的２１７密闭（在断层上盘）	１４７	１１０	１１０	－１００
12	通１１号煤层３０１火区的４７密闭（在断层上盘）	１４７	２００	２００	－３００
5	通１４号煤层原３４２４工作面进风道的４３２密闭（５０９火区下部）	１８４	５１０	５４７	－７００

＊因通过钻孔注入SF_6，在１１号层中最大流动距离只算平距；而在１４层中的取样点，则按平距加１１号、１４号层间垂距计算。

高；１１、１２号取样点捕获SF_6的时间滞后，是由于该区顶板冒落时间很久，与地面相对压能差很小，漏风风速只有０．４～０．６m/min.

第三次实验　目的是考察５０３火区与地面和其它火区的连通情况。该火区及其下部采区在羊涧沟风井通风系统影响范围内，释放点与取样点的有关情况见表５、６。



取 样 时 间 （从释放时起算），h

图 3

表5

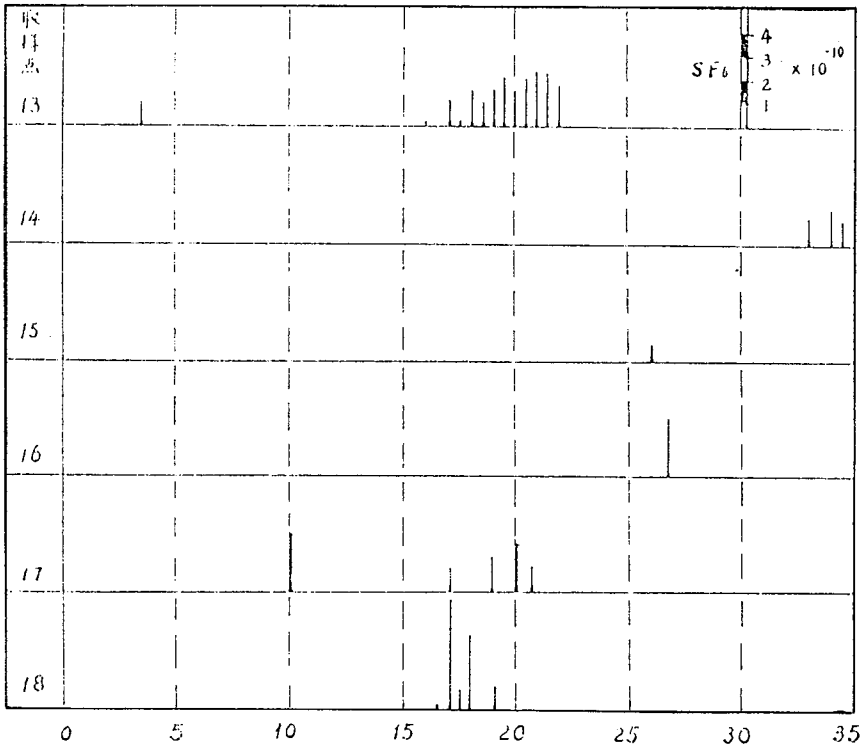
释放点号	座标 X	座标 Y	座标 Z	释放量 (l)	位置
№5	45900	49350	1400	130.4	303火区地表裂隙口

取样分析资料见图4。

取样分析资料表明。在№5释放点释放SF_6以前到释放后3.4小时内，本实验所有取样点均未检得SF_6。说明前两次实验释放的SF_6未到达本区。在第一次实验时，留在13号取样点

表6

取样点号	位置	取样点与№5释放点间距 (m) 垂距	平距	垂、平距之和	相对压能差 (Pa)
13	通统车房的漏风孔洞	191	220	411	−250
14	通11号层采空区的900号密闭	191	180	371	−1000
15	通羊洞沟风井11号层总回风巷的48号密闭	191	540	731	−3400
16	通11号总回风巷的729号密闭	191	540	731	−3400
17	通14号层采空区的216号密闭	223	260	483	−1500
18	通14号层采空区的63号密闭	223	283	506	−1500



取 样 时 间 （从释放时间起算），h

图 4

长期取样，也未捕获SF_6。因此可以确认３０３火区是孤立的，因为４０２盘区与３０３盘区间的煤柱宽达４０m，在１２号、１４号煤层开采时也注意保持盘区间煤柱的完整，起到了隔断作用。

本次实验除了１３号取样点因有直通火区的孔洞而捕获SF_6较早外，其它各取样点也均捕获SF_6，但时间较晚，漏风风速较小，从０．２m／min到０．８m／min不等，说明地表裂隙与１１、１２、１４号煤层的采空区均相通，但连通程度不高，原因是本区煤层早已采完，且多用刀柱法开采。

结 论

１１号煤层３０１、５０９、４０２三个火区，通过塌陷裂隙上通地表，下达１２号、１４号煤层的采空区，三个火区又互相连通，由于从地面大量吸风，火区长期不能熄灭，对火区下部煤层的开采造成严重威胁，在采用均压调节技术治理火区，防止烟气流入生产盘区，必须将此三个火区视作一个体系统盘考虑。１１号煤层３０３火区是一个孤立火区，与地表裂隙的连通程度不高，但１３号取样点通火区的漏风孔洞漏风明显，

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TECHNIQUE SF_6 DE DETECTION DU DEPLACEMENT D'AIR VERS D'IMPORTANTES ZONES D'INCENDIE SOUTERRAINES ET ENTRE CES ZONES

Wu Zhongli et al

Des observations ont été effectuées dans la mine de charbon de Baidong au moyen de la technique de détection par SF_6. D'importantes quantités de SF_6 sont libérées à travers des craquelures d'affaissements superficielles. Ceci a permis de découvrir que chacune des quatre zones d'incendies, de 1,5 km^2 au total environ, comporte ses propres passages d'air vers la surface et que trois d'entre elles sont communicantes, la quatrième étant séparée. Cette expérience fournit des indications très fiables permettant de développer des méthodes de prévention et d'extinction des incendies pour la totalité des mines.

SF_6 INDIKATOR-TECHNIK ZUR AUFSPÜRUNG VON LUFTVERBINDUNGEN VON UND ZU GROßEN BRANDGEBIETEN UNTERTAGE.

Wu Zhongli et al.

In der Baidong Kohlengrube werden unter Anwendung der SF_6 Indikatoren Beobachtungen gemacht. Große Mengen von SF_6 werden durch Bodenspalten freigesetzt. Dabei wurde entdeckt, daß jedes der vier brennenden Gebiete, die insgesamt eine Fläche von 1,5 km umfassen, seine eigene Luftzufuhr von der Oberfläche hat, drei der Gebiete sind miteinander verbunden, während das vierte von den übrigen getrennt ist. Die Versuche liefern sehr verläßliche Richtlinien zur Entwicklung von Methoden zur Feuerverhütung und Feuerlöschung in Gruben untertage.

МЕТОД ПРИМЕНЕНИЯ SF_6 ДЛЯ ОПРЕДЕЛЕНИЯ ПУТЕЙ ПРОХОЖДЕНИЯ ВОЗДУХА МЕЖДУ БОЛЬШИМИ ПОДЗЕМНЫМИ УЧАСТКАМИ ОХВАЧЕННЫМИ ПОЖАРОМ

Ву Жонгли и другие, КНР

Были проведены наблюдения в Байдонгской угольной шахте с использованием SF_6 как индикатора для слежения. Большие количества SF_6 были запущены через поверхностные трещины оседания. Было обнаружено, что каждый из четырех горящих участков (общей площадью полтора квадратных километра) имел свои собственные пути для прохождения воздуха на поверхность. Три из них были соединены между собой, и четвертый существовал отдельно. Проведенный эксперимент предлагает очень надежный способ для разработки методов предупреждения и тушения пожаров для всей шахты.

FIRE IN A DESCENDING AIR FLOW

T. P. Stefanov and E. E. Arsenian
Higher Institute of Mining and Geology, Bulgaria

ABSTRACT

Fires in descending air flows in the inner part of the ventilation net bring to some difficulties in forecasting and control of failures. A method of numerical modelling (numerical experiment) for finding out the peculiarities of these failures is applied. Definite mine nets are analyzed.

The influence of the speed and the temperature of air, the depression of the ventilators and the airdynamic resistances over the stability of the ventilation system is investigated. The possibilities for control of the ventilation system in case of fire for saving people and putting out fire are appreciated.

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Manuscript was not received in time to be included in Proceedings.

INCENDIE DANS UN COURANT D'AIR DESCENDANT

T. P. Stefanov, E.E. Arsenian

Il est difficile de prévoir et de contrôler les incendies qui se produisent dans les courants d'air descendants à l'intérieur des systèmes de ventilation. On applique une méthode de modelage numérique (expérience numérique) pour trouver leurs particularités. Des réseaux miniers définis sont analysés.

Nous examinons l'influence de la vitesse et de la température de l'air, de la dépression des ventilateurs et de la résistance aérodynamique sur la stabilité du système de ventilation. Nous évaluons les possibilités de contrôle du système de ventilation en cas d'incendie, pour sauver des vies humaines et éteindre le feu.

BRÄNDE BEI ABSTEIGENDER BEWETTERUNG.

T. P. Stefanov, E. E. Arsenian

Brände bei absteigender Bewetterung im inneren Teil eines Wettersystems bringen einige Probleme in der Voraussage und Kontrolle von Ausfällen mit sich. Ein numerisches Modell wird zur Ergründung der Eigentümlichkeiten dieser Ausfälle angewandt. Echte Wettersysteme werden analysiert.

Der Einfluß der Wettergeschwindigkeit, des Unterdrucks des Lüfters und des aerodynamischen Widerstandes auf die Stabilität wird untersucht. Die Möglichkeiten der Kontrolle des Bewetterungssystems im Brandfalle zur Brandbekämpfung und zur Rettung der Belegschaft werden untersucht.

ПОЖАР В НИСХОДЯЩЕМ ВОЗДУШНОМ ПОТОКЕ

Т.П. Стефанов и Э.Э. Арсенян

Пожары в нисходящих воздушных потоках внутренней части вентиляционной сети создают трудные для прогнозирования и управления аварийные ситуации. Чтобы понять особенности этих аварий, использован метод числового моделирования (числового эксперимента). Анализированы конкретные рудничные сети.

Изучено влияние скорости и температуры воздуха, депрессии вентиляторов и аэродинамических сопротивлений на устойчивость вентиляционной системы. Оценены возможности управления вентиляционными системами во время пожара для спасения людей и ликвидации пожара.

下降空气流的火灾

T. P. 司德分诺夫, E. E. 阿尔锡镍恩

在通风网路内部有下降空气流的火灾不易预测和控制。这里我们用数值模型（数值实验）来找出这种灾害的特性。在分析中我们用了某些确定的矿网。

空气流的流速和温度，通风机的抽空，及空气阻力对通风设备稳定性的影响在这里将作研讨。我们可以意识到火灾时以控制通风设备来挽救人员和扑灭矿火的可能性。

NEW FINDINGS FROM FIRE TESTS ON DIESEL-POWERED FREE-STEERED VEHICLES FOR UNDERGROUND MINES

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ABSTRACT

The tests in the Tremonia Experimental Mine were carried out in a road with continuous ventilation and in one with auxiliary ventilation, and standard cooling curves dependent of the height of the roadway were obtained. The total duration of the fire, the development of the temperature in the vehicle and the composition of the combustion gases were determined. We had also to find out whether, with auxiliary ventilation, it would be appropriate to use metal instead of plastic air ducts, making it feasible to supply fresh air to workers at the roadhead for as long as possible. Finally, we considered whether, in the case of fire, it is better to switch off the auxiliary ventilation fan or to let it continue to operate.

INTRODUCTION

The increasing use of free-steered vehicles in bituminous coal mines also increases the risk of fire. To make it possible to assess the fire hazard more effectively, the investigations carried out at the Tremonia Experimental Mine, and which had been already reported at the 21st International Conference in Sydney, were continued. In contrast to the earlier investigations, we now used greater quantities of fuel for the fires and we placed the test vehicle both in a road with continuous ventilation and in a road with auxiliary ventilation. Again, we took, as a basis, the most unfavourable case, i.e. where a narrowly limited original fire develops into a full open fire within a very short time, because the HRD extinguisher and other safety equipment on the vehicle do not function.

EXPERIMENTAL ARRANGEMENT

The position of the vehicle was varied in accordance with the conditions (fig. 1). In the tests with continuous ventilation the mock-up vehicle was placed between the 260 m and 265 m points in the roadway; in the tests with auxiliary ventilation it was placed between the 300 m and 305 m or the 280 m and 285 m marks. The experimental roadhead, simulated by air doors 1 or 2 was either at the 190 m or at the 226 m point. The different positions of the mock-up gave the various distances between vehicle and roadhead.

Between the 190th and 230th metre in the roadway, fire drift 2 had a cross-section of 7 m²; between the 230th and 320th metre, the cross-section was 16 m².

TESTS WITH CONTINUOUS VENTILATION

A total of 4 tests were carried out; in addition to the four tyres with a total weight of 320 kg, two further quantities of fuel were used and two air speeds were investigated.

Table 1 list the variable experimental conditions.

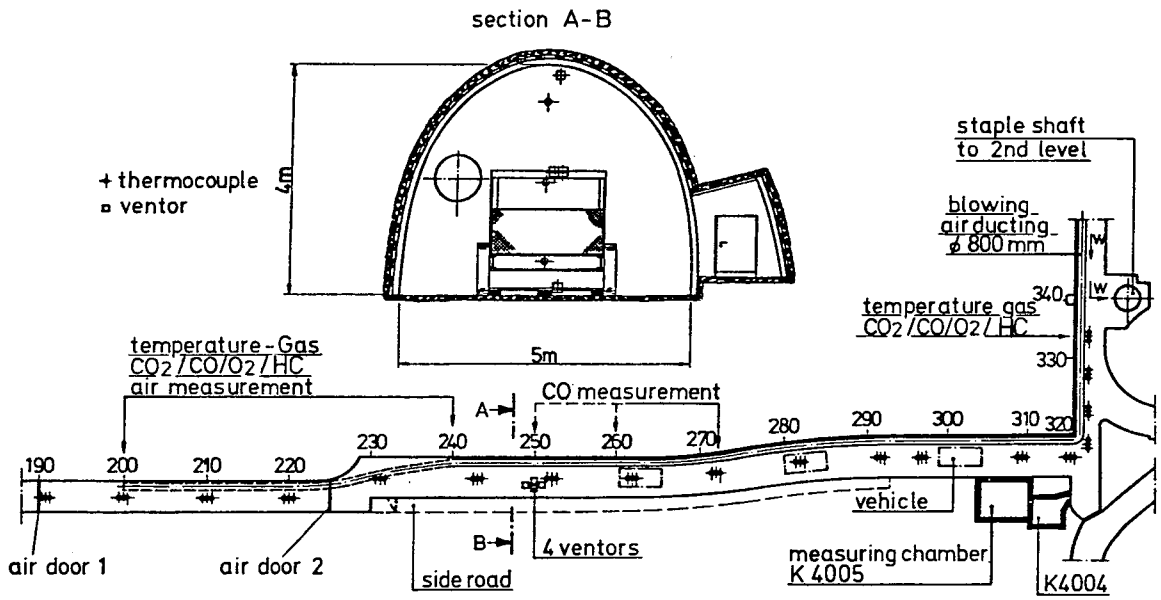


Figure 1. Experimental arrangement with various positions of mock-up vehicle in fire drift 2 in the Tremonia Experimental Mine

Table 1: Experimental conditions in the case of vehicle fires in a road with continuous ventilation

VERSUCH-NR.	DIESELKRAFTSTOFF (l)	OEL (l)	w (m/s)	$\dot{V}$ (m³/min)
7	200	-	0,7	650
8	200	-	1,4	1300
9	400	200	1,4	1300
10	400	200	0,7	650

Key:
Versuch = test
Dieselkraftstoff = diesel fuel
Oel = oil
W = airspeed
V = airflow

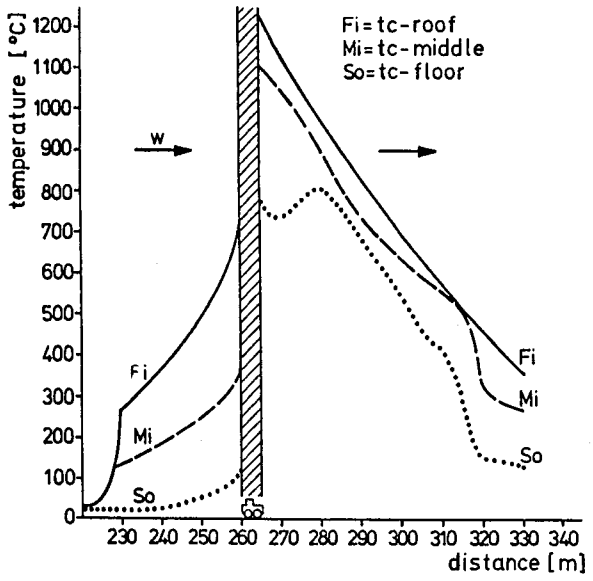


Figure 2. Maximum temperatures in a roadway with continuous ventilation as a function of the distance from the fire;
w (air speed) = 0.7 m/s; fuel for fire 600 l liquid fuel plus 320 kg tyres (test no. 10)

Temperatures

Examples of the temperatures developing in the roadway during tests no. 10 and 9 are illustrated in figs. 2 and 3.

In the case of the greater air speed of 1.4 m/s (fig. 3) the temperature stratification on the fresh air side is marked in another way. The temperature pattern in the thermocouples (tc) on the floor (So) and middle (Mi) is, in this case congruent and the temperatures do not exceed 50 °C.

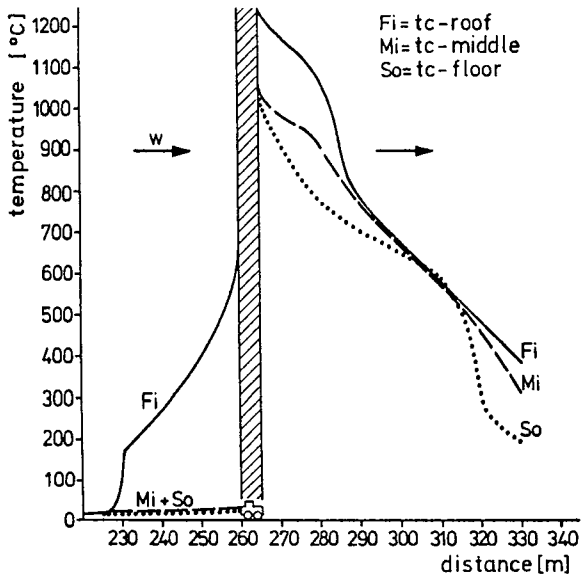


Figure 3. Maximum temperatures in a roadway with continuous ventilation as a function of the distance from the fire;
w (air speed) = 1.4 m/s;
fuel for fire 600 l liquid fuel plus 320 kg tyres (test no. 9)

The thermocouple in the roof (Fi), however, indicates a similar temperature pattern to that in test 10; the maximum temperature here is 650 °C.

Figs. 2 and 3 show, further, that the reflux of the combustion gases against the fresh air below the roof extended to the cross-section reduction at 230 m. If this reduction had not been present, the reflux below the roof would have continued to spread against the direction from which the fresh air was coming.

In the return air zone immediately behind the vehicle, temperatures of about 1200 °C occurred; that is considerably higher than those in the fresh air zone; this maximum temperature had dropped to about 600 °C at a distance of 35 m behind the vehicle. By extrapolating, we find that, for a distance of 100 m behind the vehicle in the return air zone, the temperature is still approximately 55 °C. From this distance onwards, persons could survive by using an oxygen self-rescue appliance. Such an oxygen self-rescue appliance is necessary, however, because the O_2 content of the air can decrease so far that the normal CO filter self-rescue appliance no longer affords protection.

Duration of fire

In the ideal case, a fire in a vehicle can be divided into three sections: the starting-, maximum- and dying-out phases (fig. 10). When reference is made, in the following, to "duration of fire", it should be taken to mean the starting- and the maximum phase.

Table 2: Duration of fire in a vehicle in the starting- and maximum phase with continuous ventilation

GRUND-BRANDLAST:		*320 kg REIFEN*	
200 l Dieselkraftstoff		*400 l Dieselkraftstoff 200 l Mineraloel*	
w = 0,7m/s (Versuch 7)	*w = 1,4m/s (Versuch 8)*	*w = 0,7m/s (Versuch 10)*	*w = 1,4m/s (Versuch 9)*
20 min	*15 min*	*25 min*	*20 min*

Key:
Grundbrandlast = basic quantity of fuel for fire
Reifen = tyres
Dieselkraftstoff = diesel fuel
Mineraloel = mineral oil
w = airspeed

In the tests with continuous ventilation, and using the same air speed, the duration of the fire, with increasing quantity of fuel, persisted for a longer period; this was because of the larger quantity of fuel used. It is true that the duration of the fire did not increase in direct proportion to the amount of fuel, as the burning surface for example likewise became greater because of the mineral oil flowing out in the vehicle and on the floor. On the other hand, when the air speed was increased and the amount of fuel remained the same, the fire lasted for a shorter time - this can be explained by the fact that the burning products were carried away more rapidly and because the supply of oxygen was greater.

If, to the times quoted in Table 2, we add a further 10 mins. to represent the dying-out phase then the total duration of the fire in these tests was approximately 35 mins.

Composition of the fire gases

Table 3 lists the concentrations of CO, CO_2, O_2 and the total hydrocarbons (HC) in the return air zone of the fire. To make things simpler, only the values measured under the roof are given in the table. In the centre of the roadway the values were somewhat lower as the hot combustion products, because of the thermal buoyancy, are carried along below the roof and fresh air dilutes the gases in the vicinity of the floor. We see that, with increasing fuel, the concentration of gas (except for O_2) also increase; this is immediately obvious. With regard to the air speed conditions are reversed as a greater air current dilutes the combustion gases.

Table 3: Extreme values of combustion in the return air zone of vehicle fires with continuous ventilation

BRANDGAS (%)	GRUND-BRANDLAST: 320 kg REIFEN			
	200 l DIESELKRAFTSTOFF		400 l DIESELKRAFTSTOFF 200 l MINERALOEL	
	w=0,7m/s (Versuch 7)	w=1,4m/s (Versuch 8)	w=0,7m/s (Versuch 10)	w=1,4m/s (Versuch 9)
CO	0,1	0,1	0,4	0,3
CO2	7,5	2,5	10,0	8,1
O2	10,2	17,1	9,7	10,5
HC	<0,1	<0,1	0,1	<0,1

Key:

Brandgas = combustion gas
Grundbrandlast = basic quantity of fuel
Reifen = tyres
Dieselkraftstoff = diesel fuel
Mineraloel = mineral oil
w = airspeed

TESTS WITH AUXILIARY VENTILATION

In these tests the air flow was approximately 10.8 m³/s which corresponded to an air speed of 0.7 m/s in the free cross-section of the roadway. The fuel used, in addition to the 4 tyres, was, in each case, 400 l diesel fuel and 200 l mineral oil. The roadhead was ventilated by an auxiliary ventilation system via plastic- or sheet metal ducting.

By way of example, fig. 4 shows the experimental arrangement with a sheet metal duct laid along the wall (v. table 4, test no. 6).

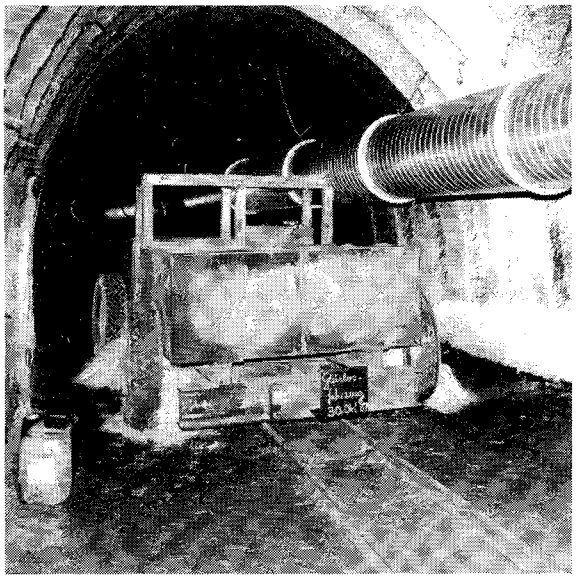


Figure 4. Auxiliary ventilation with a sheet metal duct laid along the wall (test no. 6)

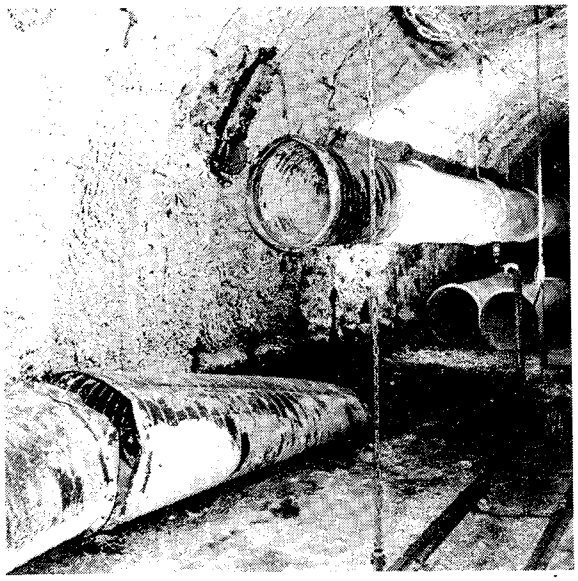


Figure 5. Sheet metal duct destroyed by the fire (test no. 6)

In this arrangement the auxiliary ventilation was discontinued in the course of the fire (fig. 5), as the sheet metal duct, like the plastic duct, was destroyed by the fire in the vehicle.

Table 4: Experimental conditions in vehicle fires in a roadway with auxiliary ventilation

VERSUCH-NR.	LUTTEN-MATERIAL	ENTFERNUNG (m) FAHRZEUG/ORT	VENTILATOR	LUTTEN-VERLEGUNG
1	Kunststoff	110	an	Stoss
4	Kunststoff	50	an	Stoss
5	Kunststoff	50	aus	Stoss
6	Blech	50	an	Stoss
11	Blech	50	an	Sohle

Key:

Versuch	= test
Luttenmaterial	= duct material
Entfernung	= distance
Fahrzeug/Ort	= vehicle/roadhead
Ventilator	= fan
Luttenverlegung	= position of duct
Kunststoff	= plastic
Blech	= sheet material
Stoss	= wall
Sohle	= floor
an	= on
aus	= off

Table 4 list the variable, experimental conditions for the 5 tests with analysable test results.

Temperatures

Experimental conditions: Intact auxiliary ventilation laying sheet metal air ducts on the floor

The fresh air flowing through the ducts past the site of the fire was heated so intensely that it was at a temperature of 174 °C when emerging from the duct (fig. 6). Flowing back through the roadway cross-section the air under the roof was further heated because of the reflux of the hot combustion gases, whilst the temperatures in the centre of the roadway and above the floor remained approximately the same.

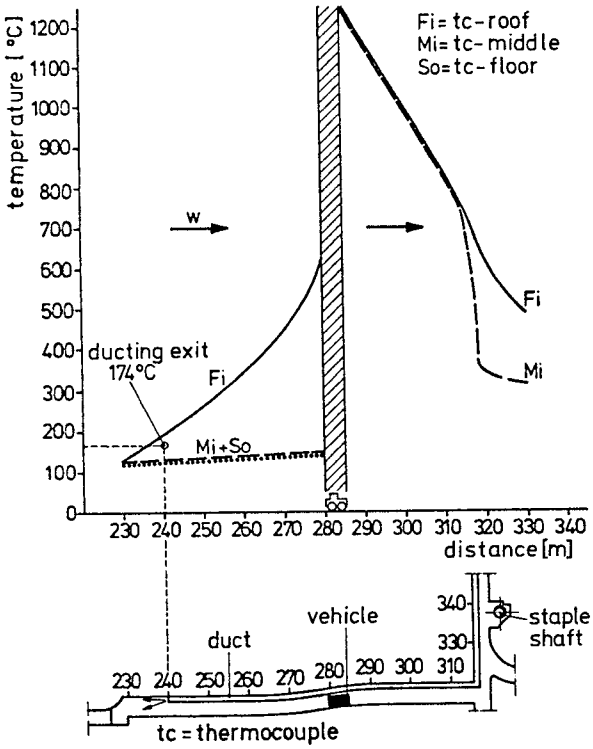


Figure 6. Maximum temperatures in a roadway with auxiliary ventilation as a function of the distance from the fire; sheet metal duct on the floor; auxiliary ventilation intact (test no. 11)

In the return air zone immediately behind the vehicle temperatures of 1200 °C occurred; at the 295 m mark

the temperature was approx. 1100 °C. No temperature stratification was detectable until 315 m. The disproportionate drop in temperature in the centre of the roadway - in the further diagrams also above the floor - can be explained by an uncontrollable air short-circuit, by means of which cool air travelled from the South and/or from the 2nd East lateral road.

Experimental conditions: Interrupted auxiliary ventilation, remains of plastic duct blow out air

The distance from the vehicle to the face was doubled to 110 m, while the part of the road with the small cross-section was included in the test zone (face at 190 m). Also, the sheet metal ducting was replaced by plastic ducting. In this case the destroyed duct blew out in the vicinity of the fire, i.e. at a distance of approx. 15 m. The hanger for the duct had become loose because of the heat; at this point the duct - otherwise functional - lay on the floor. The remaining part to sway and roll, a sort of pitching and tossing movement, so that the free jet emerged in various directions. Figure 7 shows the development of the maximum temperatures in front of and behind the fire. At the face, a temperature of 65 °C was measured.

The temperature patterns developing between 190 m and 330 m represent almost standard cooling curves, from which, for any position of the burning vehicle in this zone, the temperature at the roadhead can be obtained. At distances greater than 110 m the temperature was still changed only slightly. In the tests carried out in 1982-84 the temperature rose to similar values (table 5), although in that case slightly different cross-section conditions had been present and smaller quantities of fuel for the fire had been used.

The table shows that increase in the fuel for the fire in the vehicle had only a slight influence on the temperature at the face. The latter was

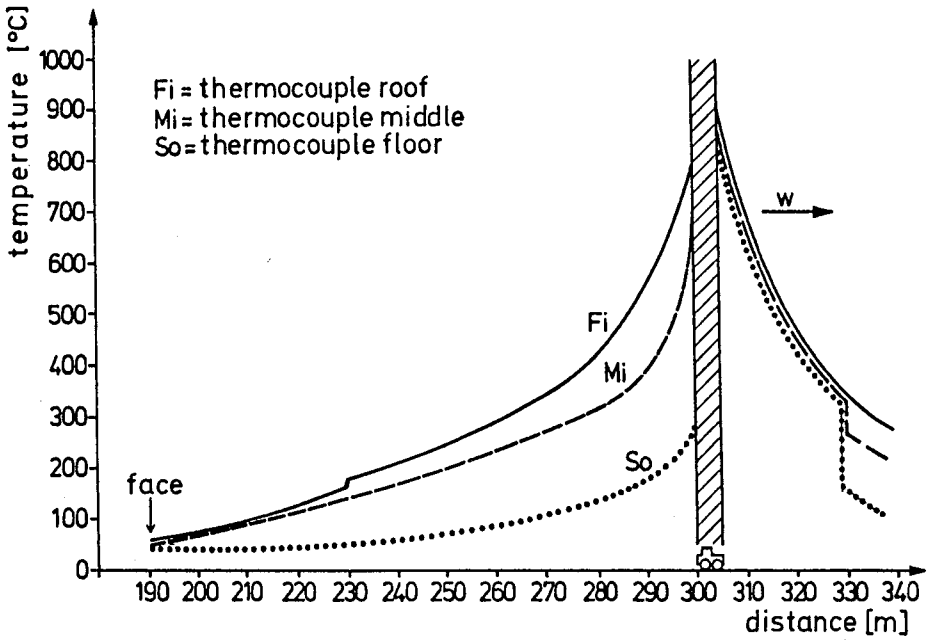


Figure 7. Maximum temperatures in a roadway with auxiliary ventilation as a function of the distance from the fire; residual duct blows air out in the vicinity of the fire (test no. 1)

therefore not so much dependent on the quantity of fuel for the fire, but was mainly affected by the distance from the fire.

Table 5: Temperature at the roadhead; 110 m from the burning vehicle; auxiliary ventilation collapsed; residual duct blows air out

GRUNDBRANDLAST 4 REIFEN = 320 kg	TEMPERATUR ORTSBRUST (°C)
200 l Dieselkraftstoff	60 (aus Versuchen von 1982-1984)
200 l Dieselkraftstoff + 200 l Mineraloel	62 (aus Versuchen von 1982-1984)
400 l Dieselkraftstoff + 200 l Mineraloel	65

Key:
Grundbrandlast = basic quantity of fuel for fire
Reifen = tyres
Temperatur Ortsbrust = temperature face
Dieselkraftstoff = diesel fuel
Mineraloel = mineral oil
aus Versuchen von 1982-84 = from tests carried out in 1982-84

Experimental conditions: Auxiliary ventilation interrupted; fan disconnected after the plastic duct had burnt trough

In spite of the fan being disconnected the fresh air from the main ventilation current reaches the fire by the following mechanism: The hot combustion gases flowed along below the roof because of the thermal buoyancy and passed over the staple shaft. During the combustion oxygen-rich air was consumed by the fire and replaced by the fresh air following after on the floor. The air movement was thus maintained by convection currents.

The fire in the vehicle was pulsating as it developed. After it had attained a certain size, it was deadened because of lack of oxygen. The result was that oxygen-containing air was able to advance so that the development pattern was repeated. Fig. 8 shows the resulting temperature development above the vehicle as a function of time.

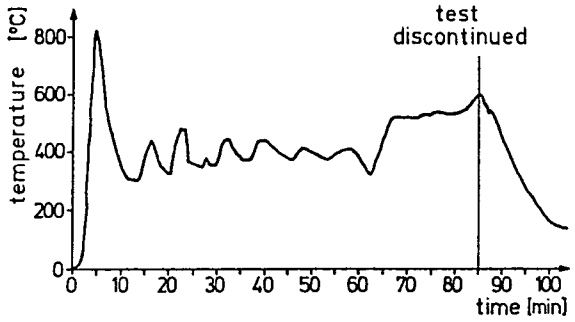


Figure 8. Temperature development above the vehicle as a function of time; auxiliary ventilation interrupted; residual duct does not blow air out
(test no. 5)

Figure 9 shows that on both sides of the fire somewhat a similar temperature stratification was present in the roadway.

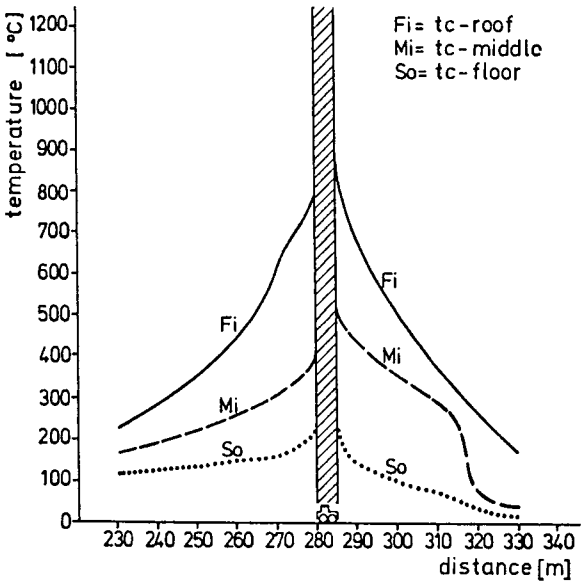


Figure 9. Maximum temperatures in a roadway with auxiliary ventilation as a function of the distance from the fire; residual duct does not blow out air (test no. 5)

Duration of fire

Table 6 represents the duration of a fire in a vehicle in a roadway with intact or interrupted auxiliary ventilation. No distinction is made as to whether the blowing-out of the residual duct is near the fire or at a distance from the fire, as the duration of the fire was somewhat similar in both cases.

Table 6: Duration of fire in a vehicle during the starting- and maximum phase, in a roadway with auxiliary ventilation

*) aus Versuchen von 1982 - 1984		GRUNDBRANDLAST: 320 kg REIFEN	
		200 l Diesel-kraftstoff	400 l Diesel-kraftstoff 200 l Mineraloel
Sonderbewetterung intakt		-	24 min (Versuch 11)
Sonderbe-wetterung unterbrochen	Restlutte blaest	23 min*)	50 min (Versuche 1, 4)
	Restlutte blaest nicht	-	>85 min (Versuch 5)

Key:

aus Versuchen von 1982-84 = from tests carried out in 1982-84
Sonderbewetterung intakt = auxiliary ventilation intact
Sonderbewetterung unterbrochen = auxiliary ventilation interrupted
Restlutte bläst = residual duct blows out
Restlutte bläst nicht = residual duct does not blow out
Grundbrandlast = basic quantity of fuel for fire
Reifen = tyres
Dieselkraftstoff = diesel fuel
Mineraloel = mineral oil
Versuche = tests

With intact auxiliary ventilation the fire persisted for 24 mins. and thus lasted as long as in the corresponding test 10 with continuous ventilation. With interrupted ventilation with the residual duct blowing out air, the fire, with approximately twice the amount of fuel, lasted approximately twice as long. It is also evident that tests with a blowing-out residual duct resulted in a shorter fire duration (50 mins.) than tests with a non-blowing residual duct when the fan was disconnected (>85 mins.).

Development of fire

The tests carried out in 1982-1984 led to the result that the temperature in the burning vehicle is not essentially increased by an increase in the amount of fuel used (cf. also table 5) but showed, however, that the maximum phase of the fire is extended. The time sequence of the temperatures measured and the mean times obtained at 5 measuring points in the vehicle is shown in fig. 10, where the fuel consisted of 320 kg tyres, 400 l diesel fuel and 200 l mineral oil.

In this case, maximum values can certainly reach 1200 °C over a period of several minutes, while minimum values can be less than the aforementioned 950 °C.

If, throughout the entire duration of the fire, i.e. including the "dying-out to 250 °C" phase, we assume a period of 90 mins., then the workers, as to the level of the air temperatures, would have a chance of survival, providing that this temperature did not exceed this 55 °C at the roadhead. This value would not be exceed only on condition that the fire occurred at a distance of more than 110 m away.

Composition of combustion gases

Table 7 lists the average concentration of the combustion gases in the case of intact and interrupted ventilation at the roadhead and in the return air zone.

It is evident that in the case of intact auxiliary ventilation no combustion gases get to the roadhead. On the other hand, high concentrations were present in the return air zone,

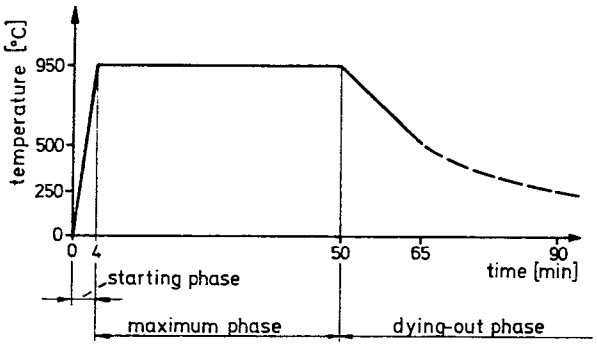


Figure 10. Model showing fire development in the vehicle as a function of the time (mean value from 5 measuring points on the vehicle; auxiliary ventilation interrupted; residual duct blows out air)

as the fire was adequately supplied with oxygen via the sheet metal duct.

Table 7: Combustion gas concentrations in the case of intact and interrupted auxiliary ventilation

Sonderbewetterung		Messposition	Brandgase (%)			
			CO	CO_2	O_2	HC
Bewetterung intakt Versuche 6/1, 11		Ort	0	0	21,0	0
		Abwetterbereich	0,3	>10,0	5,1	0
Bewetterung unterbrochen	Lutte blaest Versuche 1, 4, 6/2	Ort	0,6	9,1	8,6	0,1
		Abwetterbereich	0,2	4,4	14,2	0
	Lutte blaest nicht Versuch 5	Ort (Fi/Mi)	1,0	11,8	4,1	>0,1
		Abwetter (Firste)	0,3	4,0	15,0	0,02
		Abwetter (Mitte)	>0,1	0,6	20,0	n.b.

Key:
Sonderbewetterung = auxiliary ventilation
Bewetterung intakt = ventilation intact
Versuch = test
Bewetterung unterbrochen = ventilation interrupted
Lutte bläst = duct blows out
Lutte bläst nicht = duct does not blow out
Messposition = measuring point
Ort = face
Abwetterbereich = return air zone
Ort (Fi, Mi) = face (roof, middle)
Abwetter = return air
Brandgase = combustion gases

When the auxiliary ventilation system at the roadhead collapsed with the residual duct blowing out, fire gases flowed to the face, so that, during the maximum phase of the fire, there were high concentrations of CO and CO_2 and the oxygen content was correspondlingly reduced. In the return air zone on the other hand the combustion gases were diluted by the fresh air flowing in through the residual duct.

After the auxiliary ventilation at the roadhead, with a non-blowing residual duct, had been interrupted approximately similar combustion gas concentrations formed there at different levels. In the return air zone on the other hand there was considerable stratification, was fresh air flowed in, in the region of the floor.

FINAL CONSIDERATIONS

Continuous ventilation

In the case of small air flows of, for example, 0.7 m/s, the hot gases accumulate against the fresh air direction to about 60 m back, which means that any extinguishing operations are hindered by heat and poor visibility.

The temperature on the return air side, about 100 m behind the fire, amounts to 55 °C; here, there is a chance of survival as regards the temperature.

The total duration of the fire in the whole of a burning vehicle with 320 kg tyres and 600 l combustible liquids is about 35 mins. in the case of an air speed of 0.7 m/s. In the case of greater air speeds of up to 1.4 m/s, the fire lasts for a slightly shorter time.

The O_2 content in the combustion gases is so slight that CO filter self-rescue appliances offer a chance of survival with small amount of fuel and fairly high air speeds (for example, 200 l Diesel fuel, 1.4 m/s).

Auxiliary ventilation

It has to be stated that the replacement of plastic ducts by sheet metal ducts does not seem appropriate:

1. The sheet metal ducts, too, may be so seriously damaged by flame effects that the auxiliary ventilation is interrupted.
2. Even in the case where the sheet metal ducting laid on the floor remains functional, the fresh air in the duct, flowing past the fire becomes so heated that the workers are endangered by the high temperatures of the air, although it is adequately supplied with oxygen. Temperature measurements in the duct at the fire and at the ducting at a distance of 50 m from the fire gave values of 250 °C and 240 °C respectively. It has thus to be reckoned that even at greater distances of up to about 110 m from the burning vehicle the fresh air is at a temperature of 200 °C when it arrives at the roadhead.

When the auxiliary ventilation at the face is interrupted and when the residual duct blows air out at the fire, the workers at the face - in the matter of temperature - have a chance of survival only when the burning vehicle is at a distance of $\geqq$110 m.

The tests have also shown that it is not advisable to disconnect the fan when the auxiliary ventilation is interrupted because this greatly lengthens the duration of the fire, while the temperature and the oxygen content at the face assume similarly unfavourable, life-threatening values. The greater duration of the fire too, could cause the O_2 supply in the oxygen self-rescue appliance to be no longer sufficient. In every case the mine workers should not be allowed to remain for an unnecessary length of time in the fumes.

In the combustion gases, the poor visibility of a few decimetres might present a problem not only for the workers at the roadhead and face but also for the rescue teams.

NEUE ERKENNTNISSE AUS BRANDVERSUCHEN AN DIESELGETRIEBENEN GLEISLOSFAHRZEUGEN FÜR DEN BERGBAU UNTERTAGE

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ZUSAMMENFASSUNG

Die auf der Versuchsgrube Tremonia durchgeführten Versuche erfolgten sowohl in einer durchgehend als auch in einer sonderbewetterten Strecke, wobei höhenabhängige Standardabkühlungskurven erstellt werden konnten. Es wurden die Gesamtbranddauer, der Temperaturverlauf am Fahrzeug und die Brandgaszusammensetzung festgestellt. Weiterhin sollte geklärt werden, ob es sinnvoll ist, bei der Sonderbewetterung an Stelle von Kunststofflutten solche aus Blech zu verwenden, um die vor Ort befindliche Belegschaft möglichst lange mit Frischluft versorgen zu können. Schließlich wurde der Frage nachgegangen, ob es günstiger ist, den Ventilator für die Sonderbewetterung im Brandfalle abzuschalten oder weiterlaufen zu lassen.

EINLEITUNG

Mit zunehmendem Einsatz von Gleislosfahrzeugen im Steinkohlenbergbau erhöht sich auch die Brandgefahr. Um das Brandrisiko noch besser abschätzen zu können, sind die auf der Versuchsgrube Tremonia durchgeführten Untersuchungen, über die bereits auf der 21. Internationalen Konferenz in Sydney berichtet worden ist, fortgesetzt worden. Im Gegensatz zu den früheren Untersuchungen wurden jetzt größere Brandlasten eingesetzt, wobei sich das Fahrzeug sowohl in einer durchgehend als auch in einer sonderbewetterten Strecke befand. Wiederum wurde von dem ungünstigen Fall ausgegangen, daß sich ein örtlich eng begrenzter Entstehungsbrand innerhalb kürzester Zeit zu einem Vollbrand entwickelt, weil die bordfeste HRD-Löschanlage und andere sicherheitstechnische Einrichtungen am Fahrzeug nicht funktionieren.

VERSUCHSANORDNUNG

Der Standort des Fahrzeugs konnte entsprechend den Vorgaben variiert werden (Bild 1). Bei Versuchen mit durchgehender Wetterführung stand die Attrappe zwischen den Längenmarken 260 m und 265 m, bei Versuchen mit Sonderbewetterung zwischen den Längenmarken 300 m und 305 m oder 280 m und 285 m. Die künstliche Ortsbrust, die durch die Wettertüren 1 oder 2 simuliert wurde, befand sich entweder bei der Längenmarke 190 m oder bei 226 m. Durch die verschiedenen Standorte der Attrappe ergaben sich unterschiedliche Abstände zwischen Fahrzeug und Ortsbrust.

Die Brandstrecke 2 besaß zwischen dem 190. und 230. Streckenmeter einen Querschnitt von 7 m²; zwischen dem 230. und 320. Streckenmeter einen Querschnitt von 16 m².

VERSUCHE MIT DURCHGEHENDER BEWETTERUNG

Insgesamt wurden 4 Versuche durchgeführt, wobei zusätzlich zu den vier Reifen mit einem Gesamtgewicht von 320 kg zwei unterschiedlich große Brandlasten gewählt und zwei Wettergeschwindigkeiten berücksichtigt wurden. In Tabelle 1 sind die variablen Versuchsbedingungen aufgelistet.

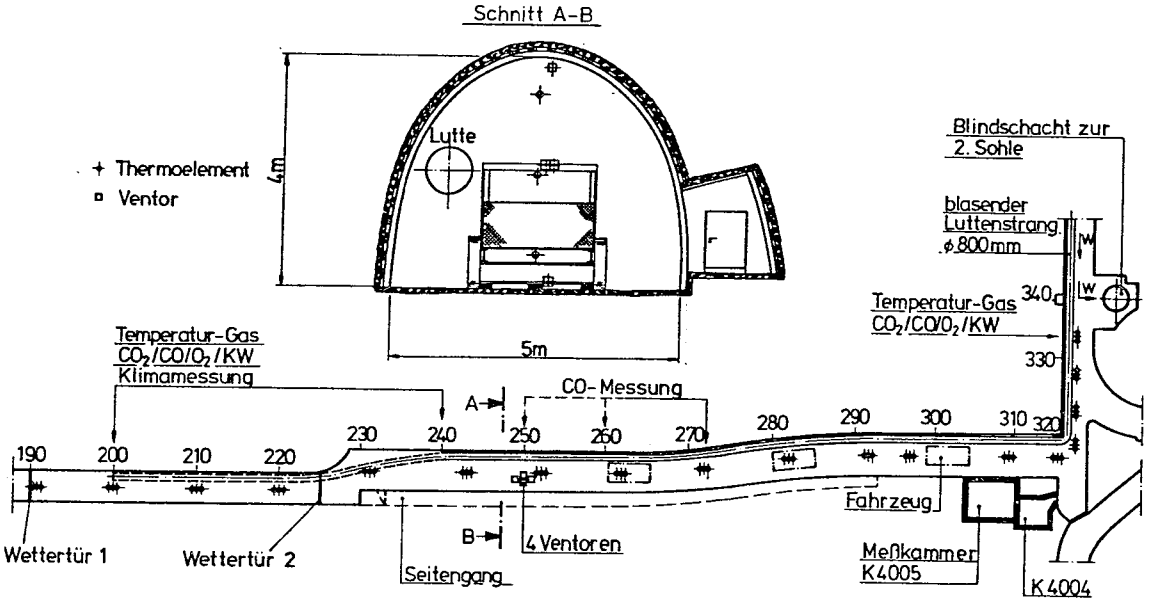


Bild 1. Versuchsanordnung mit unterschiedlichen Standorten der Fahrzeugattrappe in der Brandstrecke 2 der Versuchsgrube Tremonia

Tabelle 1: Versuchsbedingungen bei Fahrzeugbränden in einer durchgehend bewetterten Strecke

VERSUCH-NR.	DIESELKRAFTSTOFF (l)	ÖL (l)	w (m/s)	$\dot{V}$ (m³/min)
7	200	-	0,7	650
8	200	-	1,4	1300
9	400	200	1,4	1300
10	400	200	0,7	650

Temperaturen

Der Temperaturverlauf in der Strecke ist beispielhaft während der Versuche Nr. 10 und 9 in den Bildern 2 und 3 dargestellt.

Bei der größeren Wettergeschwindigkeit von 1,4 m/s (Bild 3) ist die Temperaturschichtung auf der Frischwetterseite in anderer Weise ausgeprägt. Der Temperaturverlauf an den Thermoelementen Sohle (So) und Mitte (Mi) ist in diesem Fall deckungsgleich, wobei 50 °C nicht überschritten werden. Das Firstenelement (Fi) zeigt jedoch einen ähnlichen Temperaturverlauf wie in Versuch 10; die maximale Temperatur beträgt hier etwa 650 °C.

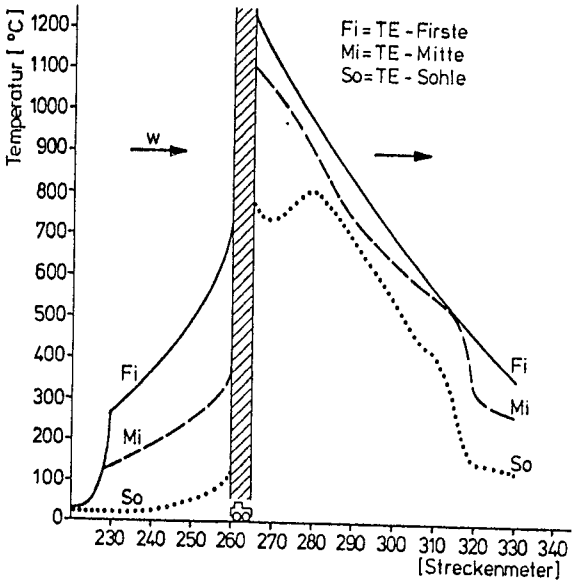


Bild 2: Maximale Temperaturen in einer durchgehend bewetterten Strecke in Abhängigkeit von der Entfernung vom Brand; w = 0,7 m/s, Brandlast 600 l Flüssigkeit plus 320 kg Reifen (Versuch Nr. 10)

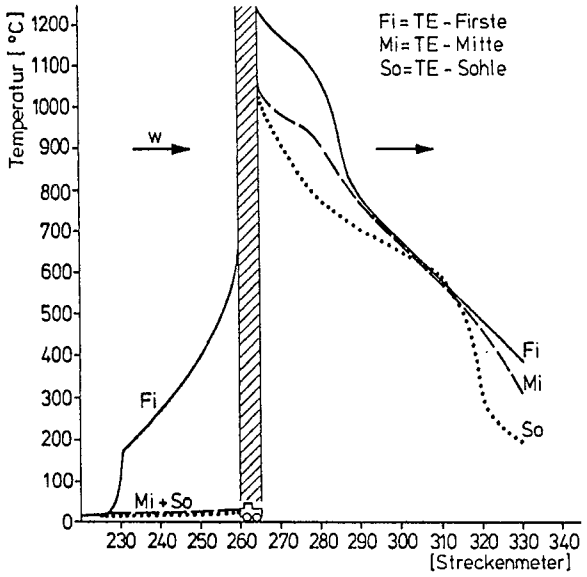


Bild 3: Maximale Temperaturen in einer durchgehend bewetterten Strecke in Abhängigkeit von der Entfernung vom Brand; w = 1,4 m/s, Brandlast 600 l Flüssigkeit plus 320 kg Reifen (Versuch Nr. 9)

Die Bilder 2 und 3 zeigen weiterhin, daß der Rückstau der Brandgase gegen die Frischwetter unter der Firste bis zur Querschnittsverengung bei 230 m reichte. Wäre diese Verengung nicht vorhanden, hätte sich der Rückstau unter der Firste weiter gegen die Frischwetterrichtung ausgebreitet.

Im Abwetterbereich unmittelbar hinter dem Fahrzeug traten mit etwa 1200 °C erheblich höhere Temperaturen als im Frischwetterbereich auf; diese Maximaltemperatur fiel 35 m hinter dem Fahrzeug auf etwa 600 °C. Werden die Temperaturmeßwerte sinnvoll extrapoliert, so ergibt sich für einen Abstand von 100 m hinter dem Fahrzeug im Abwetterbereich noch eine Temperatur von etwa 55 °C. Ab dieser Entfernung könnten Personen bei Verwendung eines Sauerstoffselbstretters überleben. Ein solcher Sauerstoff-Selbstretter ist jedoch erforderlich, weil der O_2-Gehalt in den Wettern so weit abfallen kann, daß der normale CO-Filtersebstretter keinen Schutz mehr bietet.

Branddauer

Ein Fahrzeugbrand kann im Idealfall in drei Abschnitte unterteilt werden, und zwar in die Anlauf-, die Maximal- und die Abklingphase (Bild 10). Im folgenden soll unter "Branddauer" die Anlauf- und die Maximalphase verstanden werden.

Tabelle 2: Branddauer eines Fahrzeugs in der Anlauf- und Maximalphase bei durchgehender Bewetterung

GRUND-BRANDLAST:		*320 kg REIFEN*	
200 l Dieselkraftstoff		*400 l Dieselkraftstoff 200 l Mineraloel*	
w = 0,7 m/s (Versuch 7)	*w = 1,4 m/s (Versuch 8)*	*w = 0,7 m/s (Versuch 10)*	*w = 1,4 m/s (Versuch 9)*
20 min	*15 min*	*25 min*	*20 min*

Bei den Versuchen mit durchgehender Wetterführung wurde bei gleicher Wettergeschwindigkeit die Branddauer mit steigender Brandlast wegen des größeren Brennstoffangebotes größer. Allerdings stieg die Branddauer nicht direkt proportional zur Brennstoffmenge an, da die brennende Oberfläche zum Beispiel durch ausfließendes Mineralöl im Fahrzeug und auf der Sohle ebenfalls größer wurde. Bei Vergrößerung der Wettergeschwindigkeit und gleichbleibender Brandlast dagegen wurde die Branddauer verkürzt, was mit dem schnelleren Abtransport der Verbrennungsprodukte und der größeren Sauerstoffzufuhr erklärt werden kann.

Rechnet man zu den in der Tabelle 2 genannten Zeiten weitere 10 min als Abklingphase hinzu, so beträgt die Gesamtbranddauer in diesen Versuchen etwa 35 min.

Brandgaszusammensetzung

In Tabelle 3 wurden die Konzentrationen von CO, CO_2, O_2 und Gesamtkohlenwasserstoffen (KW) im Abwetterbereich des Brandes aufgeli-

Tabelle 3: Extremwerte der Brandgaskonzentrationen im Abwetterbereich von Fahrzeugbränden bei durchgehender Wetterführung

BRANDGAS (%)	GRUND-BRANDLAST: 320 kg REIFEN			
	200 l DIESELKRAFTSTOFF		400 l DIESELKRAFTSTOFF 200 l MINERALOEL	
	w=0,7m/s (Versuch 7)	w=1,4m/s (Versuch 8)	w=0,7m/s (Versuch 10)	w=1,4m/s (Versuch 9)
CO	0,1	0,1	0,4	0,3
CO2	7,5	2,5	10,0	8,1
O2	10,2	17,1	9,7	10,5
KW	<0,1	<0,1	0,1	<0,1

stet. Dabei wurden aus Gründen der besseren Übersicht nur die unter der Firste gemessenen Werte eingetragen. In der Streckenmitte lagen die Werte etwas tiefer, da die heißen Brandgase wegen des thermischen Auftriebs unter die Firste ziehen und Frischwetter die Brandgase im Sohlenbereich verdünnen. Man erkennt, daß mit steigender Brandlast auch die Gaskonzentrationen (mit Ausnahme von O_2) steigen, was unmittelbar einleuchtet. Bezüglich der Wettergeschwindigkeit liegen die Verhältnisse genau umgekehrt, da ein größerer Wetterstrom die Brandgase verdünnt.

VERSUCHE MIT SONDERBEWETTERUNG

Der Wetterstrom betrug bei diesen Versuchen etwa 10,8 m³/s, was einer Wettergeschwindigkeit von 0,7 m/s im freien Streckenquerschnitt entsprach. Als Brandlast dienten zusätzlich zu den 4 Reifen stets 400 l Dieselkraftstoff und 200 l Mineralöl. Die Sonderbewetterung des Ortes erfolgte durch einen Kunststoff- oder Blechluttenstrang.

Das Bild 4 zeigt beispielhaft die Versuchsanordnung mit einer am Stoß verlegten Blechlutte (s. Tabelle 4, Versuch Nr. 6).

Bei dieser Anordnung wurde die Sonderbewetterung während des Brandes unterbrochen (Bild 5), da die Blechlutte, ähnlich wie die Kunststofflutte, durch den Fahrzeugbrand zerstört wurde.

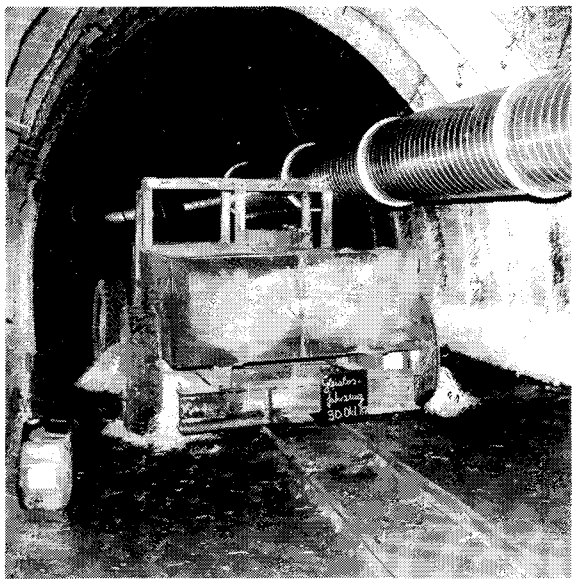


Bild 4. Sonderbewetterung mit einer am Stoß verlegten Blechlutte (Versuch 6)

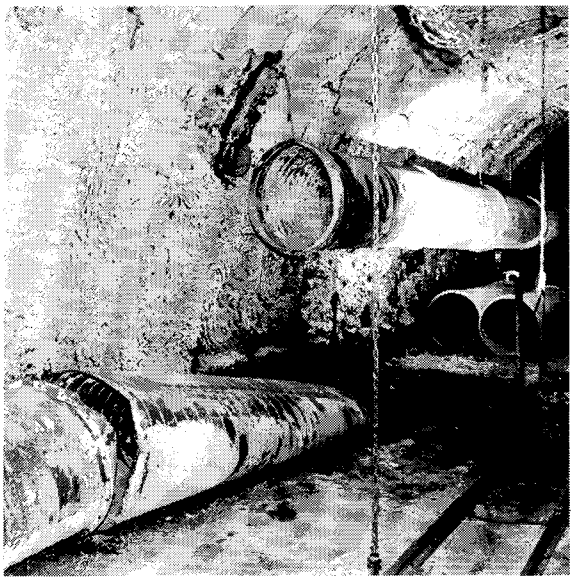


Bild 5. Durch den Brand zerstörte Blechlutte (Versuch 6)

In Tabelle 4 sind die variablen Versuchsbedingungen der 5 Versuche mit auswertbaren Versuchsergebnissen zusammengestellt.

Tabelle 4: Versuchsbedingungen bei Fahrzeugbränden in einer sonderbewetterten Strecke

VERSUCH-NR.	LUTTEN-MATERIAL	ENTFERNUNG (m) FAHRZEUG/ORT	VENTILATOR	LUTTEN-VERLEGUNG
1	Kunststoff	110	an	Stoss
4	Kunststoff	50	an	Stoss
5	Kunststoff	50	aus	Stoss
6	Blech	50	an	Stoss
11	Blech	50	an	Sohle

Temperaturen

Versuchsbedingungen: Intakte Sonderbewetterung bei Verlegung von Blechlutten auf der Sohle

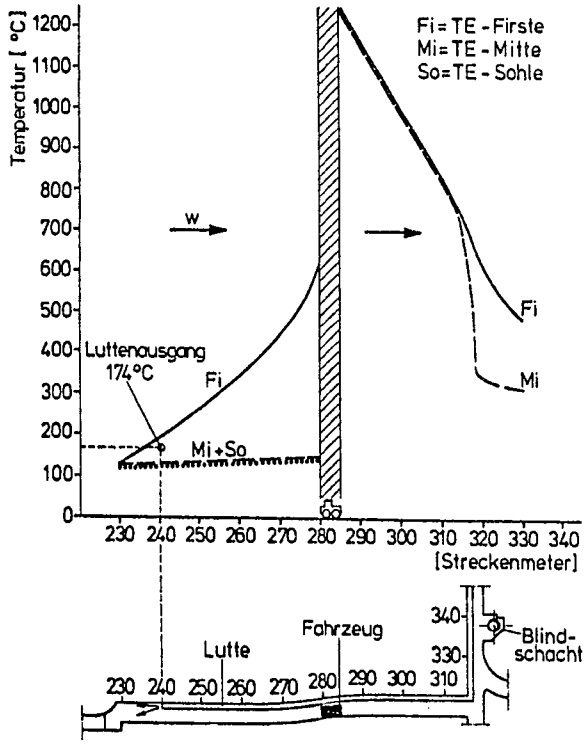


Bild 6. Maximale Temperaturen in einer sonderbewetterten Strecke in Abhängigkeit von der Entfernung vom Brand; Blechlutte auf der Sohle; Sonderbewetterung intakt; Versuch 11

Die durch die Lutten am Brandherd vorbeiströmenden Frischwetter wurden so erhitzt, daß sie mit 174 °C aus der Lutte austraten (Bild 6). Beim Zurückströmen durch den Streckenquerschnitt wurden die Wetter unter der Firste auf Grund des Rückstaus der heißen Brandgase weiter erhitzt, während die Temperaturen in der Streckenmitte und über der Sohle etwa gleich blieben. Im Abwetterbereich traten unmittelbar hinter dem Fahrzeug Temperaturen um 1200 °C auf, bei 295 m etwa 1100 °C. Eine Temperaturschichtung war bis 315 m nicht feststellbar. Der überproportionale Abfall der Temperatur in der Streckenmitte - in den weiteren Bildern auch über der Sohle - kann durch einen nicht beherrschbaren Wetterkurzschluß erklärt werden, durch den kühle Wetter aus dem Süden und/oder aus der 2. östlichen Richtstrecke auf der 4. Sohle strömten.

Versuchsbedingungen: Unterbrochene Sonderbewetterung, Rest der Kunststofflutte bläst aus

Die Entfernung vom Fahrzeug zum Ort wurde um das Doppelte auf 110 m erhöht, indem der Streckenteil mit dem kleinen Querschnitt in den Versuchsbereich einbezogen wurde (Ortsbrust bei 190 m). Außerdem wurde der Blechluttenstrang durch einen solchen aus Kunststoff ersetzt. In diesem Fall blies die zerstörte Lutte in der Nähe des Brandes aus, das heißt, in einer Entfernung von etwa 15 m. Die Aufhängeleiste der Lutte hatte sich durch Wärmeeinwirkung gelöst; die Lutte lag an dieser Stelle - ansonsten funktionsfähig - auf der Sohle. Durch die austretenden Frischwetter geriet das Reststück in eine "schlingernde" Bewegung, so daß der Freistrahl in unterschiedliche Richtungen austrat. Bild 7 zeigt den Verlauf der maximalen Temperaturen vor und hinter dem Brand. Vor Ort wurden etwa 65 °C gemessen.

Die Temperaturverläufe zwischen 190 m und 330 m stellen weitgehend angenäherte Standardabkühlungskurven dar, aus denen für jeden beliebigen Standort des brennenden Fahrzeugs in diesem Bereich die Temperatur an der Ortsbrust abgelesen werden kann. In größeren Entfernungen als 110 m änderte sich die Temperatur nur noch unwesentlich. Auch bei den 1982 -

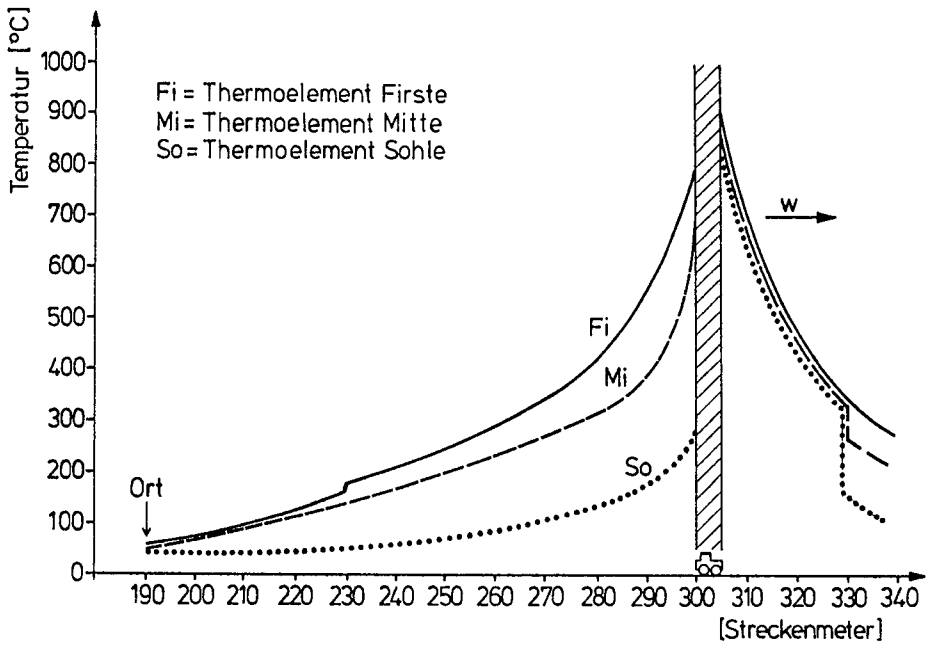


Bild 7. Maximale Temperaturen in einer sonderbewetterten Strecke in Abhängigkeit von der Entfernung vom Brand; Restlutte bläst in Brandnähe aus, Versuch 1

1984 durchgeführten Versuchen stieg die Temperatur auf ähnliche Werte (Tabelle 5), obwohl seinerzeit geringfügig andere Querschnittsverhältnisse vorlagen und geringere Brandlasten eingesetzt worden waren.

Tabelle 5: Temperaturen an der Ortsbrust; 110 m vom brennenden Fahrzeug entfernt; Sonderbewetterung zusammengebrochen; Restlutte bläst aus

GRUNDBRANDLAST 4 REIFEN = 320 kg	*TEMPERATUR ORTSBRUST (°C)*
200 l Dieselkraftstoff	*60 (aus Versuchen von 1982-1984)*
200 l Dieselkraftstoff + 200 l Mineraloel	*62 (aus Versuchen von 1982-1984)*
400 l Dieselkraftstoff + 200 l Mineraloel	*65*

Aus der Tabelle ersieht man, daß die Erhöhung der Brandlast im Fahrzeug die Temperatur an der Ortsbrust nur wenig beeinflußte. Diese war daher nicht so sehr abhängig von der Größe der Brandlast oder der Branddauer, sondern wurde hauptsächlich von der Entfernung zum Brand beeinflußt.

Versuchsbedingungen: Unterbrochene Sonderbewetterung; Ventilator nach dem Durchbrennen der Kunststofflutte abgeschaltet

Trotz abgeschalteten Ventilators erreichten die Frischwetter aus dem Hauptwetterstrom den Brand über folgenden Mechanismus:

Die heißen Brandgase strömten unter der Firste durch den thermischen Auftrieb über den Blindschacht ab. Beim Verbrennen wurden sauerstoffreiche Wetter vom Brand verbraucht, die durch auf der Sohle nachströmende frische Wetter ersetzt wurden. Die Wetterbewegung wurde also durch Konvektionsströme aufrecht erhalten.

Der Brand am Fahrzeug zeigte einen pulsierenden Verlauf. Nachdem er eine bestimmte Größe erreicht hatte, wurde er aufgrund von Sauerstoffmangel gedrosselt. Als Folge davon konnten sauerstoffhaltige Wetter nachströmen, so daß sich der Ablauf wiederholte. Bild 8 zeigt den daraus resultierenden Temperaturverlauf über dem Fahrzeug in Abhängigkeit von der Zeit.

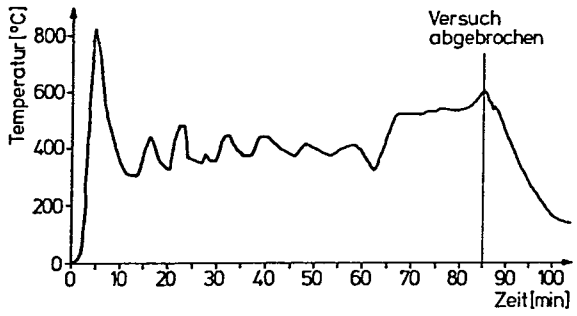


Bild 8: Temperaturverlauf über dem der Zeit; Sonderbewetterung unterbrochen; Restlutte bläst nicht aus; Versuch 5

Bild 9 zeigt, daß auf beiden Seiten des Brandes etwa gleichartige Temperaturschichtungen in der Strecke vorhanden waren.

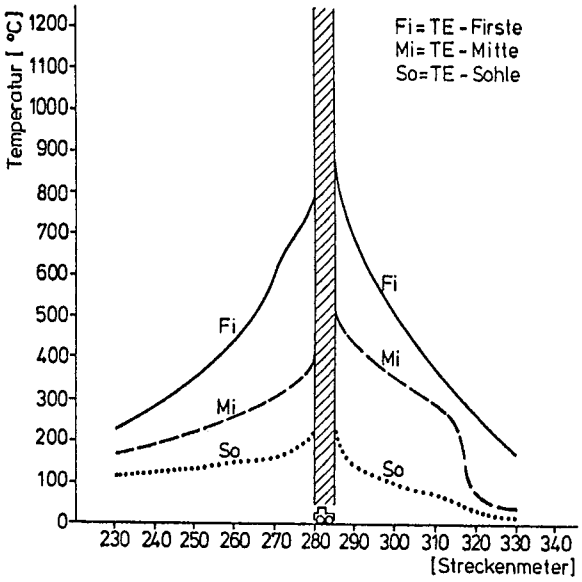


Bild 9: Maximale Temperaturen in einer sonderbewetterten Strecke in Abhängigkeit von der Entfernung vom Brand; Restlutte bläst nicht aus; Versuch 5

Branddauer

In der Tabelle 6 ist die Branddauer eines Fahrzeugs in einer Strecke mit intakter oder unterbrochener Sonderbewetterung dargestellt. Es wird nicht zwischen brandnahem und -fernem Ausblasen der Restlutte unterschieden, da die Branddauer in beiden Fällen etwa gleich war.

Die Branddauer bei intakter Sonderbewetterung betrug 24 min und war damit ebenso lang wie die beim entsprechenden Versuch 10 mit durchgehender Bewetterung. Bei unterbrochener Wetterführung mit blasender Restlutte war die Branddauer bei Versuchen mit der etwa 2-fachen Brandlast etwa doppelt so lang. Weiterhin fällt auf, daß Versuche mit ausblasender Restlutte eine kürzere Branddauer (50 min) aufweisen, als solche mit nicht blasender Restlutte bei stillgesetztem Ventilator (>85 min).

Tabelle 6: Branddauer eines Fahrzeugs während der Anlauf- und Maximalphase in einer sonderbewetterten Strecke

*) aus Versuchen von 1982 - 1984		GRUNDBRANDLAST: 320 kg REIFEN	
		200 l Diesel-kraftstoff	400 l Diesel-kraftstoff 200 l Mineraloel
Sonderbewetterung intakt		-	24 min (Versuch 11)
Sonderbe-wetterung unterbrochen	Restlutte blaest	23 min*)	50 min (Versuche 1, 4)
	Restlutte blaest nicht	-	>85 min (Versuch 5)

Brandverlauf

Die in den Jahren 1982 - 1984 durchgeführten Versuche führten zu dem Ergebnis, daß die Temperatur am brennenden Fahrzeug durch Vergrößerung der Brennstoffmenge nicht we-

sentlich gesteigert wird (vergleiche auch Tabelle 5), daß dadurch die Maximalphase des Brandes jedoch verlängert wird. Der zeitliche Verlauf der an 5 Meßstellen am Fahrzeug erfaßten und gemittelten Temperaturen geht aus Bild 10 hervor, wobei die Brandlast aus 320 kg Reifen, 400 l Dieselkraftstoff und 200 l Mineralöl bestand.

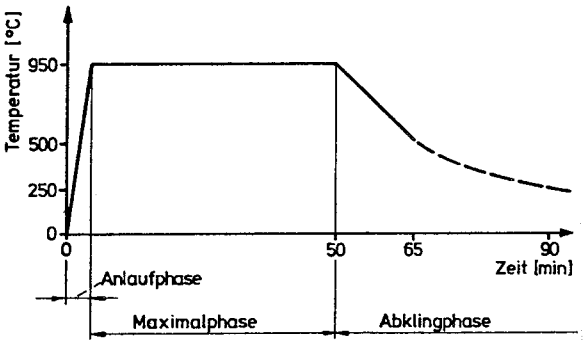


Bild 10: Modellhafter Brandverlauf am Fahrzeug in Abhängigkeit von der Zeit (Mittelwert aus 5 Meßstellen am Fahrzeug; unterbrochene Sonderbewetterung; Restlutte bläst aus)

Maximalwerte können dabei durchaus über einen Zeitraum von mehreren Minuten 1200 °C erreichen, während Minimalwerte die genannten 950 °C unterschreiten können.

Nimmt man für die Gesamtdauer des Brandes, das heißt, einschließlich der Abklingphase bis 250 °C eine Zeit von 90 min an, dann hätte die Ortsbelegschaft hinsichtlich der Höhe der Wetter-Temperaturen eine Überlebenschance, wenn diese Temperatur vor Ort 55 °C nicht überschreitet. Dieser Wert wird jedoch nur dann nicht überschritten, wenn sich der Brand in einer größeren Entfernung als 110 m ereignet.

Brandgaszusammensetzung

In Tabelle 7 wurden die durchschnittlichen Brandgaskonzentrationen bei intakter und unterbrochener Sonderbewetterung vor Ort und im Abwetterbereich aufgelistet.

Tabelle 7: Brandgaskonzentrationen bei intakter und unterbrochener Sonderbebewetterung

Sonderbewetterung		Messposition	Brandgase (%)			
			CO	CO_2	O_2	KW
Bewetterung intakt Versuche 6/1, 11		Ort	0	0	21,0	0
		Abwetterbereich	0,3	>10,0	5,1	0
Bewetterung unterbrochen	Lutte blaest Versuche 1, 4, 6/2	Ort	0,6	9,1	8,6	0,1
		Abwetterbereich	0,2	4,4	14,2	0
	Lutte blaest nicht Versuch 5	Ort (Fi/Mi)	1,0	11,8	4,1	>0,1
		Abwetterer (Firste)	0,3	4,0	15,0	0,02
		Abwetter (Mitte)	>0,1	0,6	20,0	n.b.

Man sieht, daß im Falle der intakten Sonderbewetterung keine Brandgase zur Ortsbrust gelangten. Dafür lagen im Abwetterbereich hohe Konzentrationen vor, da der Brand über die Blechlutte ausreichend mit Sauerstoff versorgt wurde. Brach die Sonderbewetterung des Ortes bei blasender Restlutte zusammen, strömten Brandgase zur Ortsbrust, so daß dort während der Maximalphase des Brandes hohe Konzentrationen an CO und CO_2 herrschten und der Sauerstoffgehalt entsprechend absank. Im Abwetterbereich wurden die Brandgase dagegen durch die über die Restlutte zuströmenden Frischwetter verdünnt.

Nachdem die Sonderbewetterung des Ortes bei nicht blasender Restlutte unterbrochen war, bildeten sich dort in unterschiedlichen Höhen annähernd gleiche Brandgaskonzentrationen aus. Im Abwetterbereich lag dagegen eine starke Schichtung vor, da Frischwetter im Sohlenbereich zuströmten.

SCHLUSSBETRACHTUNG

Durchgehende Bewetterung

Bei geringen Wetterströmen von zum Beispiel 0,7 m/s stauen sich die heißen Brandgase gegen die Frischwetterrichtung etwa 60 m zurück, was

eine Behinderung eventueller Löscharbeiten durch Hitze und schlechte Sicht bedeutet.
Die Temperatur auf der Abwetterseite beträgt etwa 100 m hinter dem Brand 55 °C; hier besteht bezüglich der Temperatur eine Überlebenschance.
Die Gesamtbranddauer eines vollständig brennenden Fahrzeugs mit 320 kg Reifen und 600 l brennbaren Flüssigkeiten beträgt etwa 35 min bei einer Wettergeschwindigkeit von 0,7 m/s. Bei höheren Wettergeschwindigkeiten bis 1,4 m/s ist die Branddauer geringfügig kürzer.
Der O_2-Gehalt in den Brandgasen ist so gering, daß CO-Filterselbstretter nur bei geringen Brandlasten und größeren Wettergeschwindigkeiten (zum Beispiel 200 l Dieselkraftstoff, 1,4 m/s) eine Überlebenschance bieten.

Sonderbewetterung

Es ist festzustellen, daß der Ersatz von Kunststofflutten durch Blechlutten aus zwei Gründen nicht sinnvoll erscheint: 1. Auch die Blechlutten können durch Flammeneinwirkung so stark beschädigt werden, daß die Sonderbewetterung unterbrochen wird. 2. Für den Fall, daß der auf der Sohle verlegte Blechluttenstrang funktionstüchtig bleibt, erhitzen sich jedoch die Frischwetter in der Lutte beim Vorbeiströmen am Brand so stark, daß die Belegschaft durch die hohen Temperaturen der genügend sauerstoffhaltigen Wetter gefährdet ist. Temperaturmessungen in der Lutte am Brand und am Luttenaustrag in 50 m Entfernung vom Brand ergaben Werte von 250 °C beziehungsweise 240 °C. Es ist damit zu rechnen, daß auch in größeren Entfernungen bis etwa 110 m vom brennenden Fahrzeug die Frischwetter mit etwa 200 °C vor Ort ankommen.

Bei unterbrochener Sonderbewetterung des Ortes und am Brand ausblasender Restlutte hat die Ortsbelegschaft - von der Temperatur her gesehen - nur dann eine Überlebenschance, wenn das Fahrzeug in einer Entfernung von $\geq$110 m brennt.

Die Versuche haben weiter gezeigt, daß es nicht ratsam ist, den Ventilator bei unterbrochener Sonderbewetterung abzuschalten, da sich dadurch die Branddauer erheblich verlängert, während die Temperatur und der Sauerstoffgehalt vor Ort aber etwa gleichartig ungünstige, lebensbedrohende Werte annehmen. Die längere Branddauer könnte zudem bewirken, daß der O_2-Vorrat der Sauerstoffselbstretter dann nicht ausreicht. In jedem Fall sollte vermieden werden, daß sich die Bergleute unnötig lange in den Brandschwaden aufhalten.

Nicht nur für die Belegschaft vor Ort, sondern auch für die Rettungsmannschaften dürften die geringen Sichtweiten von wenigen Dezimetern in den Brandgasen ein Problem darstellen.

NOUVEAUX RESULTATS D'ESSAIS D'INCENDIE SUR DES VEHICULES DIESEL A LIBRE DIRECTION DESTINES A L'EXPLOITATION MINIERE PROFONDE

Heyn, W. W., Foit

Les essais d'incendies de véhicules, déjà présentés lors de la 21ème Conférence Internationale des Instituts d'Essais sur la Sécurité Minière, se sont poursuivis dans une voie ventilée par un système d'appoint et dans une voie continuellement ventilée sous une charge de feu plus importante.

Dans le cas de la voie sous ventilation continue, les gaz d'in-cendie se sont refroidis à tel point qu'à une distance de 100 m, des personnes munies d'appareils d'auto-sauvetage à oxygène pouvaient y survivre. La durée totale de l'incendie a été évaluée à 35 mn.

Dans le cas de la voie ventilée par le système d'appoint, on peut établir des courbes standard de refroidissement variant selon la hauteur; ces courbes permettent de prendre la température in situ de chaque emplacement de véhicule entre le front de taille de l'avancée et une distance de 110 m. A partir de 110 m, les températures ne constituaient plus un danger de mort, mais des appareils d'auto-sauvetage à oxygène auraient été nécessaires. La durée totale de l'incendie d'un conduit de plastique résiduel détruit à proximité du véhicule était de 90 mn environ. Si on éteignait le ventilateur, la durée totale de l'incendie se prolongeait jusqu'à 125 mn, avec une température in situ et une composition gazeuse aussi peu favorables. Des conduits de plaques métalliques ont également été détruits au cours de l'incendie. S'ils demeuraient intacts, l'air frais contenu dans les conduits s'échauffait beaucoup, atteignant une température de sortie in situ de 200°C environ.

НОВЫЕ ДАННЫЕ ИЗ РЕЗУЛЬТАТОВ ПОЖАРНЫХ ИСПЫТАНИЙ ДИЗЕЛЬНЫХ ТРАНСПОРТНЫХ СРЕДСТВ ДЛЯ ШАХТ ГЛУБОКОГО ЗАЛЕГАНИЯ

В. Хейн и В. Фойт

Данные о пожарах транспортного оборудования были представлены на 21-ой международной конференции по технике безопасности в шахтах. Испытания в этой области были продолжены в условиях постоянно вентилируемых штреков и штреков со вспомогательной вентиляционной системой применительно к более сильным пожарам. В условиях постоянной вентиляции шахтеры с переносными кислородными приборами могут выжить, т.к. воздух достаточно охлаждается на расстоянии 100 м от очага пожара. Пожар в эксперименте продолжался 35 минут. Для штреков со вспомогательной вентиляционной системой кривые охлаждения могут быть определены для любого места нахождения оборудования в пределах 110 м от передового забоя. За пределами 110 м температуры не являются опасными для жизни, однако кислородные дыхательные аппараты все же необходимы. Пожар, продолжавшийся приблизительно 90 минут, уничтожил работавший недалеко от оборудования пластмассовый воздуховод. При выключенной вентиляционной системе продолжительность пожара увеличивалась до 125 минут, при этом температура и состав газов почти не отличались; металлические воздуховоды были также разрушены пожаром. В случаях, когда воздуховоды не разрушались, температура воздуха на выходе достигала 200 С.

深井矿工业自由转向柴油车着火试验新研究结果

W. 海恩，及 W. 福埃特

车辆着火曾在第２１届矿业安全研究所国际会议中提出，现在继续对这问题在辅助通风和连续通风的行车道迁有更大的火灾情矿之下作进一步的研究．

在连续通风情况下，火灾气体在１００米处以外会冷却下来，使带有氧气自救护器的人员可以不受其害．整个火灾过程历时３５分钟．

辅助通风的行车道上我们可以从开采工作面到１１０米距离之间测定很多点就地的温度来作出与高度有关的冷却曲线．１１０米以外则无生命危险，但氧气自救护器仍必须带着．如果被焚毁的残余塑料通风管在车辆附近，则火灾整个过程约９０分钟．如果通风扇关掉，则在同样的不利的就地温度和火灾气体成份情况之下，火灾过程会延长到１２５分钟．薄钢板通风管也被火烧毁．如果没有烧毁则通风管里的新鲜空气会强烈烧热，可达到约２００ C 的出口温度．

RELATIONSHIPS BETWEEN SMOKE AND CARBON MONOXIDE AND THEIR IMPLICATION TOWARD IMPROVED MINE FIRE DETECTION

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ABSTRACT

Tests were conducted to determine the relative levels of carbon monoxide (CO) and smoke produced from smoldering and flaming mine combustibles, including wood, coal, and conveyor belting. The data indicated that smoldering combustibles generate a very low level of CO compared to the level of smoke produced. Even though during flaming, the level of CO relative to that of smoke increases, it is found that a "typical" alarm level for smoke is comparable to a CO level of about 6 ppm. For smoldering the equivalent CO level is in the range of 2 ppm. The implications of these findings in regard to early warning mine fire detection are discussed. The overall results clearly indicate that developing mine fires can be detected earlier using smoke sensors rather than CO sensors.

INTRODUCTION

Mine fires pose a severe hazard to underground personnel. If not detected rapidly and reliably in their initial stages of development, fires can increase in intensity eventually reaching levels sufficient for flame propagation. Further, as the fire intensity increases, so do the levels of smoke and CO. These smoke levels may reach critical levels where smoke obscuration can reduce, if not eliminate, a miner's chance for escape. Similarly, CO levels can increase to dangerous levels thus further reducing chances for successful evacuation.

The sources of fuel available for fires can vary significantly from one location to the next in an underground mine. Fires of liquid or gaseous fuels develop very rapidly and those areas of mines where such fuels exist should be protected by automatic detection systems coupled with automatic suppression capabilities. Fires of solid combustibles usually take longer to develop, and for these types of fires, a well-planned distribution of fire sensors throughout the entries to be protected can provide for warning in sufficient time to safely evacuate and often extinguish the fire locally before it has a chance to spread.

For these types of fires, the solids involved may be wood, coal, conveyor belting, electrical cables, and others, or combinations of these solids. Further, these types of fires may undergo prolonged periods of smoldering prior to the actual development of flames. To detect these fires, CO fire sensors are currently used with alarm thresholds of 10 or 15 ppm of CO above ambient. However, the sensitivity of CO fire sensors to overheating conveyor belts is inadequate in many instances due to the formation of low levels of CO, which when diluted by the ventilation airflow are not high enough to produce sensor alarm. Similar types of occurrences could be expected for other combustible solids.

It is observed, however, that these incidents produce high levels of smoke. As a result, if smoke detectors were used, these types of occurrences would most always be detected. The following paragraphs discuss the formation of CO and

smoke and provide insight as to how these two types of fire sensors can most efficiently be utilized for fire detection in underground mines.

SMOKE AND CO FORMATION

In general, smoldering fires produce both smoke and CO. However, the mechanisms responsible for their production are quite different. CO is produced via a chemical reaction by which the carbon in the fuel combines with oxygen in the surrounding environment. Further, the rate at which the CO is produced depends upon the concentrations of carbon in the fuel (% of carbon, usually by weight) and the temperature of the fuel. This rate may also depend upon how the carbon is bound chemically to other molecules within the fuel. From coal and wood, CO can be formed at relatively low temperatures ~150-200° C (even lower for spontaneous heatings). For synthetic materials, such as conveyor belts, the temperatures to produce significant levels of CO are higher due both to the carbon content of belting and to the ways carbon is chemically bound.

Smoke particles, on the other hand, are produced by means of thermal processes whereby the fuel surface heats and expels both solid particulate matter into the environment as well as high molecular weight gases which combine to form smoke particles. Most, if not all combustible solids will produce smoke particles at temperatures in the range of 100° to 200° C. And as the solid surface temperature rises, the rate of emission of smoke particles also increases.

During flaming combustion, the high flame temperatures are sufficient to promote the formation of CO. Also, during flaming, the CO production rate will increase as the oxygen to fuel ratio decreases. This is because the formation of CO_2 (carbon dioxide) diminishes due to reduced oxygen availability and the fuel carbon is mostly used to form CO.

During flaming at high oxygen to fuel ratios (typical of fires that are over ventilated), the formation of CO is directly related to the fire intensity. Further, these rates of formation at high oxygen to fuel ratios represent minimum values. As the fire intensity increases and the oxygen to fuel ratio decreases, the rates of formation of CO per unit fire intensity also increase.

Smoke in a flame is produced primarily as soot particles which recombine and grow in particle size as they move away from the flame. Solid flames usually burn at or near a limit flame temperature (~1500° K) and the yellowish color is due to the burning soot particles as they emit radiant energy. Just as for CO, as the oxygen to fuel ratio decreases, the rates of formation of smoke will increase. And, the rates of formation of smoke at high oxygen to fuel ratios represent approximate minimum rates.

There do exist differences, then, in the ways in which smoke and CO are formed both in smoldering and flaming fires. It should also be noted that the rates of formation divided by the ventilation airflow rate in the area in which either is formed is a dilution effect which also serves to reduce the smoke and CO levels. This effect must always be taken into account when discussing detectable levels of either smoke or CO.

The units for measurement of CO common to all CO fire sensors are parts per million (ppm) of CO. 1 ppm of CO equals 2.69×10^{19} molecules of CO per cubic meter of air, or 1.25 milligram of CO per cubic meter of air. Smoke may be measured in terms of the number of smoke particles per unit volume of air or the mass of smoke particles per unit volume of air. However, for smoke fire sensors, the units of measurement are usually in terms of the optical density (OD) of the smoke. The optical density is a

measure of the ability of the smoke to obscure visibility. The greater the optical density, the greater the obscuration. Optical density is a function of the number, or mass, of smoke particles and the average diameter of the smoke particles.

EXPERIMENTAL

To determine the CO and smoke levels that are produced from smoldering and flaming mine combustibles, experiments were conducted in an intermediate-scale fire tunnel. The tunnel, described in detail in previous reports (Egan, 1986, Egan, 1987), is 10 meters long with a square cross-section of 0.64 m^2. It is completely lined with fire brick and a fan at the exhaust end of the tunnel provides for ventilation air velocities of 0.20 to 2.0 m/sec.

With the exception of coal, all flaming fire tests were conducted using a natural gas burner igniter mounted on the tunnel floor. Once flaming had been achieved, the burner was turned off and the combustible sample allowed to burn. For coal, ignition was achieved using three electrical strip heaters buried in a pile of coal. For all the flaming tests, the data reported below are for steady-state burning of the combustibles.

For all the smoldering fire tests, the combustibles were heated to a point of steady-state smoldering using electrical strip heaters. All data reported below are for the steady-state smoldering. For conveyor belting, smoldering tests were conducted at strip heater surface temperatures of 275° C, 450° C, and 600° C. For all other combustibles, smoldering tests were conducted at a standard heater surface temperature of 450° C only.

Gases and smoke were extracted from the tunnel at a point ~8.0 m downstream of the fire where the combustion products are completely mixed with the ventilation airflow. The exception to this procedure is the measurement of smoke obscuration which is made with a three wavelength probe (Cashdollar, 1979) mounted within the tunnel approximately 0.15 m upstream of the gas and smoke sampling probes. Smoke obscuration is measured at three wavelengths, 0.45 μm, 0.63 μm, and 1.0 μm. For the data reported here, the smoke optical densities, expressed in units of m^{-1} are the averages of the values measured at 0.45 μm and 0.63 μm.

All CO measurements were made using an eCOlyzer Model 2100 CO Analyzer with dual ranges of 0 to 100 and 0 to 500 ppm. This analyzer has an accuracy of ±1% of the full-scale reading. Smoke mass concentrations were made using a Tapered Element Oscillating Microbalance (TEOM) (Patashnick, 1986) with an accuracy of ±5% of the full-scale reading, which was typically set at 200 mg/m^3.

Air velocities were measured using a bi-directional flow probe (McCaffrey, 1976) with an accuracy of ±7% of the indicated reading.

All gas, smoke, and velocity data were recorded using a PDP 11/44 computer and subsequently transferred to files of a larger VAX 11/780 for reduction and analysis.

TEST RESULTS

Simultaneous measurements of ppm of CO, smoke mass concentration, M_o, and smoke optical density, OD, allow for the data to be expressed in terms of ratios. Three ratios are of particular interest:

1) The first ratio, denoted by R1, is the ratio of smoke optical density, OD, in units of m^{-1}, to smoke mass concentration, M_o, in units of mg/m^3. The ratio R1 is a measure of the ability of the smoke to obscure visibility per unit mass concentration. The larger the value of R1, the more efficient is the smoke in reducing visibility. The

smaller the value of R1, the less efficient is the smoke in reducing visibility.

2) The second ratio, R2, is the ratio of smoke optical density to ppm of CO. This ratio is important because it provides a measure of the relative effectiveness of smoke and CO fire sensors with fixed alarm thresholds.

3) The third ratio, R3, is the ratio of smoke mass concentration to the mass concentration of CO, both quantities being expressed in units of mg/m^3. This ratio is important because it represents a measure of the relative levels of smoke and CO produced from different combustibles and during different stages of combustion (flaming or smoldering).

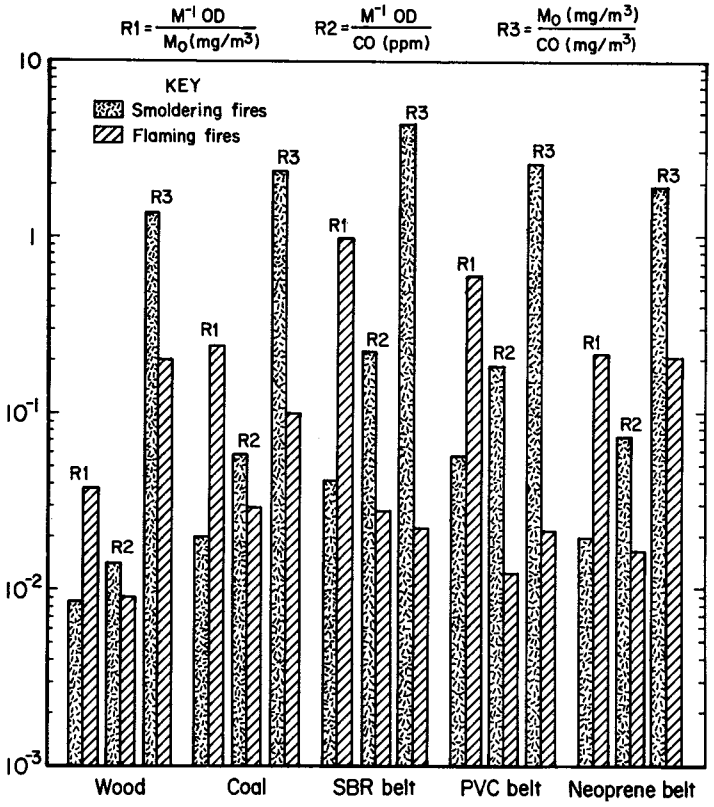


Figure 1. - Values of the three defined ratios for combustibles during both smoldering and flaming.

The average data obtained during these tests for materials which underwent flaming as well as smoldering combustion are shown in Figure 1. In Figure 1, the open vertical bars represent the average data for smoldering while the solid vertical bars represent the average data for flaming. Several aspects of these data that are noteworthy include:

1) All of the combustibles tested have higher R1 values during flaming than during smoldering, indicating that smoke from flaming fires is more efficient in obscuring visibility.

2) All of the combustibles show lower R2 values during flaming than during smoldering. This is reasonable, since at the higher temperatures reached during flaming more CO should be formed.

3) All combustibles tested yielded much higher R3 values during smoldering than during flaming. Again, this is reasonable since at lower temperature smoldering, less CO is produced.

In addition to these data, data from flaming fires of transformer fluids and data from smoldering fires of PVC line brattice were also obtained. For flaming transformer fluid fires, the ratio values were:

R1 = 0.27
R2 = 0.067
R3 = 0.20

For smoldering PVC brattice tests, the ratio values were:

R1 = 0.020
R2 = 0.046
R3 = 1.84

RELEVANCE TO FIRE DETECTION

The data presented in the previous section allow for the estimations of:

1) The expected average smoke optical density at a CO fire sensor alarm threshold of 15 ppm and the resultant obscuration level; and

2) The expected average CO level at a smoke fire sensor alarm threshold of 0.10 m^{-1} OD.

Table 1 lists the % obscurations over a 3.0 m path that would exist at the 15 ppm alarm threshold for CO fire sensors.These comparisons indicate that, on the average, at CO alarm levels, the smoke obscurations can be significant, even for flaming fires. For smoldering fires, these smoke levels reduce visibility to a very low level.

Table 2, below list the CO levels that would exist at typical smoke sensor alarm thresholds of 0.10 m^{-1} OD.

Table 2 is a clear indication of the superiority of smoke fire sensors to CO fire sensors. Both for flaming and smoldering fires, smoke fire sensors will, on the average, alarm before the CO reaches the 15 ppm threshold.

Table 3, below lists the average mass concentration of both smoke and CO (both in mg/m^3) at the smoke sensor alarm threshold of 0.10 m^{-1} OD.

This table indicates that during flaming the smoke mass concentrations at alarm are much, much lower than the CO mass concentrations. For smoldering, the smoke mass concentrations are higher than the CO mass concentrations by an average factor of ~2.5. However, during smoldering much less CO is produced.

Table 1: Relative values of smoke obscuration over a 3 meter path at the CO alarm level of 15 ppm

Fuel	% Obscuration over a 3 m path	
	Flaming	Smoldering
Wood	33	47
Coal	73	92
SBR belt	70	100
PVC belt	42	100
Neoprene belt	51	96
Transformer fluid	95	-
PVC brattice	-	87

Table 2: Relative values of CO at a smoke sensor alarm level of 0.10 m^{-1} OD

Fuel	ppm CO Flaming	ppm CO Smoldering
Wood	11.1	7.1
Coal	3.4	1.8
SBR belt	3.7	0.5
PVC belt	8.3	0.6
Neoprene belt	6.3	1.4
Transformer fluid	1.5	-
PVC brattice	-	2.2

Table 3: Mass concentration (mg/m^3) of both smoke and CO at a smoke sensor alarm level of 0.10 m^{-1} OD

Fuel	Flame		Smolder	
	Smoke Mass	CO Mass	Smoke Mass	CO Mass
Wood	2.7	13.9	11.9	8.9
Coal	0.42	4.3	5.0	2.2
SBR belt	0.10	4.6	2.4	0.57
PVC belt	0.17	10.4	1.8	0.69
Neoprene belt	0.48	7.8	5.3	1.7
Transformer fluid	0.37	1.9	-	-
PVC brattice	-	-	5.0	2.7

DISCUSSION

During smoldering combustion, typical mine combustibles produce about 2.5 times more smoke than CO. As fires develop to their flaming stages, the trend reverses and about 8 times as much CO is produced as smoke. However, the resultant optical densities per unit mass concentration of smoke increases from an average value of 0.027 during smoldering to an average value of 0.38 during flaming. The data indicate that smoke produced from flaming fires is much more efficient in attenuating light and reducing visibility. Further, much lower levels of smoke are required to produce optical densities equal to 0.10 m^{-1} OD alarm thresholds. The results of these experiments is straight forward. Typical alarm levels for smoke occur at equivalent CO levels significantly less than CO alarm levels. This means that a typical smoke detector would detect a fire earlier than a typical CO sensor. This is particularly true for smoldering fires. Consequently, the use of smoke sensors in underground mines has the potential to improve fire detection capabilities and increase the level of fire safety.

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RAPPORTS ENTRE LA FUMEE ET LE CO - LEUR PORTEE SUR L'AMELIORATION DE LA DETECTION DES INCENDIES MINIERS

C.D. Litton

Des essais ont eté effectués pour déterminer les niveaux relatifs de CO et de fumée produits par des combustibles miniers incandescents ou en flammes. Les données obtenues indiquent que de nombreux types de convoyeurs à courroies produisent une quantité très faible de CO comparée à la quantité de fumée produite. Ces résultats rejoignent ceux de rapports concernant des courroies incandescentes qui ne produisent pas suffisamment de CO pour déclencher les systèmes de détection d'incendies. Au cours des incendies, le niveau de CO augmente par rapport à celui de la fumée, mais un niveau d'alarme "typique" de fumée est comparable à un niveau de CO de moins de 5 ppm. Pour des combustibles incandescents, le niveau équivalent de CO est de 1 ppm environ. Ces résultats ont d'importantes répercussions sur les systèmes de détection d'incendies à l'avance et notre rapport les examine en détail.

ZUSAMMENHANG ZWISCHEN RAUCH UND CO UND IHRE ROLLE IN DER VERBESSERUNG DER GRUBENBRANDENTDECKUNG.

C. D. Litton

Untersuchungen zur Bestimmung der relativen Menge von CO und Rauch, die in Schwelbränden und offenen Bränden von brennbaren Materialien in Gruben entstehen, wurden durchgeführt. Die Ergebnisse zeigen, daß viele Sorten von Fördergurten einen geringen Anteil von CO, verglichen mit dem erzeugten Rauch, erzeugen. Diese Erkenntnisse stimmen mit Berichten überein, nach denen schwelende Gurte nicht genügend CO produzierten, um das Feueralarmsystem zu aktivieren. Während des offenen Brandes erhöht sich der Anteil des CO relativ zum Rauch, doch ergab sich, daß eine typische 'Alarmstufe' für Rauch einem CO Anteil von weniger als 5 ppm vergleichbar ist. Bei Schwelbränden würde der entsprechende CO Gehalt um etwa 1 ppm oder darunter liegen. Diese Erkenntnisse haben wichtige Auswirkungen auf die Brandfrüherkennung und werden im Vortrag weiter behandelt.

ИСПОЛЬЗОВАНИЕ СООТНОШЕНИЯ ДЫМА И ОКИСИ УГЛЕРОДА (CO) ДЛЯ УСОВЕРШЕНСТВОВАНИЯ СПОСОБОВ ОБНАРУЖЕНИЯ ПОЖАРА В ШАХТАХ

К.Д. Литтон

Испытания были направлены на определение относительных уровней CO и дыма, появляющихся при тлении и воспламенении горючих элементов в шахте. Данные, полученные при испытаниях, показывают, что многие типы транспортерных лент порождают очень малое, по сравнению с дымом, количество углерода. Эти выводы согласуются с более ранними результатами, когда в случаях тления транспортерных лент выделялось столь несущественное количество окиси углерода, что не срабатывала противопожарная система. В процессе горения уровень CO по отношению к дыму возрастает, но было установлено, что "типичный" опасный уровень дыма может быть приравнен к уровню CO 5 г/т. В процессе тления опасным уровнем считается 1 г/т CO или меньше. Эти результаты могут найти важное применение в разработке системы своевременного предупреждения пожара. Они будут более подробно рассмотрены в самом докладе.

烟和一氧化碳之间的关系及它们对矿火测定改进的关连

C．D． 列顿

我们举行了测定从矿里可燃物熏烧和燃烧所产生的一氧化碳和烟的相对含量的试验．试验的结果指出很多种输送机皮带产生的一氧化碳还较烟为低．这种现象与皮带熏烧时所产生的一氧化碳往往不足以驱动测火器报警的实际情形相符合．燃烧时一氧化碳的含量较烟增加得快．烟的"典型"报警程度相当于百分之五以下的一氧化碳含量．熏烧时一氧化碳含量在百分之一以下．这些数据对于及早测火警告系统有很重要的关系．在本文中将有详细的讨论．

Session II

CONVEYOR FIRES

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RESEARCH ON THE TEMPERATURE RISE OF MINE CONVEYOR BELTS DUE TO FRICTION AND THE REGULARITY OF GAS GENERATION DURING BELT FIRES

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SYNOPSIS

Although the fire-resistant belt conveyors have been widely used, the belt fires due to friction frequently occurred. In the study of this paper, three kinds of mine conveyor belts made in Japan and one kind of mine conveyor belt made in China were used as the test samples to determine their thermal properties and to examine the temperature rise and gas generation during friction tests of these belts over the driving drum. In the tests, the melt and breakdown were occurred in case of three kinds of Japenese belts, the temperature of belt melt and breakdown varied from 208°C to 332.5°C and a large amount of gases were produced. The Chinese belt has strong resistance to friction and the temperature in tests was only up to 135°C. On the basis of test results, this paper gives the predicting indications and prevention measures for mine conveyor belt fires.

I. INTRODUCTION

The belt conveyors have been widely used as a transporting tool in underground mines and open-pit mines. With the deepening of mining and excavation and the increase of productive capacities, the heavy and high speed belt conveyors have been developed. Although a variety of fire control measures have been taken by many countries, the burning, melt, breakdown and fire incidents of belts still occorred frequently. As many conveying belts were quentially set up in mine intake roadways and once a belt fire happened, the fire would quickly spread, the high temperature air stream and fume containing noxious and harmful

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gases would flow along roadways, resulting in extremely serious incidents. In 1950, a disastrous belt fire in Creswell Colliery in U. K. killed 80 men(1). In January 1984, a serious belt fire in Miike Colliery in Japan caused 83 deaths and 16 injuries (2). In December 1984, there was a belt fire in Wilberg Colliery in U. S. A. when 27 men lost their lives(3). In China's Coal mines, serious belt fires incidents also occurred.

Therefore, many researchers in major coal producing countries began to research on the causes of belt fires and their preventive measures from 60's. It has been found from the rerearch results that the natural rubber and synthetic rubber catch fire very easily. For this reason, the fire-resistant conveying belts have been developed and widely used. Despite all this, the belt fires still occurred occasionally. According to the statistical data from European Community, the belt fire incidents account for about 10 to 15 per cent of total mine fire incidents(1, 3). So it is of great significance to conduct further research into the ignition posibility of fire-resistant belts.

It has been well documented that the major ignition cause of rubber belts is the slipping friction of the belt over the driving drum(4, 5), i.e. the high speed friction that produced betweem the belt and the driving drum when the belt does not run synchronously with the driving drum due to overload or it is in stopping state.

In the study of this paper, three kinds of fire-resistant belts made in Japan and one kind of fire-resistant belt made in China were used as test samples. The friction tests of the driving drum with a belt that was in stopping state were carried out in the process of the measurement of basic heat characteristics of the belt. At the same time, investigations into the temperature rise and the properties of gases produced during friction test were made by means of changing the tension of the belt and the rotation speed of the driving drum.

II. BELT SAMPLES AND THEIR BASIC HEAT CHARACTERISTICS

The properties of belt samples are lised in Table 1. Before the starting of test, the basic heat characteristics of a belt were examined firstly, mainly including the thermogravimatric analysis and the measure-

Table 1 The properties of belt samples

Sample	Type	Number of canvas plies	Width (mm)	Thickness (mm)				Weight (kg/m^2)	Remarks
				Total	Top rubber ply	Down rubber ply	Canvas ply		
A	fire-resistant	5	900	14	5	5	4	15.9	made in Japan
B	fire-resistant	5	900	15	5	5	5	16.3	made in Japan
C	fire-resistant	5	900	13	4	4	5	15.6	made in Japan
D	fire-resistant	8	805	15	3	2	10	20.7	made in China

ment of calorific capacity.

1. Thermogravimatric Analysis

In the thermogravimatric analysis, the change of the temperature of an object to be measured was controlled by the preset temperature program and at the same time the change of the weight of this object was measured, which was regarded as the function of the change of temperature. The apparatus used for this analysis is TGA-41H Thermogravimatric Analyser made by Shimadzu Corporation, Japan.

In order to measure the content of organic matter and the content of non-combustible matter at relatively low temperature, the measurements were carried out first in nitrogen (the first stage) and then in air (the second stage) on the basis of the following test conditions.

In the first stage, the flow rate of nitrogen was 50 ml/min., the object was heated from the atmospheric temperature to 600°C at the rate of 30°C/min, then retaining this temperature for 30 minutes. (the thermolysis of organic matter contained in the rubber)

In the second stage, the flow rate of air was 50 ml/min and the temperature rised from 600°C to 800°C at the rate of 20°C/min, then retaining this temperature for 30 minutes. (the combustion of carbon black)

The test results obtained are summarized in Table 2

Furthermore, in order to examine the pyrolytic characteristics of en object to be measured in air, It was pyrolized in the low temperature stage according to the following test conditions.

The flow rate of air was 50 ml/min, and the object was

Table 2 The results of thermogravimatic analysis

Sample	Content of organic matter (1st stage)	Decrement of combustible matter (2nd stage)	Content of non-combustible matter
A	65(210-575°C)	30	5
B	65(220-560°C)	29	6
C	64(210-580°C)	28	8
D	61(190-600°C)	28	11

heated from the atmospheric temparature to 800°C at the rate of 30°C/min, then retaining this temperature for 30 minutes.

Fig. 1 shows the thermogravimatric curves of four belt samples measured in air. Under above test conditions, four belt samples were quickly pyrolyzed at the temperature of 200 to 240°C, but the slope of G - T curve of belt D was more gentle than that of others.

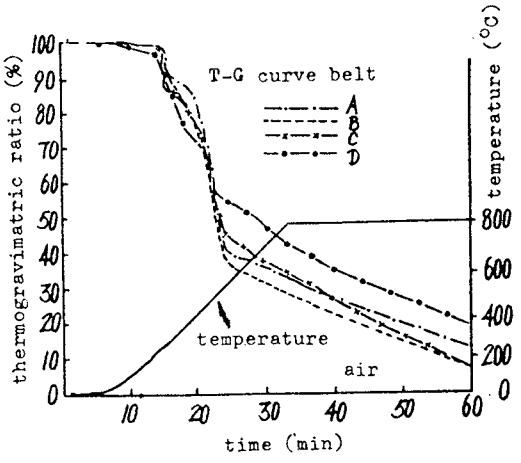


Fig. 1 G-T curves of four belts measured in air

The calculated activation energies from the results of thermogravimatric analysis are listed in Table 3. It can be seen from this Table that there is a considerable difference between belt D and belt A, B and C.

Table 3 The calculated results of activation energies

Sample	A	B	C	D
Activation energy (kcal/mole)	15.6	16.5	15.8	11.2

2. Calorific Capacity

The calorific capacity is also an important parameter for evaluating combustible matter. The measurement was conducted with CA-4 Automatoc Calorimeter made by Shimadzu Corporation, Japan. Table 4 gives the measured results. The calorific capacities for belt A, B and C are about 8000 Kcal/kg and that for belt D is 6290 Kcal/kg, which is much lower than that of belt A, B and C.

Table 4 The results of calorific capacities

Sample	A	B	C	D
Calorific capacity (kcal/kg)	8130	7990	8360	6290

III. FRICTION TESTS OF BELTS

The friction tests were

carried out by rubbibg a fixed belt over a driving drum. The test device shown in Fig. 2 consists of a variable-speed motor, a driving drum and a temperature measuring and recording apparatus. The steel drum is 234 mm in diameter and 400 mm long. The temperature measuring system comprises a CA thermocouple, an A/D converter and a computer.

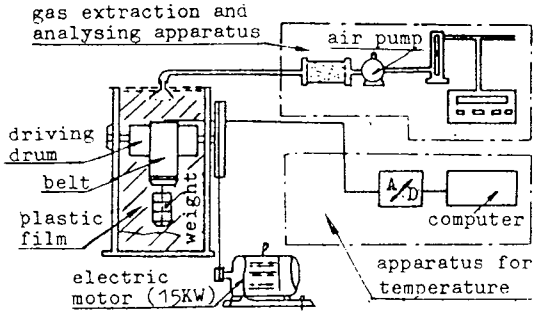


Fig. 2 The device for friction tests of belts

The gases generated during friction test were analysed with Riken GX-110B harmful gas analyser made in Japan, and CO, O_2, H_2S and hydrocarbon gases were continuously detected. For the measurement of H_2S and CO gases, the potentiometric electrolysis technique was applied and for the measurement of hydrocarbon gases, the contact combustion technique was applied.

1. Test Method

The belt piece for tests was 900 mm long, 150 mm wide (134 mm wide for belt D) and with original thickness. The length direction was consistent with the advancing direction of the belt. Fig. 3 shows the arrangement of the belt piece and thermocouples in the measurement of temperature. At the middle part of the belt piece, five small holes were drilled, each was apart from 30 mm and 10 mm deep. The thermocouples with diameter of 1.6 mm were inserted in these holes. Another one thermocouple was set up on the surface (middle part) for measuring the surface temperature. The wrap angle of belt and the driving drum was 180°. One end of the belt was fixed and to the other end weights were applied.

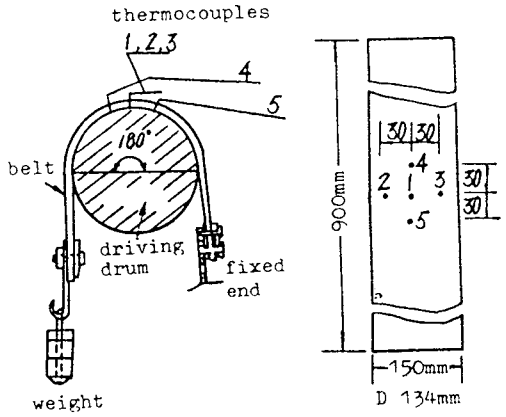


Fig. 3 The arrangement of the belt piece and thermocouples

The upper part of the test device was covered with plastic film so as to extract the gases generated during friction test. The gases were sucked out by a gas pump at flow rate of 26

L/min. In the whole process of friction test, the change of belt shape and the fume generation were observed while the measurements of gases and temperatures were made.

The running velocity of conveyor belts used underground was generally 2 to 5 m/sec. So the peripheral velocities varied in three steps of 2, 3 and 5 m/sec and the weights that applied to the belt varied in 5, 10 and 15 kg during tests. When the weight was 20 kg, the friction ceased because the motor could not be started.

2. Test Results

(1). The influence of peripheral velocity of the driving drum

Tests were conducted with 15 kg weight by changing the peripheral velocities of the driving drum. The results obtained are shown in Table 5. Belt A, B and C melted and broke in 6 to 18 minutes after the starting of friction and at this time their maximum temperatures were 208 - 276.5°C, 291 - 332.5°C and 256 - 308°C respectively. When the peripheral velocity increased, the time of belt melt and breakdown became shorter and the maximum temperature became higher. But when the peripheral velocity of belt D increased to 5 m/sec, on the contrary, the maximum temperature reduced.

When the friction started in one minute, thick fume presented; when temperature reached 150°C,

Table 5 The results of friction tests (I)

Sample	Peripheral velocity (m/sec)	Time of belt breaking	Maximum temperature (°C)	Maximum rate of temperature rise (°C/min)	The time and temperature at which CO began to be detected		Maximum gas concentration		
					time	temperature (°C)	CmHn (%)	H_2S (ppm)	CO (ppm)
A	2	16'50"	276.5	19.5	5'00"	76-110	-	77	108
	3	12'00"	269.6	25.5	3'00"	104-150	-	188	200
	5	7'05"	208.0	45.0	2'40"	79-95	-	25	123
B	2	17'40"	291.2	18.5	5'40"	65-105	0.15	124	235
	3	14'35"	320.3	23.9	2'20"	85-143	0.15	119	114
	5	8'50"	332.5	32.7	2'00"	86-152	0.10	54	110
C	2	14'00"	256.3	18.0	5'40"	94-177	-	51	75
	3	7'40"	282.0	33.1	2'40"	67-126	-	141	114
	5	5'20"	308.0	55.4	1'20"	77-107	-	55	55
D	2	-	135.3	5.7	3'20"	42-67	-	-	3
	3	-	126.4	6.2	10'20"	92-110	-	-	4
	5	-	132.4	4.5	33'00"	109-128	-	-	2

the fume generation became more intense, simutaneously it could be seen that the belt began to soften and to be stripped. After that the friction extended into the depth of the belt, a lot of belt fragments flew apart in all directions. CO and H_2S gases were detected in 2 to 5 minutes after the starting of friction.

Fig. 4 (a), (b) and (c) show the variations of temperatures and concentrations of gases produced by belt A, B and C with time (It was counted from the beginning of friction, similarly hereinafter) at the peripheral velocity of 3 m/sec and with a 15 kg weight. With the rise in temperature, the concentrations of produced gases reached their peak values. When the temperature of belt A was up to 150°C, a

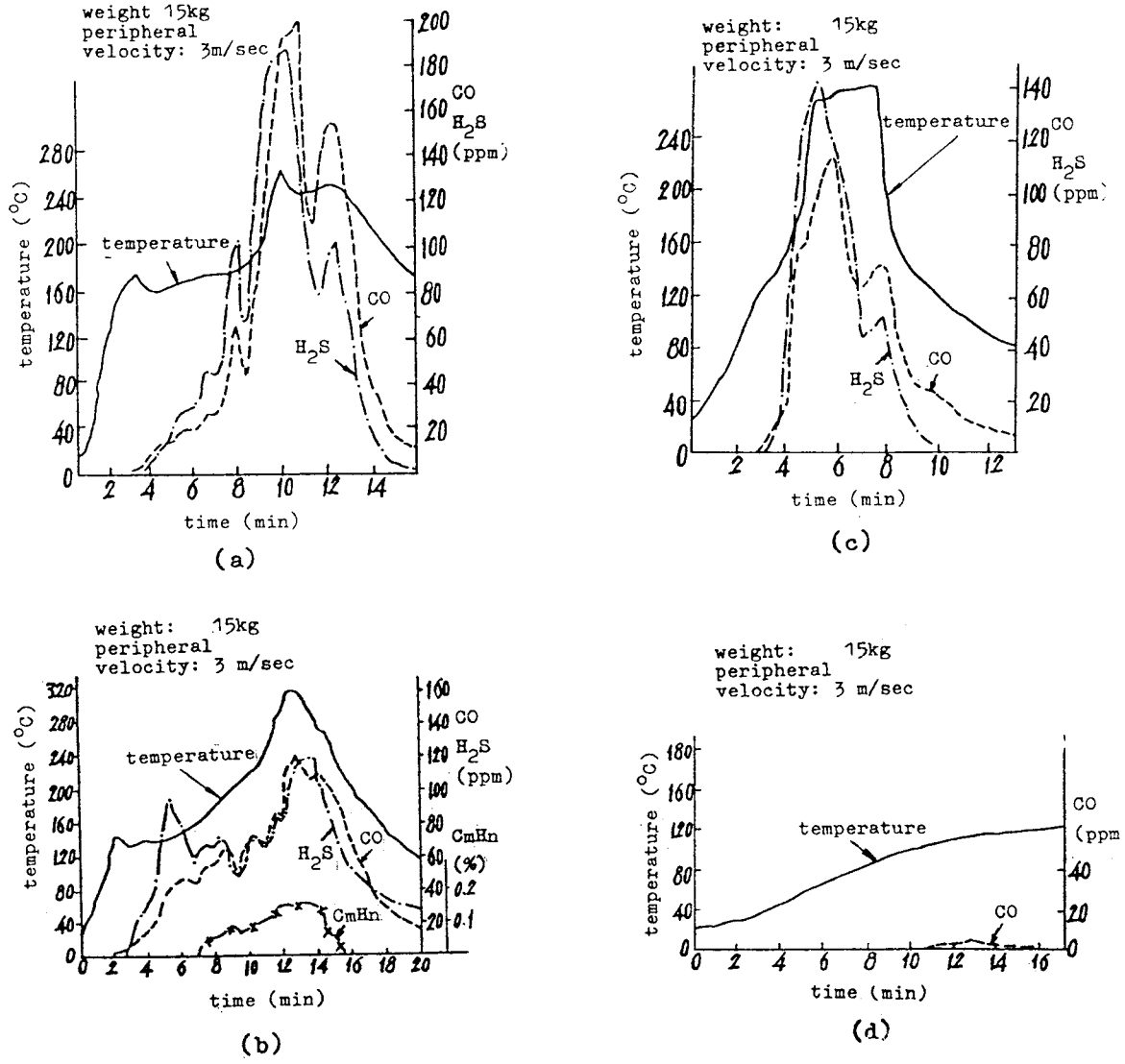


Fig. 4 The temperatures and concentrations of produced gases versus the time of friction

large amount of gases began to arise and when the temperature of belt B was between 120°C and 180°C, hydrocarbon gases were detected, but the rate of temperature rise was slower and the concentrations of these gases were lower as compared with belt A and C. For the test of belt C, when its temperature reached 120 °C, the concentrations of produced gases sharply increased, and moreover the maximum concentrations of gases were obtained more earlily than the maximum temperatures.

The maximum temperature of belt D was 135°C during friction test, and there were no stripping, softening, melt and breakdown when the belt was continuously rubbed for two hours. there was only a little carbonization on the rubbed surface and a small amount of gases were produced, accompanied by very thin and discontinuous smoke. Fig. 4(d) shows the variations of temperature and concentrations of gases produced by belt D with time at the peripheral

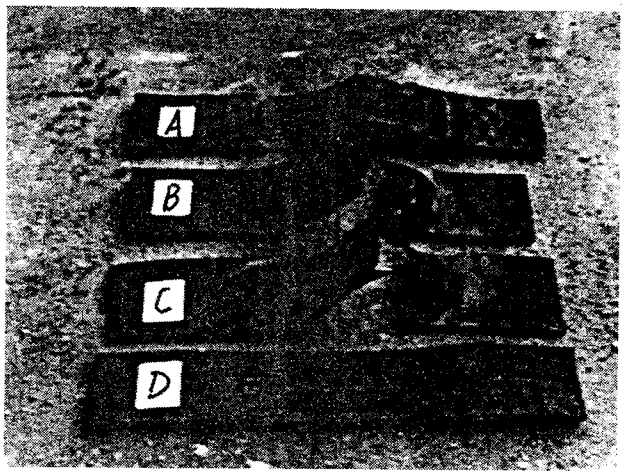


Fig. 5 The belts after friction tests

velocity of 3 m/sec and with a 15 kg weight. The temperature rised slowly and the maximum temperature only reached 135°C to 140°C.

Fig. 5 shows the state of belt A, B, C and D after they were subjected to rubbing.

(2). The influence of tension

Table 6 summarizes the test results of belt A when the peripheral velocity was 2 m/sec and the weights were 5, 10 and 15 kg respectively. The time of belt melt and breakdown was between 15'50" and 49'30", and it became

Table 6 The results of friction tests (II)

Sample	Weight (kg)	Time of belt breaking	Maximum temperature (°C)	Maximum rate of temperature rise (°C/min)	The time and temperature at which CO began to be detected		Maximum gas concentration		
					time	temperature (°C)	CmHn (%)	H_2S (ppm)	CO (ppm)
A	5	49'30"	234.6	4.6	11'20"	64-79	0.05	81	136
	10	26'20"	275.8	11.6	7'40"	90-109	0.10	30	73
	15	16'50"	276.5	19.5	5'00"	76-110	-	77	108

shorter while a heavier weight was applied to this belt. When the maximum temperature was between 234.6 and 276.5°C, a huge amount of CO and H_2S gases produced, and when the weights were 5 and 10 kg, hydrocarbon gases also produced.

IV. THE ANALYSIS OF TEST RESULTS

1. The Influence of Peripheral Velocity and Tension

The quantity of heat produced in the process of a belt rubbing over the driving drum can be calculated by the following equation(6).

$$Q = \mu VP \qquad (1)$$

where Q - quantity of heat per square meter per second, $J/(m^2.sec)$;
V - friction speed, m/sec;
P - friction pressure, Pa;
μ - coefficient of friction.

The quantity of heat produced during friction test is directly proportional to the friction speed and friction pressure, and the coefficient of friction is its proportionality factor. The rubbing force is directly proportional to the tension and when the rubbing area is constant, more heat is produced within unit friction time and the temperature rises more quickly.

Fig. 6 shows the plot of the maximum rates of temperature rise of belts against the peripheral velocities of the driving drum when the weight was 15 kg. The curves of belt A, B and C are almost linear and the temperature of belt C appears to rise most quickly. There is almost not any change for the curve of belt D and the rate of its temperature rise is 4.5 - 6.2°C.

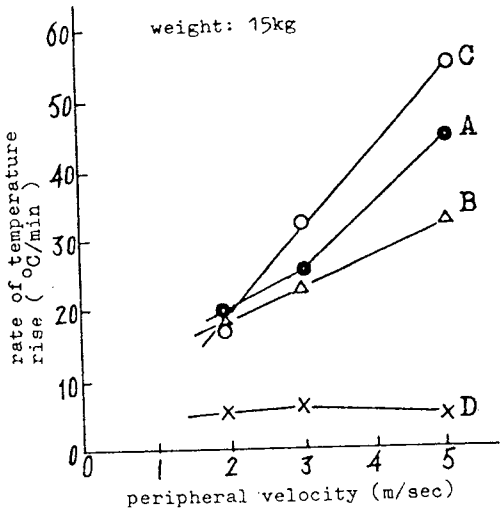


Fig. 6 Relation between peripheral velocity and maximum temperature

The influence of tension was similar to that of peripheral velocity shown in Table 6. Therefore, the quantity of heat produced during belt friction test can expressed by equation (1).

The melt and breaking time of belt A, B and C is reversely proportional to the peripheral velocity of the driving drum as

shown in Fig. 7.

The oxygen content in ambient atmosphere is always 20.3 - 21%, but the concentrations of CO and H_2S gases are relatively high (see Table 5 and 6), the most are up to 100 ppm or more. When the temperature is between 65 and 110°C, CO and H_2S gases begin to be detected, and with the rise of temperature the concentrations of these gases become higher.

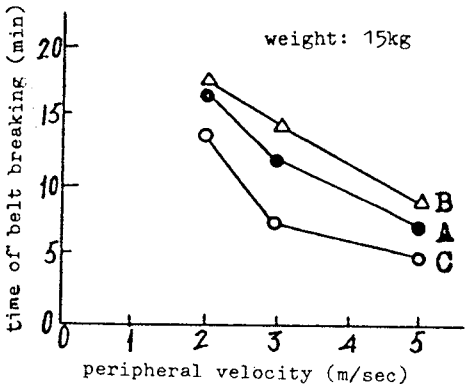


Fig. 7 Relation between peripheral velocity and breaking time of belts

2. The Comparison of Gases Produced During Belt Friction Tests

Fig. 8 shows the variations of the temperatures of belt A, B and C with the concentrations of gases produced during friction test at the peripheral velocity of 3 m/sec and with a 15 kg weight. It can be seen from Fig. 8 that the temperature of

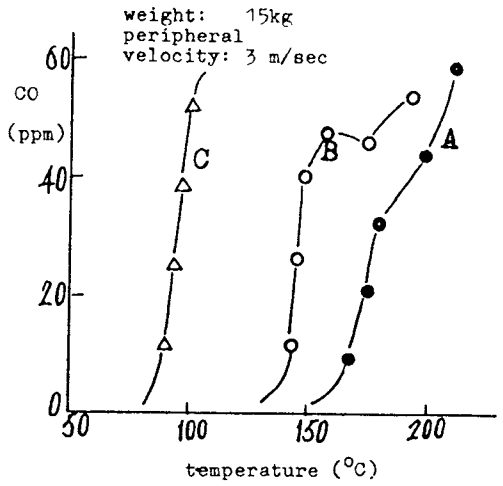


Fig. 8 The concentration of CO gas versus temperature

CO gas generation of belt C is rather low. Fig. 9 and 10 show the variations of CO and H_2S gases produced during friction test with time under the same test conditions. It is obvious that the gases generate from belt C ealier than from others.

3. The Relation of Friction and Basic Heat Values

The calorific capacities and

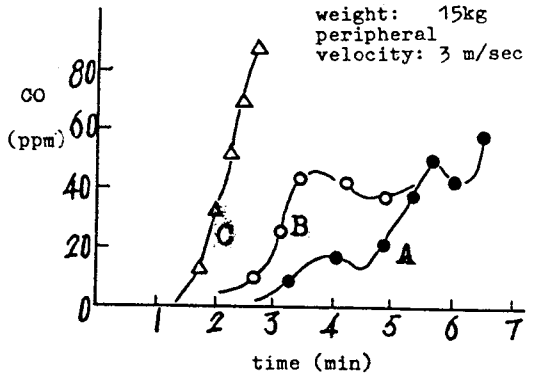


Fig. 9 The concentration of CO gas versus time

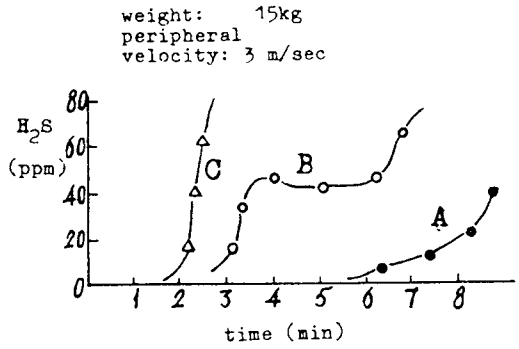


Fig. 10 The concentration of H_2S gas versus time

activation energies of belt A, B and C were greater than that of belt D. In the rubbing tests, belt A, B and C melted and broke, their maximum temperatures were in the range of 208°C - 332.5°C, and also a huge amount of CO and H_2S gases generated. For belt D, no melt and breakdown presented, the temperature was relatively low and little combustible gas arised. This shows that there were qualitative relations between the basic heat values of belts and the breakdown, the temperature rise and the properties of produced gases.

V. CONCLUSIONS

By the friction tests conducted with three kinds of Japanese fire-resistant belts and one kind of Chinese fire-resistant belt, we further examined the basic heat characteristics of these conveyor belts, the temperature rise and the properties of gases produced during belt rubbing tests. Conclusions are drawn as follows.

(1) It can be seen from the content of organic matter, the content of non-combustible matter, the activation energy and the calorific capacity measured in nitrogen that there are considerable differences between Japanese fire-resistant belt A, B and C and Chinese belt D. This is due to the different materials of belts.

(2). Some fire-resistant belts may melt and break during rubbing tests, but some may not. When the belt A, B and C melted and broke, their temperatures varied from 208°C to 332°C.

(3). The belts that may melt and break can produce a huge amount of gases while temperature rises. So it is possible to predict the belt fires by detecting these gases.

(4). Belt D has good fire resistance and wear resistance. So it is an important means for the prevention of belt fires to change the compositions of fire-resistant belts to improve their fire resistance. The rubbing tests of a belt against a driving drum is one of the major methods

for testing the fire resistance of conveyor belts.

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矿用运输胶带摩擦升温及其火灾气体发生规律的研究

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摘　要

难燃性胶带虽已广泛应用，但摩擦引起胶带火灾事故仍不断发生。文中以中国、日本制的四种难燃胶带为试样，测定热的特性值，考察其与主动滚轴摩擦时温度上升和气体发生的特性。有三种胶带发生熔断，温度达208～332.5°C，并伴有大量的CO和H_2S气体发生。而另一种（中国）胶带耐摩性强，最高温度只达135°C。根据试验结果，提出了胶带火灾预测指标和防治措施。

一、前言

胶带作为运输的手段已广泛应用在矿山的地面和井下。随着矿井采掘深度的加大，生产能力的提高，胶带输送机正向高速化、大型化发展。虽然许多国家对胶带已采取了种种防火措施，但运输胶带的燃烧、熔断及火灾事故仍不断地发生。由于许多胶带在矿井的进风巷内连续设置，一旦发生火灾，就迅速蔓延，含有有毒有害气体的高温气流、烟雾沿着巷道流动往往导致极其重大的事故。1950年英国Greswell煤矿的胶带火灾致使80人死亡[1]。1984年1月，日本三井三池煤矿发生胶带火灾致使83人死亡，16人重伤[2]。1984年12月，美国Wilberg煤矿因胶带火灾，造成27人死亡[3]。中国也曾发生过矿井运输胶带火灾的重大事故。

因此，世界主要产煤国家的学者从60年代起，开始研究胶带火灾的原因和预防措施。从研究的成果得知，天然橡胶和合成橡胶的胶带极容易发生火灾，因而，研制出难燃性胶带，并已被广泛采用。尽管这样，胶带的火灾事故依然时有发生，据欧州共同体的调查，胶带火灾的次数约占总发火次数的10～15%[1,3]，为此重新对难燃性胶带发火的可能性进行研究仍具有重大的意义。

根据许多文献资料分析认为，胶带发火的主要原因是胶带与主动滚轴之间产生滑动摩擦[4,5]，亦即胶带由于过载等原因，不能和主动轴同步移动而处于停止状态时，和主动滚轴之间产生的高速摩擦。

本研究中，把日本的三种难燃胶带和中国的一种难燃胶带作试样，在测定热的基础特性的同时，进行了主动滚轴和停止状态的胶带之间的摩擦试验，改变胶带的张力和主动滚轴的旋转速度，以考察摩擦时的温升和产生气体的有关特性。

二、胶带试样和热的基础特性

胶带试样特性如表1。

试验前，首先调查胶带热的基础特性，主要进行了热重量分析和发热量的测定。

1、热重量分析

热重量分析中预先给定调节温度的程序使被测物的温度变化，同时测定该物的重量变化测定时使用的仪器是日本岛津TGA-41H型热重分析装置。

为了在较低温度下测定有机物量和不可燃物的量，首先在氮气中（第一阶段），接着在空气中（第二阶段）进行测定，其试验条件分别如下：

第一阶段：氮气流量50毫升／分，从常温以30°C／分的速度升温，升至600°C，恒温30分钟。（橡胶质有机物的热分解）

第二解段：空气流量50毫升／分，由600°C开始以20°C／分的速度升至800°C，恒温30分钟。（碳黑的燃烧）

测定结果如表2。

表1　　胶带试样特性

试样	种类	帆布层数	宽度（mm）	厚度（mm）				重量（kg/m²）	备注
				全厚	上层胶	下层胶	帆布层		
A	难燃	5	900	14	5	5	4	15.9	日本制
B	难燃	5	900	15	5	5	5	16.3	日本制
C	难燃	5	900	13	4	4	5	15.6	日本制
D	难燃	8	805	15	3	2	10	20.7	中国制

表2　　热重量（%）分析结果

试样	有机物量（第1阶段）	可燃物减量（第2阶段）	不可燃物量
A	65(210～575°C)	30	5
B	65(220～560°C)	29	6
C	64(210～580°C)	28	8
D	61(190～600°C)	28	11

其次，为了研究在空气中的热分解特性，低温阶段也在空气中进行了分解，其试验条件如下：

空气流量50毫升／分，由常温以30°C／分的速度升至800°C，恒温30分钟。

在空气中所测定的热重量变化如图1，在上述试验条件下，四个试样均在220～240°C开始急剧的热分解，但是胶带D比其它三种胶带的T－G曲线的斜率要小。

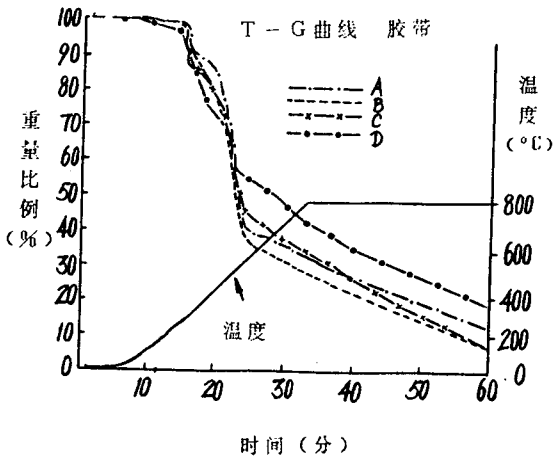


图1　在空气中的T－G曲线

表3　　活化能计算结果

试样	A	B	C	D
活化能（千卡／摩）	15.6	16.5	15.8	11.2

由热重量分析的结果来计算活化能如表3由表3看出，胶带D和胶带A、B、C之间有相当大的差异。

2、发热量

发热量也是评价可燃物的重要特性，我们采用日本岛津CA-4型燃研式自动热量计进行了该性能的测定。测定结果如表4。胶带A、B C的发热量为8000千卡／公斤左右，而胶带D为6290千卡／公斤，要低得多。

表 4　　发热量测定结果

试　　样	A	B	C	D
发　热　量 （千卡／公斤）	8130	7990	8360	6290

三、胶带摩擦试验

本试验是驱动滚轴和停止胶带间的摩擦试验。试验装置（如图 2 ）由可变速电动机，驱动滚轴、温度测定和气体分析装置组成。钢制滚轴的直径为234 毫米，宽度为400 毫米。温度测定部分由CA热电偶，A／D 转换器，微型计算机组成。产生的气体由日本理研GTX-1108型有害气体检测仪分析，O_2、CO、H_2S 及碳氢化合物等气体都是连续测定。H_2S 、CO的测定用的是电位电解法原理。碳氢化合物的测定用的是接触燃烧法原理。

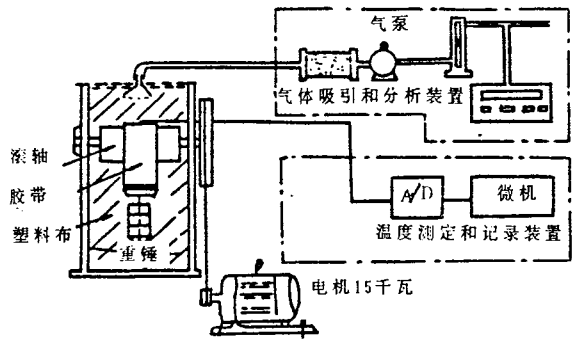


图 2　胶带摩擦试验装置

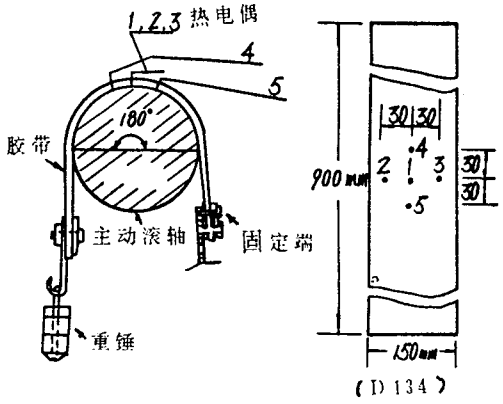


图 3　胶带试验片和热电偶的配置

1 、试验方法

供试验用的胶带片尺寸为长900 毫米，宽150 毫米（试料D 的宽为134 毫米），厚度为原厚，长度方向和胶带前进方向一致。温度测定时如图 3 所示，在试验片的中央部位钻 5 个小孔，其间隔为30毫米，深10毫米，将直径为1.6 毫米的热电偶分别插入孔内。在胶带表面（中央）设置一支热电偶测定胶带表面温度，胶带和滚轴的围包角为180° 。胶带的一端固定，另一端悬挂重锤。

为了抽出摩擦时所产生的气体，用塑料布覆盖试验装置的上半部分。气体由气泵吸出，其流量为26升／分。在整个摩擦过程中，连续测定气体和温度的同时，对胶带的形状变化、发烟等情况进行观察。

井下胶带的行走速度一般为 2 ～ 5 米／秒，所以试验时，滚轴的园周速度按 2 、 3 、 5 米／秒三个等级变化，锤重按 5 、10、15公斤变化。试验时，锤重增加到20公斤时，因电机启动不了，而摩擦中止。

2 、试验结果

（ 1 ）主动滚轴圆周速度的影响

重锤的重量一定（15公斤），变化园周速度进行试验的结果如表 5 。胶带 A 、 B 、 C 在摩擦开始 6 ～18分钟内即被熔断。这时的最高温度分别为208 ～276.5°C 、291 ～332.5°C，256 ～308°C 。当园周速度加大时，其熔断时间变短，最高温度变高。但胶带 A 在园周速度增至 5 米／秒时，最高温度反而变低。

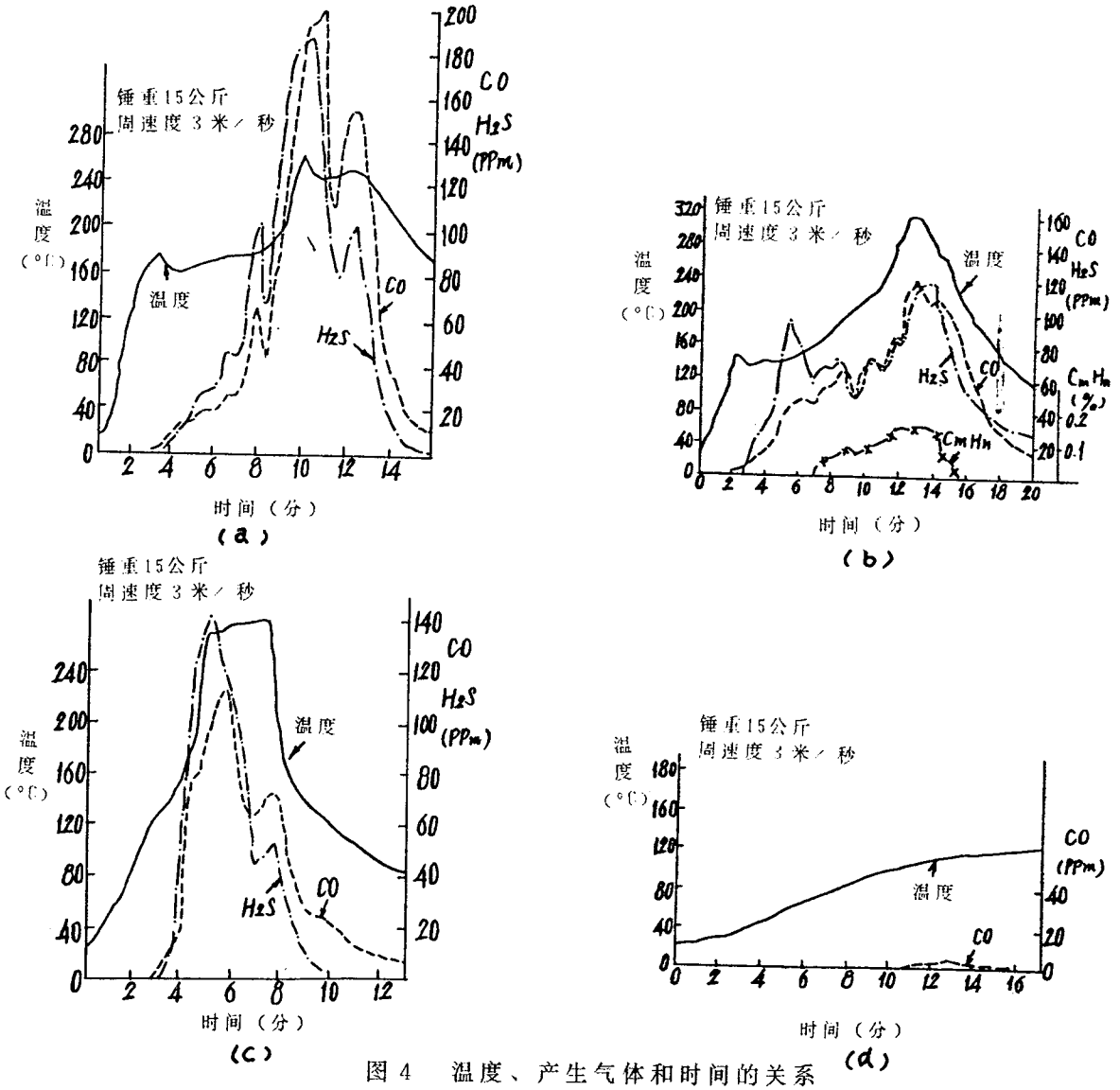


图4　温度、产生气体和时间的关系

a、b、c、d——分别为对胶带A、B、C、D作出的试验结果

表6　　摩擦试验结果（2）

试样	锤的重量（kg）	熔断时间	最高温度（°C）	最大温升速度 °C/分	CO气体最先被检测到时的		最高气体浓度		
					时间	温度(°C)	C_mH_n (%)	H_2S (ppm)	CO (ppm)
A	5	49′30″	234.6	4.6	11′20″	64～79	0.05	81	136
	10	26′20″	275.8	11.6	7′40″	90～109	0.10	30	73
	15	16′50″	276.5	19.5	5′00″	76～110	--	77	108

表5　　摩擦试验结果（1）

试样	圆周速度 (m/s)	熔断时间	最高温度 (°C)	最大温升速度 °C/分	开始检测到CO时的时间和温度		最高气体浓度		
					时间	温度 (°C)	C_mH_n (%)	H_2S (ppm)	CO (ppm)
A	2	16′50″	276.5	19.5	5′00″	76 ～110	--	77	108
	3	12′00″	269.6	25.5	3′00″	104 ～150	--	188	200
	5	7′05″	208.0	45.0	2′40″	79 ～95	--	25	123
B	2	17′40″	291.2	18.5	5′40″	65 ～105	0.15	124	235
	3	14′35″	320.3	23.9	2′20″	85 ～143	0.15	119	114
	5	8′50″	332.5	32.7	2′00″	86 ～152	0.10	54	110
C	2	14′00″	256.3	18.0	5′40″	94 ～177	--	51	75
	3	7′40″	282.0	33.1	2′40″	67 ～126	--	141	114
	5	5′20″	308.0	55.4	1′20″	77 ～107	--	55	55
D	2	--	135.3	5.7	3′20″	42 ～67	--	--	3
	3	--	126.4	6.2	10′20″	92 ～110	--	--	4
	5	--	132.4	4.5	33′00″	109 ～128	--	--	2

胶带A、B、C在摩擦开始1分钟后，即产生浓烟；当温度达150°C时，发烟更加激烈同时可以看到胶带开始软化，并被剥离，之后摩损向深部发展，大量高温的胶带剥离片向四周飞散。摩擦开始后2～5分钟内检测到有CO和H_2S气体。

圆周速度3米/秒，锤重15公斤时胶带A、B、C的温度、产生气体与时间（从摩擦开始时计起，下同）的关系分别如图4a、4b、4c。随着温度的上升，产生气体的浓度达到最大值。胶带A当温度达到150°C时，开始产生大量的气体。胶带B在温度120～180°C间检测到碳氢化合物，但和胶带A、C相比，其温升速度和产生气体的浓度要小。胶带C的试验中，当温度达到120°C时其产生气体的浓度就急剧地增大，而且产生气体的浓度最大值的时间比温度达到最大值时间早些。

胶带D摩擦时的最高为135°C，连续摩擦2小时，也没有发生剥离、软化、熔断等现象，只是在摩擦面有少许炭化现象，并且发生气体也较少，只是间断地出现了非常清淡的烟雾。当圆周速度3米/秒，锤重15公斤时，温度、产生气体和时间的关系如图4d所示。温度上升非常缓慢。最高温度达到135～140°C，以后就停止上升。

胶带A、B、C、D的试验片摩擦后的状态如图5。

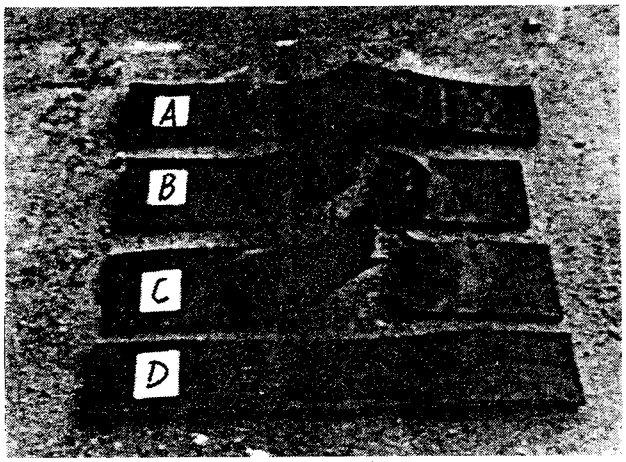


图5　摩擦试验片的样片

（2）张力的影响

主动滚轴的圆周速度为2米/秒，重锤的重量分别为5、10、15公斤时，胶带A的试验结果如表6，熔断时间在16分50秒至49分30秒之间；当重锤重量增加时，熔断时间变短。最高温度234.6～276.5°C时，产生大量CO和H_2S气体，重锤的重量为5、10公斤时，还产生碳氢化合物。

四、试验结果的分析

1、圆周速度、张力的影响

胶带和滚轴之间的摩擦所产生的热量，可以用下式来表示[6]：

$$Q = uvp \qquad (1)$$

式中：Q—单位时间内单位面积上的发热量，焦/(米^2 ·秒)；

v—摩擦速度，米/秒；

p—摩擦压力，帕；

u—摩擦系数。

即摩擦所产生的热量与摩擦压力、摩擦速度成正比，其比例的系数是摩擦系数。摩擦压力与张力成正比，当摩擦面积一定时，每单位摩擦时间内所产生的热量多，则升温速度快。

锤重15公斤时，胶带摩擦最大温升速度和滚轴圆周速度之间关系如图6，胶带A、B、C的曲线几乎是线性关系，胶带C的温升速度最快。而胶带D几乎没有什么变化，其温度速度为4.5 ～6.2°C/分。

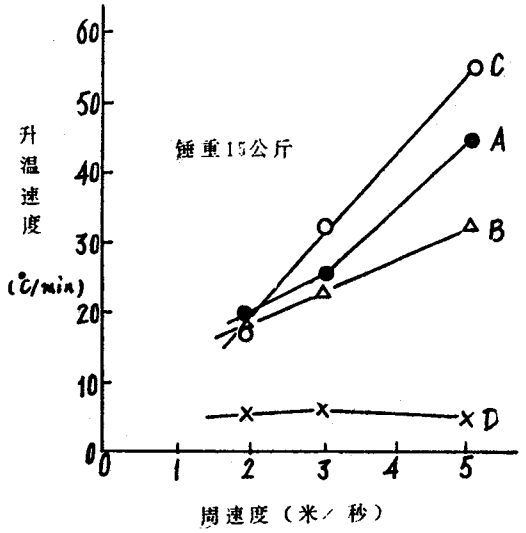


图6 圆周速度和最大升温速度的关系

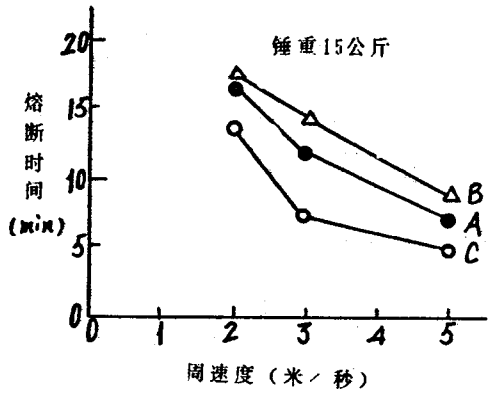


图7 圆周速度和熔断时间的关系

张力的影响如表6，同圆周速度的影响呈现出同样的倾向。因此，胶带摩擦时所产生的热量可用（1）式来表示。

胶带A、B、C的熔断时间和滚轴圆周速度的关系如图7所示，呈反比例关系。

胶带摩擦时环境气体中O_2浓度始终为20.3～21%，而CO、H_2S的浓度却相当高，如表5、6，浓度达到100ppm以上者相当多。开始被检测到CO和H_2S气体时的温度是65～110°C，随着温度的上升，产生气体的浓度变高。

2、摩擦时产生气体的比较

在主动滚轴的圆周速度为3米/秒，锤重15公斤条件下，胶带A、B、C摩擦时的温升和产生气体关系的比较如图8，从开始产生CO气体的温度来看，胶带C较低。图9、图10是胶带A B C在同样条件下产生CO和H_2S气体时间的比较，从图中可以看出，仍然是胶带C产生气体较早。

3、摩擦和热特性值的关系

从发热量和活化能来说，胶带A、B、C比胶带D大得多。在摩擦试验中，胶带A、B、C出现熔断现象，其最高温度为208 ～332.5°C，同时产生大量的CO和H_2S气体。而胶带D没有出现熔断现象，温度也较低，几乎没有产生可然性气体。由此可见，胶带的基础特性值与摩擦时的熔断、温升、产生气体特性之间存在着定性的关系。

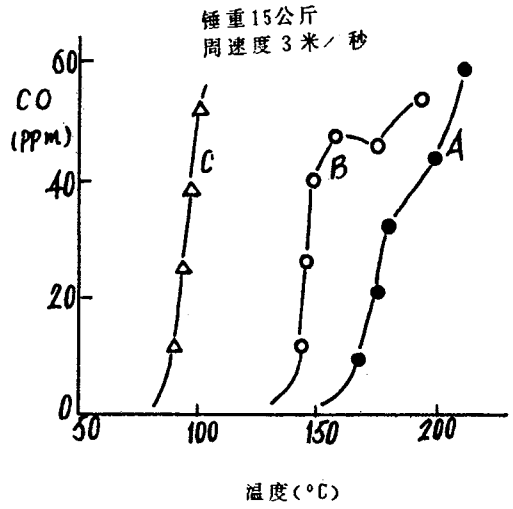


图8　产生CO气体和温度的关系

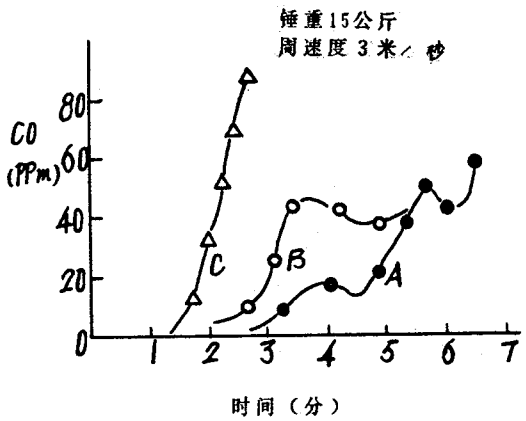


图9　产生CO气体和时间的关系

五、结论

以日本和中国制造的难燃性胶带作为试样，作滚轴摩擦试验并进一步探讨胶带的热基础特性及摩擦时温升和产生气体的特性，得到了以下结论：

（1）在N_2气体中所测得的有机物含量和不燃性物量，活性化能及发热量值来看，胶带A、B、C（日本制）和胶带D（中国制）之间存在着相当大的差异，这是由于材质不同所致。

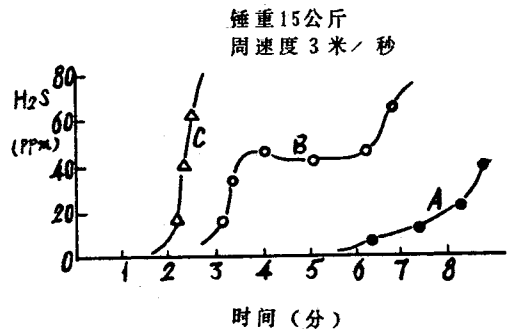


图10　产生H_2S 气体和时间的关系

（2）难燃性的胶带在摩擦中有发生熔断和不发生熔断的区别，胶带A、B、C在熔断时的温度为208 ～332.5°C 。

（3）产生熔断现象的胶带，在摩擦温度上升的同时，产生大量的CO和H_2S 气体，因此，通过检测这些气体来预测胶带火灾是可能的。

（4）胶带D的阻燃、耐摩性能好。因此，从材质上提高难燃胶带的难燃程度是防止胶带火灾的重要措施之一。胶带和滚轴摩擦试验是检验难燃性的主要方法之一。

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RECHERCHES SUR L'ELEVATION DE TEMPERATURE DES COURROIES DE CONVOYEURS MINIERS CAUSEE PAR LA FRICTION, ET REGULARITE DU DEGAGEMENT GAZEUX AU COURS D'INCENDIES DE COURROIES

Jiang Shi-cai, Takeshi KOMAI, Takehiro ISEI

Même si l'usage de courroies de convoyeurs ignifugées est très répandu, des incendies causés par les frictions des courroies se produisent continuellement. Notre rapport examine trois types de courroies de convoyeurs fabriqués au Japon et un type fabriqué en Chine; ces courroies, qui servent d'échantillons d'essais, permettent d'établir leurs propriétés thermiques et d'examiner l'élévation de la température et la production de gaz au cours d'essais de friction avec des tambours d'entraînement. Au cours de ces essais, les trois types de courroies japonaises se sont rompues par coulée à une température entre 208 et 333°C, avec production d'une grande quantité d'oxyde de carbone et d'hydrogène sulfuré. Les courroies chinoises ont présenté une forte résistance à la friction, et la température au cours des essais ne s'est élevée que jusqu'à 135°C. Sur la base de ces résultats, notre rapport donne des indications de calcul et des mesures de prévention des incendies de courroies de convoyeurs miniers.

FORSCHUNGSARBEIT ÜBER TEMPERATURANSTIEGE AN FÖRDERBÄNDERN DURCH REIBUNG UND ÜBER DIE REGELMÄßIGKEIT VON GASERZEUGUNG BEI FÖRDERBANDBRÄNDEN.

Jiang Shi-cai, Takeshi Komai, Takehiro Isei,

Trotz des weitgehenden Einsatzes von flammenfesten Förderbändern kommt es immer wieder zu Bränden, die durch Reibung erzeugt werden. Der Vortrag befaßt sich mit den Ergebnissen von vier verschiedenen Fördergurten, von drei Gurten aus Japan und einem aus China, die auf ihre thermischen Eigenschaften und ihre Gaserzeugung während eines Reibungstestes auf einer Antriebstrommel untersucht wurden. Die drei Gurte aus Japan zeigten eine Schmelztrennung bei einer Temperatur zwischen 208°C und 333°C, wobei eine große Menge von Kohlenmonoxid und Schwefelwasserstoff freigesetzt wurde. Der Gurt aus China zeigte eine hohe Resistenz gegen Reibung, und die Temperatur stieg nur auf 135°C. Der Vortrag gibt einige Warnungszeichen und einige vorbeugende Maßnahmen zur Verhütung von Bränden an Grubenförderbänden an, die sich auf die genannten Ergebnisse gründen.

ИЗУЧЕНИЕ ПОВЫШЕНИЯ ТЕМПЕРАТУРЫ КОНВЕЙЕРНЫХ ЛЕНТ ВСЛЕДСТВИЕ ТРЕНИЯ И ОБРАЗОВАНИЯ ГАЗА ВО ВРЕМЯ ПОЖАРОВ

Джианг Ши-Кай, КНР; Такеши Комаи и Такехиро Исеи, Япония.

Пожары на конвейерных лентах по причине трения продолжают возникать, несмотря на широкое использование огнеупорных лент. При работе по подготовке настоящего доклада, были использованы три вида конвейерных лент, произведенных в Японии и один вид, сделанный в Китае. Целью эксперимента являлось определение термальных свойств и изучение повышения температуры и генерации газа в процессе трения между конвейерными лентами и приводным барабаном. В процессе эксперимента, разрыв по плавлению произошел при температуре от 208 до 333оС в случае трех видов японских конвейерных лент, а также образовалось большое количество углекислого газа и сероводорода. Китайская конвейерная лента обладала сильной сопротивляемостью к трению, и температура в процессе эксперимента достигла только 135оС. На основании результатов этого эксперимента, мы предлагаем прогнозные индикаторы и меры предотвращения пожаров на конвейерных лентах.

FLAMMABILITY TESTS OF CONVEYOR RUBBER BELTS WITH FABRIC CARCASE USED IN JAPANESE UNDERGROUND COAL MINES

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ABSTRACT

Small-scale flame test, oxygen index test and laboratory-scale gallery test were conducted on the "flame-resistant" rubber conveyor belts with 5-ply fabric carcase used in Japanese underground coal mines. The results of the oxygen index test were compared with those of the other two tests respectively, and the correlation coefficients were calculated. In conclusion, the critical oxygen indices of belt samples are found to be relatively consistent with results of the laboratory-scale gallery test, although correlation between the results of oxygen index test and those of small-scale flame test seems not to be so clear. Besides, it is confirmed that the critical oxygen index becomes higher with increase of sample width before its width reaches to a certain value, which depends on whether the fabric carcase of the belt is "flame-resistant" or not.

INTRODUCTION

In underground mines, it is desirable to use fire-resistant conveyor belts from the standpoint of prevention of underground mine fires. The "flame-resistant" rubber conveyor belts, which satisfy the specifications prescribed in JIS K 6324, have been almost commonly used in Japanese underground coal mines since publication of this standard in 1966. The fire-resistance test prescribed in this standard consists only of the small-scale flame test for conveyor belt pieces with covers. The specifications are given only to the mean duration of flame after removal of the Bunsen burner and also to non-reappearance of flame after applying the air current. Thus it is titled as "Qualitative Standard for Flame Resistance of Conveyor Belts", which specifies only resistance to flame of the Bunsen burner qualitatively but not the quantitative grade of fire-resistance of belting materials.

Oxygen index test, on the other hand, is widely adopted as the well-known quantitative method for fire-resistance test on the polymeric materials. The method of oxygen index test is also described in JIS K 7201, titled as "Testing Method for Flammability of Polymeric Materials Using the Oxygen Index Method". This standard can be applied to flammability test for single and homogeneous polymeric materials, such as plastics, rubbers, textiles and so on. A conveyor belt with fabric carcase, however, consists of rubber for covers, textile reinforcement and adhesive rubber. The oxygen index of such composite materials seems to be difficult to determine exactly since flammability of each material the belt is composed of might be rather different. In spite of such difficulty, oxygen index test, as well as small-scale flame test, has been recommended as a quality assurance test for conveyor belts in European Community countries(ECC Safety and Health Commission 1978), and in West Germany it has been accepted as a part of approval test for fire-resistant conveyor belts(Anderson 1984). In Canada, moreover, the oxygen index testing

method was reviewed in detail, and the adequate sample width was proposed as 25mm(Mintz 1983).

On the other hand, several kinds of in-gallery flammability tests for conveyor belts were proposed, and some of them have been standardized in some countries(Anderson 1984). Through such in-gallery flammability tests, we can assess not only ignitability but also flame propagation properties of conveyor belts under certain airflow conditions, which seems to be more significant than any other kind of smaller-scale flammability tests. The laboratory-scale gallery test proposed in West Germany(DIN 22 118 Entwurf 1978) seems to be available since it is reported that there is a certain correlation between the results from this kind of laboratory-scale gallery test and those from the large-scale gallery test(Grumbrecht 1976), and in addition, the testing cost for laboratory-scale gallery test is much smaller.

Authors have conducted the small-scale flame test, the oxygen index test and the laboratory-scale gallery test, for the rubber conveyor belts with fabric carcase, manufactured for use in Japanese underground coal mines, in order to evaluate these flammability test methods. The small-scale flame test was conducted, almost based on the method prescribed in JIS K 6324-1977, and in addition, the duration of afterglow, as well as that of flame, was measured for belt samples both with and without covers. On the other hand, the oxygen index test was carried out, almost in accordance with the method of Canadian Explosive Atmospheres Laboratory(Mintz 1983). Laboratory-scale gallery test, moreover, was conducted, almost in accordance with the method prescribed in the draft of DIN 22 118, and in addition, the ignitability and flame propagation speed on the belt samples were determined. The following parts of this report describes the method and results of these flammability tests.

SMALL-SCALE FLAME TEST

Small-scale flame test was conducted on 6 different conveyor belt samples with 5-ply fabric carcase for use in Japanese underground coal mines (Nakagawa and Takahashi 1987). Every belt sample tested is regarded as "flame-resistant", in accordance with JIS K 6324. This standard prescribes that the flame-resistant test shall be conducted on 3 conveyor belt pieces with covers only, and the specifications are given only to mean duration of flame after removal of the Bunsen burner as within 1 minute and also to non-reappearance of flame after applying the air current of 1.5m/s.

It is said, however, that prevention of afterglow is relevant for underground coal mines where the presence of glow may ignite a methane/air explosion (Szymanski and Mintz 1986). In addition, International Standard (i.e. ISO 340-1982), titled as "Flame retardation of conveyor belts -- Specification and method of test", has a note which says, "provision is made for performing the test on test pieces without covers, as covers of belts may be ripped off accidentally in service."

Thus authors have also measured the duration of afterglow, and furthermore, conducted the flame test on the belt pieces without covers, in order to assess fire-resistance of the "flame-resistant" belts used in Japanese underground coal mines, from the different view point.

Table 1 Thickness of conveyor belt samples for the small-scale flame test.

Sample	Full thickness (mm)	Approximate thickness of carcase (mm)
A	15.2 --- 15.4	5.0
B	13.8 --- 14.0	4.5
C	12.8 --- 13.1	5.0
D	12.2 --- 12.3	5.0
E	13.1 --- 14.7	5.0
F	11.2 --- 13.6	4.5

Conveyor Belt Samples

Conveyor belt pieces, 150mm long and 25mm wide, were cut parallel to either warp or weft for each sample, thickness of which is shown in Table 1. In this table, Samples A, B and C are unused, and the others are already used ones.

Testing Procedure

The conveyor belt piece prepared for this test was placed above the Bunsen burner with top opening diameter of about 11mm, as prescribed in JIS K 6324-1977, and the height of inner and outer cone of flame from the burner was adjusted to about 50mm and 160mm respectively.

Flow rate of town gas was carefully adjusted so that the copper wire with the diameter of about 0.7mm may melt within 6 seconds after being inserted into the same position as the belt piece's.

The flame from the burner was applied to the belt piece for 1 minute, and then the duration of flame on the piece was measured after removal of the burner. Then in about 30 seconds after flameout, air current of about 1.5m/s was applied to the belt piece, both in order to check whether the flame reappear or not and to measure the duration of afterglow while the piece is exposed to airflow.

The duration of flame was measured for 12 pieces (6 parallel to warp, 6 parallel to weft) and that of afterglow for 8 pieces (4 parallel to warp, 4 parallel to weft) of each belt sample with covers. On the other hand, 8 pieces for each sample without covers are prepared for measurement of duration of flame.

Test Results

Mean value and standard deviation of duration of flame and those of afterglow for each belt sample with covers are shown in Table 2. Every measured value of duration of flame for each sample with covers was within 5 seconds, and its mean value was within the range between 0.0 and 2.0 seconds. Reappearance of flame was not recognized for any pieces at all during exposure to the airflow. Mean duration of afterglow for each sample, however, distributed within the range between 204 and 3309 seconds, quite longer than duration of flame. This fact shows that each belt sample tested is "flame-resistant" enough to satisfy the specification in JIS K 6324-1977, considering the requirement that mean duration of flame be within 60 seconds, but that afterglow is hardly inhibited.

Table 3 shows mean value and standard deviation of flame and those of afterglow for the 3 samples without covers. Every piece of the other 3 samples burnt themselves out during measurement of duration of flame for more than 3 minutes after removal of the burner. This fact shows that the carcases of Samples B, C and D are made of "non-flame resistant" materials. Besides, only one amomg 8 pieces of Sample E burnt itself out, thus duration of afterglow was measured for 7 of them. Mean duration of afterglow for each 3 sample without covers is distinctly shorter than that for the same sample with covers, which implies that inhibition of afterglow on the covers is less effective than that on the carcase.

Table 2 Mean durations of flame and afterglow for each conveyor belt sample with covers, showing standard deviations.

Sample	Duration of flame (s)		Duration of afterglow (s)	
	Mean value	Standard deviation	Mean value	Standard deviation
A	1.7 (12)	1.0	557 (8)	148
B	1.4 (12)	1.1	846 (8)	271
C	1.5 (12)	0.8	639 (8)	134
D	2.0 (12)	1.3	641 (8)	124
E	1.3 (12)	0.5	204 (8)	46
F	0.0 (12)	0.0	3309 (8)	684

Note : The numbers of conveyor belt pieces are shown in the brackets following the mean values.

Table 3 Mean durations of flame and afterglow for each conveyor belt sample without covers, showing standard deviations.

Sample	Duration of flame (s)		Duration of afterglow (s)	
	Mean value	Standard deviation	Mean value	Standard deviation
A	1.5 (8)	1.4	106 (8)	29
E	14.0 (7)	6.8	7 (7)	19
F	10.6 (8)	5.0	377 (8)	90

Note 1 : The numbers of conveyor belt pieces are shown in the brackets following the mean values.
Note 2 : All pieces of Samples B, C and D burnt themselves out during measurement of duration of flame.

OXYGEN INDEX TEST

In recent years in Canada, it has been suggested that the adequate sample size for this sort of test should be 25mm wide by full thickness(Mintz 1983,1985), in order to obtain good reproducibility of the test results, although Commission of the European Communities has recommended 50mm wide by full thickness(ECC Safety and Commission 1978). On the other hand, the commonly-referred standards for oxygen index test, such as ASTM D 2863-77 and JIS K 7201-1976, ordinarily prescribe that the size of physically self-supporting specimen shall be 6.5±0.5mm wide and 3.0±0.5mm thick. Thus the oxygen index test was conducted, in order not only to determine the critical oxygen index (hereinafter, referred to as COI) of conveyor belts used in Japanese underground coal mines, but also to confirm the adequate width of sample.

Sample Preparation

For each belt sample, pieces with various width, at least including the range between 2mm and 25mm, and full thickness, cut parallel to warp, were prepared for the oxygen index test. The length of the pieces cut from the belt samples was about 150mm, and each piece was tested repeatedly unless its length became less than 70mm after having cut off the burnt and charred part.

Testing Procedures

The test was conducted using the oxygen index measuring apparatus, which consists of a test column, gas flow measurement and control devices, as shown in JIS K 7201-1976.

The test column is made of heat-resistant transparent glass having the inner diameter of about 75mm. The flow rate of oxygen/nitrogen mixture through this column was regulated to be at a certain oxygen concentration and at the total flow rate of 11.4l/min before the test. Oxygen concentration of the mixture in the column was checked using a galvanic cell type oxygen analyzer, and then the flowmeters of the apparatus are calibrated. As the igniting source, liquified petroleum gas was used, and the length of flame from the ignitor was regulated to be about 50mm when the fuel gas was flowing vertically upward from the nozzle of the ignitor. Then the flame was applied to the belt piece clamped vertically in the test column either for, at most, about 45 seconds or until the entire top surface of the piece flamed.

The duration of flame from the removal of the ignitor was measured. Either in case it was less than 3 minutes or burnt length of the belt piece was less than 50mm, the oxygen concentration was increased in 0.5% for the following test, and vice versa. The data were arranged in accordance with the modified Bruceton up-and-down method(Dixon 1965), as adopted at the Canadian Explosive Atmospheres Laboratory(Mintz 1983), and then the critical oxygen index for each belt sample was determined.

Test Results

For each belt sample, the relationship between the sample width and COI was determined, as shown in Fig.1 for Samples A, E and F, and also in Fig.2 for Samples B, C and D. It is because, as the result of the small-scale flame test, the carcases of Samples A, E and F are considered to be "flame-resistant" but those of Samples B, C and D are "non-flame resistant" that authors have classified the belt samples into two categories.

Figs.1 and 2 show that COI of the belt samples is dependent on the sample width when it is relatively small. When the width is under a certain value, the smaller it becomes, the lower the COI tends to be. On the contrary, when the width is more than this value, COI appears to become almost constant. For

example, in case of Sample A, COI becomes higher with increase of sample width until its width reaches to about 5mm, and on the other hand, in case of Samples B, C and D, until it reaches to about 15mm, which shows that "flame-resistance" of the fabric carcase makes great influence on COI of the belt samples.

The results on COI of Samples A, B and C, in particular, were compared with those obtained for both fire-resistant and non-fire resistant rubber belts with fabric carcase in Canada(Mintz 1983, 1985) in Fig.3. From this figure, it is confirmed that the sample width of 25mm is adequate for the oxygen index test on conveyor belts with fabric carcase also in the light of the reproducibility of the data.

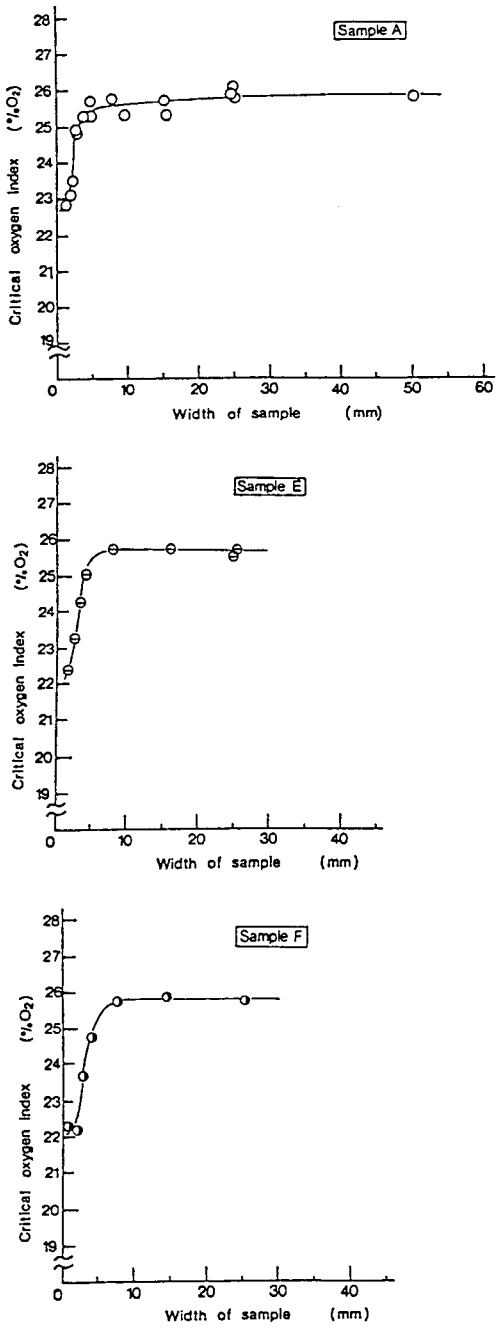


Fig.1

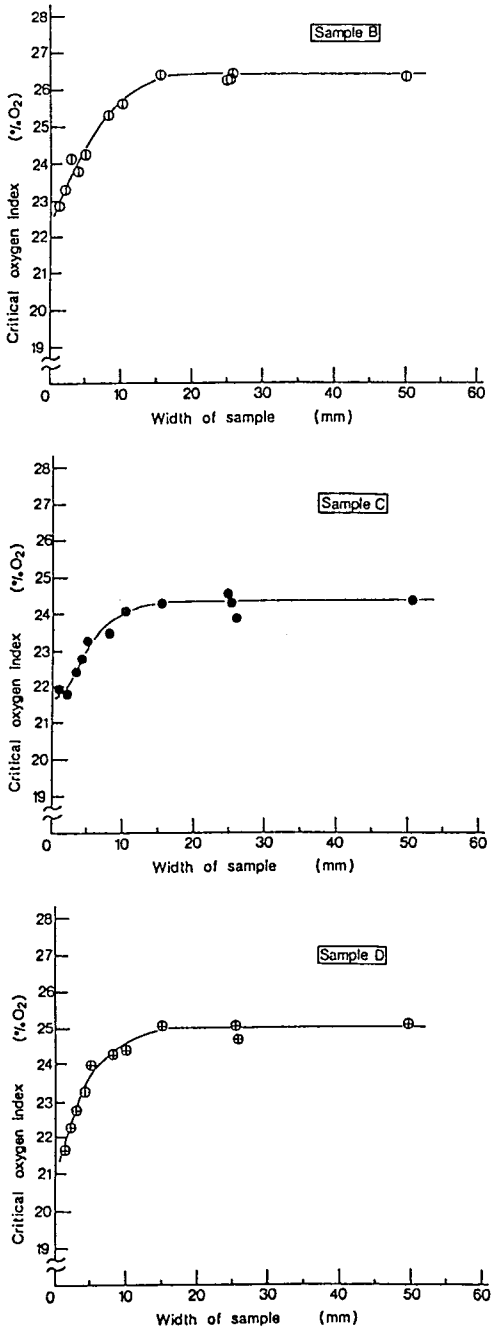


Fig.2

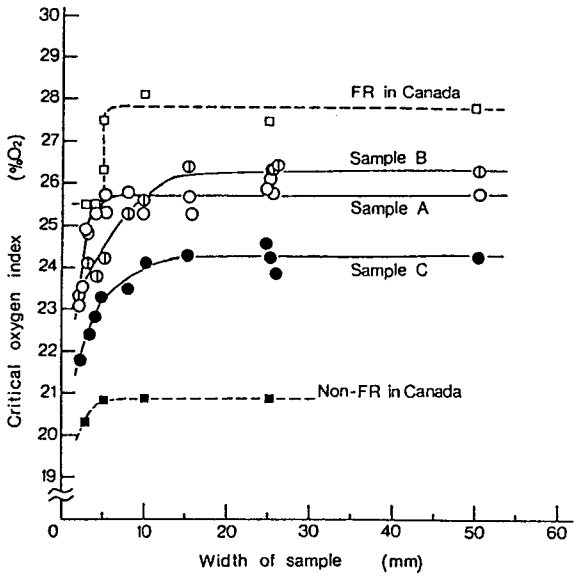


Fig.3

Table 4 Representative results of oxygen index test for each belting sample.

Sample	Width (mm)	Thickness (mm)	Room temperature (°C)	Critical oxygen index
A	2.9	15.1	18	24.9
	3.0	15.3	22	24.8
	24.9	15.1	11	26.1
	24.8	15.2	19	25.9
	25.2	15.2	23	25.8
B	3.2	13.8	19	24.1
	25.2	13.8	11	26.3
	25.4	13.8	19	26.3
	25.8	13.8	23	26.4
C	3.3	13.0	18	22.4
	25.0	13.1	11	24.3
	24.6	13.0	17	24.6
	25.7	13.1	24	23.9
D	2.8	12.3	19	23.3
	26.0	12.3	17	24.7
	25.3	12.3	23	25.1
E	3.3	14.6	22	24.3
	25.0	14.7	11	25.8
	24.8	14.6	24	25.6
F	3.1	13.0	24	23.8
	25.4	13.4	11	25.8
	25.7	13.4	24	25.8

Table 5 Results of oxygen index test for cover rubber of each belting sample.

Sample	Width (mm)	Thickness (mm)	Room temperature (°C)	Critical oxygen index
A	6	5.0	21	24.6
	25	5.0	23	23.9
B	6	4.8	22	23.8
	25	4.8	23	24.4
C	6	4.2	22	22.7
	25	4.1	24	22.8
D	6	3.9	23	23.0
	25	3.8	24	22.3
E	6	4.5	19	24.4
	25	4.5	20	24.3
F	6	4.3	19	25.1
	25	4.3	20	24.8

Table 4 shows the values of COI for each 6 belt samples in case of the width of about 3 and 25mm respectively. COI for about 25mm wide samples ranges between 23.9 and 26.3, and on the other hand, that for about 3mm wide ones ranges between 22.4 and 24.9. It is found that the former is a little bit higher (about 1.0-2.5% in oxygen concentration) than the latter. The difference in COI between about 3mm wide pieces and about 25mm wide ones seems to be dependent on the "flame-resistance" of fabric carcases, which must be caused by the difference of the shape of curve on COI versus sample width.

Table 5 shows the COI of the rubber for covers on the belts. COI of about 25mm wide piece is compared with that of about 6mm wide ones for each sample. Its value is almost independent of the width of the specimen, and relatively smaller than that of 25mm wide belt piece with fabric carcase.

LABORATORY-SCALE GALLERY TEST

This kind of test is adopted as a fire-resistance test for conveyor belts especially for use in underground coal mines in West Germany, and has been proposed as DIN 22 118 since 1978, as mentioned above. This standard is still a draft, but this kind of flammability test is available for assessing both ignitability and flame propagation characteristics on the stationary belt sample placed in the airflow through the duct. Thus authors have conducted the laboratory-scale gallery test, using the apparatus having the model gallery.

This chapter represents the method and results of this kind of laboratory-scale gallery test, which was modified from that prescribed in the draft of DIN 22 118.

Testing Apparatus

The apparatus for this laboratory-scale gallery test consists of three major parts; i.e.

model gallery, supporting frame and scrubber, as shown in Fig.4. The inner cross section of the model gallery is square-shaped and its one side is approximately 350mm long. The length of the gallery is about 2.5m, and this gallery, mounted on the supporting frame, is angle-adjustable within the range between 0 and 20°. The operation of the heat-resistive electric blower installed inside the scrubber can make suction airflow either ascensionally or horizontally in the model gallery. In this gallery, the belt sample, at most 1.5m long and 10cm wide, can be mounted on the small-scale trestle, and the propane burner is placed at the position of about 50mm from the suction opening of the model gallery during the test.

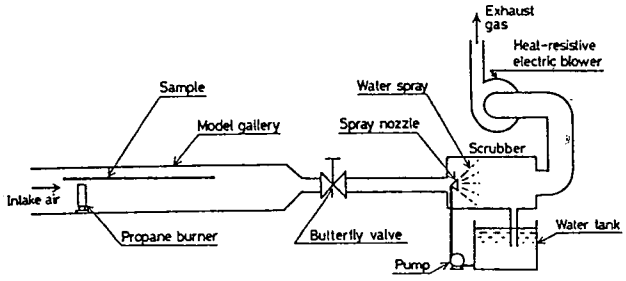


Fig.4

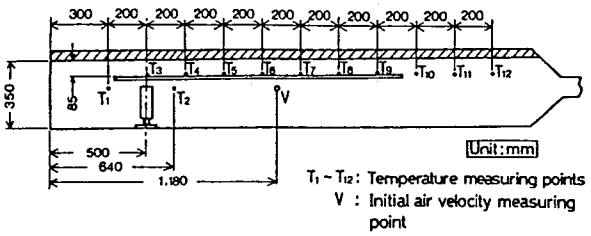


Fig.5

Testing Procedure

Prior to the tests, the measuring points were arranged, as shown in Fig.5. Temperatures were measured at 12 locations in the model gallery. In addition, the sensing device of hot-wire anemometer was inserted into the definite location before igniting the burner, in order to monitor and adjust the airflow speed to about 0.5m/s by regulating the butterfly valve. Subsequently this sensing device was pulled out of the model gallery and the burner was ignited.

The belt samples, about 1500mm long and 90mm wide, were tested, exposed to the flame from the propane burner. During the test, the flow rate of the propane gas was regulated to about $2.5l/min$, and the flame was applied to the belt sample for about 15 minutes. The temperatures were recorded until the flame on the belt sample desappeared.

Test Results

Every belt sample tested continued to flame after its ignition, and there was no intact part of the sample after the test. Time necessary for flaming of the belt sample (t_{ign}) and flame propagation speed (V_f) on each sample were shown in Tables 6 and 7. Table 6 shows the data under horizontal ventilation condition and Table 7 shows those under ascensional one.

In these tables, t_{ign} is defined as the elapsed time from ignition of the burner until T_4 comes up to 300°C, and V_f is, moreover, determined from the elapsed time until T_9 comes up to 300°C.

From these results, it is found that t_{ign} comes to be longer under ascensional ventilation condition than under horizontal one, but that V_f comes to be smaller. This must be caused by the larger thermal diffusivity in the model gallery in case of ascensional ventilation, where buoyancy caused by the combustion gas from the burner and belt sample increases the airflow speed.

Table 6 Results of the laboratory-scale gallery test on conveyor belts under horizontal ventilation condition.

Sample	Thickness (mm)	Ambient temperature (°C)	Relative humidity (%)	t_{ign} (min)	V_f (m/min)
A	15.4	14	67	6.0	0.103
B	13.6	14	37	7.1	0.096
C	12.8	11	63	3.9	0.127
D	12.4	15	48	3.7	0.125
E	13.6	17	39	4.6	0.110
F	12.8	17	51	4.2	0.096

Table 7 Results of the laboratory-scale gallery test on conveyor belts under ascensional ventilation condition (Inclination:20°).

Sample	Thickness (mm)	Ambient temperature (°C)	Relative humidity (%)	t_{ign} (min)	V_f (m/min)
A	15.2	16	69	14.8	0.625
B	13.8	14	37	13.5	0.106
C	12.8	13	55	11.5	0.769
D	12.3	14	57	11.2	0.667
E	14.4	14	57	11.4	0.909
F	12.0	15	68	13.9	0.400

DISCUSSIONS

Correlations among the flammability test results obtained were determined and compared one another. In order to quantify and signify the correlations, both sample correlation coefficient (denoted by r) and rank correlation coefficient (denoted by r_s) were calculated in each case.

Figs.6 and 7 show the relationships between COI and the results of small-scale flame test for each 25mm wide belt sample. The calculated values of r and r_s between COI and duration of flame were -0.29 and -0.26 (Fig.6), and those between COI and duration of afterglow were 0.22 and 0.26 (Fig.7) respectively. From these data, it is found that correlation between the results of small-scale flame test and those of oxygen index test is not so clear.

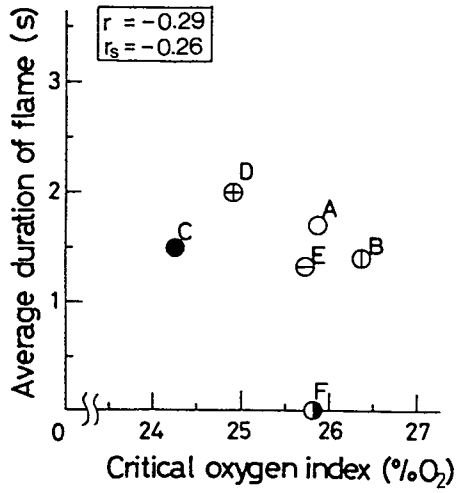


Fig.6

Correlations between COI and the results of laboratory-scale gallery test were also determined. Fig.8 shows the relationships between COI and the results of the laboratoty-scale gallery test under the horizontal ventilation condition, and Fig.9 shows the ones between COI and the similar results under ascensional ventilation condition. According to these calculated results of correlation coefficients (the range of absolute value of r is between 0.67 and 0.94, and that of r_s is between 0.66 and 0.90), it is found that correlations are relatively high between these test results.

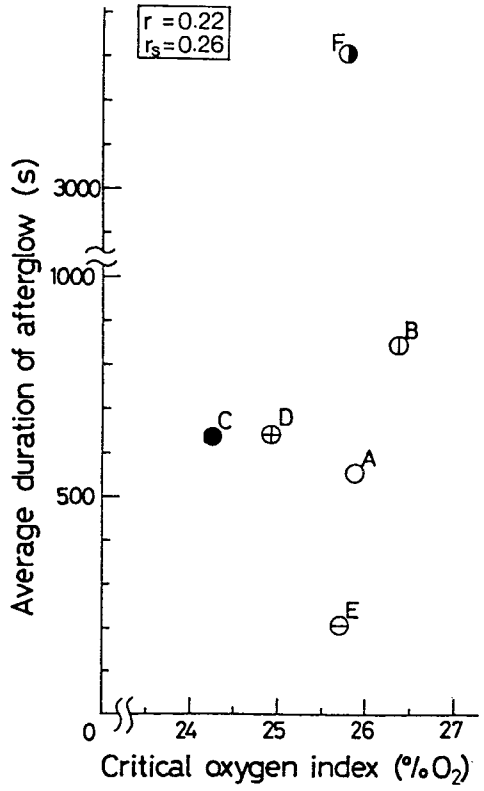


Fig.7

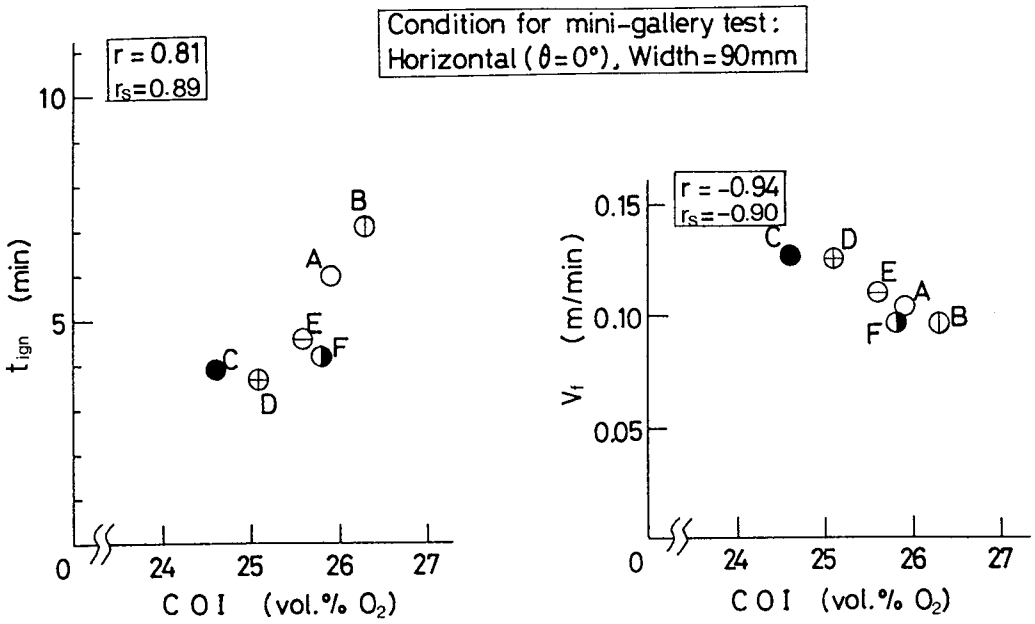

Condition for mini-gallery test:
Horizontal (θ=0°), Width=90mm
r = 0.81
rs=0.89
tign (min)
COI (vol.% O2)
r = -0.94
rs=-0.90
Vf (m/min)
COI (vol.% O2)

Fig.8

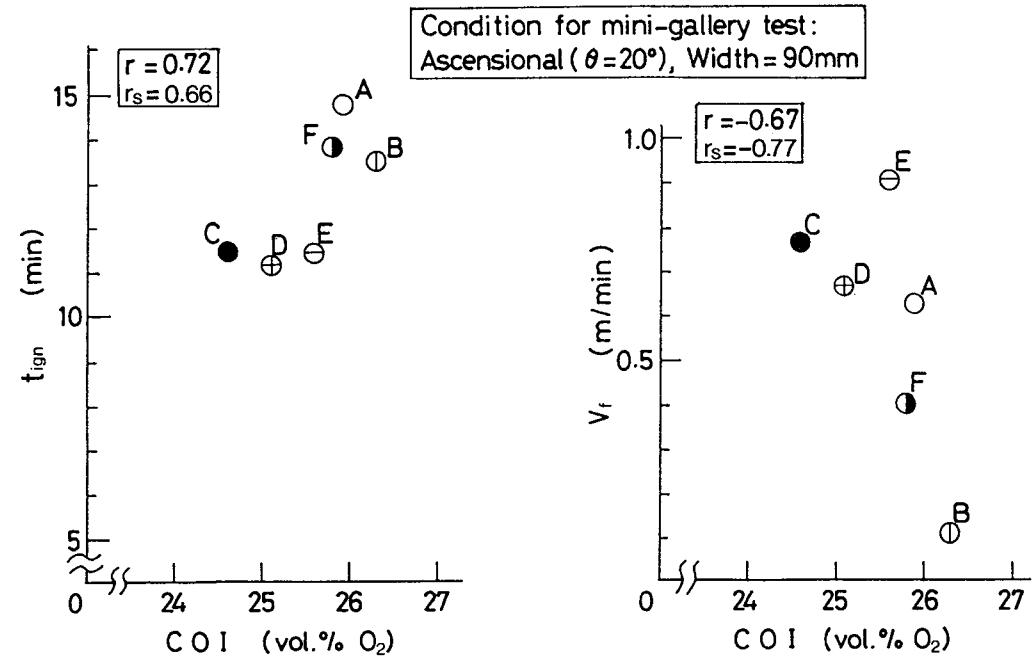

Condition for mini-gallery test:
Ascensional (θ=20°), Width=90mm
r = 0.72
rs=0.66
tign (min)
COI (vol.% O2)
r =-0.67
rs=-0.77
Vf (m/min)
COI (vol.% O2)

Fig.9

CONCLUSIONS

Through the small-scale flame test, it has been made clear that the carcases of "flame-resistant" rubber conveyor belts used in Japan are not always made of "flame-resistant" materials, and that some of the "flame-resistant" rubber belts do not inhibit afterglow so effectively.

The relationship between sample width and COI of the rubber belts was found to be dependent on "flame-resistance" of the fabric carcases, and it was confirmed that adequate sample width for oxygen index test was 25mm, as suggested in Canada.

On the other hand, the results of oxygen index test are not so consistent with those of small-scale flame test. The results on the laboratory-scale gallery test, however, are relatively consistent with COI of the belts.

ACKNOWLEDGEMENT : Data on the oxygen index test were arranged, referring to the proposal presented by Dr. K.J.Mintz, research scientist for Canada Centre for Mineral and Energy Technology, during his visit to our institute. We are greatly thankful to him for his kind instruction. Besides, the belt samples were provided by Mr.T.Komai from the Coal Mine Safety Research Center, Kyushu, and the sample cutting works were implemented by Mr.T.Matsumoto, Mr.M.Iimura, Mr.T.Kizawa and Mr.K.Nakazawa, technicians of Engineering Service Division of our institute. We are also as much grateful to them for their good favour and fulfilment of such laborious works.

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ESSAIS D'INFLAMMABILITE DE COURROIES DE CONVOYEURS EN CAOUTCHOUC A CARCASSE DE TISSU UTILISEES DANS LES MINES SOUTERRAINES DE CHARBON JAPONAISES

Yuichi NAKAGAWA, Masayoshi TAKAHASHI, Shin-ichiro YAMAO, Makoto KOHNO

Des essais de flammes à petite échelle, d'indice d'oxygène et des essais de galeries en laboratoire ont été effectués sur des courroies de convoyeurs en caoutchouc "ignifugé", comportant des carcasses de 5 épaisseurs de tissu, utilisées dans les mines souterraines de charbon japonaises. On a comparé les résultats des essais d'indice d'oxygène à ceux des deux autres essais respectivement, et calculé les coefficients de corrélation. En conclusion, les indices critiques d'oxygène des courroies échantillons concordent relativement bien avec les résultats des essais de galerie effectués en laboratoire, bien que la corrélation entre les résultats des essais d'indice d'oxygène et ceux des essais à petite échelle ne semble pas être aussi claire. Il a en outre été confirmé que l'indice d'oxygène critique s'accroit avec l'augmentation de la largeur de l'échantillon jusqu'à ce que cette largeur atteigne une certaine valeur, qui varie selon que la carcasse de tissu de la courroie est ou non "ignifugée".

FLAMMTESTS MIT GUMMIFÖRDERGURTEN MIT GEWEBEEINLAGEN, DIE IN JAPANISCHEN GRUBEN IM EINSATZ SIND.

Y. Nakagawa, M. Takahashi, S. Yamao, M. Kohno

Kleine Flammtests, Sauerstoff - Index - Tests und ein Test in einer Laborstrecke wurden an 'flammfesten' Gummigurten mit fünf Gewebeeinlagen, wie sie in Japan eingesetzt werden, durchgeführt. Die Ergebnisse des Sauerstoff - Index - Tests wurden mit den Ergebnissen der beiden anderen Tests verglichen, und ein Korrelationsfaktor wurde gebildet. Es zeigte sich, daß die kritischen Sauerstoff - Indexe der Probegurte relativ gut mit den Ergebnissen aus den Untersuchungen auf der Laborstrecke übereinstimmen, daß jedoch die Korrelation der Ergebnisse der Sauerstoff - Indexe mit denen der kleinen Flammentests nicht so ausgeprägt war. Außerdem kann bestätigt werden, daß die kritischen Sauerstoff - Indexe mit der Verbreiterung des Gurtes steigen, bis dieser eine gewisse Breite erreicht hat, die von der Flammfestigkeit der Gewebeeinlage abhängig ist.

ИСПЫТАНИЯ ПОЖАРОСТОЙКОСТИ РЕЗИНОВЫХ КОНВЕЙЕРНЫХ ЛЕНТ, ИЗГОТОВЛЕННЫХ НА ТКАНЕВОЙ ОСНОВЕ, КОТОРЫЕ ИСПОЛЬЗОВАЛИСЬ В УГОЛЬНЫХ ШАХТАХ ЯПОНИИ

Юичи Нагакава, Мазайоши Такахаши, Шин-ичиро Ямао и Макото Коно

Были проведены мелкомасштабные испытания огнем, испытания по кислородному показателю и лабораторные испытания на "огнеупорных" резиновых конвейерных лентах на пятислойной тканевой основе, использующихся в японских подземных угольных шахтах. Результаты испытаний по кислородному показателю были сравнены с результатами двух других испытаний, и были вычислены коэффициенты корреляции. В заключение было определено, что критические кислородные показатели для образцов лент являются относительно сопоставимыми с результатами закрытых лабораторных испытаний. В то же время, корреляция между результатами испытаний по кислородному показателю и испытаний огнем не представляется столь отчетливой. В дополнение было подтверждено, что критический показатель кислорода возрастает с возрастанием ширины образца до определенной величины, которая зависит от огнеупорных качеств тканевой основы.

日本地下煤矿中应用的纤维心橡皮输送机皮带的可燃性试验

中川汤一， 高桥正义
山尾新一郎 及 高野真 （均译音）

小型火焰试验，氧气指数试验和实验室型通道试验均以在日本地下煤矿中应用的五层纤维心的"耐火"橡皮输送机皮带来实施。氧气指数试验的结果用来与其他两种试验结果相比较，以计算它们之间的相关系数。其结果是皮带试验品的临界氧气指数与实验室型通道试验有相当一致的关连，而与小型火焰试验关连的情形不甚明确。此外，临界氧气指数随着试样宽度的增加而增高，直到试样宽度达到某一定值时，这值与皮带的纤维心是否耐火有关。

BRANDSCHUTZ AN UNTERTÄGIGEN GURTBANDFÖRDERERN

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ZUSAMMENFASSUNG

Auf Veranlassung der staatlichen Bergaufsicht der DDR wurde vom Institut für Bergbausicherheit im Zusammenwirken mit der Kaliindustrie ein umfangreiches Programm von Großbrandversuchen an einer untertägigen Bandanlage durchgeführt. Es wurden vor allem der Verlauf derartiger Brände untersucht sowie Verfahren zu ihrer Beherrschung bzw. der Begrenzung ihrer Auswirkungen erprobt.
Die Ergebnisse wurden Bestandteil einer neuen Brandschutzvorschrift der Obersten Bergbehörde beim Ministerrat der DDR. Für die Bewertung der Brandgefährdung und die Festlegung von Brandschutzmaßnahmen an untertägigen Gurtbandförderern wurde eine Anleitung ausgearbeitet.

EINLEITUNG

Im Kalibergbau der DDR sind in den vergangenen Jahrzehnten grundlegende Modernisierungen der angewendeten Bergbautechnologien erfolgt. Diese haben aber auch dazu geführt, daß der Einsatz brennbarer Stoffe vielfältigster Art drastisch zugenommen hat. Beispielsweise sind in den Kaligruben der DDR etwa 300 km Bandanlagen im Einsatz, deren Gummigurte allein ein Brandpotential bis ca. 50 kg/m Streckenlänge darstellen. Damit wurden sowohl das Brandrisiko als auch die möglichen Auswirkungen von Brandhavarien bedeutend erhöht. Schwere Bandbrände haben dies nachdrücklich unterstrichen, so daß der Brandschutz an diesen Anlagen und die Gewährleistung der Selbstrettung der Grubenbelegschaft unter solchen Bedingungen von besonderer bergbausicherheitlicher Relevanz sind.

Auf Veranlassung der staatlichen Bergaufsicht und unter Mitwirkung der Kaliindustrie wurde vom Institut für Bergbausicherheit ein umfangreiches Untersuchungsprogramm zur Klärung der grundlegenden Zusammenhänge derartiger Brände sowie zur Erprobung neuer Methoden und Verfahren zu ihrer Beherrschung bearbeitet.

Erste Grundlagenuntersuchungen (Florschütz 1989) und eine Analyse des internationalen Erkenntnisstandes haben zu der Schlußfolgerung geführt, daß umfassende und reale Erkenntnisse nur aus Untersuchungsbedingungen zu gewinnen sind, die den praktischen Gegebenheiten des untertägigen Grubenbetriebes entsprechen. Deshalb wurde das Programm in der Hauptsache auf der Grundlage einer Serie von Großbrandversuchen in einer Kaligrube realisiert.

Über die wichtigsten Ergebnisse

und Erkenntnisse wird im vorliegenden Beitrag zusammengefaßt berichtet.

VERSUCHSBEDINGUNGEN

Die Großbrandversuche wurden in einer annähernd söhligen und im Steinsalz aufgefahrenen Bandstrecke von 1,5 km Länge und 6 m x 3,5 m = 21 m^2 Querschnitt durchgeführt. Die Bandkonstruktion wurde jeweils mit Ober- und Untergurt belegt, je nach Versuchsinhalt betrug die Gurtbreite 800 oder 1200 mm, und die abgebrannten Bandlängen betrugen bis 258 m. Die Gurte waren des weiteren unterschiedlich in Dicke, Verschleißgrad und Zusammensetzung, so daß auch eine deutlich unterschiedliche Brennwilligkeit vorlag. Die spezifische Brandmasse betrug ca. 22 bzw. 40 kg/m, maximal wurden bei einem Versuch 11,4 t Gurt abgebrannt. In der Bandstrecke wurden Wettergeschwindigkeiten von 1,6 - 2,75 m/s gewählt, woraus sich Wettermengen bis 3500 m^3/min ergaben.
Diese Bedingungen entsprechen weitgehend den realen Praxisverhältnissen, und sie wurden für die verschiedensten Untersuchungsanforderungen differenziert angewendet.

VERLAUF EINES BANDBRANDES

Es wurden folgende, den Verlauf eines Bandbrandes signifikant charakterisierende Ergebnisse erzielt:

- Vor allem in Abhängigkeit von der Wettergeschwindigkeit gerät der Fördergurt bei entsprechendem Zündinitial innerhalb weniger Minuten in Brand und breitet sich sehr schnell aus. Bereits nach 15 Minuten sind bis zu 30 m Bandanlage vom Brand erfaßt, und die mittlere Brandausbreitungsgeschwindigkeit beträgt bei Fortschreiten des Brandes bis 18 m/min.

- Die Intensität und der Verlauf des Brandes werden maßgeblich vom möglichen Brandmasseumsatz bestimmt. Mit dem Erreichen des höchsten Masseumsatzes von über 200 kg/min verringerte sich der O_2-Gehalt auf 3 - 4 Vo.-%, die CO-Konzentration betrug etwa 3 Vol.-% und die CO_2-Konzentration bis 13 Vol.-%. Den Verlauf der ermittelten Gasanteile der Brandwetter zeigt Bild 1.

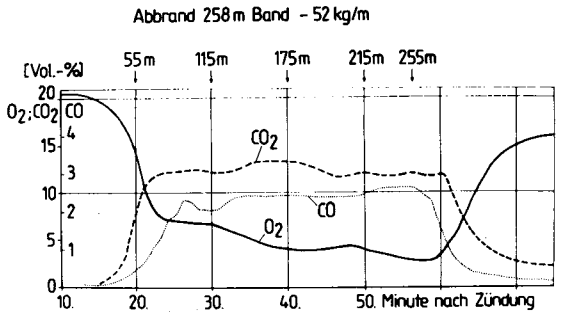


Bild 1

- Die während des Brandes auftretenden Temperaturen betragen unmittelbar im Brandherd mehr als 1200 °C und im Mittel über den gesamten Grubenbauquerschnitt bis zu ca. 850 °C.
Abwetterseitig tritt ein starker Temperaturabfall ein. 20 m vom Brandende sinkt die Temperatur auf ca. 500 °C und in 350 m Entfernung beträgt sie nur noch 80 °C (Bild 2) und verringert sich hier innerhalb von 3 Stunden auf 50 °C.

- Die höchste Brandintensität und die stabile Brandphase werden erst bei einer Bandlänge von etwa 200 m erreicht. Es bildet sich in Abhängigkeit von der zugeführten Wettermenge eine begrenzte Vollbrandzone aus, z.B. 80 m, die sich im angegebenen Geschwindigkeitsbereich ausbreitet.

- In Abhängigkeit von der Bewetterung kann sich der Brand auf große Längen ausdehnen, unter Umständen auf mehrere Kilometer.

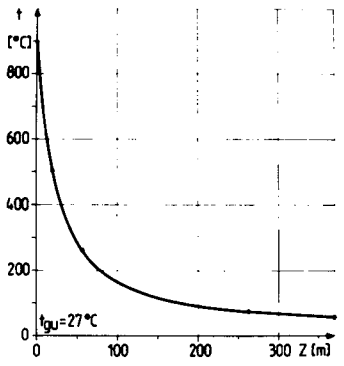


Bild 2

BRANDBEKÄMPFUNG

Der Brandverlauf bestätigt, daß bei einem bereits entwickelten Bandbrand eine direkte Brandbekämpfung mit herkömmlichen Mitteln und Methoden nicht möglich ist.

Zunächst wurden ein neues Verfahren zur Brandverzögerung durch eingebaute Sollbruchstellen sowie eine speziell für Bandanlagen konzipierte Wasserlöschanlage erprobt.
In Auswertung von untertägigen Bandbränden und aus den Ergebnissen der Versuche zum Brandverlauf wurde die Schlußfolgerung gezogen, daß ein Bandbrand auf einen bestimmten Abschnitt begrenzt und in relativ kurzer Zeit zum Erliegen gebracht werden kann, wenn es gelingt, den Wetterstrom und damit die Sauerstoffzufuhr sehr schnell und in hinreichendem Umfang zu drosseln. Hierfür wurde eine selbstauslösende Wettersperre entwickelt und ihre Wirksamkeit ebenfalls unter den angegebenen Bedingungen erprobt.
Zu diesen Verfahren ist im wesentlichen folgendes mitzuteilen:

1. Sollbruchstellen

Unter Sollbruchstellen sind thermisch instabile Konstruktionselemente zu verstehen, die aus miteinander verklebten Stahlhülsen oder Laschenpaaren bestehen (Bild 3) (Florschütz 1984). Sie werden in die Tragelemente der an der Firste aufgehängten Gurtbandförderer eingebaut. Die Instabilität wird durch das Schmelzen eines organischen Klebers oder von Weichlot bei Temperaturen von ca. 100 °C erreicht.

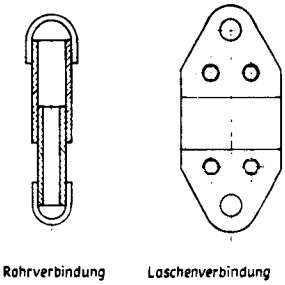


Bild 3

Durch Ansprechen der Sollbruchstellen fällt die Bandkonstruktion auf die Sohle, wodurch die Brandintensität erheblich sinkt. Bei 180 m Versuchsbandlänge, in die ab 50 m Sollbruchstellen in die Tragketten eingebaut waren, wurden folgende Ergebnisse erzielt:

- Die Brandausbreitungsgeschwindigkeit betrug bei einer Wettergeschwindigkeit von 1,8 m/s nur 0,4 m/min gegenüber 18 m/min ohne diesen Eingriff.

- Durch den wesentlich verringerten Brandmasseumsatz sanken die Temperaturen der Brandwetter erheblich und lagen deutlich unter den Werten eines Vollbrandes. Damit waren auch die thermischen Auswirkungen

auf den Grubenbau bedeutend geringer, 4 m vom Brand entfernte Kabel blieben unversehrt.

- Die Rauchentwicklung war stark herabgesetzt, so daß sich deutlich verbesserte Bedingungen für die Selbstrettung ergeben.

- Die Länge der mit Sollbruchstellen auszurüstenden Bandanlage ist in Abhängigkeit von der Wettergeschwindigkeit festzulegen. Im Bereich von 0,25 - 2 m/s soll die Länge 50 - 150 m betragen.

Mit diesen erreichbaren Effekten sind die Sollbruchstellen vor allem im Zusammenwirken mit anderen Maßnahmen zu nutzen.

2. Wasserlöschanlage

Es wurde eine Wasserlöschanlage (Bild 4) entwickelt und erprobt, die speziell auf Grubenbedingungen ohne zentrales Leitungsnetz zugeschnitten ist.

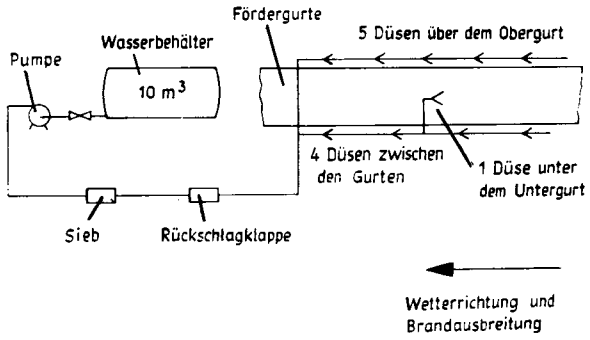


Bild 4

Sie besteht aus 2 Stahlrohren mit insgesamt 10 Düsen und setzt die Bandanlage auf 10 m Länge einem dichten und intensiven Wassersprüh aus.
Weitere Daten:
. Wasserdurchsatz 250 l/min
. Wasserdruck 0,6 - 0,8 MPa
. Wasservorrat 10 m³
. Rohrdurchmesser NW 50
. Automatische Auslösung durch Ansprechen von Auslöseschlauch oder Schmelzlot, wenn sich der Brand auf 5 m an die Löschanlage genähert hat.

Die Löschanlage ist für folgende Bedingungen in söhligen Bandstrecken erfolgreich erprobt worden und einsetzbar:
. Bandbreite bis 800 mm,
. Wettergeschwindigkeit bis 2 m/s,
. Grubenbauquerschnitt ca. 20 m², Streckenhöhe > 3 m,
. Abstand zwischen Bandanlage und Firste mindestens 2 m,
. Kabelverlegung am gegenüberliegenden Stoß.

Durch die Anwendung von Sollbruchstellen frischwetterseitig vor der Löschanlage kann der Löscherfolg gesichert bzw. der Einsatzbereich erweitert werden.

WETTERSPERRE

Die selbstauslösende Wettersperre stellt eine neuartige und besonders wirkungsvolle Lösung zur Beherrschung von Bandbränden dar (Schmidt/Martin/Hoffmann 1987). Ein Ausführungsbeispiel zeigt Bild 5. Am Anfang des Halteseiles von ca. 70 m Länge ist ein thermisches Auslöseelement angeordnet, welches bei einer Temperatur von 175 °C anspricht und die Sperrenklappe in Wirkstellung bringt. Zum Zeitpunkt der Auslösung hat der Brand einen Abstand zur Sperre von 150 m, so daß diese nicht unzulässig thermisch beansprucht wird.

Mit dem Schließen der Sperre kommen folgende entscheidende Faktoren zur Wirkung:

- starke Drosselung des Wetterstromes und entsprechende Reduzierung der Sauerstoffzufuhr,

- Wegfall des Anfacheffektes durch die gegen Null gehende Wetterbewegung und die starke

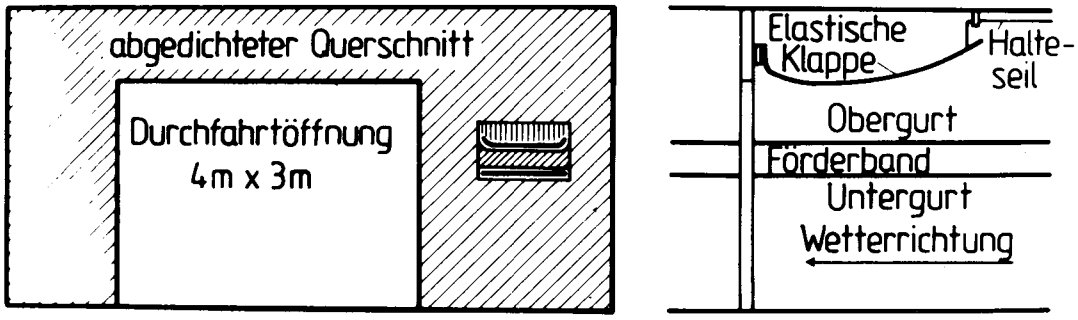


Bild 5

Verminderung der Turbulenzen in der Brandzone,

- schnelle Verringerung der Brandintensität sowie
- deutliche Herarbsetzung der thermischen Aufbereitung des noch nicht brennenden Fördergurtes.

Bei Versuchsbränden an einer 200 m langen Bandanlage (Bild 6) gelang es die Brände zum Verlöschen zu bringen, wobei nur 35 - 45 m Band abgebrannt sind.

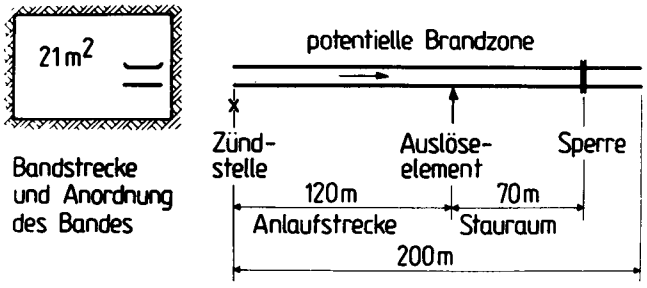


Bild 6

Für die Wirksamkeit der Wettersperre wurden für zunächst söhlige Grubenbaue von ca. 20 m² Querschnitt folgende Kriterien ermittelt:

- Der Wetterstrom muß so stark gedrosselt werden, daß sich die Wettergeschwindigkeit auf

$$v_{Sp} < 0{,}25 \text{ m/s}$$

verringert, wobei die an der Sperre auftretende Druckdifferenz 200 Pa nicht überschreiten soll. Die erforderliche Dichtigkeit der Sperre ergibt sich aus dem Drosseldiagramm in Bild 7 mit $m < 0{,}024$.
Der Wetterstrom wird damit auf < 300 m³/min gedrosselt.

- Die Anwendung ist auch bei größeren Grubenbauquerschnitten möglich, z.B. 30 und 40 m². In diesen Fällen ist der gedrosselte Wetterstrom durch geringere Druckdifferenzen an der Sperre und durch eine höhere Dichtigkeit zu begrenzen. Zum Beispiel bei 30 m² Querschnitt auf $\Delta p = 150$ Pa, $m = 0{,}02$ und bei 40 m² Querschnitt auf $\Delta p = 100$ Pa, $m = 0{,}016$ entsprechend Bild 7. Die angegebenen Druckdifferenzen und Dichtigkeiten wurden an bereits errichteten Wettersperren ohne weiteres erreicht.
- Der Abstand zwischen Wetter-

sperre und Auslöseelement soll mindestens 70 m und zu frischwetterseitigen Verbindungen mit parallelen Wetterwegen mindestens 150 m betragen. Dazwischenliegende Wetterverbindungen müssen abgedichtet werden.

- Ob die Wettersperre die angegebenen Kriterien erfüllt, ist durch Kaltversuche zu überprüfen.

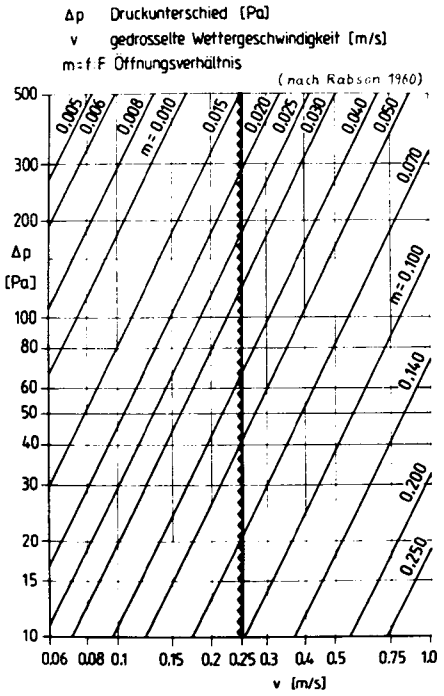


Bild 7

Unter Bewetterungsbedingungen, bei denen an der Sperre sehr hohe Depressionen von Δp = 1000 Pa aufgetreten sind, hat sich der zunächst gestoppte Brand mit geringer Intensität und Ausbreitungsgeschwindigkeit fortgesetzt und führte zum Durchbrennen der Sperre.

Der Löscherfolg einer Wettersperre kann zusätzlich gesichert werden, wenn frischwetterseitig ein Schnelldamm errichtet oder durch Eingriff in die Wetterführung die Wetterzuführung zur Brandstrecke unterbunden wird.

Weitere Arbeiten sind darauf gerichtet, die Anwendung der Wettersperren für geneigte Grubenbaue mit einfallender und ansteigender Bewetterung und die Auslösung der Sperren noch universeller und über große Bandlängen zu ermöglichen.

AUSBLICK

Die gewonnenen Ergebnisse und erprobten Brandschutzmaßnahmen wurden Bestandteil einer neuen Brandschutzvorschrift der Obersten Bergbehörde beim Ministerrat der DDR. Diese trat am 1. 4. 1989 in Kraft.
Als Anleitung für die Bergbaubetriebe wurde ein Programm für die Bewertung der Brandgefährdung und zur Festlegung von Brandschutzmaßnahmen an untertägigen Gurtbandförderern entwickelt (Bild 8).

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Programm für die Bewertung der Brandgefährdung und zur Festlegung von Brandschutzmaßnahmen an untertägigen Gurtbandförderern

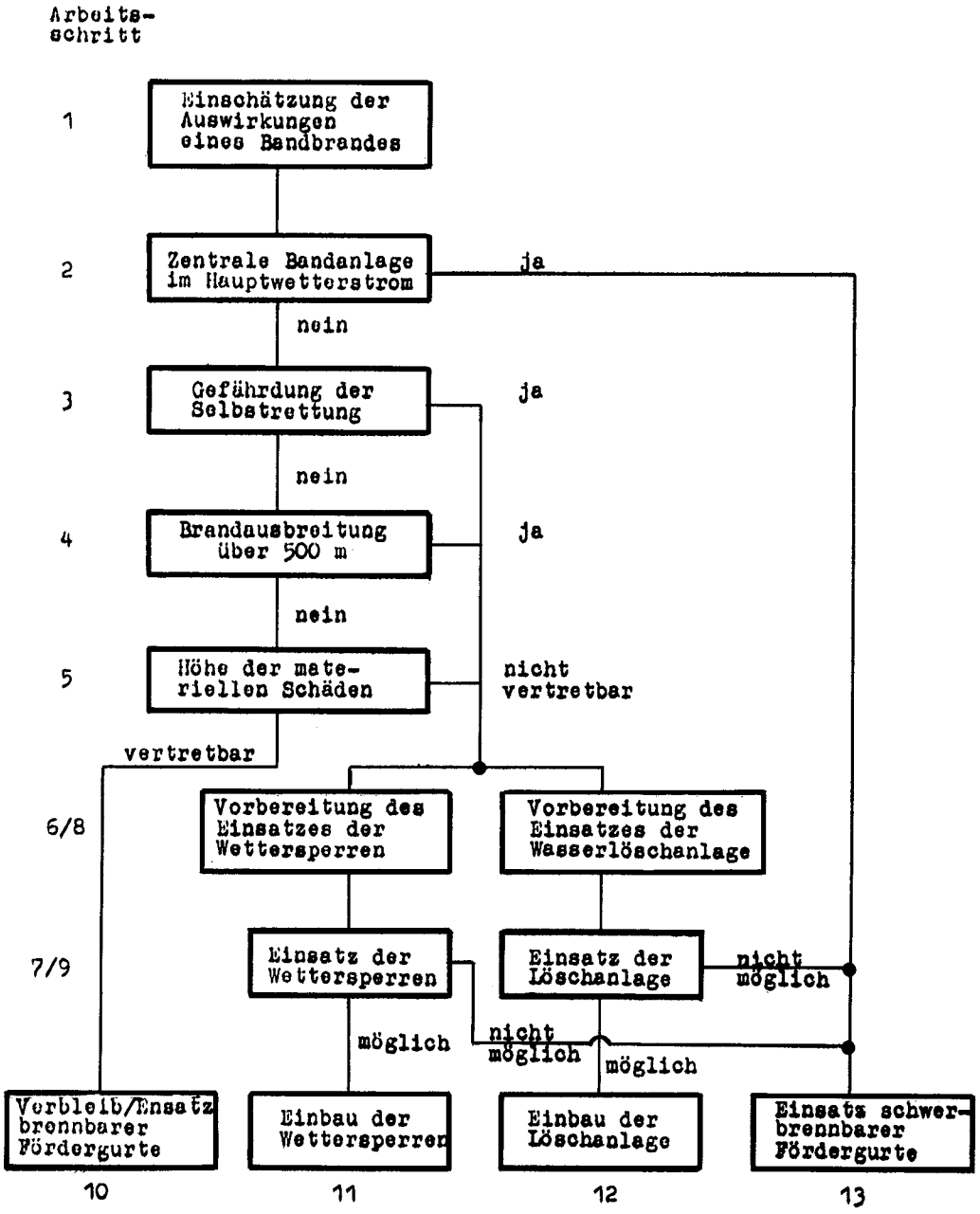


Bild 8

FIRE PROTECTION OF UNDERGROUND BELT CONVEYORS

P. Florschutz, R. Hoffmann, K. Haisler

During the last years, considerable use has been made of belt conveyors in the GDR as a very effective haulage method in underground mines. However, the advantages of this technology are counteracted by the increased fire hazard introduced by this method. At the Institut fur Bergbausicherheit (Institute of Safety-in-Mines) investigations were therefore made on the ignition and propagation of belt fires as a function of conditions such as environmental temperature, belt troughing, initial ignition intensity, and air velocity. The tests showed that in 5 minutes after ignition, full combustion is reached and rubber mass transformations of 200 kg/minute at a maximum occur. Fire propagation velocities of more than 10 m/minute were recorded. Quenching tests were successfully made with specified fracture points for the reduction of flame velocity, and with ventilation barriers and automatically triggered water-extinguishing equipment. Based on numerous tests, a program was established aimed at determining the fire hazard of belt conveyors and at specifying the necessary fire protection measures.

PROTECTION ANTI-INCENDIE DES CONVOYEURS A BANDES MINIERS

P. Florshutz, R. Hoffman, K. Haisler

Au cours des dernières années, on a utilisé de manière extensive les convoyeurs à bandes dans les mines souterraines de la RDA, car ils constituent un moyen de chargement très efficace. Aux avantages de cette technologie correspondent toutefois une augmentation des risques d'incendie que présente cette méthode. En conséquence, l'Institut fur Bergbausicherheit (Institut de la Sécurité Minière) a effectué des recherches sur la combustion et la propagation des incendies de bandes en fonction de certaines conditions telles que la température de l'environnement, les rigoles des bandes, l'intensité de la combustion initiale et la vélocité de l'air. Ces essais ont permis de démontrer que la combustion atteint son maximum 5 mn après le début d'un incendie; des modifications de la masse de caoutchouc se produisent également à 200 kg/mn maximum. On a enregistré des vélocités de propagation de flammes de plus de 10 m/mn. Des essais de refroidissement à points de fractures spécifiques ont été effectués pour diminuer la vélocité des flammes, ainsi que des essais de barrages de ventilation et de systèmes d'extinction d'eau à déclenchement automatique; ces essais ont réussi. Puis, un programme a été développé sur la base de nombreux essais pour tenter de détecter les risques d'incendie des convoyeurs à bandes et pour spécifier quelles sont les mesures de protection anti-incendie nécessaires à prendre.

ПРОТИВОПОЖАРНАЯ ЗАЩИТА ЛЕНТОЧНЫХ КОНВЕЙЕРОВ

П. Флоршутц, Р. Хоффманн и К. Хайзлер

В течение последних лет относительно широкое распространение в ГДР получили ленточные конвейеры, которые являются эффективным методом транспортировки в условиях подземных шахт. Применение ленточных конвейеров, однако, увеличивает опасность пожара. Институтом безопасности в шахтах проведены исследования по возгоранию конвейерных лент и распространению огня в зависимости от таких факторов, как температура воздуха, прогибание ленты, интенсивность начального очага возгорания и скорость потока воздуха. Испытания показали, что через 5 минут после возгорания происходит полное сгорание и максимальная трансформация резиновой массы в 200 кг/мин. Скорость распространения огня достигает 10 м/мин. Успешно проведены испытания по тушению пожара с определенными разбивающими точками, которые снижают скорость распространения пламени; с использованием вентиляционных барьеров и автоматически включающихся водяных огнетушителей. На основе многочисленных испытаний была внедрена программа, направленная на определение огнеопасности ленточных конвейеров. Эта программа также определяет соответствующие противопожарные меры.

地下皮带输送机火灾保护

P．弗洛舒兹， R．霍夫曼， K．海士勒

最近这几年民主德国地下煤矿曾充分利用皮带输送机作为非常有效的运输工具，但这种设备的优点也因为带来了增加火灾危害的可能性而抹刹掉一部分．在矿业安全研究所我们对于皮带火灾的起始和传播以环境温度，皮带凹槽，最初着火强度，及空气速度作为因素作了研究．研究结果指出在着火五分钟后，燃烧便达到充分的程度，而且橡皮变质达到每分钟２００公斤的最高速度．火焰传播速度曾超过每分钟１０公尺．我们用在固定的破裂点减低火焰速度及通风障碍和自动作用的水式灭火设备来做的熄灭实验都很成功．根据很多实验结果我们建立了一项方案以测定皮带输送机的火灾危害和规定必需的防火设备．

FLAME SPREAD ALONG HORIZONTAL CONVEYOR SYSTEMS

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ABSTRACT:

Fire tests of varying scales have long been used to screen materials for use in hazardous environments such as an underground mine. Generally the test methods do not allow quantification of the hazard and cannot be applied to scales of fire larger than the test. A general approach is outlined which enables wind assisted fire spread and other fire hazards along conveyor-like structures to be predicted. The basic approach is to assess decoupled fire behaviour on an instrument like the NBS Cone Calorimeter and use this quantitative data in theoretical models of fire spread. These models have to be validated against full scale tests. Examples of full scale fire tests are given for a horizontal configuration at various ventilation rates and compared with data from the NBS Cone Calorimeter.

INTRODUCTION.

The development of quantitative models of the underground fire phenomenon will enable colliery managers the opportunity to safely control the mine environment in a way not previously thought possible; the risk and consequences of fire can be assessed prior to the event for any area of the mine from a knowledge of the geometry and the materials in that area; the siting and frequency of detectors can be tailored to suit any mine; the most effective methods of control of a fire can be established before the event or in real-time; the safest egress routes for a given fire can be estimated.

Although the ability to achieve this end depends on the development of faster computers than currently available, the success of such models relies on the prediction of flame spread along one or more material surfaces. Recent developments in fire dynamics have got some way to predicting flame spread under natural convection on vertical surfaces (Quintiere 1988, Delichatsios 1988). This paper proposes a method for predicting flame spread under wind assisted conditions over horizontal surfaces from small scale tests like the NBS Cone Calorimeter (Babrauskas, 1982) or NBS Lift apparatus (Quintiere et al, 1986).

CONE CALORIMETRY.

The NBS Cone Calorimeter was developed as a test method that would give quantitative data on material behaviour under different fire conditions (Babrauskas,1982). Typically the heat release rate, mass loss rate, gas species, smoke densities (all measured as a function of time), the time to ignition and the soot yield are measured at a number of incident heat fluxes. The important parameters as far as flame spread predictions are concerned are the time to ignition, the heat release rate and the mass loss rate.

According to transient heat conduction theory for a semi-infinite solid, the time to ignition can be related to the incident heat flux (Quintiere,1988) by

$$(\tau_{ig}-1) = B\,[\,1 - \exp(t_{ig}).\mathrm{erfc}(t_{ig}^{1/2})\,] \qquad 1)$$

where $\tau_{ig} = T_{ig} / T_o$, $B = q''_e / hT_o$ and $t_{ig} = h^2 t_{ig} /kpc$, T_{ig} is the surface ignition temperature, q''_e,

is the incident surface heat flux, h is the surface heat transfer coefficient, k is the thermal conductivity, p is the density and c is the thermal heat capacity of the solid, t_{ig} is the time to ignition.

Equation 1 predicts a critical value of q"e (q"crit) below which the material will not ignite. The time to ignition over a range of incident heat fluxes allows T_{ig}, h, kpc and q"crit to be evaluated. Table 1 compares these values for PMMA, particle board and some conveyor belt materials evaluated by two methods. One method is that used by Quintiere (Quintiere,1988) while the other is being developed by the authors. The latter method utilizes the expansion of Equ. 1 as a power series in the limit of t tending to infinity.

$$1/q''_e = a_o + a_1 t^{-1/2} + a_2 t^{-3/2} + a_3 t^{-5/2} + \ldots \quad 2)$$

where $a_o = 1/ q''_{crit} = 1/ h(T_{ig}-T_o)$, $a_1 = 2h/(\pi kpc)^{1/2}$ and $h = h_c + \sigma\varepsilon(T_{ig}^4 - T_o^4)/(T_{ig}-T_o)$. $h_c = 0.0115 \pm 0.0045$ $kWm^{-2}K^{-1}$ has been measured for the centre's Cone Calorimeter, σ is the Stefan-Boltzmann constant and ε the surface emissivity.

This latter method gives lower values of q''_{crit} and kpc compared with the standard method. These q''_{crit} values are in good agreement with Cone Calorimeter experiments in which the time to ignition was measured over periods of an hour, rather than the standard 10 minutes (Babrauskas,1982). The kpc values seem low although not excessively so (Drysdale,1985), and this may be the value at this limiting heat flux or could be due to strong coupling between the T_{ig} and kpc values in the analysis.

Table 2 gives the values of the heat of gasification, L, and heat of combustion, H_c, for some materials. These were obtained from correlating the mass loss rate, m", with incident heat flux, q"e, and total heat release rate, Q";

$$q''_e = Lm'' + (q''_{rr} - q''_{fl}) \text{ and } Q'' = H_c m'' \quad 3)$$

q''_{rr} is the heat flux reradiated from the surface and q''_{fl} is the heat flux at the surface due to the flame.

Generally the heat release rate is not constant with time. In Fig. 1 the heat release rate for

Material	q''_{crit} kWm^{-2}		T_{ig} ^oC		kp $kW^2m^{-4}K^{-2}s$	
	1	2	1	2	1	2
PMMA (black)	18.8	11.9	438	352	0.43	0.14
PMMA (clear)	15.7	11.0	403	338	0.54	0.13
Particle Board	22.0	12.0	470	353	0.46	0.14
Partical Board[a]	18.1	11.7	431	348	0.52	0.13
Conveyor A	24.5	16.4	493	411	0.75	0.13
Conveyor D	27.0	16.0	514	406	0.96	0.22
Conveyor E	26.2	13.8	508	378	0.34	0.1
Conveyor F	23.7	10.4	486	328	1.08	0.43
Conveyor G	28.4	13.1	526	368	0.29	0.09

1) Quintiere's method
2) Authors' method
a) Conditioned for 2 weeks at 20 ^oC,10%RH

Table 1 Ignition Properties of some Materials

Material	L	H_c	$q''_{rr} - q''_{fl}$
	MJ kg^{-1}	MJ kg^{-1}	kWm^{-2}
PMMA (Black)	2.22	27.1	-12
PMMA (Clear)	2.79	28.4	-18
Particle Board	5.98	12.3	-24
Particle Board [a]	7.20	12.6	-39

a) Conditioned for 2 weeks at 20°C, 10%RH

Table 2 Heat of Gasification and Combustion.

PMMA is shown for a number of incident heat fluxes. This material is well behaved with the surface temperature remaining constant until the thermal wave reaches the back surface of the sample after which the heat release rate starts to increase. In Fig. 2, the heat release rate for particle board is shown. This material chars. After the initial surface pyrolysis, the heat release rate declines from its peak value as the surface char layer builds up. The rate remains constant until a) the thermal wave hits the back surface and b) the char layer burns off exposing unburned material. Both these effects result in an increased rate of heat release. As a consequence, care must be taken in evaluating the average values for the materials.

SINGLE SURFACE SPREAD.

Measurements of horizontal single surface flame spread on PMMA under wind assisted conditions have been undertaken using the Londonderry Occupational Safety Centre fire gallery. These experiments (Apte et al 1989) have demonstrated the existence of the two modes of fire spread sketched in Fig. 3. In the first mode (Fig. 3a), the flame is confined within a boundary layer. The second mode (Fig. 3b) occurs

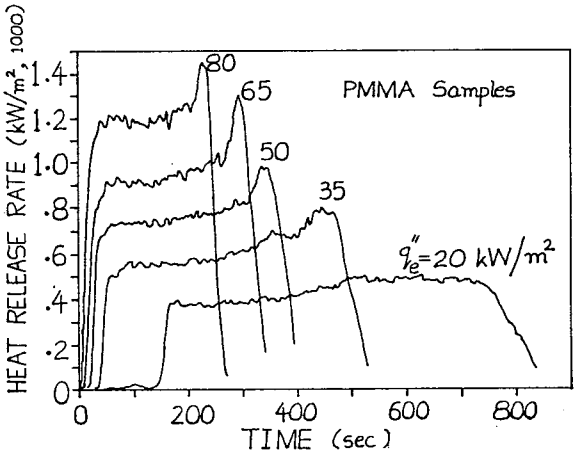


Fig. 1 PMMA Heat Release Rates.

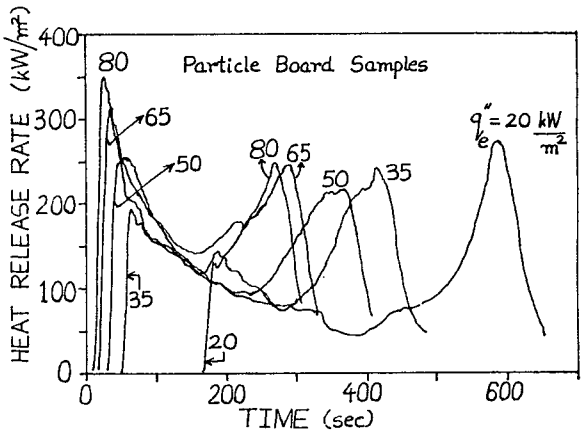


Fig. 2 Particle Board Heat Release Rates.

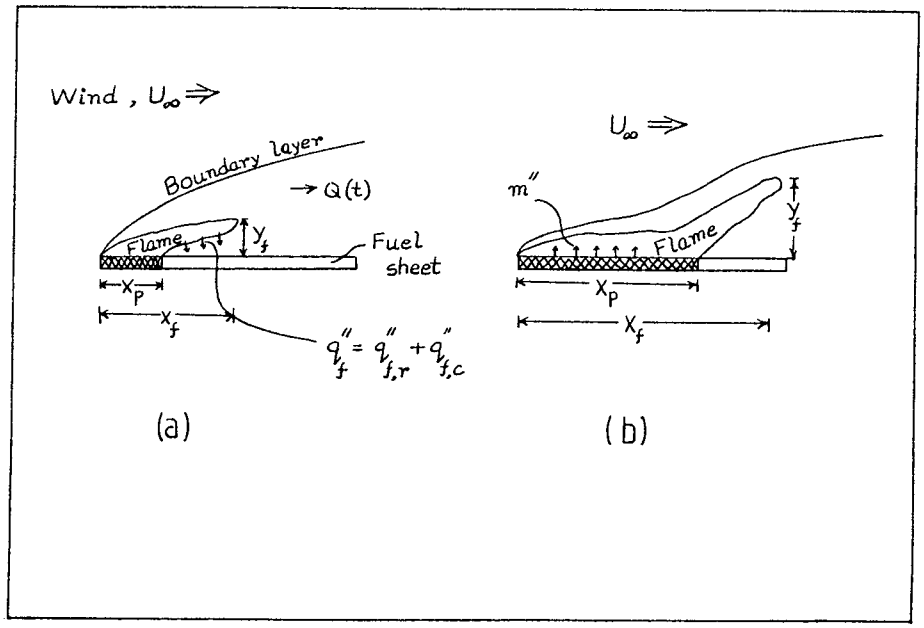


Fig. 3 Flame Spread over a Single Surface.

when the flame has sufficient buoyancy and blowing from the pyrolysing surface to break out of the boundary layer.

Fire spread occurs due to the preheating of the fuel surface ahead of the ignition front by radiative and convective heat transfer from the flame. An expression for the flame spread velocity, defined here as the pyrolysis front advancement rate, has been derived in the following form (Quintiere,1988)

$$V_p = (x_f - x_p)(q''_f)^2 / (\pi/4)kpc(T_{ig} - T_o)^2 \qquad 4)$$

where $(x_f - x_p)$ is the unburnt fuel surface heated by the flame, q''_f is the net incident surface heat flux from the flame.

The burning rate is controlled by the net surface heat flux available for fuel vapourization. The principle heat flux components to and from the surface can be equated as follows:

$$q''_f = q''_{f,r} + q''_{f,c} = q''_{r,r} + m''L \qquad 5)$$

to the surface / from the surface

Conduction and radiation into the interior of the fuel and the reflection of the flame radiation from the surface have been neglected here. The net surface flux from the flame for PMMA is then

$$q''_f = 10.3 + 1650m'' \qquad 6)$$

as the latent heat of gasification for PMMA is 1650 $kJkg^{-1}$ (Jackson,1988).

The flame radiative flux, $q''_{f,r}$, can be evaluated to a first approximation from

$$q''_{f,r} = Q\chi_r / 2x_{fw} \qquad 7)$$

where Q is the total rate of heat release, χ_r it's radiant fraction, w is the fuel sheet width and x_{fw} is the flame area projected normal to the surface. $Q\chi_r$ is therefore the rate of radiant heat emanating from a total flame surface are of $2x_{fw}$.

The convective component, $q''_{f,c}$ is calculated from Equ. 5. by difference.

The measured flame length, x_f, and flame thickness, y_f, as shown in Fig. 4, increase approximately linearly with the pyrolysis front position, x_p. The flame length position is independent of wind speed in the first regime (1.2m)

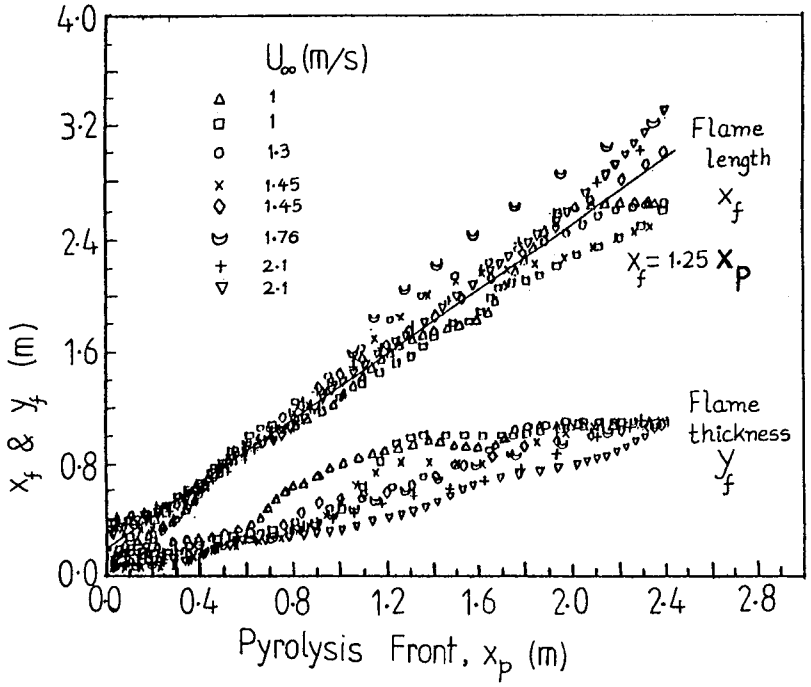


Fig. 4 Measured Flame Length and Thickness.

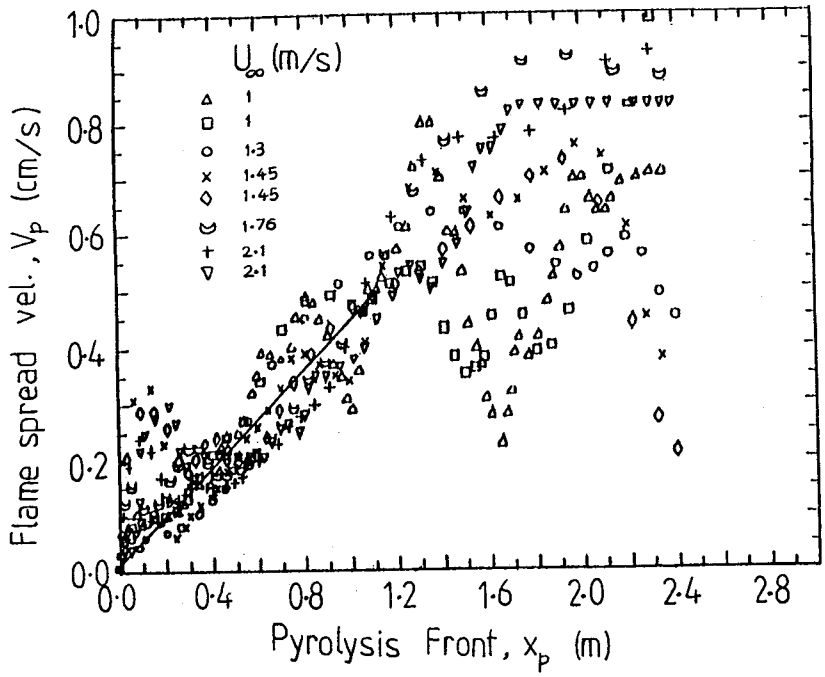


Fig. 5 Measured Flame Spread Velocity.

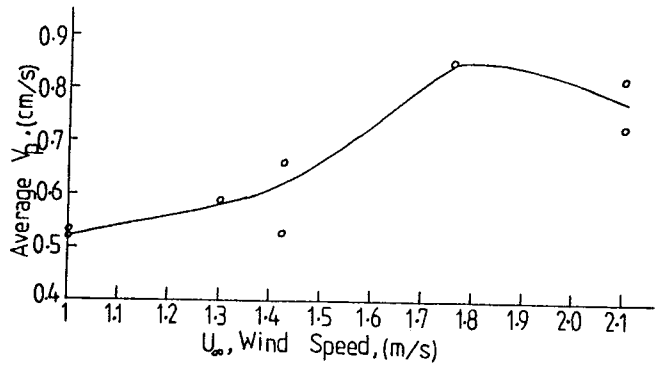


Fig. 6 Average Flame Spread Rates.

but is slightly dependent on wind speed in the second regime.

The measured flame spread velocity, V_p, is shown in Fig. 5 as a function of the pyrolysis length, x_p. V_p increases linearly within the first regime. For x_p>1.2m, V_p appears to be higher for larger wind velocities. This is clearly seen in Fig. 6 where V_p has been averaged from x_p= 1.2 to 2.4m. V_p reaches a maximum of 0.0085ms^{-1} at a wind velocity of 1.76 ms^{-1}.

The heat release flux and pyrolysis flux are shown in Fig.7 and show similar correlations with the wind velocity in this regime. In Fig. 8 the total net surface heat flux given by Equ. 6 is shown together with the radiant and convective fractions. In the boundary layer regime the radiant fraction is about 0.45, but increases in the second regime towards 1.0 when the full PMMA sheet is burning.

The measured and predicted flame front velocities are shown in Fig. 9 as a function of pyrolysis length, x_p. Two values of kpc are used. The boundary layer regime is well predicted by a kpc value of 0.25 kW^{2}m^{-4}K^{-2}s but is under predicted for a value of 1.02 kW^{2}m^{-4}K^{-2}s (Quintire,1988). The kpc value of 0.25 is nearer to the 0.13 obtained by the author's method than the 0.54 value obtained by Quitiere's method from the cone calorimeter data (Table 2). The former value overpredicts flame spread while the latter value underpredicts it.

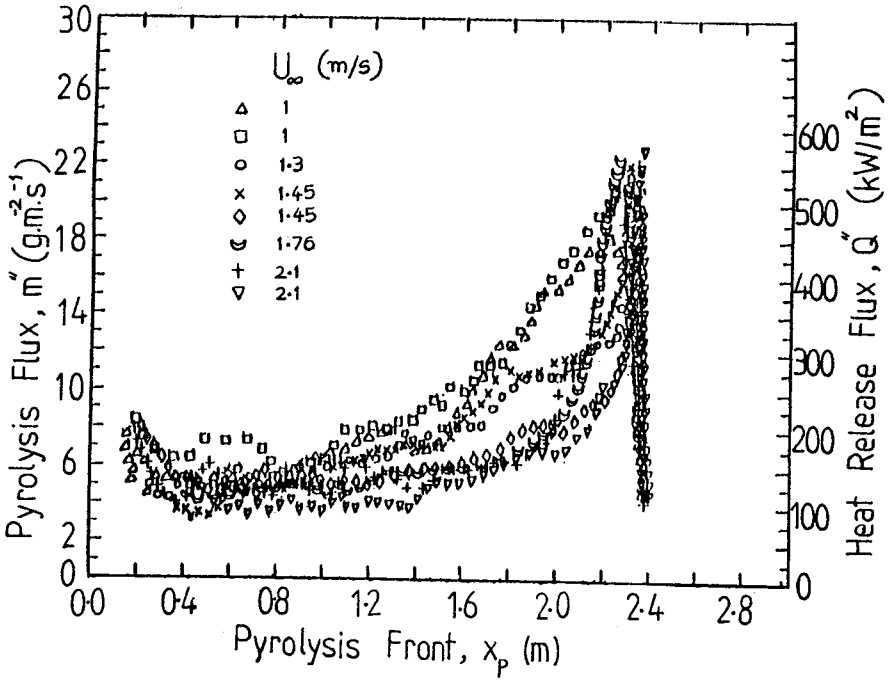


Fig. 7 Profiles of Pyrolysis and Heat Release Fluxes.

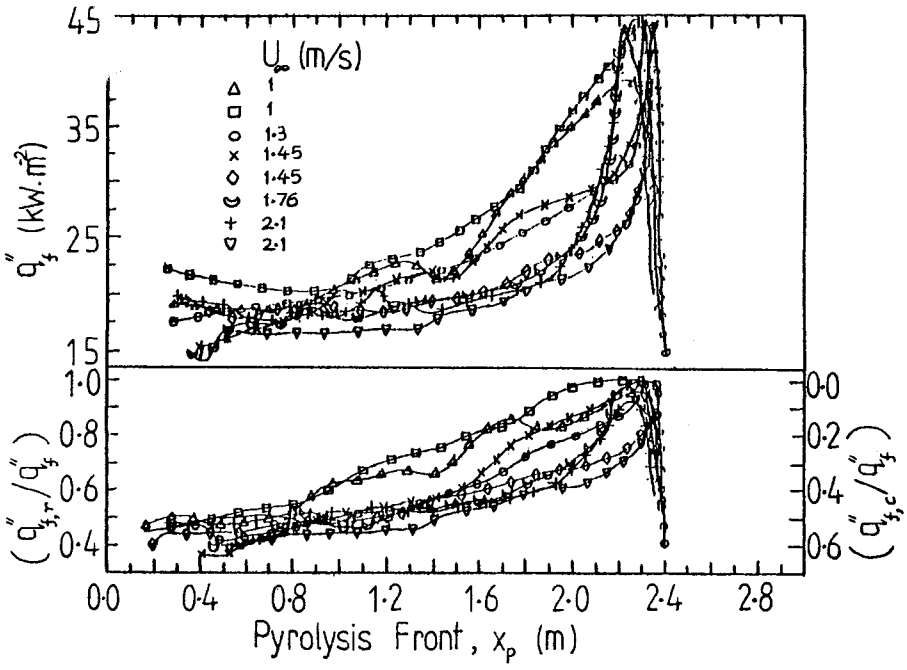


Fig. 8 Estimated Flame-Surface Heat Fluxes.

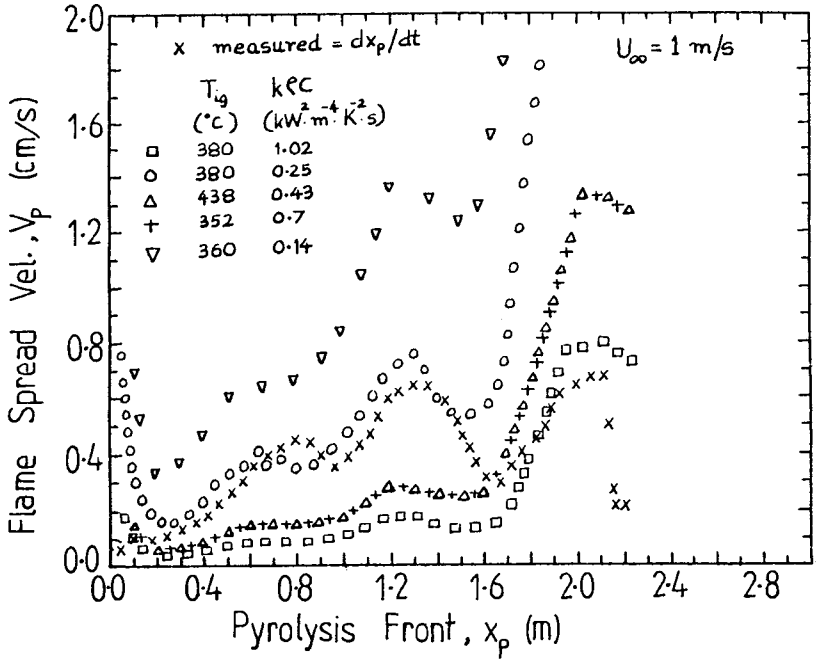


Fig. 9 Predicted Flame Spread Rate.

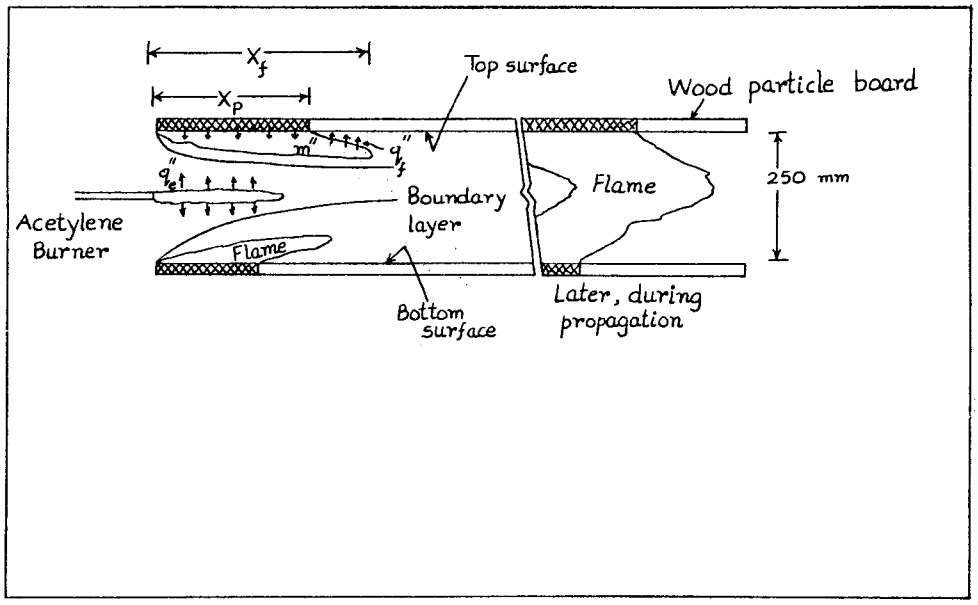


Fig. 10 Fire Spread between Two Parallel Surfaces.

DOUBLE LAYER SPREAD.

Measurements were made of ventilated flame spread and burning over parallel horizontal particle boards (0.13m thick x 1.m wide x 4.8m long) as sketched in Fig. 10. An acetylene burner was used for ignition which produced a 0.3m flame sheet extending over the full width of the boards and radiating about 33kWm^{-2} onto both surfaces. Fire from one surface acts as an external source of heat for the other surface, assisting in a sustained flame spread not obtainable over a single wood surface. For wind velocities between 0.5 to 2.ms^{-1}, measurements of pyrolysis front (observed as a blackening of the surface) and the flame front were correlated quite well by

$$x_f = 1.33x_p \qquad 8)$$

A comparison of the measured (dx_p/dt) and the predicted (V_p) spread rate using Quintiere's formula (Equ. 4) is shown in Fig. 11. Two sets of kpc and T_{ig} values estimated from Cone Calorimeter data was used (Table 1). In this case, the data obtained from Quintiere's method appears to predict the flame spread better than the author's method.

The surface heat flux, q''_f in Equ. 4 was, to a first approximation, estimated by assumming that it was one half the radiant flux emanating from a single flame midway between the two surfaces, with an area of x_f x 1.m^2. Thus

$$q''_f = 0.2Q / 2x_f \qquad 9)$$

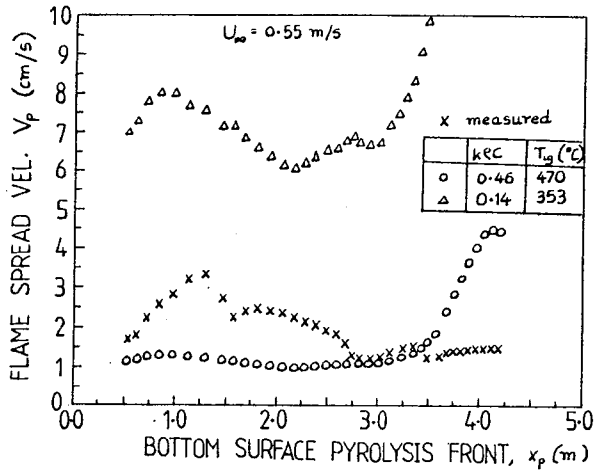


Fig. 11 Predicted Flame Spread.

Q, the total heat release rate was calculated from measurement of the combustion product gas concentrations. The radiant fraction was taken top be 0.2.

DISCUSSION.

The use of Cone Calorimetry data looks promising as a means of predicting full scale fire spread. The values of Tig, kpc, L and Hc obtained give reasonable agreement with measurements of fire spread over one and two surfaces. In this analysis these parameters are assumed to be constant over the range of incident heat flux obtained in a fire. In reality, T_{ig}, kpc and L are transient and are asymptotic for a given incident heat flux (Jackson,1988). An alternative interpretation of Cone Calorimeter data is actively being investigated to improve the predictions of flame spread and other fire properties.

Further full scale experiments are being conducted to elucidate the feedback mechanisms on under-surface spread and between surfaces in different configurations.

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PROPAGATION DU FEU LE LONG DE SYSTEMES HORIZONTAUX DE CONVOYEURS

A.R. Green, R.W. Bilger, V. Apte

On effectue depuis longtemps des essais d'incendie à échelles variées pour tester des matériaux dans des environnements sujets aux risques, tels que les mines souterraines. Les méthodes d'essais ne permettent généralement pas de quantifier les risques et ne peuvent être appliquées à des échelles d'incendies supérieures à celles utilisées lors des essais. Nous présentons une méthode générale permettant de calculer la propagation des incendies par le vent et les autres risques d'incendies de structures du type convoyeurs. La méthode de base consiste à évaluer le comportement d'un incendie découplé au moyen d'un dispositif tel que le calorimètre NBS à cone et d'appliquer ces données quantitatives sur des modèles théoriques de propagation d'incendies. Ces modèles sont ensuite validés par des essais à grande échelle. Nous donnons des exemples d'essais effectués sur des PNMA, des panneaux d'agglomérés et des matériaux de convoyeurs à configuration horizontale, selon des taux de ventilation variables; nous les avons comparés aux données obtenues par le calorimètre NBS à cone. La propagation du feu est plus rapide pour les essais à double couches; ceci est dû à l'augmentation de la réaction des radiations vers la surface.

BRANDAUSBREITUNG AN HORIZONTALEN FÖRDERUNGSSYSTEMEN ENTLANG

A. R. Green, W. Bilger, V. Apte

Brandversuche in verschiedenen Ausmaßen dienen seit langem zur Untersuchung von Material, das in explosionsgefährdeten Räumen wie Kohlegruben untertage eingesetzt wird. Im allgemeinen lassen diese Tests keine Quantifizierung der Gefahr zu und können nicht auf Brände übertragen werden, die größer sind als das Testfeuer. Ein Lösungsweg, die wetterbedingte Ausbreitung von Bränden sowie andere mit Feuer zusammenhängende Gefahren entlang förderähnlichen Strukturen vorauszusagen, wird vorgeschlagen. Die Untersuchungen beruhen auf der Bewertung des Verhaltens vereinzelter Feuer mit einem NBS Kegel Kalorimeter und der Benutzung dieser quantitativen Daten in theoretischen Modellen der Feuerausbreitung. Diese Modelle müssen in Großversuchen bestätigt werden. Beispiele für Großversuche mit PMMA, Spanplatten und Gurtmaterial sind für waagerechte Anordnungen unter verschiedenen Wetterraten vorhanden und wurden mit Daten von dem NBS Kegel Kalorimeter verglichen. Die Flammenausbreitung findet in doppellagigen Experimenten sehr viel rascher statt, bedingt durch die erhöhte Rückstrahlung zur Oberfläche.

РАСПРОСТРАНЕНИЕ ОГНЯ ВДОЛЬ ГОРИЗОНТАЛЬНЫХ КОНВЕЙЕРНЫХ СИСТЕМ

А.Р. Грин, Р.В. Билгер и В. Апте

Отборочные испытания возгораемости материалов, применяемых в таких опасных условиях, которые существуют на подземных шахтах, проводятся уже много лет. Обычно испытательные методы не позволяют количественно определить степени опасности, и они не применимы к пожарам большим, чем те, с которыми проводились испытания.. В докладе описывается общий подход, который позволяет предсказать распространение пожара вдоль конвейеро-подобных конструкций с помощью ветра и других схожих условий. В основном для оценки поведения пожара используется конусный калориметр типа НБС. Полученные количественные данные применяются для построения теоретических моделей распространения огня. Эти модели должны быть подтверждены в испытаниях большего масштаба. Приводятся примеры крупномасштабных испытаний на огнестойкость ПММА, плит из прессованных опилок и конвейерных материалов для горизонтальных конфигураций в различных вентиляционных условиях, их результаты сравниваются с данными, полученными из НБС конусного калориметра. Показывается, что в двухслойных экспериментах огонь распространяется быстрее в связи с увеличением отражения тепловой радиации на поверхность.

沿水平输送设备火灾扩展

A．R．葛林

R．W．别尔葛， V．艾普特

为了鉴别在象地下煤矿一样有危险的环境所用的材料，各种大小型火灾试验都曾做过．普通这种试验不能用数量来表示危害的程度．而且不能应用于比试验规模更大的火灾．这里我们用一种通用的方法来预测风助的火灾扩展和其他沿类似输送设备的火灾危害．这方法的基础是用类似（美国）国家标准局（NBS）锥式量热器来测定去耦的火灾情况，然后利用这种数据在火灾扩展的理论模型里．这些理论必须要以实际规模的试验来验证．以有机玻璃（PMMA）压缩颗粒木板，和输送材料进行的实际规模的火灾试验里用了不同的通风率得到的结果用来与国家标准局锥式量热器的数据作比较．因为表面的辐射回复增加的关系，火焰扩展在双层实验里较快．

CONVEYOR BELT FLAMMABILITY TESTS: COMPARISON OF LARGE-SCALE GALLERY AND LABORATORY-SCALE TUNNEL RESULTS

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Bureau of Mines, Pittsburgh Research Center, USA

ABSTRACT

The U.S. Bureau of Mines conducted flammability tests of synthetic rubber and polyvinyl chloride conveyor belts in a large-scale fire gallery and a laboratory-scale ventilated tunnel. The gallery was 27-m-long, 3.8-m-wide and, 2.5-m-high with a 7.5-m-square cross sectional area. For the gallery tests, 9.1-m-lengths of belting were placed on the top rollers of a conveyor frame and thermocouples installed to measure flame spread rates. The airflow was 1.52 m/s and the ignition source was a liquid fuel tray fire. The laboratory-scale tests were conducted in a 1.8-m-long by 0.46-m-square insulated tunnel at an airflow of 1.02 m/s. A 1.52-m-long by 0.23-m-wide belt sample, fastened to a metal rack, was used and the igniter was a 12-jet gas torch applied to the upstream end. A belt was judged to have passed the gallery test if a portion of the sample was undamaged. The same criterion was also applied to the laboratory-scale tunnel test. Of 17 belts examined, 1 belt that underwent surface charring in the gallery test and failed, passed the tunnel test and another belt that passed the gallery test, failed the tunnel test. The results for the remaining 15 belts were in complete agreement.

INTRODUCTION

A conveyor belt fire in an underground coal mine is a serious threat to life and property. To minimize the hazard of belt fires, the U.S. Code of Federal Regulations for underground coal mines requires approved fire-resistant belting, automatic fire suppression systems for belt conveyor drive areas, automatic fire sensor and warning device systems along belt haulageways, waterlines installed parallel to the entire length of belt conveyors, belt slippage and sequence switches, and special ventilation requirements along belt haulage entries (U.S. Code of Federal Regulations, Part 75, 1988). Despite these precautions, about 20% of the reportable U.S. underground coal mine fires from 1983 through 1988 - fires lasting more than half an hour after discovery, or causing injury - involved conveyor belting. In one instance, a belt fire destroyed the belting along 370 meters of belt entry in less than 24 hours, with one fatality caused by a stress induced heart attack from fighting the fire (Bondra 1987). In another case, a fire starting in the drive area of a belt line spread rapidly and resulted in sealing and abandonment of the entire mine (Wilhem 1988).

The current U.S. flammability test for acceptance of fire-resistant belting for underground coal mines is specified in the Code of Federal Regulations (U.S. Code of Federal Regulations, Part 18, 1988) and is conducted by the Department of Labor, Mine Safety and Health Administration (MSHA). The test is performed in a 0.53 m cubical test chamber using 4 samples 152-mm-long by 12.7-mm-wide by the belt thickness. The sample is positioned horizontally in the chamber, with the transverse axis inclined at 45°, and one end exposed to the flame from a Bunsen type burner for 1 min in still air. At the end of 1 min, the burner flame is removed and the ventilating fan turned on to give an

air current of 1.52 m/s. The duration of the belt flame/glowing is measured. A belt passes the test if 4 samples of the same belt do not exhibit either duration of flame exceeding an average of 1 min, or afterglow exceeding an average of 3 min duration. Because of the nature of this small-scale test, it is impossible to predict the flammability behavior of the belting under realistic large-scale test conditions.

Several other countries employ more severe large-scale flammability belt tests, such as the German inmine gallery test and the propane grid gallery test (Anderson 1984 and Green 1984). Belt fire damage is used as a pass/fail criterion for these tests. Heyn and Linhart (Heyn 1983) demonstrated the difficulty in correlating the German gallery test with several laboratory-scale fire tests and concluded that the large-scale gallery test could not be discontinued. The advantages of developing a simple laboratory-scale fire test for conveyor belting that would reproduce large-scale fire gallery results are obvious in cost and time saved. Such a test could readily be used by approval agencies, flammability testing laboratories and conveyor belt manufacturers to reliably evaluate fire-resistant belting.

This paper describes a study conducted by the Bureau of Mines, in cooperation with MSHA, to assess the flammability behavior of conveyor belting in a large-scale gallery test. The results were also utilized to develop an improved laboratory-scale ventilated tunnel fire test for belting.

LARGE-SCALE FIRE GALLERY TEST

The large-scale fire gallery test was conducted in the Bureau of Mines surface fire gallery located at Lake Lynn Laboratory. The fire gallery consists of a 27.4-m-long tunnel constructed of masonry block walls, a metal arched roof, and a concrete floor. The tunnel is coupled to a 1.8-m-diameter, 3500-m^3/min axivane fan via a 6-m-long tapered transition section. The ventilation flow can be varied between 0.5 m/s and 10 m/s by adjusting the pitch of the fan blades and/or by throttling the fan intake. A schematic of the gallery is shown in figure 1. The cross-sectional area of the tunnel is 7.5 m^2. The interior walls and roof of the tunnel are covered with ceramic blanket insulation. Except where noted, tunnel distances are measured from the junction of the fire tunnel and transition section, designated as the 0-meter mark. A typical conveyor belt frame, 21-m-long and 1.5-m-wide, is centered in the tunnel. The frame consists of a 0.4-m-diameter tail pulley and 0.13-m-diameter troughed idler assemblies at 1.2-m intervals.

A liquid fuel tray fire is the ignition source. The tray, 0.6-m-long by 1-m-wide by 0.3-m-deep, is just downstream of the tail pulley at a tunnel distance of 4.5 m and elevated 20 cm above the floor. The tray contains water on which fuel is floated. The ignition region is shielded from the ventilation flow by a steel plate and concrete blocks.

The gallery is instrumented with thermocouples to measure belt and gas temperatures. An array of 12 thermocouples, connected in parallel and distributed over the cross-sectional area of the tunnel, is located at 23.4 m to measure the average temperature of the stratified gas exit stream.

For these tests, a 9.1-m-length of conveyor belting was cut from a roll, weighed, and bare beaded thermocouples embedded just below the top surface of the belt. The thermocouples were positioned at equally spaced distances from one end of the belt sample along the centerline and near each edge. Typically, 20 belt thermocouples are used to monitor belt temperatures and calculate flame spread rates.

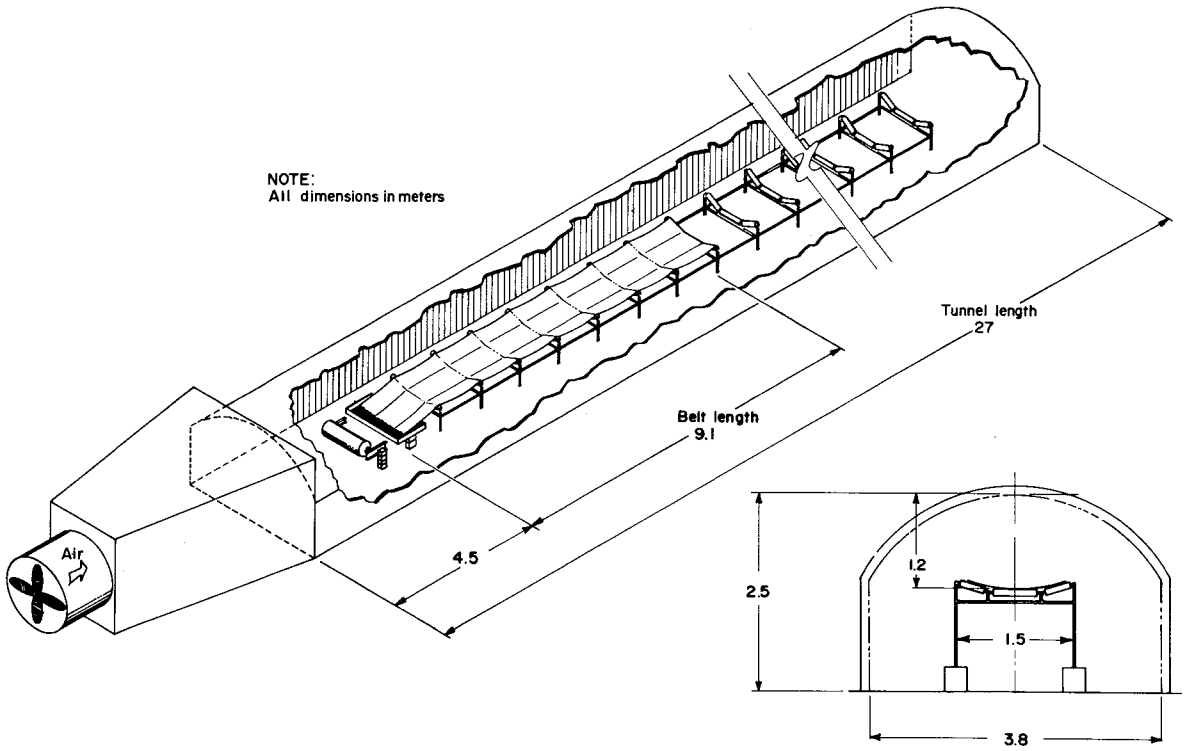


Figure 1. - Schematic of surface fire gallery.

The belt sample was then placed on the rollers of the conveyor belt frame with the top (load carrying) cover up, if applicable, and one end bent downward into the shielded ignition area. The distance of the belting from the tunnel roof was about 1.2 m.

The tunnel airflow was adjusted to 1.52 m/s. The airflow was measured by a handheld vane anemometer at three locations along the sample length at a height of 25 cm above the belt. The average airflow near the exit of the tunnel was also measured. The airflow fluctuated slightly, but was within $\pm$ 10% of the set value. Previous studies in the gallery on the effect of ventilation on conveyor belt fires (Lazzara 1987 and Verakis 1988) demonstrated that flame propagation at these test conditions was most likely to occur at an airflow of 1.52 m/s.

A fuel mixture of 1.9 L of unleaded gasoline and 5.7 L of kerosene was poured into the ignition tray. The tray fire with this fuel loading (7.6 L) burned for 5 to 6 min, with a peak fire size of about 700 kW. The flames enveloped the top and bottom surfaces of about 1.5 m of belting.

BELT SAMPLES

Nine synthetic rubber (R) belts and 8 polyvinyl chloride (P) belts were tested. Table 1 describes the belting. All the belts, except R3 and R7, were new. Belts R3 and R7 were obtained from mines and were slightly worn.

All the belts, except R1, passed the current U.S. acceptance test for fire-resistant belting. Belt R1 is thus considered to be non fire-resistant and would not be permitted in U.S. coal mines. However, it could be used in underground noncoal mines. Belts R4, R10, P2, P3, P4, P6 and P9 also passed the extended time (50 min) propane grid burner test.

RESULTS OF LARGE-SCALE GALLERY TEST

The results for the large-scale gallery test are given in table 2. The flame spread rates were determined from the time-temperature traces obtained from the belt thermocouples and the peak fire size was estimated from the temperature increase of the thermocouple array (Lazzara 1987). The time after the ignition of the tray fire at which the peak fire intensity occurred and the maximum gas temperature measured by a thermocouple near the roof at the exit of the tunnel are also given.

Four types of flammability behavior were observed: (1) rapid flame spread, >4m/min, followed by the entire sample burning, (2) rapid flame spread that charred the entire top surface of the belting, but left the bottom surface undamaged, (3) a slowly propagating flame, with spread rates ranging from 0.3 to 1.3 m/min, that completely consumed the belting (4) and a nonpropagating fire with a portion of the 9.1 m length sample left undamaged.

Large quantities of smoke were generated by the burning belts, and vision was severely limited except for a narrow region near the gallery floor. Belt samples that were entirely consumed fell from the rollers and continued to burn on the floor; the residue usually consisted of ashes and small portions of badly charred belting.

Of the rubber belts tested, R1 (non fire-resistant), R2, R3, R7, R9, and R11 were completely consumed by propagating fires, while R4 (chloroprene) and R10 (SBR-chloroprene blend, 1 ply) did not propagate flame and portions of the 9.1 m samples were undamaged. The entire top surface of belt R6 was lightly charred. For belt R2, the time-temperature traces obtained from the belt thermocouples are shown in figure 2. The tray fire ignited the belt, and a rapid flame spread (5.8 m/min) occurred over most of the top surface of the sample about 6 min after the start of the test. This initial flame, however, was not sustained but was followed by a steadily propagating flame front, with a rate of 0.9 m/min over the last 5 m of the sample, that destroyed the belting. The 9.1-m-long sample was totally consumed in about 25 min.

Of the polyvinyl chloride belts tested, P1 was entirely consumed, while the entire top surfaces of P2, P4, and P6 were deeply charred.

Belts P3, P7, P8 and P9 did not propagate flame over the entire sample length, although for P9 the fire damage (light charring) extended to the 7.6 meter point, which was the maximum damage for any of the belts that passed the test. Several of the belts (R3, P1, P3, and P4) were tested more than once with similar results.

The simple criterion of belt fire damage, excluding blistering, was selected to decide whether a belt passed or failed the large-scale gallery test. A belt passed this gallery test if fire damage did not extend to the end of the 9.1-m-long sample and a portion of the sample was undamaged across its width. Applying this criterion to the results given in table 2, 11 of the 17 belts tested failed the large-scale gallery test and 6 passed.

LABORATORY-SCALE VENTILATED TUNNEL FIRE TEST

The laboratory-scale ventilated tunnel was similar to that proposed by the Bureau of Mines in an earlier study (Sapko 1981 and Perzak 1982) and is shown in figure 3. The test chamber is 1.8-m-long by 0.46 m square and is constructed from 2.5-cm-thick refractory material. The square chamber is connected to round exhaust ducting via a stainless steel transition section. The inner surface of the transition section is lined with a 13-mm-thick ceramic fiber blanket. The main exhaust is

constructed from 30.5-cm-diameter galvanized steel ducting. The igniter is a commercial 12 jet methane gas burner (two rows of 6 jets). The belt sample is fastened to a steel rack constructed of slotted angle iron.

Table 1: Conveyor Belting Description

Belt	Construction	Width, m	Thickness mm	Weight, kg/linear-m
R1	4-plies SBR[1] 7 mm top cover 2 mm bottom cover	1.07	15	17.8
R2	4-plies SBR 5 mm top cover 2 mm bottom cover	1.07	13	16.9
R3	4-plies SBR 6 mm top cover 3 mm bottom cover	1.02	14	18.6
R4	chloroprene solid woven	1.07	9	14.3
R6	SBR solid woven 3 mm top cover 2 mm bottom cover	1.07	9	12.7
R7	3-plies SBR 6 mm top cover 2 mm bottom cover	1.16	12	17.8
R9	chloroprene/SBR[2] 3-plies 3 mm top cover 2 mm bottom cover	1.07	10	14.0
R10	chloroprene/SBR[2] solid woven 3 mm top cover 2 mm bottom cover	1.07	10	12.2
R11	3-plies SBR 5 mm top cover 2 mm bottom cover	1.07	11	14.9
P1	PVC[3] solid woven	1.07	11	14.2
P2	Do.	1.07	11	13.8
P3	Do.	1.05	9	11.4
P4	Do.	1.07	10	13.9
P6	Do.	1.07	11	14.2
P7	Do.	1.07	13	18.6
P8	Do.	0.91	13	14.7
P9	PVC[4] solid woven with covers	1.22	13	21.9

[1] SBR - styrene-butadiene rubber
[2] Belts R9 and R10 contained the same ratio of chloroprene to SBR, but differed in construction.
[3] PVC - polyvinyl chloride
[4] The PVC cover and carcass compounds were different blends.

Table 2: Large-Scale Fire Gallery Test Results for Synthetic Rubber and Polyvinyl Chloride Belts

Belt	Flame rate, m/min	Peak fire size (time), Mwatt (min)	Max. gas temp.[1], °C	Belt damage[2], m	Pass or Fail[3]	Comments
R1	7.6	5.7 (12.0)	428	9.1	F	Rapid flame spread followed by entire sample destruction.
R2	0.9 (5.8)	3.9 (18.5)	330	9.1	F	Non-sustained rapid flame (5.8 m/min) followed by a steady propagating flame consuming belt.
R3	0.3	0.6 (25.5)	315	9.1	F	Slow steady propagating flame that consumed the belt.
R4	NP[4]	0.7 (4.5)	130	1.5	P	Non-propagating flame that damaged belt in ignition zone.
R6	6.1	1.9 (3.8)	271	9.1	F	Rapid flame spread that charred top surface of belt.
R7	4.6	6.3 (12.0)	448	9.1	F	Rapid flame spread followed by entire sample destruction.
R9	1.3	2.7 (13.5)	287	9.1	F	Slow flame spread that completely consumed belt sample.
R10	NP	1.3 (3.8)	163	2.4	P	Non-propagating flame that damaged belt in ignition zone.
R11	5.4	5.1 (13.5)	391	9.1	F	Rapid flame spread followed by entire sample destruction.
P1	6.7	3.0 (4.0)	394	9.1	F	Rapid flame spread followed by entire sample destruction.
P2	6.6	2.3 (4.8)	275	9.1	F	Rapid flame spread that deeply charred top surface of belt.
P3	NP	0.7 (4.3)	164	0.5	P	Non-propagating flame that damaged belt in the ignition zone.
P4	9.1	2.0 (3.2)	411	9.1	F	Rapid flame spread that deeply charred top surface of belt.
P6	4.6	2.8 (2.8)	310	9.1	F	Rapid flame spread that deeply charred top surface of belt.
P7	NP	0.7 (4.0)	129	1.8	P	Non-propagating flame that damaged belt in the ignition zone.
P8	NP	0.8 (4.5)	128	2.4	P	Non-propagating flame that damaged belt in the ignition zone.

Table 2: Large-Scale Fire Gallery Test Results for Synthetic Rubber and Polyvinyl Chloride Belts -- Continued

Belt	Flame rate, m/min	Peak fire size (time), Mwatt (min)	Max. gas temp.[1], °C	Belt damage[2], m	Pass or Fail[3]	Comments
P9	NP	1.3 (5.0)	196	7.6	P	0.6 meter consumed and 7.0 meters charred on top surface.
TRAY FIRE		0.7 (5.0)	106	---	-	No belt, 7.6 liters of fuel.

[1]Gas temperature increase as measured near the roof at 27 m.
[2]Belt damage includes charring, but not blistering.
[3]The pass criterion is no fire damage across belt width at the end of sample.
[4]Non-propagating flame; does not reach the end of the 9.1-m-long sample.

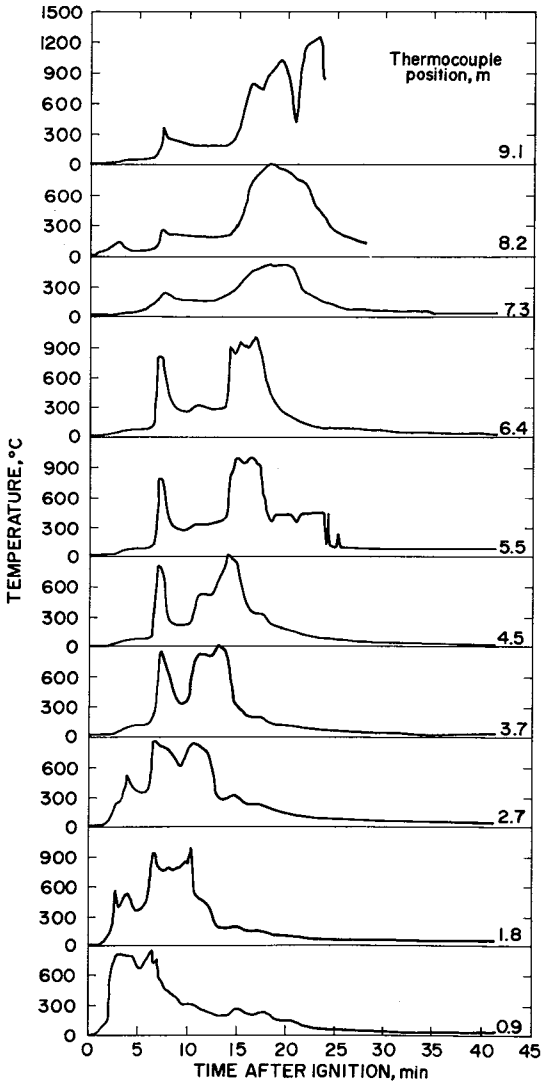


Figure 2. - Time-temperature traces of belt thermocouples for test of belt R2 at 1.52 m/s airflow.

In preliminary experiments to establish a set of standard test conditions, several variables were examined, including airflow, sample width, distance of rack to tunnel roof, and burner duration. Several of the belts that exhibited different burning characteristics in the large-scale gallery test were used for these tests. The laboratory test conditions were adjusted to give similar fire damage results to the large-scale gallery test for these belts. The test conditions that were finalized: sample size, 1.52-m-long by 0.23-m-wide by nominal belt thickness (6 to 19 mm); distance of sample rack from tunnel roof, 20 cm; tunnel airflow, 1.02 m/s; duration of gas igniter, 5 min; methane flow to burner, 35 L/min at 22° C and 1.01 kPa.

To conduct a test, a belt sample is fastened with the top cover up, if applicable, to the steel rack with 1.4-mm-diameter cotter pins and thin washers to prevent the belt from shrinking away from the burner. The rack is placed in the tunnel and the airflow (1.02 m/s) set. The airflow is measured by a vane anemometer placed on the belt surface about 30 cm from the front of the tunnel. The methane burner is ignited and the flame allowed to stabilize. The burner is then applied to the front edge of the belt sample with the flames impinging equally on the top

and bottom surfaces of the sample. After 5 min, the burner is removed, and the belt sample allowed to burn until the flames are out. If a portion of the sample remains on the rack, the rack is removed and the extent of any undamaged belting, across the width of the sample, is measured.

A belt is judged to have passed the laboratory-scale test if, in three trials, there remains a portion of the 1.52 meter length sample that is undamaged across its width, excluding blistering. A belt fails the test if in any single trial, fire damage extends to the end of the sample.

RESULTS OF LABORATORY-SCALE VENTILATED TUNNEL FIRE TEST

All 17 belts that were examined in the large-scale fire gallery test were subjected to the laboratory-scale fire test described above. More than the required number of three trials were made for several of the belts to check repeatability. The results are given in table 3. For most of the belts, the fire damage was similar to that found in the large-scale fire gallery test. Of the 9 rubber belts tested, 7 failed the laboratory-scale test by being totally consumed and 2 passed.

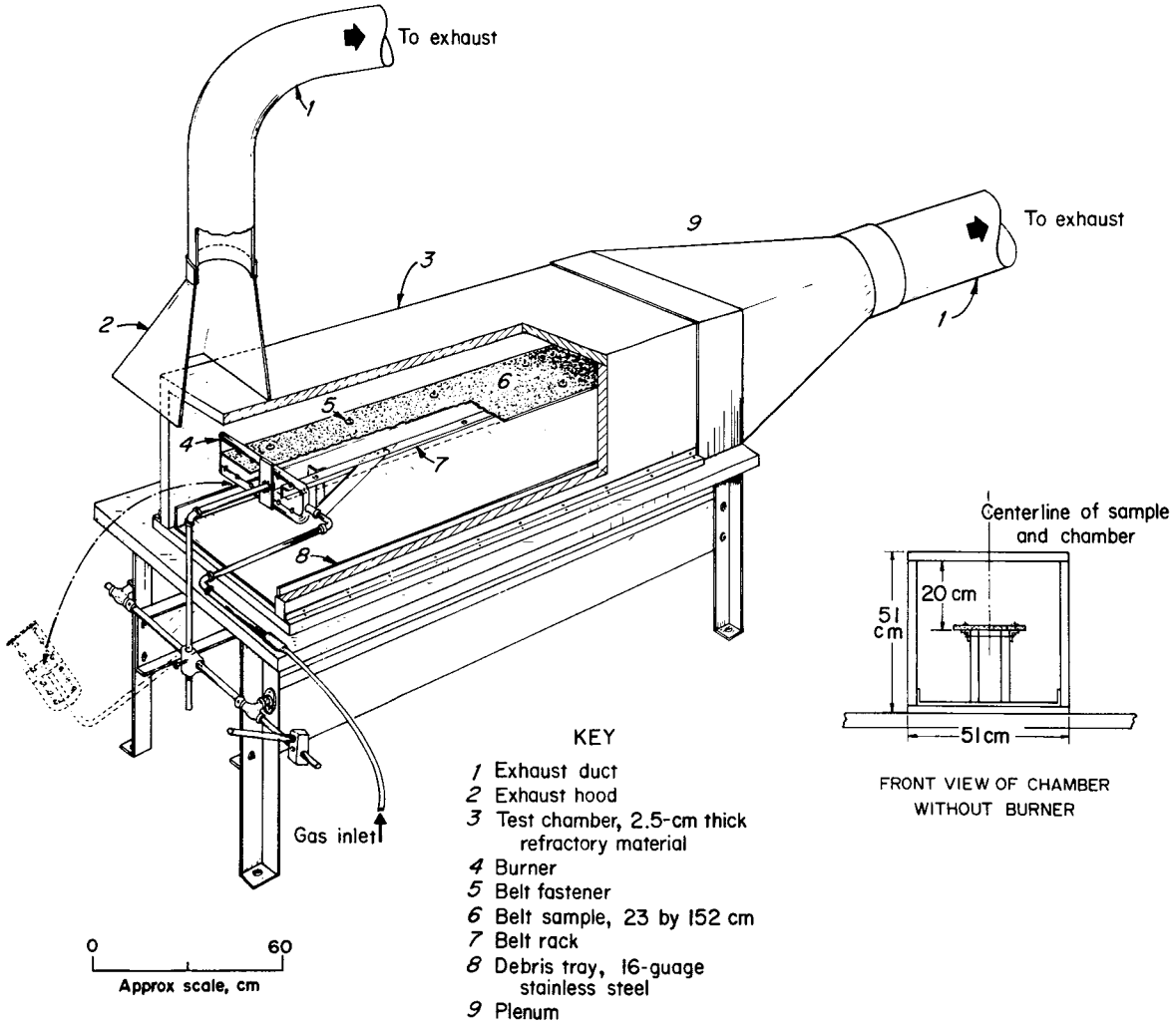


Figure 3. - Schematic of laboratory-scale fire tunnel.

For the 8 polyvinyl chloride belts, 4 failed the laboratory-scale test (P1 by being totally consumed and P2, P4 and P8 due to charring) and 4 passed. The repeatability of the test results for the poor and very good fire-resistant belts (defined by total destruction or fire damage limited to the ignition area in the large-scale gallery test, respectively) was excellent.

COMPARISON OF LARGE-SCALE GALLERY AND LABORATORY-SCALE FIRE TEST RESULTS

A comparison of the pass/fail results of the large-scale gallery and laboratory scale fire tests for the 17 belts shows that they are in very good agreement. For the 9 rubber belts, the results are in complete agreement, with the same 7 belts failing the gallery and laboratory scale test, and 2 belts (R4 and R10) passing both tests. As noted earlier, belt R1 is non fire-resistant while the 8 other belts passed the current U.S. acceptance test for fire-resistant belting. Belts R4 and R10 also passed the extended time propane grid burner test. For the 8 polyvinyl chloride belts, the pass/fail results for 6 of the belts were the same, with 3 belts (P1, P2, P4) failing the large-scale gallery and laboratory-scale fire tests and 3 belts (P3, P7, P9) passing both tests. There are discrepancies in the comparison for belts P6 and P8.

Table 3: Laboratory-scale Ventilated Tunnel Fire Test Results

Belt	Comments	Failure ratio[1]	Belt damage[2], m	Pass[3] or Fail
R1	Total belt destruction.	1/1	1.52	F
R2	Do.	3/3	1.52	F
R3	Do.	3/3	1.52	F
R4	Top belt surface partially charred.	0/6	0.91	P
R6	Total belt destruction.	1/1	1.52	F
R7	Do.	1/1	1.52	F
R9	Do.	2/2	1.52	F
R10	Top belt surface partially charred.	0/3	0.81	P
R11	Total belt destruction.	2/2	1.52	F
P1	Total belt destruction.	3/3	1.52	F
P2	Top surface completely charred in one trial.	1/3	1.52	F
P3	Top belt surface partially charred.	0/5	0.89	P
P4	Top surface completely charred in one trial.	1/3	1.52	F
P6	Top belt surface partially charred.	0/3	1.04	P
P7	Do.	0/3	0.64	P
P8	Top surface completely charred in one trial.	1/3	1.52	F
P9	Top belt surface partially charred.	0/3	0.74	P

[1] The failure ratio is the number of samples that were damaged over the entire sample length (1.52 m) divided by the number of trials.

[2] The fire damaged length for the most extensively damaged sample in all the trials. Belt damage includes charring, but not blistering. For belts that charred, the top surface damage was always more extensive than bottom surface damage.

[3] The pass criterion is no fire damage across sample width at end of sample in 3 separate trials.

Belt P6 failed the large-scale gallery test but passed the laboratory-scale test; belt P8 passed the large-scale test but failed the laboratory scale test. As previously stated, all the polyvinyl chloride belts passed the current U.S. test for fire-resistant belting and belts P2, P3, P4, P6 and P9 also passed the extended time propane grid burner test. The two discrepancies are not completely understood, however, belt P6 sustained the greatest amount of fire damage (1.04 m) than any other belt that passed the laboratory-scale test. Belt P8 was the narrowest belt (0.91-m-width) tested in the large-scale gallery and this may have influenced the results; a narrower belt would be more likely to pass the gallery test since energy losses from the burning belt would be more than for a wider belt, all else being equal.

Overall, of the 17 belts examined, there was agreement for 15 of the belts between the pass/fail results of the large-scale gallery test and the laboratory-scale tunnel test.

CONCLUSIONS

Based on the flammability data obtained from a large-scale gallery test of 17 conveyor belts, a relatively simple and inexpensive laboratory-scale fire test was developed for conveyor belting. The laboratory-scale test results were in very good agreement with those obtained in the large-scale gallery based solely on the criterion of belt fire damage. MSHA plans to revise its current conveyor belt flammability acceptance tests to include the Bureau's new laboratory-scale fire test for belting (minutes of public meeting, 1989).

ACKNOWLEDGEMENTS

The authors wish to acknowledge the invaluable assistance of personnel from the Mine Safety and Health Administration's Approval and Certification Center in conducting the large-scale gallery tests. The cooperation of conveyor belt manufacturers is greatly appreciated.

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ESSAIS D'INFLAMMABILITE DE COURROIES DE CONVOYEURS: COMPARAISON DES RESULTATS D'ESSAIS DE GALERIES A GRANDE ECHELLE ET D'ESSAIS DE TUNNELS EN LABORATOIRE

C.P. Lazzara, F.J. Perzak

Le Bureau des Mines Américain a effectué des essais d'inflammabilité sur des courroies de convoyeurs en caoutchouc et en chlorure de polyvinyle dans une galerie à grande échelle et dans un tunnel ventilé de laboratoire. Lors des essais en galeries, on a placé des courroies de 9,1 m de long sur les rouleaux supérieurs d'un châssis de convoyeur et installé des thermocouples pour mesurer le taux de propagation des flammes. Le déplacement d'air dans la galerie était de 1,5 m/s et la source d'allumage consistait en un feu de plateau de combustible liquide. Les essais en laboratoire ont été effectués dans un tunnel carré isolé de 1,8 m de long sur 0,46 m de côté avec déplacement d'air de 1,0 m/s. On a utilisé une courroie échantillon de 1,52 m de long sur 0,23 m de large, attachée à une grille de métal; l'allumeur était un chalumeau à gaz de 12 jets appliqué à l'extrémité amont. Au cours des essais à grande échelle, nous avons observé quatre types de comportements: (1) propagation rapide de la flamme, suivie de la combustion de tout l'échantillon; (2) propagation rapide de la flamme, calcinant la surface supérieure de la courroie; (3) propagation lente du feu, détruisant l'échantillon; enfin, (4) un feu sans propagation, avec dégâts limités de la courroie. Si une portion de la courroie n'était pas endommagée, elle avait réussi l'essai. Le même critère de réussite a été appliqué à l'essai de tunnel en laboratoire. Sur 17 courroies examinées, 2 qui avaient été calcinées superficiellement au cours de l'essai en galerie et avaient donc échoué, ont réussi l'essai en tunnel. Les résultats donnés par les 15 courroies restantes se sont parfaitement rejoints.

FÖRDERBAND-FLAMMTESTS: VERGLEICHE ZWISCHEN GROßANGELEGTEN BRANDTESTS IN FÖRDERSTRECKEN UND IM LABOR DURCHGEFÜHRTEN TUNNELTESTS.

C. P. Lazzara, F. J. Perzak

Das U. S. Bureau of Mines führte Entflammbarkeitsversuche mit Gummi - und Polyvinylchlorid (PVC) Gurten in Brandstrecken und im Labor in einem bewetterten Tunnel durch. In den Brandstreckenversuchen wurde ein 9,1 Meter langer Gurt auf Bandrollen eines Förderrahmens angeordnet, danach wurden Thermoelemente zur Messung der Brandausbreitungs-Geschwindigkeit angebracht. Die Wettergeschwindigkeit lag bei 1,5 m/s, als Zündquelle wurde eine Wanne mit entzündetem Brennstoff verwendet. Der Labortest fand in einem 1,8 Meter langen isolierten Tunnel mit 0,46 m Querschnitt statt, mit einer Luft-Geschwindigkeit von 1,0 m/s. Ein an einem Metallrahmen befestigter Gurt von 1,52 Meter Länge und 0,23 Meter Breite wurde von einem 12-flammigen Gasbrenner auf der Luftzufuhrseite entzündet. In den Großversuchen konnten vier Verhaltensmuster beobachtet werden: (1) rasche Flammenverbreitung gefolgt von einer Entflammung der gesamten Probe; (2) rasche Flammenverbreitung gefolgt von der Verkohlung der Gurtoberfläche: (3) langsam fortschreitendes Feuer mit Zerstörung des Gurtes; und (4) ein sich nicht ausbreitendes Feuer mit begrenzter Beschädigung des Gurtes. Der Gurt bestand den Test, wenn ein Teil der Probe nicht beschädigt war. Die gleichen Kriterien wurden auch für die Tunnelversuche im Labor angewendet. Von 17 Proben, die untersucht wurden, versagten 2 Gurte mit Oberflächenverkohlung im Brandstreckentest, bestanden aber den Tunneltest. Die Ergebnisse der 15 übrigen Gurte stimmten genau überein.

ИСПЫТАНИЯ КОНВЕЙЕРНЫХ ЛЕНТ НА ВОСПРАМЕНЯЕМОСТЬ: СРАВНЕНИЕ РЕЗУЛЬТАТОВ ЛАБОРАТОРНЫХ ИСПЫТАНИЙ С БОЛЕЕ КРУПНЫМИ ИСПЫТАНИЯМИ, ПРОВОДИВШИМИСЯ В ЗАКРЫТОЙ ШТОЛЬНЕ

К.П. Лаззара и Ф.Дж. Перзак

Горным Бюро США были проведены испытания на возгораемость резиновых и поливинил-хлоридовых транспортерных лент в условиях штольни и экспериментального вентилируемого туннеля. При испытаниях в штольне транспортерные ленты длиной 9,1 м были положены на ролики конвейерной рамы, а также были установлены термопары для измерения скорости распространения пламени. Скорость потока воздуха в штольне составляла 1,5 м/сек, источником зажигания было жидкое топливо, содержавшееся в поддоне. Лабораторные испытания проводились в закрытом туннеле размером 1,8 м x 0,46 кв.м при скорости потока воздуха 1 м/сек. Образец транспортерной ленты 1,52 м в длину и 0,23 м в ширину прикреплялся к металлической раме, а воспламенителем служил 12-струйный газовый факел, направленный вверх. При испытаниях в штольне было зарегистрировано четыре типа поведения пламени: 1) быстрое распространение пламени, сопровождавшееся полным сгоранием образца; 2) быстрое распространение пламени, при котором обугливалась внешняя поверхность ленты; 3) медленное распространение пламени, при котором лента полностью сгорала; 4) нераспространяющееся пламя, которое не нанесло ленте большого вреда. Образец считался прошедшим испытание, если часть его оставалась несгоревшей. Из семнадцати испытанных транспортерных лент, две из них, внутренняя поверхность которых обуглилась при испытаниях в штольне, штольневые испытания не прошли, но испытания в лаборатории выдержали. Сравнение результатов штольневых и лабораторных испытаний для оставшихся пятнадцати транспортерных лент не показало никакого различия.

输送机皮带可燃性试验：大尺度平巷和实验室尺度隧道试验的比较

C．P．拉沙辣　及　F．J．帕儿沙克

美国矿业局用橡皮和聚氯乙烯输送机带做了大尺度平巷和实验室尺度通风隧道的可燃性试验．在平巷试验里，９．１米长的带料放在输送机架的滚柱上，火焰扩展速度便由带上安置的热电偶测定．平巷空气流动的速度为每秒１．５米．用液体燃料盆火引燃．实验室尺度试验则在１．８米长０．４６米见方绝缘的隧道里进行，隧道里的空气流速为每秒１．０米．一条１．５２米长０．２３米宽的带料固定在金属架上，用１２个火焰喷枪在上流引燃．在大尺度试验里，我们观察到下列现象：

（１）当整个试样烧着后，火焰扩展得非常快，（２）迅速的火焰扩展会将带料上层烧焦；（３）缓慢传播的火焰会将试样全部烧毁；（４）不传播的火焰会对带料的损坏较小．如果试样还有一部份没有烧毁，则带料便算通过试验．实验室尺度试验通过的标准也一样．十七件试验的带料中有二件在平巷试验里上面烧焦未通过，但通过了隧道试验．其他十五件带料则在两种试验中结果一致．

Session III

SPONTANEOUS COMBUSTION

FIRE PREVENTION IN WORKINGS WITH RIDER SEAMS PRONE TO SPONTANEOUS COMBUSTION

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Abstract

Spontaneous combustion fires are relatively easy to control if suitable countermeasures are taken in time. However, CO-make in certain rider seams falling into the goaf during the extraction of the seam below it are problematic. Initially, it is not discernible in the working whether a harmless extensive oxidation or a spontaneous combustion fire will develop. In these workings comprehensive sealing measures have to be carried out to reduce stray currents. A continuous intensified CO-monitoring of the overall ventilation current and of the methane drainage system as well as a regular check of the gas composition in the goaf are required. Apart from the decisive CO-make the determination of hydrogen and higher hydrocarbons, in particular unsaturated hydrocarbons, gain in importance for the assessment of the oxidation condition. Preventive inertization of the goaf may be carried out through pipes in the roadside pack of the intake roadway and through pipes advancing with the face line. Stray currents and nitrogen currents may be monitored by tracer gases.

Introduction

Spontaneous combustion fires are relatively easy to control if suitable countermeasures are taken in time. However, CO-make in certain rider seams falling into the goaf during the extraction of the seam below it are problematic.

In the Ruhr coal mining district these are, for example, the seams Katharina above Hermann-Gustav, Röttgersbank above Wilhelm, Wassserfall above Sonnenschein and Zollverein seams with other mining activities below them. Depending on the prevailing geologic conditions also other rider seams may exibit this characteristic feature.

Spontaneous combustion or extensive oxidation occurred if the distance between the Katharina rider seam and the worked Hermann-Gustav seam was approx. 10 m or less. It is known that increased CO-make occurred in the Sonnenschein seam if the Wasserfall rider seam also had a distance of less than 10 m. CO-make doubled if the distance between the rider seam and the worked seam was reduced to 5 m. There are several indications showing that rock temperatures are increasing in line with the depth favour extensive oxidation. Initially, it is, however, not discernible in the working whether a harmless extensive oxidation or a spontaneous combustion fire will develop in the goaf.

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Both phenomena were also observed in the actual rider seams, in particular if stray ventilation flows through the lossened rider seam into workings above.

Fire prevention measures during extraction

In workings with these conditions comprehensive sealing measures are required to reduce stray currents.

- Roadway drivage and backfilling of the roadway supports with packbuilding materials.

- Securing of the face edges of the rise heading.

- Construction of tight roadside packs in line with the face advance.

- Filling of the spandrel between roadside pack and goaf with dry pulverous material (e.g. electrostatic precipitator ash). The filling material has to be entered as soon as the caved strata settled (approx. 80 to 150 m behind the face line). Dry pulverous filling materials have the advantage of penetrating into fissures, cracks, and other cavities when being pressed together.

- Shotcrete application in the roadway after the rear abutment pressured ceased.

The sealing measures in the gateroads are shown in Fig. 1.

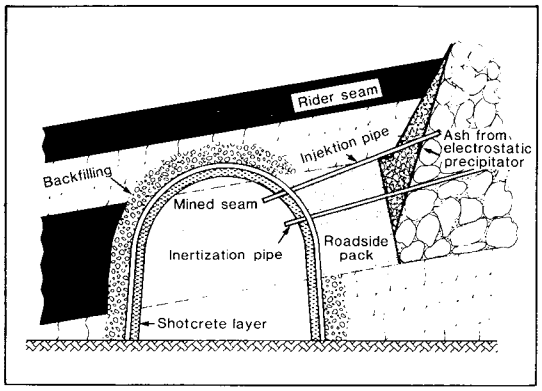


Fig. 1: Sealing measures in gateroads

Assessing the extensive oxidation and the size of the fire

An oxidation process underground is generally detected by the CO-content of the ventilation current. In this context it is not the CO-content in ppm but the CO-production in l/min which is important for the assessment. Even small CO-flows may be already characteristic for an incipient fire (fine coal smouldering fire below a conveyor belt). A CO-production of more than 10 l/min (warning value) requires a detailed investigation into the cause in any case. At CO-flows of 20 l/min (alert value) measures have to be taken to reduce CO-production.

Extensive oxidation sometimes produces CO-flows of more than 20 l/min without a fire (spontaneous combustion) having occurred. In these cases the assessment requires additional parameters like the behaviour of CO-make on days without coal production or increases and reductions of the daily output, the pyrolysis products hydrogen and higher hydrocarbons, in particular unsaturated hydrocarbons, as well as the Graham fire coefficient. Reduced CO-make on days without coal production or reduced

output is a clear indication for an extensive oxidation. In analogy the CO-flows have to increase with increasing output or face advance as well as on weekends (Friday).

As long as hydrogen and hydrocarbons, which are released at temperatures above 150°C, are not traced in the gas samples taken in the goaf, the CO-make at a constant high CO-flow of more than 20 l/min can also be characterized as extensive oxidation. If, in contrast, the mentioned gases occur in higher concentrations the stage of spontaneous combustion has been reached or exceeded. By means of gas chromotographs hydrogen and higher hydrocarbons, in particular unsaturated hydrocarbons, can be detected already in the ppm range. The available measuring data are, however, not yet sufficient to indicate reliable critical limits for an individual gas component or for the reaction between the gas components. In contrast, the overall analysis provides for a better assessment. The answer to the question whether or not an ignition source has to be expected in the goaf continuous to be of decisive importance. Moreover, the fire coefficient (FC) as the parameter for the oxidation intensity has to be also referred to in the assessment. The fire coefficient is a dimensionless figure indicating the ratio of the CO-content produced by a fire to the oxygen content consumed by the oxidation source and changes with the temperature at the fire source. Contrary to the results of laboratory examinations the FC can, however, hardly be used under practical conditions to draw a reliable conclusion on the temperature at the oxidation source. A change in the FC of gas samples taken at the same sampling point clearly points out to an increase respectively decrease of the oxidation intensity. In seams prone to spontaneous combustion fires fire coefficients of up to 1 were stated over longer periods in several cases without the oxidation process developing into a mine fire. In the German coal mining industry fire coefficients of up to 30 were determined for concealed fires and of up to 90 for open fires. In contrast to laboratory examinations the fire coefficient can hardly be related directly to the temperature in the goaf.

If a spontaneous combustion is detected the CH_4 content in the goaf directly behind the face support and in the spandrel area behind the face and the return airway is important for the assessment of the overall situation and in particular the explosion hazard. If required, explosion coefficients (EC) have to be established at suitable sampling points and their development has to be monitored carefully. For gas contents in the explosive range a CH_4 ignition may not be excluded in presence of a fire source. It may be avoided if the gas composition in the goaf is low in oxygen (< 12 % by vol.). If hydrogen (H_2) or higher hydrocarbons occur in higher concentrations in addition to methane (CH_4) the oxygen content has to be reduced to values < 6 % vol. A means to keep the oxygen content in the goaf low is the preventive inertization using nitrogen (N_2).

Monitoring by measurements during coal extraction

For the assessment of the oxidation behaviour an intensified CO-monitoring of

the total ventilation current and the methane drainage system (if existing) as well as a regular check of the gas composition in the goaf is required.

In the intake air the base load and at the end of the return airway the total CO-load has to be determined. Moreover, the volumetric CO-flow has to be established if a methane drainage system is operated. Both CO-flows form the total CO-production.The difference of these two flows to the base load forms the CO-production of the working.

Moreover, additional CO-measuring instruments may be installed in certain intervals (150 m to 300 m) in the return gateroad or manual measurements may be taken to establish the CO-loads (profiles) in the roadway sections. In this way it may be detected where the CO is added and at which point an increase does no longer occur (Fig. 2).

In the return gateroad snifter pipes are installed at the face edge and in 10 to 30 m intervals along the goaf to take gas samples, which are analyzed in respect to CO, CH4, H2, hydrocarbons, 02 and C02, at first in daily and later on in longer intervals. As the face advances only the more significant of the snifter pipes further outbye may be permanently monitored (indicator snifter pipes). A fire gas measuring installation is suitable for the simultaneous monitoring of several measuring points. By means of a controlable measuring point switch it may determine the concentrations of CH4, CO, H2, 02, and C02 in the mine gases taken at four sampling points and transmit the results to the surface providing for an overall assessment of the CO-make, the explosibility of the gas mixtures, and the inertization success. Using the same measuring point switch also the intrinsically safe gas chromatrograph might be operated to monitor several sampling points. The methane drainge system, too, has to be permanently monitored in view to CO-concentrations. Apart from the gas sample from the collector mains of the working gas samples have also to be taken from individual gas boreholes.

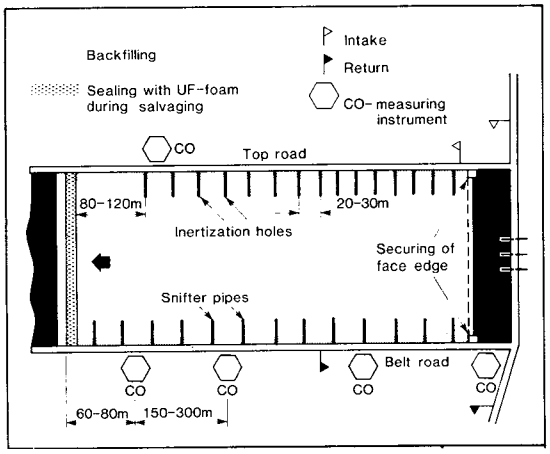


Fig. 2: Sealing and monitoring by measurements

Preventive inertization

If the CO-production indicates that a level of more than 20 - 30 l/min has to be expected in the long run a preventive inertization of the goaf is advisable. To this end 6 to 12 m boreholes are drilled into the goaf in 20-30 m intervals. Pipes are installed through which the nitrogen is supplied. The nitrogen should be injected 80 - 120 m behind the face line so that only little nitrogen enters into the open face area. If a new borehole is supplied with nitrogen with increasing face advance, the borehole with the greatest distance to the

face line has to be shut down afterwards.

This systematic procedure guarantees that nitrogen is always supplied to several boreholes at a distance of 80 to 200 m from the face line and that the nitrogen may be favourably distributed in the goaf behind the face. In this way it is possible to keep the oxygen content in the goaf which has not yet consolidated at low levels (<12 % by vol.) by supplying only small amounts of nitrogen. If required, the 02-content may be reduced further (< 6 % by vol.) by supplying more nitrogen.

To monitor the inertization success the snifter pipes installed for gas sampling on the return side are used. Fig. 3 shows the average gas profiles (C0, 02) over the life of the working.

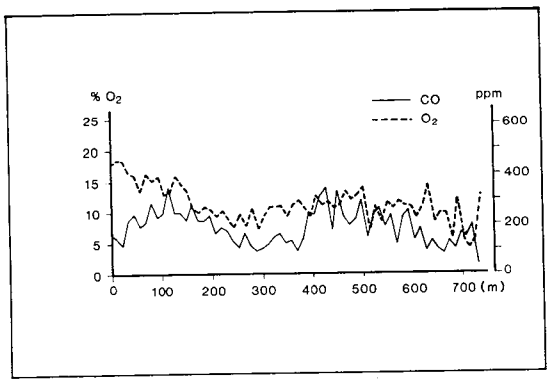


Fig. 3: Average gas profiles measured over the life of the working

To be in a position to find out which way the nitrogen takes with the stray currents through the goaf and which time is required a tracer gas (like sulphur hexafluoride SF6) may be added to the nitrogen. Following the evaluations of the measurements taken in the face and at the snifter pipes in the return airway the nitrogen may be controlled to produce the best possible effect as regards the injection points and the quantities supplied.

Fire protection measures during salvaging

When extracting coal in workings with rider seams which are prone to spontaneous combustion the C0-make in the goaf is reduced in the compacted areas of the goaf. If the operations in the workings cease the subsequent compaction stops so that stray currents may flow over coal lying in the goaf over a longer period during the salvaging operation. For this reason an adequately dimensioned and travelable airway has to be produced and kept open along the cavity line during the salvaging operation. Moreover, the following measures have to be taken:

- the spandrel between roadside pack and goaf has to be filled up to the face
- the line of shields has to be sealed against the goaf using UF-foam (like Iso-schaum)
- the face has to be supported with square work after salvaging the shields, the free area towards the goaf has to be filled with UF-foam and sealed.

If preventive inertization has to be continued pipelines may be installed in the face in larger intervals 60 to 80 m before reaching the final face position. The pipe spacing in the face should amount to 20 - 40 m. The final length of the piping should vary, e.g. 30,50 and 60 m (Fig. 4). Apart from the inertization pipes also special pipes with e.g. 20 m length may be installed in the face to measure the gas com-

position and the inertization condition in the goaf. The piping has to consist of individual pipes with a length corresponding to the advance units of the face support (e.g. 0.75 m). The endpipe has to be protected by a protective cap against clogging and squeezing and has to be secured by a bolt in the floor. Thus, the piping may not be advanced in line with the face. Prior to advancing the support the piping is extended. The pipes used to date had a diameter of 43 mm with a wall thickness of 6 - 7 mm. The pipes were connected by round-threaded collars (Fig. 4).

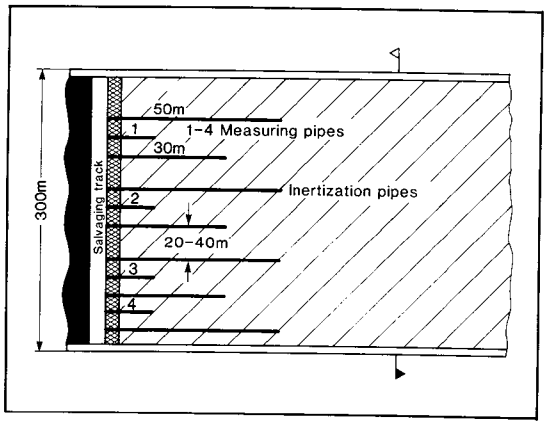


Fig. 4: Inertization pipes in the face

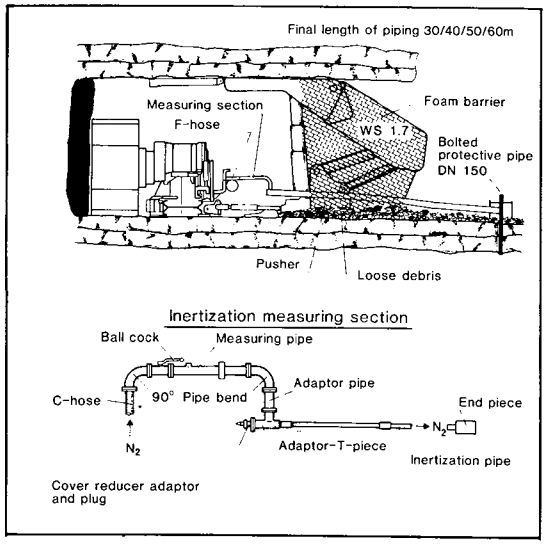


Fig. 5: Face inertization line

By supplying compressed air it may be checked whether the piping is open. For measurements an adapter piece may be installed at the face side end of the piping. After salvaging the shield support the piping has to be extended into the open travelable airway providing for the N2 supply through a circuit line installed in both gateroads. The piping is designed for N2-flows of 2 - 4 m3/min.

Conclusion

At a colliery in the Ruhr district four workings were successfully operated according to this method between 1985 and 1989. A fifth working is being extracted at present, and five other panels are scheduled for extraction (Fig. 6). At other collieries the method has been has been applied in a modified form under similar conditions.

By means of this method CO-make rider seams falling into the goaf during the extraction of deeper seams may be controlled.

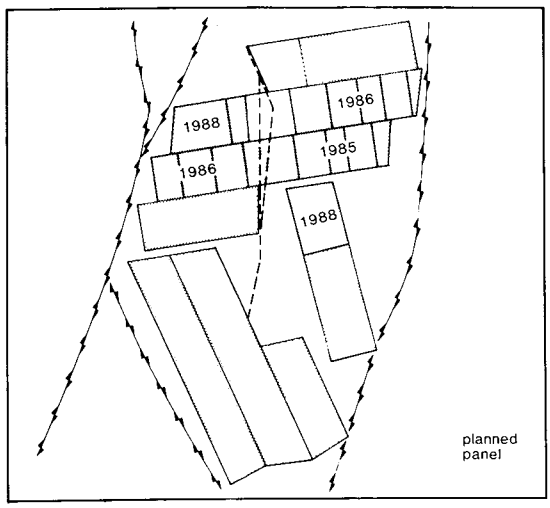


Fig. 6: Working plan of the Sonnenschein seam

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BRANDVERHÜTUNG IN ABBAUBETRIEBEN MIT ZUR SELBSTENTZÜNDUNG NEIGENDEN BEGLEITFLÖZEN

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Zusammenfassung

Selbstentzündungsbrände sind durch rechtzeitiges Einleiten gezielter Gegenmaßnahmen weitgehend beherrschbar. Problematisch sind jedoch CO-Entwicklungen in bestimmten Begleitflözen, die beim Abbau des tiefer liegenden Flözes in den Bruchraum fallen. Für den Betrieb ist zunächst nicht erkennbar, ob sich eine unbedenkliche großflächige Oxidation oder ein Selbstentzündungsbrand entwikkeln wird. In solchen Abbaubetrieben sind umfangreiche Abdichtungsmaßnahmen zur Verminderung der Schleichwetterströme durchzuführen. Eine kontinuierliche verstärkte CO-Überwachung des Gesamtwetterstroms und der Gasabsaugung sowie eine regelmäßige Kontrolle der Gaszusammensetzung des Bruchraums sind erforderlich. Neben der entscheidenden CO-Entwicklung erhalten die Bestimmung von Wasserstoff und höheren Kohlenwasserstoffen, insbesondere ungesättigten Kohlenwasserstoffen für die Beurteilung des Oxidationszustandes Bedeutung. Durch Rohrleitungen im einziehenden Streckenbegleitdamm und durch Rohrstränge, die im Streb mitgeführt werden, kann der Bruchraum im Alten Mann vorbeugend inertisiert werden. Mit Markierungsgasen können Schleichwetterströme und Stickstoffströme verfolgt werden.

Einleitung

Selbstentzündungsbrände sind durch rechtzeitiges Einleiten gezielter Gegenmaßnahmen weitgehend beherrschbar. Problematisch sind jedoch CO-Entwicklungen in bestimmten Begleitflözen, die beim Abbau des tiefer liegenden Flözes in den Bruchraum fallen.

Im Steinkohlenbergbau an der Ruhr handelt es sich z. B. um die Flöze Katharina über Hermann-Gustav, Röttgersbank über Wilhelm, Wasserfall über Sonnenschein und unterbaute Zollvereinflöze. Je nach geologischer Beanspruchung können auch andere Begleitflöze diese Charakteristik aufweisen.

Selbstentzündungen bzw. großflächige Oxidationen sind dann aufgetreten, wenn der Abstand des oberhalb liegenden Flözes Katharina zum gebauten Flöz Hermann-Gustav ca. 10 m und weniger betrug. Von Flöz Sonnenschein ist bekannt, daß erhöhte CO-Produktionen auftraten, wenn das Begleitflöz Wasserfall ebenfalls einen geringeren Abstand als 10 m hatte; eine Verdoppelung der CO-Produktion trat auf, wenn der Abstand des Begleitflözes zum gebauten Flöz auf 5 m zurückging.

Es gibt eine Reihe von Hinweisen dafür, daß die mit der Teufe zunehmenden Gebirgstemperaturen die großflächigen Oxidationen begünstigen.

Für den Betrieb ist aber zunächst nicht erkennbar, ob sich im Bruchraum eine unbedenkliche großflächige Oxidation oder ein Selbstentzündungsbrand entwickeln wird. Beide Erscheinungen sind auch in den Begleitflözen selbst beobachtet worden, und zwar insbesondere dann, wenn Schleichströme durch das aufgelockerte Begleitflöz in noch darüberliegende Abbaue strömen.

Brandverhütungsmaßnahmen während des Abbaus

In Abbaubetrieben mit diesen Verhältnissen sind umfangreiche Abdichtungsmaßnahmen zur Verminderung der Schleichwetterströme durchzuführen:

- Auffahren der Strecken mit Hinterfüllung durch Baustoffe.
- Sicherung der Anlaufkanten des Aufhauens.
- Mitnahme von dichten Streckenbegleitdämmen.
- Verfüllen des Zwickels zwischen Streckenbegleitdamm und Bruchraum mit trockenen, staubförmigen Stoffen (z. B. Elektrofilterasche). Die Verfüllstoffe sind dann einzubringen, wenn der Bruch sich bereits abgesetzt hat (ca. 80 - 150 m hinter dem Strebstand). Trockene, staubförmige Verfüllstoffe haben den Vorteil, daß sie beim Zusammendrücken weiter in die Klüfte, Spalten und sonstige Hohlräume eindringen.
- Wölben des Streckenmantels in dem Bereich, in dem der nacheilende Gebirgsdruck abgeklungen ist.

Die Abdichtungsmaßnahmen in den Abbaustrecken sind in Bild 1 dargestellt.

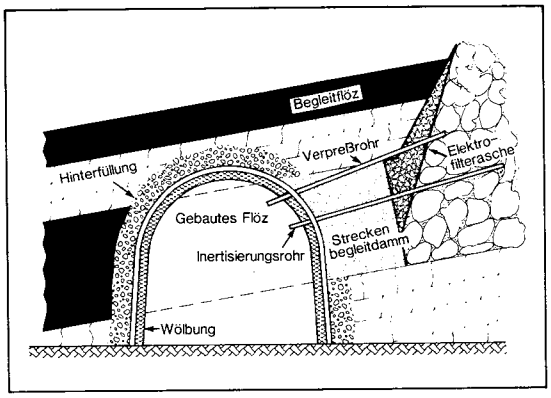


Bild 1: Abdichtungsmaßnahmen in Abbaustrecken

Beurteilung der großflächigen Oxidation und des Brandumfangs

Ein Oxidationsvorgang unter Tage wird in der Regel am CO-Gehalt des durchgehenden Wetterstroms erkannt. Dabei ist für die Beurteilung des Oxidationsumfangs nicht der CO-Gehalt in ppm, sondern der CO-Zustrom in l/min von Bedeutung. Selbst geringe CO-Ströme können bereits für einen beginnenden Brand charakteristisch sein (z. B. Kohlenklein-Glimmbrand unter einem Förderband). CO-Ströme von mehr als 10 l/min (Warnwert) bedürfen in jedem Fall der näheren Untersuchung der Ursache; bei CO-Strömen von 20 l/min (Alarmwert) sind Maßnahmen zur Verringerung des CO-Zustromes zu treffen.

Großflächige Oxidationen bringen mitunter höhere CO-Ströme als 20 l/min, ohne daß ein Brand (Selbstentzündung) aufgetreten ist. In solchen Fällen müssen daher weitere Kennwerte für die Beurteilung herangezogen werden. Dazu eignen sich das Verhalten der CO-Produktion an Tagen der Betriebsruhe sowie bei Aufstockung und Verminderung der Tagesfördermenge, die Pyrolyseprodukte Wasserstoff und höhere Kohlenwasserstoffe, insbesondere die ungesättigten Kohlenwasserstoffe sowie die Brandkennziffer nach Graham. Fallen die CO-Ströme an Tagen der Betriebsruhe und bei Rückgang der Fördermenge, so ist das eindeutig ein Hinweis auf eine großflächige Oxidation. Entsprechend müssen die CO-Ströme bei Erhöhung der Fördermenge oder des Abbaufortschritts sowie am Wochenende (Freitag) ansteigen.

Solange Wasserstoff und Kohlenwasserstoffe, die oberhalb einer Temperatur von 150 °C freigesetzt werden, in den Gasproben aus dem Bruchraum nicht nachgewiesen werden, kann die CO-Entwicklung bei konstantem hohen CO-Strom oberhalb 20 l/min ebenfalls als großflächige Oxidation charakterisiert werden. Treten dagegen die genannten Gase in höheren Konzentrationen auf, ist das Stadium der Selbstentzündung erreicht oder überschritten. Durch den Einsatz von Gaschromatographen können Wasserstoff und höhere Kohlenwasserstoffe, insbesondere ungesättigte Kohlenwasserstoffe,schon im ppm-Bereich festgestellt werden. Die bisher vorliegenden Meßdaten reichen

jedoch noch nicht aus, um zuverlässige kritische Grenzwerte für eine einzelne Gaskomponente bzw. für das Verhältnis der Gaskomponenten untereinander anzugeben. Dagegen gibt die Gesamtanalyse die Möglichkeit einer besseren Beurteilung. Entscheidend bleibt die Beantwortung der Frage, ob mit einer Zündquelle im Alten Mann gerechnet werden muß oder nicht. Weiterhin muß die Brandkennziffer (BKZ) als Kennwert der Oxidationsintensität zur Beurteilung mit herangezogen werden. Die Brandkennziffer ist eine dimensionslose Zahl, die dem Verhältnis des von einem Brand erzeugten CO-Gehaltes zu dem vom Oxidationsherd verbrauchten Sauerstoffgehalt entspricht und die sich mit der Temperatur am Brandherd ändert. Allerdings läßt die BKZ - entgegen den Ergebnissen von Laboratoriumsuntersuchungen - in der Praxis kaum einen zuverlässigen Schluß auf die am Oxidationsherd herrschende Temperatur zu. Dagegen zeigt die Veränderung der BKZ von Gasproben derselben Probenahmestelle eindeutig den Anstieg bzw. Rückgang der Oxidationsintensität. In zur Selbstentzündung neigenden Flözen wurden verschiedentlich über längere Zeit Brandkennziffern bis zu 1 festgestellt, ohne daß der Oxidationsprozeß sich zu einem Grubenbrand entwickelte. Im deutschen Steinkohlenbergbau wurden bei verdeckten Bränden Brandkennziffern bis zu 30 und bei offenen Bränden bis zu 90 ermittelt. Eine direkte Zuordnung der Brandkennziffer zur Temperatur im Bruchraum ist im Gegensatz zu Laboratoriumsuntersuchungen kaum möglich.

Wird eine Selbstentzündung festgestellt, dann ist für die Beurteilung der Gesamtsituation, insbesondere der Explosionsgefahr, der CH_4-Gehalt im Bruchraum des Alten Mannes unmittelbar hinter dem Strebausbau und im Zwickelbereich vom Streb zur Ausziehstrecke von Bedeutung. Gegebenenfalls sind an geeigneten Probenahmestellen Explosionskennziffern (EKZ) zu ermitteln und deren Entwicklung sorgfältig zu beobachten. Liegen die Gasgehalte im explosionsfähigen Bereich, dann kann bei Vorhandensein eines Brandherdes eine CH_4-Zündung nicht ausgeschlossen werden. Diese kann verhindert werden, wenn die Gaszusammensetzung im Alten Mann sauerstoffarm ist (< 12 Vol.-%). Treten zusätzlich zum Methan (CH_4) Wasserstoff (H_2) oder höhere Kohlenwasserstoffe in höheren Konzentrationen auf, muß der Sauerstoffgehalt auf Werte < 6 Vol.-% verringert werden. Eine Maßnahme, den Alten Mann sauerstoffarm zu halten, ist die vorbeugende Inertisierung mit Stickstoff (N_2).

Meßtechnische Überwachung während des Abbaus

Für die Beurteilung des Oxidationsverlaufs ist eine verstärkte CO-Überwachung des Gesamtwetterstroms und der Gasabsaugung (wenn vorhanden) sowie eine regelmäßige Kontrolle der Gaszusammensetzung im Alten Mann erforderlich.

Im einziehenden Wetterstrom ist die jeweilige Grundbelastung und am Ende der Ausziehstrecke die CO-Gesamtbelastung zu ermitteln. Außerdem ist der CO-Volumenstrom in der evtl. vorhandenen Gasabsaugung festzustellen. Beide CO-Ströme bilden den Gesamt-CO-Zustrom. Die Differenz dieser Ströme zur Grundbelastung ergibt den CO-Zustrom des Betriebs.

Ferner können in bestimmten Abständen (150 m - 300 m) in der ausziehenden Abbaustrecke zusätzliche CO-Meßgeräte installiert oder aber Handmessungen vorgenommen werden, um in den Streckenabschnitten die CO-Belastungen (Profile) zu ermitteln. Dadurch kann erkannt werden, an welchen Stellen die CO-Mengen zuströmen und an welchem Punkt eine weitere Zunahme nicht mehr auftritt (Bild 2).

In der ausziehenden Abbaustrecke werden an der Anlaufkante und in Abständen von 10 - 30 m Schnüffelrohre zum Bruchraum eingebracht, an denen zunächst täglich und zu späteren Zeitpunkten in größeren Zeitabständen Gasproben entnommen werden, die auf CO, CH_4, H_2, Kohlenwasserstoffe, O_2 und CO_2 analysiert werden. Mit fortschreitendem Abbau können von den weiter zurückliegenden Schnüffelrohren nur noch die bedeutsamen ständig überwacht werden (Indikator-Schnüffelrohre). Eine Brandgas-Meßanlage eignet sich für die gleichzeitige Überwachung mehrerer Meßstellen. Sie kann über einen steuerbaren Meßstellenumschalter von vier Probenahmestellen

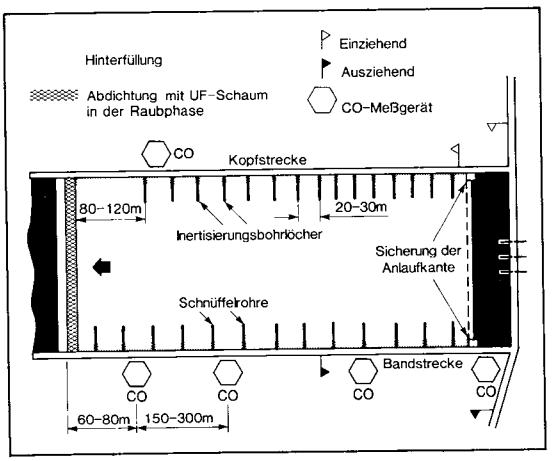


Bild 2: Abdichtungsmaßnahmen und meßtechnische Überwachung

die Konzentrationen von CH_4, CO, H_2, O_2 und CO_2 in den Grubengasen bestimmen und nach über Tage fernübertragen, so daß auch hierüber eine Gesamtbeurteilung der CO-Entwicklung, der Explosionsfähigkeit der Gasgemische und des Inertisierungserfolgs möglich ist. Mit dem selben Meßstellenumschalter könnte auch der eigensichere Gaschromatograph zur Überwachung mehrerer Probenahmestellen betrieben werden. Die Gasabsaugung ist ebenfalls ständig auf CO-Konzentrationen zu überwachen. Neben der Gasprobe aus der Sammelleitung des Abbaubetriebs sind auch Gasproben aus einzelnen Gasbohrlöchern zu entnehmen.

Vorbeugende Inertisierung

Läßt die CO-Entwicklung erkennen, daß über Dauer mit einer höheren CO-Produktion als 20 - 30 l/min gerechnet werden muß, so ist eine vorbeugende Inertisierung des Bruchraums im Alten Mann ratsam. Zu diesem Zweck werden in Abständen von 20 - 30 m Bohrlöcher von 6 - 12 m Länge in den Alten Mann gestoßen und verrohrt. Über diese Rohre wird der Stickstoff aufgegeben. Die Stickstoffaufgabe sollte 80 - 120 m hinter dem Strebstand vorgenommen werden, damit in den offenen Strebraum nur wenig Stickstoff austritt. Wenn mit fortschreitendem Abbau ein neues Bohrloch mit Stickstoff beaufschlagt wird, ist anschließend das vom Abbau am weitesten entfernt liegende Bohrloch abzuschlagen.

Durch diese Systematik wird gewährleistet, daß immer mehrere Bochlöcher im Abstand von 80 - 200 m dem Streb folgend mit Stickstoff versorgt werden, und daß der Stickstoff im Alten Mann hinter dem Streb günstig verteilt wird. So kann die Gaszusammensetzung im Bereich des noch aufgelockerten Bruchraumes durch geringe Stickstoffströme sauerstoffarm (< 12 Vol.-%) gehalten werden. Der O_2-Gehalt kann bei Bedarf durch Aufgabe zusätzlicher Stickstoffmengen noch weiter gesenkt werden (< 6 Vol.-%).

Der Überwachung des Inertisierungserfolgs dienen die für die Gasprobenahme auf der Ausziehseite eingebrachten Schnüffelrohre. Bild 3 stellt die mittleren Gasprofile (CO, O_2) über die gesamte Laufzeit eines Abbaubetriebes dar.

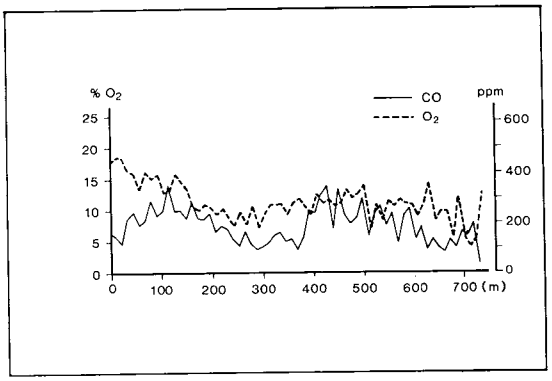


Bild 3: Mittlere Gasprofile aus Schnüffelrohren gemessen über Laufzeit des Abbaubetriebes

Um erkennen zu können, welchen Weg der Stickstoff mit den Schleichwettern durch den Alten Mann nimmt und welche Zeit für das Durchströmen benötigt wird, kann dem Stickstoff ein Markierungsgas (z. B. Schwefelhexafluorid SF_6) beigegeben werden. Nach Auswertung der Messungen im Streb und an den Schnüffelrohren der Ausziehstrecke kann der Stickstoff optimal hinsichtlich Aufgabeort und -menge gesteuert werden.

Brandschutzmaßnahmen während der Raubphase

Beim Abbau von Betrieben mit zur Selbstentzündung neigenden Begleitflözen geht die CO-Produktion im Alten Mann dort zurück, wo der Bruchraum verdichtet ist. Bei Stillstand des Betriebes hört die nachfolgende Verdichtung auf, so daß während der Raubphase über einen längeren Zeitraum Schleichwetter über die im Bruchraum liegende Kohle fließen kann. Aus diesem Grunde muß in der Raubphase an der Auslaufkante ein ausreichend großer und befahrbarer Wetterweg hergestellt und aufrecht erhalten werden. Ferner müssen folgende Maßnahmen getroffen werden:

- Die Zwickel zwischen Streckenbegleitdamm und Bruchraum sind bis zum Streb zu verfüllen.
- Die Gesamtschildsäule zum Bruchraum ist mit UF-Schaum (z. B. Isoschaum) abzudichten.
- Der Streb ist nach dem Rauben der Schilde mit Türstockausbau zu sichern; der freie Raum zum Alten Mann ist mit UF-Schaum zu verfüllen und abzudichten.

Muß die vorbeugende Inertisierung beibehalten werden, kann man in größerer Entfernung 60 - 80 m vor Erreichen der Endstellung des Strebs Rohrleitungsstränge im Streb einbringen. Der Abstand der Rohrstränge im Streb sollte 20 - 40 m betragen. Die Endlänge der Rohrstränge sollte unterschiedlich bemessen werden, z. B. 30, 50 und 60 m (Bild 4). Neben den Inertisierungsrohren können für das Messen der Gaszusammensetzung und des Inertisierungszustandes im Bruchraum auch gesonderte Rohrstränge von z. B. 20 m Länge im Streb verlegt werden. Der Rohrstrang muß aus Einzelrohren, die dem Schrittmaß des Strebausbaus entsprechen (z. B. 0,75 m Länge), bestehen. Das Endrohr ist gegen Verstopfen und Zusammendrücken durch eine Schutzkappe besonders geschützt und durch einen Anker im Liegenden gesichert. Dadurch kann der Rohrstrang nicht mitgezogen werden. Vor dem Rücken des Ausbaus werden die Rohrstränge jeweils verlängert. Die bisher eingesetzten Rohre hatten einen Durchmesser von DN 43 mm, die Wand-

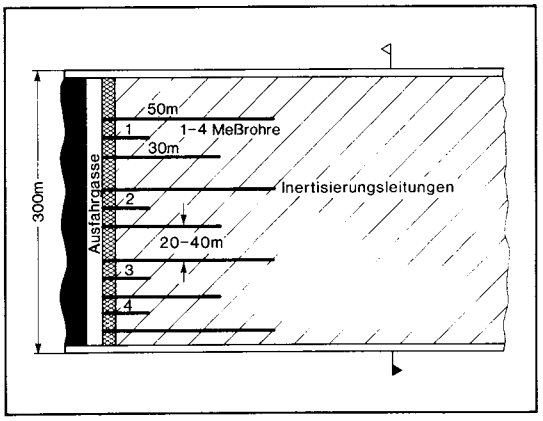


Bild 4: Inertisierungsleitungen im Streb

stärke betrug 6 - 7 mm. Die Verbindung der Rohre erfolgte durch Muffen mit Rundgewinde (Bild 5).

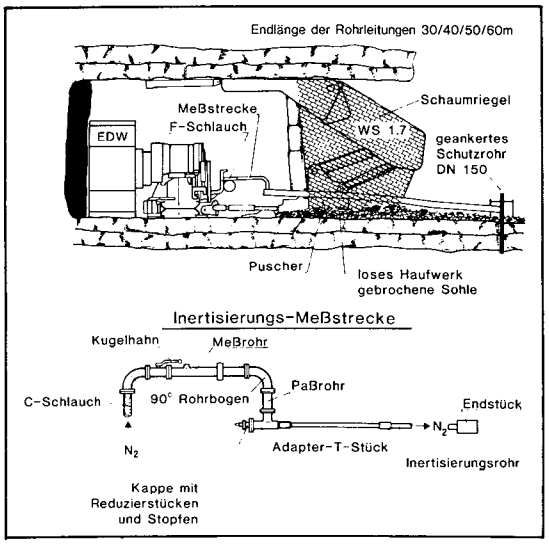


Bild 5: Streb-Inertisierungsleitung

Durch Aufgabe von Druckuft kann die Durchgängigkeit des Rohrstrangs überprüft werden. Für Messungen kann am strebseitigen Ende des Rohrstrangs jeweils ein Adapterstück angebracht werden. Die Rohrstränge sind nach Rauben des Schildausbaus bis in den offenen befahrbaren Wetterweg zu verlängern. Der befahrbare Wetterweg erlaubt die N_2-Zufuhr über beide Abbaustrecken durch eine Ringleitung. Die Rohrstränge nehmen N_2-Ströme von 2 - 4 m³/min auf.

Schlußbemerkung

Auf einem Bergwerk des Ruhrgebiets wurden nach diesem Verfahren vier Abbaubetriebe in den Jahren 1985 bis 1989 erfolgreich zu Felde geführt; ein fünfter Abbaubetrieb ist z. Z. im Verhieb, und fünf weitere Bauhöhen sind für den Abbau vorgesehen (Bild 6). Auf anderen Bergwerken ist das Verfahren bei ähnlichen Verhältnissen in abgewandelter Form zur Anwendung gekommen.

Durch dieses Verfahren können CO-Entwicklungen in Begleitflözen, die beim Abbau des tieferliegenden Flözes in den Bruchraum fallen, beherrscht werden.

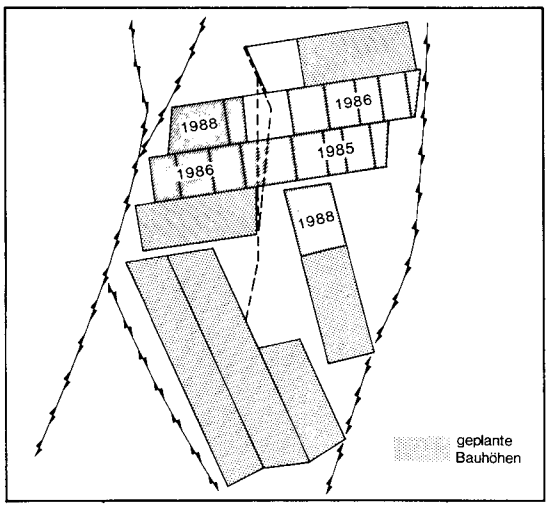


Bild 6: Abbaugrundriß Flöz Sonnenschein

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PREVENTION DES INCENDIES DANS LES CHANTIERS A COUCHES SUJETS AUX COMBUSTIONS SPONTANEES

M. Funkemeyer, F.J. Kock

La plupart des incendies provoqués par des combustions spontanées peuvent être contrôlés par l'application de mesures appropriées. Toutefois, des problèmes surviennent en raison de la présence de CO dans des nerfs de couches spécifiques s'effondrant dans les remblais au cours d'excavations effectuées dans les couches inférieures. En ce qui concerne les chantiers, il n'est généralement pas possible de prévoir s'il se produira une oxydation prolongée inoffensive ou un incendie provoqué par une combustion spontanée. Des mesures complexes d'obturation doivent donc y être prises pour empêcher les fuites d'air : remblayage des voies de portes au cours du creusement, assujettissement des parois, construction de murs de remblais hermétiques, remplissage des écoinçons entre les tassements et les remblais, consolidation des voies.

Il est nécessaire d'effectuer une surveillance intensive continue du CO dans tout le courant de ventilation et le système de drainage des gaz, ainsi qu'une analyse régulière de la composition des gaz dans le remblai. Outre le CO, il est également important de déterminer la teneur en hydrogène et en hydrocarbures pour évaluer l'état d'oxydation. Des tubes de murs de remblais sont installés du côté admission; de surcroît, les tubes installés dans le support de protection peuvent être installés dans le front de taille. Une inertisation préventive peut être effectuée, si cela s'avère nécessaire, par ce système de pipe-line. Des gaz révélateurs permettent de suivre les courants vagabonds et les courants d'azote.

ПРОТИВОПОЖАРНЫЕ МЕРЫ ДЛЯ ПРЕДУПРЕЖДЕНИЯ САМОПРОИЗВОЛЬНОГО ВОЗГОРАНИЯ В ТОНКИХ СЛОЯХ УГЛЯ

М. Функемайер и Ф.-Дж. Кок

Большинство самопроизвольных пожаров может быть предотвращено своевременным применением контрмер. Наибольшую проблему представляет собой выделение CO в определенных наклонных слоях угля, соединяющихся с выработанным пространством при разработке нижнего слоя. В начальной стадии разработки неизвестно, будет ли в наклонном пласте происходить безопасное окисление или самовозгорание. В таких условиях должен быть предпринят комплекс мер по герметизации: засыпка штреков, укрепление целиков, устройство плотных боковых закладок, укрепление штолен. Необходимы постоянный контроль над содержанием CO, который принимается здесь за решающий фактор, а также определение содержания водорода и углеводородов для оценки уровня окисления. В вентиляционных выработках устанавливаются закладные трубы; кроме того, могут быть установлены трубопроводы между опорами проходческого щита. В случае необходимости эти воздуховоды помогут произвести предупредительную инертизацию. Случайные потоки воздуха и азота могут быть прослежены с помощью газов-индикаторов.

易于自燃的夹石煤层工作区的防火问题

M. 丰根美益，执照工程师，F. J. 卡克，执照工程师

多数的矿井自然火灾都可由及时地采取适当的对策来控制。但是，在煤层开采过程中，顶上悬支的会产生一氧化炭的煤层落入采空区时的火灾就会发生问题。在夹石煤层中工作时，是否仅是在一阵无害的氧化，还是自燃的矿火开始，在起初时不易辨别出来。在这种情况下，必须施行繁琐的密封措施以隔绝空气：掘进时采空区运输巷的填土，夹层的卡紧，路边充填带的建造，充填带与采空区拱肩的填塞，及行车路的巩固。必须不断进行整个通风气流和排气系统中一氧化炭的强化监测和采空区气体成份的分析。除此之外，氢和碳氢化合物的一氧化炭合成的测定，对氧化程度的判断也是很重要的。在巷道进口，装上通风管；另外，掩护支架上的通风管可安装在工作面上。有必要的话，可通过这套管道系统进行及时的防护性惰性化作业。用过扩散通风和氮气流后，再用示踪气体。

КОМПЛЕКСНАЯ ЗАЩИТА ГОРНЫХ РАБОТ ОТ САМОВОЗГОРАНИЯ УГЛЯ

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РЕЗЮМЕ

Освещены результаты исследований влияния теплофизических параметров угля на интенсивность процесса самовозгорания. Установлено, что из всех рассмотренных факторов особую значимость имеет природная (естественная) влажность угля, которая предопределяет сорбционную способность последнего к кислороду воздуха и, следовательно, обуславливает продолжительность инкубационного периода. Показано,что успешное решение проблемы предупреждения эндогенных пожаров в угольных шахтах возможно только при комплексном применении профилактических мер с использованием так называемых трех защитных линий, которые реализуются соответственно на стадии проектирования, в процессе ведения очистных работ и после их завершения. Такая схема составляет по существу основу разрабатываемых в каждом угольном бассейне СССР инструкций по предупреждению и тушению эндогенных пожаров в шахтах.

Согласно современным воззрениям самовозгорание угля представляет собой совокупность физико-химических и теплофизических процессов, протекающих в угольном скоплении. Известно, что уголь является высокопористым веществом, весьма сложным по составу и химической структуре, и представляет по существу систему "уголь-вода-метан". При ведении горных работ равновесное состояние данной системы нарушается, вследствие чего уголь выделяет метан и сорбирует кислород. Интенсивность протекания указанных процессов во многом зависит от содержания в угле влаги, заполняющей трещины, микро- и макропоры. Особую значимость этот фактор имеет в начальный период окисления, когда происходит зарождение и развитие очага самовозгорания угля. Исследованиями ВостНИИ установлено, что по характеру изменения влажности угля процесс его самовозгорания во времени возможно разделить на три стадии: разогревание, ин-

тенсивное испарение влаги из угля и углубление зоны испарения. Первая стадия характеризуется незначительной скоростью сушки, вследствие чего она протекает в течение длительного времени (2 мес) и сопровождается снижением влажности на 0,8–1,0 % (рис.1, кривая 1) и повышением температуры угля на 10–12 К по сравнению с естественной (рис.1, кривая 2).

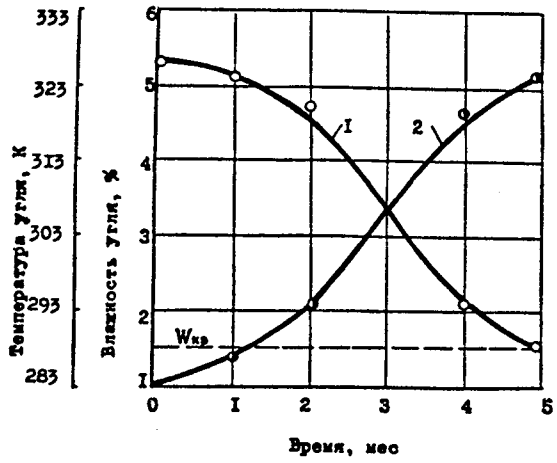


Рис.1.

В основном окислению подвергается свежеобнаженная поверхность угля, при этом вследствие аккумуляции свыше 50 % генерируемого тепла происходит разогревание угольного скопления. Указанный процесс возможен даже при незначительном притоке воздуха, соответствующем ламинарному режиму. Вторая стадия протекает также длительное время и характеризуется резким повышением величины влагосодержания воздуха в исходящей из угольного скопления струе. С уменьшением влажности в угольном веществе происходит интенсивное раскрытие макро- и микропор и увеличение реагирующей поверхности, к которой имеется доступ кислорода, что обуславливает интенсификацию реакции окисления и повышение температуры угля до 318 К. Скорость движения воздуха в данной стадии соответствует переходному режиму.

При снижении влажности угольного вещества на 60 % от начальной наступает третья стадия, особенностью которой является уменьшение интенсивности влагообмена вследствие перемещения зоны испарения внутрь кусков угля и, следовательно, увеличения пути диффузии влаги. Это сопровождается также уменьшением количества аккумулированного тепла, однако, в связи с концентрацией его в относительно малом объеме скопления температура угля может резко повыситься и произойти его самовозгорание. При достаточном притоке воздуха переход процесса самонагревания угля в возгорание происходит в таких условиях в течение 5–7 суток. Стадия протекает при турбулентном режиме движения воздуха через угольное скопление. Рассмотренные стадии составляют так называемый инкубационный период самовозгорания, который при прочих равных условиях тем продолжительнее, чем выше природная (естественная) влажность угля и меньше его сорбционная способность к кислороду. Особо следует отметить влияние на продолжительность инкубационного периода начального перепада температур между фильтрующимся воздухом и углем, определяющим направленность и интенсивность тепло- и массообменных процессов. Для бурых углей, характеризующихся большим содержанием влаги (35–38 %), нежесткой молекулярной структурой и меньшей плотностью по сравнению с каменными углями, указанные выше стадии протекают значительно быстрее, вследствие чего инкубационный период самовозгорания составляет 0,5–1,0 мес. В соответствии с рассмотренным механизмом формирования очагов самовозгорания угля в основу решения проблемы предупреждения эндогенных пожаров на шахтах СССР заложен принцип комплексного применения профилактических мер с использованием трех защитных линий.

I. Первая линия реализуется на стадии проектирования горных работ и предусматривает выполнение следующих требований.

I.I. Вскрытие и подготовку крутых мощных пластов угля, склонного к самовозгоранию, осуществлять с помощью полевых штреков и промежуточных квершлагов. Причем полевые штреки должны закладываться на таком расстоянии от нижележащего в свите пласт, чтобы исключалась возможность подработки их и образования аэродинамической связи с выработанным пространством. Согласно исследованиям ВостНИИ это расстояние составляет от 6 до 42 м на откаточном и от 8 до 73 м на вентиляционном горизонте в зависимости от показателя литологического состава пород лежачего бока, мощности и угла падения надрабатывающего пласта и наклонной высоты этажа.

I.2. Отработку пластов самовозгорающегося угля производить односторонними по простиранию выемочными полями с ведением очистных работ к вскрывающей выработке и неотработанному массиву крыла шахтного поля. При этом не допускаются вскрытие и подготовка выемочных полей (блоков) с главных квершлагов, прорезка околоквершлажных профилактических целиков между выемочными полями, а также приведение угольных целиков у капитальных выработок в неудароопасное состояние с нарушением их воздухонепроницаемости.

I.3. Системы разработки и технологические схемы очистных работ использовать только в таких горно-геологических условиях, которые соответствуют техническим требованиям их применения.

I.4. Поддерживать наименее опасные в пожарном отношении режимы проветривания шахт и выемочных участков за счет применения рациональных схем и способов вентиляции, позволяющих сократить прососы воздуха через выработанные пространства, между выемочными полями, пластами, действующими выработками и земной поверхностью. Принятая схема проветривания шахты (или крыла) должна обеспечивать направление исходящей струи воздуха в сторону неотработанной части массива и исключать возможность поступления воздуха в изолированные поля. Проветривание очистных забоев необходимо осуществлять, как правило, по возвратноточной схеме. Прямоточные схемы проветривания допускается применять на газообильных участках при отсутствии геологических нарушений и выемке пласта на полную мощность в один слой.

2. В процессе ведения очистных работ необходимо осуществить вторую линию защиты, которая включает в себя выполнение следующих профилактических мероприятий: возведение искусственных изоляционных полос и барьеров, объемную обработку выработанных пространств действующих выемочных столбов (лав), особенно на участках пластов с геологическими нарушениями, растворами ингибиторов, снижение окислительной способности угля в надштрековых и околобремсберговых (околоуклонных) целиках, обеспечение рационального распределения вентиляционных давлений с помощью отрицательных и положительных регуляторов и др. Для реализации указанных мероприятий ВостНИИ разработан ряд новых способов предупреждения самовозгорания угля, которые прошли стадию опытной эксплуатации и внедряются на шахтах восточных районов СССР.

2.I. Способ выравнивания давлений воздуха для борьбы с подземными пожарами и газом. Способ основан на создании в ограниченном объеме горных выработок (с включением выработанных пространств) с помощью шлюзовых устройств и вспомогательных подземных вентиляторов такого вентиляци-

онного давления, которое снижает утечки воздуха через отработанное пространство до пожаробезопасного значения и одновременно выносит из него газ, ликвидируя местные скопления метана. С применением этого способа отработано свыше 50 выемочных полей на мощных крутых пластах угля в Прокопьевско-Киселевском районе Кузбасса.

2.2. Способ торможения процесса самонагревания угля в выработанных пространствах действующих выемочных полей с применением антипирогенов в виде аэрозолей. Сущность его заключается в том, что из очистного забоя сонаправленно с путями утечек воздуха в выработанное пространство с помощью обособленной вентиляционной струи вводится антипироген в виде аэрозолей (рис.2). Он предназначен для

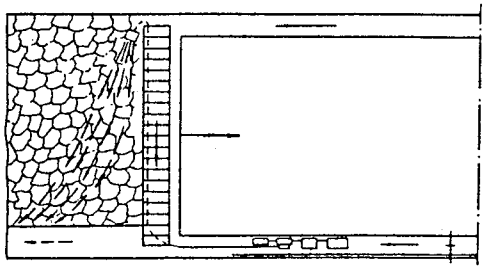


Рис.2

предупреждения эндогенных пожаров в действующих выемочных полях на пологих и наклонных пластах без остановки очистных работ и без изменения их вентиляционных параметров. Способ прошел промышленные испытания на шахтах "Нагорная" ПО "Прокопьевскуголь", "Распадская", им.В.И.Ленина и им.Шевякова ПО "Южкузбассуголь", где с применением его добыто более 2 млн.т угля в зонах повышенной эндогенной пожароопасности без признаков самонагревания угля. Результаты испытаний показали основное преимущество данного способа по сравнению с известными техническими решениями - возможность включать профилактическую обработку выработанного пространства в технологический процесс добычи угля. В настоящее время аэрозольная профилактика эндогенных пожаров применяется на шахтах Кузбасса и других угольных месторождений восточных районов страны.

2.3. Способ профилактической обработки межгоризонтного целика порошковым антипирогеном при щитовой системе разработки (рис.3). Практически он реали-

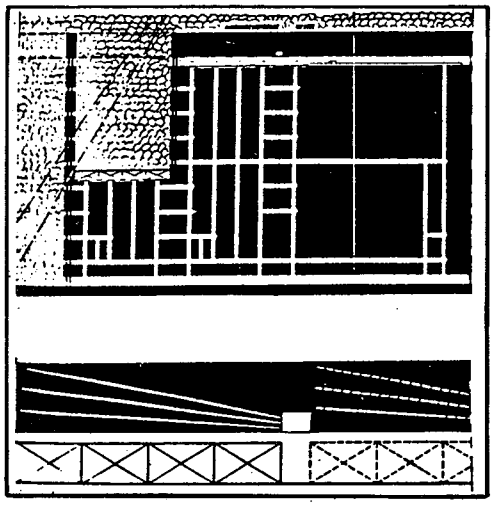


Рис.3

зуется следующим образом. В специально пробуренные скважины подается под давлением порошковый антипироген в объемах, определяемых в зависимости от размеров обрабатываемого массива. При обрушении межгоризонтного целика происходит перемешивание угля с антипирогеном. В случае повышения температуры угля до 30-50°С антипироген разлагается с поглощением тепла и выделением инертных газообразных продуктов, в результате чего скорость реакции окисления замедляется, процесс растягивается во времени, и самовозгорание не происходит.

Промышленные испытания способа проводились на шахтах "Черкасовская" ПО "Киселевскуголь", "Зиминка", им.Калинина и им.Ворошилова ПО "Прокопьевскгидроуголь". С применением его безаварийно отработано более 50 щитовых выемочных столбов с поло-

жительным эффектом в сложных горно-геологических и горно-технических условиях шахт Прокопьевско-Киселевского района Кузбасса.
Применение антипирогенов в виде аэрозолей и порошков на шахтах восточных районов СССР регламентируется отраслевым "Руководством по применению способов торможения развития самонагревания угля в выработанных пространствах выемочных полей шахт" (Кемерово, 1986).
2.4. Профилактика эндогенных пожаров с использованием гелеобразующих составов (рис.4). Способ предназначен

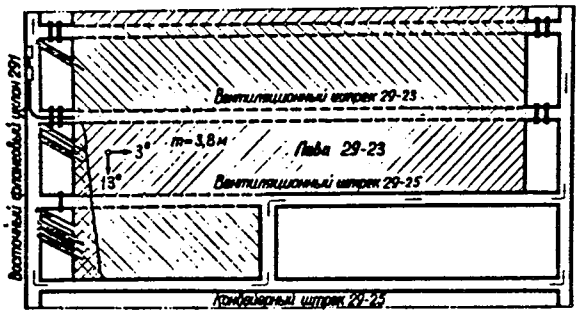


Рис.4

для предупреждения эндогенных пожаров за счет снижения утечек воздуха через выработанное пространство и дезактивации концентрированных потерь угля в нем. Эти составы представляют собой смесь компонентов, при смешивании которых в определенном соотношении образуется гелеобразная смесь, обладающая антипирогенными свойствами. Гелеобразующий состав способен вспениваться и загустевать через определенное время, что и определяет область его применения. Состав может подаваться в монтажные и демонтажные камеры для снижения утечек воздуха через выработанное пространство, в пространство между двойными перемычками, а также использоваться с целью тампонажа трещин в приконтурных зонах перемычек и тушения эндогенных пожаров как пеногель на основе инертного газа. Способ прошел промышленные испытания на шахтах "Распадская", им.В.И.Ленина, "Томская", "Зыряновская", "Байдаевская" и "Новокузнецкая" ПО "Южкузбассуголь". Испытания показали, что способ является технологичным и более эффективным по сравнению с глинистой пульпой. В настоящее время он успешно применяется на шахтах восточных районов СССР.
3. После окончания производства очистных работ реализуется третья защитная линия, которая включает: качественную и своевременную изоляцию отработанных участков от действующих горных выработок и земной поверхности в регламентированные сроки, устанавливаемые нормативными документами; организацию контроля за ранними признаками самовозгорания угля и состоянием изоляции.
Своевременная реализация в полном объеме схемы комплексного решения проблемы профилактики эндогенных пожаров за счет осуществления рассмотренных выше защитных линий, особенно при переходе шахт на нижние горизонты, дает возможность практически полностью исключить опасность возникновения очагов самовозгорания при ведении горных работ на пожароопасных пластах. Эта схема составляет по существу основу разрабатываемых в каждом угольном бассейне инструкций по предупреждению и тушению эндогенных пожаров в шахтах.

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COMPLEX SPONTANEOUS COAL IGNITION PROTECTION MEASURES IN MINING

V.M. Abramov, V.B. Popov, and L.P. Belavencev

When coal mines go deeper and coal seams subject to spontaneous combustion are worked, controlling endogenous gob fires of active mines is of particular importance. To reduce fire hazard in USSR coal mines, a complex approach to spontaneous coal ignition is adopted:

1. During mine design, particular attention is paid to ventilation flow stability, to openings to surface, and to lateral drift layout.
2. During installation and/or dismounting, chambers are filled with gelling compounds; antipyrogen aerosols are applied when pillars (face) are robbed, especially in geologically disrupted seams, stump and inclined drift coal pillars are treated with antipyrogens in aqueous solution or powders. Much attention is paid to monitoring gas composition and temperature of the mine atmosphere for early signs of spontaneous ignition.
3. Abandoned workings are isolated and monitored for signs of spontaneous combustion. A number of spontaneous coal heating prevention methods are being adopted at the mines of eastern regions. These developments essentially constitute the basis for the spontaneous mine fire prevention and fire fighting instructions under development in every coal basin.

MESURES COMPLEXES DE PROTECTION CONTRE LA COMBUSTION SPONTANEE DU CHARBON DANS LES MINES

V.M. Abramov, V.B. Popov et L.P. Belavencey

A mesure que l'on creuse plus profondément dans les mines de charbon et que l'on effectue des excavations dans les couches de charbon exposées aux combustions spontanées, il devient crucial de contrôler les incendies endogènes dans les remblais des mines en activité. Pour diminuer les risques d'incendie dans les mines d'URSS, nous avons adopté une méthode d'approche complexe de la combustion spontanée du charbon, comportant les éléments suivants:

1. Au cours de la phase du plan de la mine, on attache une attention particulière à la stabilité du courant de ventilation, aux ouvertures vers la surface, ainsi qu'à la disposition des galeries latérales.
2. Pendant l'installation et/ou le démontage, on emplit les fourneaux de composés gélifiants; on applique des aérosols antipyrogènes lorsque les piliers (fronts de taille) sont écrémés, particulièrement dans les couches géologiques perturbées; on traite les tronçons des piliers et les piliers des galeries inclinées avec des antipyrogènes en solutions aqueuses ou en poudres. Enfin, on surveille tout particulièrement la composition gazeuse et la température de l'atmosphère minière afin de détecter les signes avant-coureurs d'une combustion spontanée.
3. On isole les tailles abandonnées et on les surveille afin d'observer tout signe de combustion spontanée. Un certain nombre de moyens de prévention de l'échauffement du charbon ont été adoptés dans les mines des régions de l'Est. Ces méthodes nouvellement développées forment la base des mesures de prévention des incendies miniers spontanés ainsi que des directives de lutte anti-incendie dans chaque bassin houillier.

KOMPLEXE METHODEN ZUM SCHUTZ GEGEN SELBSTENTZÜNDUNG IN KOHLEGRUBEN

V. M. Abramov, V. B. Popov, L. P. Belavencey

Wenn die Kohlegruben in größere Tiefen vorstoßen und zur Selbstentzündung neigende Flöze abgebaut werden, ist die Beherrschung der von innen entstandenen Brände besonders wichtig. Um die Feuergefahr in sowjetischen Kohlegruben zu vermindern, wurde ein komplexer Plan zur Lösung des Problems der Selbstentzündung angenommen:

1. Während der Grubenauslegung wird besonders auf die Bewetterungsstabilität, auf Tagesöffnungen und Streckenplanung geachtet.
2. Während der Installation und/oder des Raubens werden Hohlräume mit Gelsubstanzen gefüllt, feuerhemmende Aerosole werden beim Rauben von Pfeilern angewendet, besonders in geologisch gestörten Lagerstätten. Blindstrecken und Kohlepfeiler im Einfallen werden mit feuerhemmenden Mitteln in wässeriger Lösung oder Stäuben behandelt. Es wird zum Zweck der Brandfrüherkennung sorgfältig auf die Überwachung der Gaszusammensetzung und der Temperatur der Grubenwetter geachtet.
3. Abgeworfene Grubenbaue werden abgedämmt und auf Anzeichen einer Selbstentzündung überwacht. In den Gruben der östlichen Regionen wird zur Zeit eine Reihe von Methoden zur Verhütung der Selbstaufheizung der Kohle eingeführt. Diese Entwicklungen bilden die Grundlage für die Verhütung von Selbstentzündungsbränden und die noch in Entwicklung stehenden Brandbekämpfungs-Vorschriften für jede Kohlelagerstätte.

煤矿中综合的煤炭自燃防止措施

V．M．阿勃拉莫夫，V．B．朴卜夫
及 L．P．贝拉凡赛

当开矿深度增加，而且煤层易发生自燃现象时，控制正在开掘煤矿里废石的内生火灾特别重要。为减少苏联煤矿的火灾危害，我们进行了下列综合煤炭自燃防止措施：

1．煤矿开采设计时特别注意通风流通的稳定性，通至地面的通道，及横坑道的布置。

2．开筑或拆除孔道时，必须先注满液胶剂；当支柱（或煤层）被移去时，须要喷防火悬浮剂，特别在地质断层处，柱墩及斜孔道支柱必须以水溶液或粉状的防火剂处理。气体成分及矿内大气温度的检定都可作为自燃迹象的预测，须特别注意。

3．遗弃的矿井应隔离并检定是否有自燃的迹象。在东部矿区我们曾采用了一些防止煤碳自热的方法。这些主要形成了各煤区预防自发矿火及救火细则的基楚。

DETECTION AND CONTROL OF FIRES AND HEATINGS IN SHALLOW, ABANDONED COAL MINES

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ABSTRACT

Heatings and fires in shallow, abandoned coal mines create an environmentally undesirable hazard in the Witbank area in South Africa, as well as locations in Europe and North America.

A research programme was set up in South Africa to detect and control the occurrence and extent of subsurface heatings and fires. Prior to any remedial action being taken to control or extinguish a heating or fire, it is essential to evaluate underground conditions in order to determine the most effective control method. Normally, such workings cannot physically be entered due to poor ground conditions and the presence of heat and toxic gases. Two novel detection methods have been developed by the Chamber of Mines Research Organisation (COMRO) for the purpose of identifying the nature and extent of such heatings remotely, via surface boreholes. Temperature monitoring allows for the detection of heating intensity and location. To determine areas of uncontrolled air infiltration into the workings, tracer gas technology is used. In addition, a method for controlling a fire which has been successfully used in South Africa is described.

INTRODUCTION

The occurrence of underground heatings and fires in shallow, abandoned workings are a legacy of early mining practices. Poor mining control has lead to the partial collapse of supposedly sealed workings, thus allowing a readily available supply of fresh air to enter. Piles of broken carbonaceous material can react with the renewed oxygen supply and an exothermic oxidation reaction takes place. The heat produced allows the reaction speed to increase, if it is not dissipated. This mechanism, together with mining and geological factors gives rise to the establishment of a heating.
If this continues unchecked, an underground fire will result. This in turn will affect the surface, with the possibility of affecting many hectares of otherwise valuable land.
In South Africa, many attempts have been made to contain or extinguish such locations affected by this problem. Some have been successful, but the operations were both time - consuming and costly. By using these inexpensive heating detection methods, remedial action to control the heating would only be required at the pinpointed heating location or air infiltration points..

In an attempt to combat this occurrence the Coal Mining Laboratory (CML)of COMRO developed a method to detect the nature and extent of a heating or fire by means of temperature monitoring and isothermal plotting. To ascertain airflow patterns within the workings, tracer gas studies have been undertaken. By examining the results of these detection methods in conjunction, a heating location and intensity may be found together with fresh air paths which feed these locations. Once this information is available, a

decision could then be made to utilise the most effective control method to contain or extinguish the heating or fire.

The workings in which the heatings occur are normally inaccessible and the only way of accessing information from the workings is by means of placing instrumentation in surface boreholes.
In order to develop and test the validity of these methods, it was necessary to obtain a site affected by spontaneous combustion which was inaccessible and could be monitored via surface boreholes. Such a site was found in the Witbank area. It consisted of abandoned bord and pillar workings at a depth of 20 metres. The area of the site was approximately 60000 square metres. The site was isolated on all four sides by a trench and as such there was no threat of a fire affecting nearby current workings. Twelve boreholes had been drilled into the workings to allow access to the underground environment for monitoring instrumentation. Following a meeting with mine management early in 1987, it was agreed that CML would be able to carry out field investigations at this site.

LOCATING UNDERGROUND HEATINGS

Method 1 - Temperature Monitoring and Isothermal Plotting

Using thermocouples and computing facilities at the test site it was possible to accurately locate the position and extent of underground heatings. Thermocouples were placed in boreholes and temperatures were monitored weekly for approximately eighteen months. Figures 1 and 2 show the advancement of heatings over this period. It can be seen initially that three distinct areas of heating activity were located. Ambient temperature was found to be 25 C. Each heating had a central temperature of between 50 and 80 C. Figure 2 illustrates how the three heatings have merged into one, more intense heating, with central temperatures in excess of 250 C.



Figure 1. Heating Locations,

Fire Test Site, March 1987

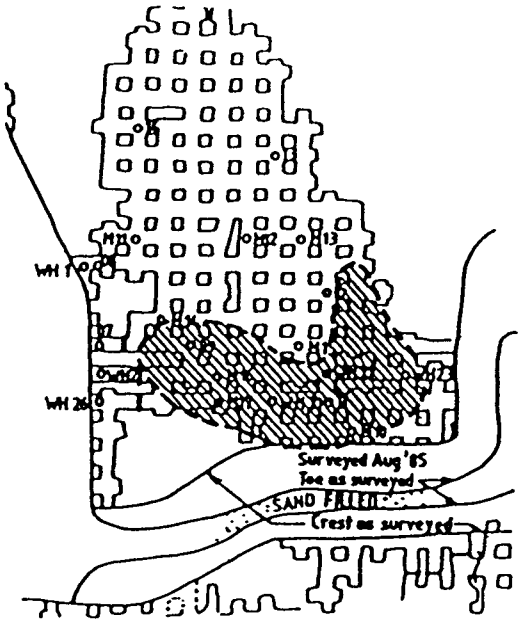


Figure 2. Heating Locations,

Fire Test Site, August 1988

Method 2 - Airflow Pattern Determination and Airflow Distribution

Airflow distribution is complex in a shallow abandoned mine where a heating and air leakage exists. The effects of convection, due to temperature differentials within

the workings, would probably induce air into the area, as well as create extensive recirculation. Thus, to evaluate this complex airflow system, tracer gas studies have been undertaken, in particular tracer gas decay.

As its name implies, a tracer gas is a substance which is mixed with the ventilating airstream in order to trace progress of the airstream through the mine workings. By studying the tracer gas concentration as a function of space and time, certain deductions can be drawn about airflow distribution. Sulphur Hexafluoride has been used extensively in North America and Europe for this purpose, and preliminary studies carried out by the Industrial Hygiene Laboratory of COMRO have further proved its applicability to tracer studies in the South African mining industry (Van der Linde, Gardiner, 1988).

Tracer Gas Decay

Tracers may be used to determine the rate of dilution or exchange of air in a semi-enclosed space, by observing the rate at which the tracer concentration diminishes with time. This is known as the tracer decay rate and conforms to the Exponential Dilution Law:

$$C2 = C1.e^{-Lt} \qquad (1)$$

where L = air exchange rate
C1 = tracer concentration at time t1
C2 = tracer concentration at time t2
t = time interval between t1 and t2

Rearranging equation (1) gives:

$$t = \frac{1}{L} \cdot \ln \frac{C1}{C2} \qquad (2)$$

An example, carried out by CML, illustrates an attempt to determine airflows in abandoned workings affected by spontaneous combustion. A quantity of Sulphur Hexafluoride was injected (using nitrogen) down a surface borehole into the workings, and samples were taken at an adjacent borehole. Sampling continued until it was considered that all the injected tracer gas had passed the sampling point.

Plotting the natural logarithm of the decay results (Figure 3), it can be seen that a straight line decay did not occur. The actual curve indicates that a certain amount of recirculation was taking place in the vicinity of the sampling point. If no recirculation had occurred, then the results would approximate to a straight line.

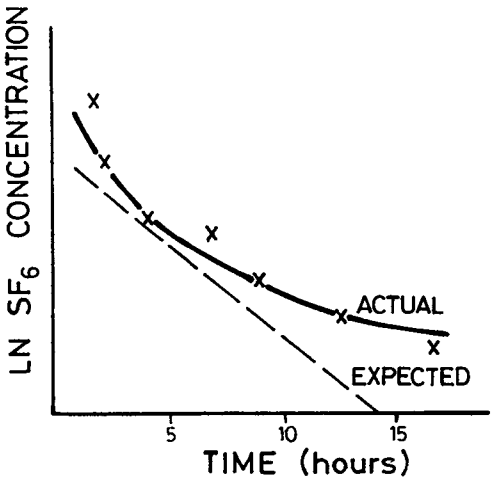


Figure 3. LN Tracer Gas Decay Against Time.

Therefore, in an attempt to quantify the amount of recirculation taking place,a further tracer gas study was undertaken at the test site, again using the tracer decay technique. In this case a large number of boreholes were sampled around the release borehole. Figure 4 illustrates the general airflow patterns found from that study. It can be seen from this Figure that two major recirculation paths occur in the southern part of the site. The main throughflow of air was found to travel in a generally south-north direction. The amount of recirculation was found to be approximately 80 to 90 percent of total airflow through the workings. Airflow velocity in the northern part of the site was found to be between 5 and 7 metres/minute. If these results are compared with those of the temperature monitoring studies (Figure 2), it can be seen that the main recirculation paths coincide with the main heating location, again in the southern part of the site. This implies that

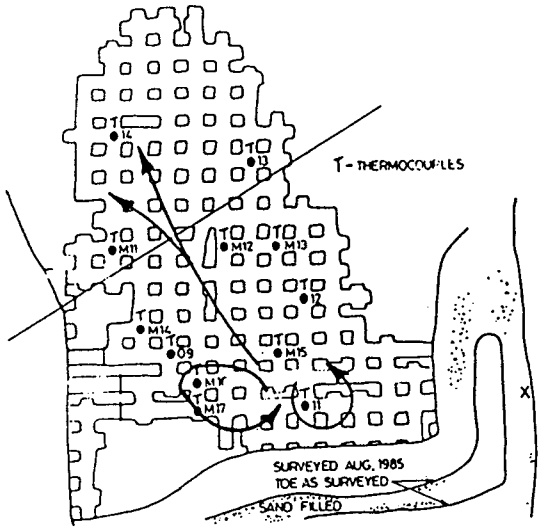


Figure 4. Airflow Patterns within Fire Test Site

heating activity is drawing fresh air into the workings and creating large thermal plumes. The oxygen - deficient air, now heated somewhat, is then drawn through the workings to the northern area, where it escapes via fissures. This will also create a "chimney " effect within the site, and more fresh air is then induced into the southern area.
From these results therefore, it is possible to pinpoint the areas of air leakage into and out of the workings. This method would be extremely useful if sealing techniques were to be considered as a control method. In addition, tracer gas decay could be used to determine whether a particular control method has been effective.

WET FLUSHING CONTROL OF AN UNDERGROUND HEATING IN ABANDONED MINE WORKINGS

Although not directly related to the detection methods noted previously in this paper, one particular control measure was so successful that it is worthy of mention. During a land rehabilitation scheme in the Witbank area, an abandoned mine with extensive workings was found to present a threat to building operations. The workings had a history of spontaneous combustion outbreaks and as such, no construction could take place on surface, until these workings were made safe. Certain criteria had to be met by the fire control operators. Firstly, the workings had to be made completely safe from spontaneous combustion outbursts. Next, no trenching could take place, as this would affect the land rehabilitation plans . The surface was to be left unchanged. Thirdly, the workings had to be prevented from bord collapses, which again would affect the surface. And finally, an unsightly fly ash dump (approximately 2 million tons) was to be removed. And so, an opportunity presented itself to the control operators to not only make safe the underground workings but also to remove a large eyesore from the horizon.
Several methods are available by which a heating or fire may be contained or extinguished. In this case however, a method of flushing or stowing material in the workings would be employed. By slurrying material underground, any voids in the workings would be filled. The slurry consisted of fly-ash mixed with water, which was gravity-fed down a 36 metre shaft. Figure 5 illustrates this. Incombustible fly-ash mixed with clinker arrived at the site from a dump approximately 800 metres distant.

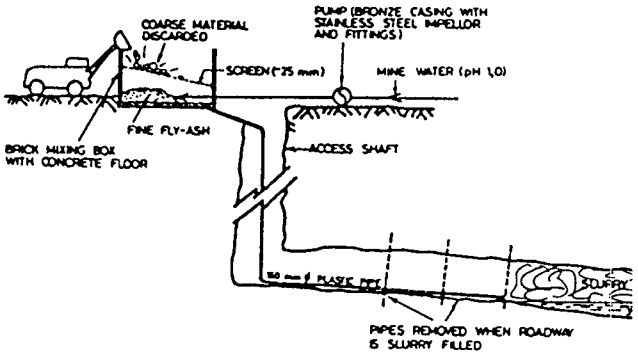


Figure 5. Gravity Feeding of Incombustible Material into Abandoned Workings

This material was then deposited into a brick-walled, concrete-floored hopper by means of a front end loader. The material was screened to minus 25 mm, with the coarse clinker being discarded. It was found that material above 25 mm did slurry effectively. As there was no readily available water

supply, the fines were mixed with water pumped from an adjacent underground mine. Since the water was extremely acidic, having a pH of 1,0, a bronze-cased pump with stainless steel impeller and fittings had to be used, to minimize corrosion. 150 mm diameter plastic pipes were used for both water and slurry delivery. The water mixed with the fly ash in the mixing box was then gravity-fed from this hopper into the accessible workings. Groundwater had accumulated to the roof in roadways nearest the barrier pillar, as the workings lay down-dip from the shaft bottom. Pipes were laid to the edge of this water and slurrying commenced. Once the slurry had filled to the roof, the last pipe was removed and slurrying continued until the last pillar near the shaft bottom was reached. The pipes were then laid in the next roadway and slurrying was recommenced. Having filled all the roadways in this manner, the pipes in the shaft were removed. The shaft was then infilled with the previously coarse clinker and other debris. The slurried material would set into a solid, firm mass, which would not only exclude air entering the workings (thereby preventing the occurrence of spontaneous combustion) but also prevented the workings from collapse. The whole operation was inexpensive compared with other control methods, and surface rehabilitation could continue as planned.

CONCLUSIONS

The problem of spontaneous combustion in shallow, abandoned workings is a legacy of early mining practices. Todays mining engineers have been left with finding quick and inexpensive solutions to this problem. However, prior to any control measures being used to contain or extinguish an underground fire,it is necessary to determine the nature and extent of such heatings. Failure to do this could allow the heating to continue unabated or even increase in intensity. In this light therefore, this paper has illustrated developments in the detection field. It has been shown that temperature monitoring and isothermal plotting can indicate the locations of heatings in shallow, abandoned workings. The movement of heatings can also be identified, as shown from the isothermal plots taken over an 18 month period. Once heating locations have been established, it would then be possible to utilise the most effective control method to isolate or extinguish these heatings. This would most certainly lead to large cost savings during remedial work.

Although temperature monitoring indicates the locality of a heating, it does not pinpoint areas of air infiltration into the workings. For a heating to propagate, fresh air is essential. In order to find the main leakage points in inaccessible abandoned workings, studies have been carried out with tracer gases for the purpose of finding such leakage points. In the study at the Witbank test site, large recirculation paths were found to occur, probably induced by thermal effects of a large heating, whose position was located by means of temperature detection. The main throughflow of air was in a south-north direction and having a velocity of between 5 and 7 metres/minute. Thus, main leakage points were found in both the southern and northern extremities of the test site. Once these locations are known, it would then be possible to exclude air from the workings by localized sealing. If abandoned workings were to be excavated, these two novel detection methods would enable an operator to determine how much recoverable coal he would be likely to mine.

Although not related to the detection methods mentioned, a specific control measure carried out at a mine in the Witbank area was discussed. Not only was land made safe for surface development but an environmentally undesirable waste dump was removed from public view.

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ACKNOWLEDGMENT

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DETECTION ET CONTROLE DES INCENDIES ET DES ECHAUFFEMENTS DANS LES MINES DE CHARBON ABANDONNEES

P. Sullivan

Les échauffements et les incendies se produisant dans les mines de charbon peu profondes abandonnées créent un risque potentiel de danger, peu souhaitable pour l'environnement. Ces effets, qui se manifestent en surface, sont habituellement limités aux pays, y compris l'Afrique du Sud, qui ont exploité autrefois des filons de charbon peu profonds. Ces zones d'incendie sont généralement inaccessibles et ne peuvent normalement être examinées qu'au moyen de trous de mine percés à la surface.

Nous avons étudié l'utilisation de systèmes de surveillance de la température permettant de détecter à distance les emplacements et les déplacements de ces échauffements. Nous présentons quelques résultats, ainsi que des comparaisons effectuées entre cette méthode de détection et les méthodes d'analyse des gaz et de thermographie aérienne à rayons infra-rouge.

Ce rapport examine également l'utilisation de gaz révélateurs pour déterminer les voies de passage d'air à l'intérieur de ces chantiers et calculer les emplacements non contrôlables d'infiltration d'air.

Enfin, il décrit certaines méthodes de contrôle utilisées en Afrique du Sud, comprenant l'installation de protections anti-incendie et diverses opérations de creusage et d'obturation.

ENTDECKUNG UND KONTROLLE VON BRÄNDEN UND AUFHEIZUNGEN IN FLACHEN, STILLGELEGTEN GRUBEN.

P. Sullivan

Aufheizungen und Brände in stillgelegten, flachen Gruben stellen eine potentielle Gefahr, auch für die Umwelt, dar. Die Auswirkungen dieser Vorkommen, die sich auf der Erdoberfläche bemerkbar machen, sind auf Länder wie Südafrika beschränkt, die schon immer flache, nahe an der Oberfläche befindliche Kohleflöze ausgebeutet haben. Die Brandgebiete sind meistens nicht zugänglich und können nur durch Bohrlöcher in der Oberfläche untersucht werden.

Die Anwendung von Temperaturüberwachungs-Einrichtungen wird untersucht, um aus der Ferne die Lage und Verlagerung der Brandherde zu überwachen. Einige der Ergebnisse werden aufgeführt und mit den Ergebnissen der Gasanalyse und der Luftaufklärung mit Infrarot-Thermographie verglichen.

Der Vortrag befaßt sich auch mit der Anwendung von Tracer-Gasen zur Bestimmung des Wetterflusses innerhalb unzugänglicher Gruben, sowie der Vorhersage der Herkunft von unkontrollierten Wetterströmen.

Der Vortrag schließt mit der Beschreibung von Methoden, die in Südafrika benutzt werden, einschließlich der Installation von Feuerbarrieren und von verschiedenen Graben- und Abdichtungsmethoden.

ОБНАРУЖЕНИЕ И КОНТРОЛЬ НАД ПОЖАРАМИ И ПОВЫШЕННЫМ НАГРЕВОМ В НЕГЛУБОКИХ ЗАБРОШЕННЫХ УГОЛЬНЫХ ШАХТАХ

П. Салливан

Повышение температуры и пожары в выработанных неглубоких угольных шахтах создают потенциально опасную и угрожающую для окружающей среды ситуацию. Их последствия очевидны в таких странах, как Южная Африка, где традиционно вырабатывали неглубокие угольные пласты. Такие очаги пожаров обычно недоступны и, как правило, изучаются только через скважины.

В настоящей работе были исследованы методы дистанционного контроля температуры с целью определения местоположения и направления распространения зон повышенной температуры. Представлены некоторые результаты и приводится сравнение данного метода с методами газового анализа и воздушной инфракрасной термометрии. Доклад также освещает использование газов-индикаторов для определения путей движения воздуха внутри недоступных выработок, а также предсказание мест неконтролируемой инфильтрации воздуха.

В заключение описываются методы контроля, применяемые в Южной Африке, включая установку огнезащитных заслонов и траншей, и герметизацию.

废弃浅井煤矿中矿火的侦察和控制

P．沙利文

在废弃的浅井煤矿中灼热和矿火会酿成非常危险并且对环境很不良的灾害．这种问题的影响通常仅限制在惯于开采浅井的国家（包括南非）中发生，这种矿火区域通常不易进入，而只能从地面钻孔来侦察．

我们曾用温度侦测器以遥控的办法测定灼热的位置和动向．这里我们将把用这种办法测得的结果与用气体分析及空中红外线自记温度计所得的结果互相比较．

本文亦叙述使用示踪气体测定不易进入矿中的空气流动情形，并预测自由气流渗透的地区．

最后我们将叙述一些在南非所用的控制方法，包括矿火阻挡层的设置和一些挖沟和封闭的办法．

НАУЧНЫЕ ОСНОВЫ И ПРАКТИЧЕСКАЯ РЕАЛИЗАЦИЯ МЕТОДОВ И СРЕДСТВ ПРОФИЛАКТИКИ ЭНДОГЕННЫХ ПОЖАРОВ

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АННОТАЦИЯ

С углублением горных работ увеличивается эндогенная пожароопасность угольных предприятий. Традиционные методы борьбы с пожаром путем заиливания выработанных пространств не обеспечивают в настоящее время необходимый уровень пожарной профилактики. В связи с этим был разработан комплексный подход к предупреждению эндогенных пожаров, включающий: методы локального и текущего прогнозирования эндогенной пожароопасности выемочных полей и участков, определение ранних стадий самовозгорания угля и пенный способ профилактики эндогенных пожаров, для реализации которого в Советском Союзе разработана и широко внедрена, особенно на шахтах Кузбасса, криогенная и пеногенераторная техника. Ежегодно более 40 выемочных участков отрабатываются с профилактической обработкой выработанных пространств инертными пенами и вспененными суспензиями, что обеспечивает добычу более 15 млн.т угля из наиболее пожароопасных угольных пластов.

Защита шахт от эндогенных пожаров сводится к реализации мер по их профилактике, обнаружению, локализации и тушению. Однако, несмотря на большие капитальные затраты, комплекс реализуемых на шахтах Минуглепрома СССР мер с середины семидесятых годов перестал давать ощутимые результаты. Стала очевидной недостаточная научная проработка вопросов прогноза и обнаружения эндогенных пожаров. Так, например, 10% пожаров в эти годы зафиксировано на пластах угля, не отнесенных к категории самовозгорающихся, а почти 70% пожаров обнаружено по появлению в горных выработках огня, дыма и повышенных концентраций оксида углерода. В Кузбассе на глубинах более 300 м резко снизилась эффективность наиболее широко распространенного способа профилактики эндогенных пожаров заилением выработанных пространств глинистой пульпой.

С учетом создавшегося положения в СССР в последние годы выполнен комплекс научных исследований, включающий разработку новых методов прогнозирования эндогенной пожароопасности выемочных полей, способов обнаружения эндогенных пожаров в ранней стадии, способов предупреждения самовозгорания угля путем нагнетания в выработанное пространство водовоздушных и инертных пен и вспененных суспензий.

Практика борьбы с эндогенными пожарами свидетельствует о том, что их профилактика эффективна тогда, когда она тесно увязана с пожароопасностью защищаемого объекта. Поэтому прогноз пожароопасности является той базой, на которой строится вся система защиты от эндогенных пожаров, и в первую очередь их предупреждение.

Наличие горючего материала, тепловой импульс и среда, поддерживающая горение – обязательные условия для возникновения пожара. Поэтому объективный прогноз эндогенной пожароопасности выемочных полей требует учета химической активности угля, предопределяющей генерацию тепла, а также тех природных

и горно-технических факторов, которые способствуют притоку кислорода и накоплению тепла в его скоплениях.

Используемый в настоящее время в СССР хроматографический метод определения химической активности (склонности угля к самовозгоранию) основан на окислении угля молекулярным кислородом. Метод состоит в том, что испытуемый уголь фракции 0,5...0,35 мм загружается в хроматографическую колонку, которая постоянно продувается инертным газом-носителем, например гелием. При определенной температуре в термостате хроматографа в поток газа-носителя импульсно вводится порция кислорода, который по мере движения вдоль хроматографической колонки частично хемосорбируется, а часть его, не вступившая во взаимодействие с углем, при выходе из колонки фиксируется детектором. Константа скорости сорбции кислорода углем определяется из выражения

$$K = \frac{2,3}{t} \lg \frac{Q_{вх.}}{Q_{вых}} \qquad (1)$$

где K – константа скорости кислорода углем, $с^{-1}$;

t – среднее время пребывания пробы в колонке, с;

$Q_{вых}$ и $Q_{вх}$ – соответственно количество кислорода на выходе и входе из колонки, мл.

Другим показателем, характеризующим склонность углей к самовозгоранию, является критическая температура самовозгорания. При исследовании связи окислительной способности углей с температурой (рис.1) установлено, что при температурах 340...420 К происходит резкое изменение процесса взаимодействия кислорода с углем всех марок. Эта температура принята в качестве критической.

Оценка эндогенной пожароопасности выемочных полей производится для тех пластов, уголь которых хроматографическим методом отнесен к склонным к самовозгоранию. Количественным критерием пожароопасности в новом методе является вероятность возникновения пожара в течение года. В основу метода была положена рабочая гипотеза об эндогенном пожаре как о сложном физико-химическом процессе, на который влияют многие количественные (мощность пласта, угол падения, содержание летучих, количество подаваемого на участок воздуха, скорость подвигания очистного забоя и др.) и качественные факторы (полнота выемки, способ подготовки, схема проветривания, система разработки и др.).

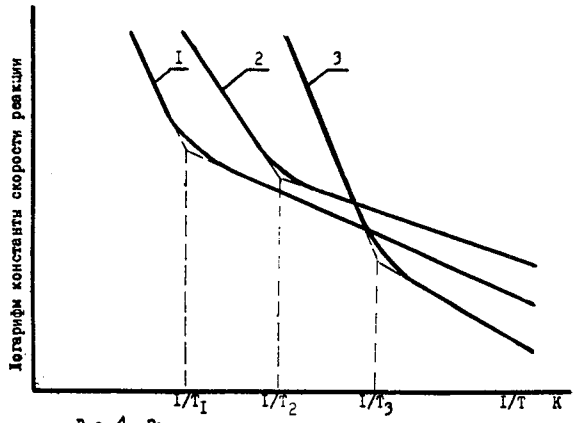


Рис. 1. Зависимость константы скорости окисления угля от температуры в аррениусовских координатах

Вероятность возникновения эндогенного пожара определяется из выражения:

$$P = 1 - e^{\lambda_ф T} \qquad (2)$$

где P – вероятность возникновения пожара;

$\lambda_ф$ – интенсивность возникновения эндогенных пожаров, определяемая по результатам ретроспективного анализа;

T – время отработки выемочного поля, мес.

Для условий Донецкого бассейна в зависимости от вероятности возникновения пожара в течение года принята следующая классификация выемочных полей по эндогенной пожароопасности: I группа – особо опасные (0,75–1,0); II группа – опасные (0,45–0,75); III группа – малоопасные (менее 0,45).

В Кузбассе, с учетом значимости влияния на разделение полей по степени пожарной опасности, при прогнозе учитываются следующие факторы: склонность угля к самовозгоранию, мощность пласта, наличие пожаров в пограничных полях, горелых пород, запасов на выходах пласта, сближенных пластов в кровле отрабатываемого, угол падения пласта, глубина горных работ, тип пород кровли. Зависимость вероятности возникновения эндогенного пожара в выемочном поле от перечисленных природных факторов описывается уравнением

$$P = \frac{1}{2} + \Phi\left(\frac{X - 0{,}222}{0{,}048}\right) \qquad (3)$$

В уравнении (3) второе слагаемое является интегральной функцией Лапласа, а безразмерная координата X равна средней арифметической частоты пожаров по всем девяти учитываемым факторам ($X = \frac{1}{9}\sum P_i^*$).

По результатам ретроспективного анализа установлено, что эндогенная пожароопасность выемочных полей в Кузбассе снижается при увеличении скорости отработки выемочных полей по простиранию, сокращении потерь угля, выравнивании давления воздуха в шахте и на поверхности, уменьшении размеров выемочных полей по падению и гидравлической закладке выработанного пространства. Однако ни одна из перечисленных мер не снижает вероятности возникновения эндогенного пожара до нуля. Поэтому вопрос обнаружения пожаров на ранней стадии развития не потерял своей актуальности.

Первоначально основой обнаружения эндогенных пожаров являлся контроль за содержанием оксида углерода в атмосфере горных выработок и выработанного пространства. Применение в последние годы высокочувствительных газоанализаторов и хроматографов позволило оценить возможность использования при контроле за самонагреванием угля других индикаторных газов (предельных и непредельных углеводородов, водорода) и показателей на их основе.

В процессе исследования продуктов термической деструкции угля при его нагревании наряду с другими газами были обнаружены этилен и ацетилен, содержание которых в рудничной атмосфере составляет около 10^{-7}%. При повышении температуры угля содержание этилена растет значительно быстрее, чем ацетилена, вследствие чего соотношение этих компонентов симбатно изменяется с изменением температуры угля. Типичные кривые зависимости отношений содержаний этилена к ацетилену от температуры для донецких углей различных стадий метаморфизма представлены на рис. 2.

Сущность метода определения ранних стадий самовозгорания угля по соотношению непредельных углеводородов заключается в отборе и анализе на содержание этилена и ацетилена проб рудничного воздуха и определении на этой основе по заранее установленной зависимости температуры угля.

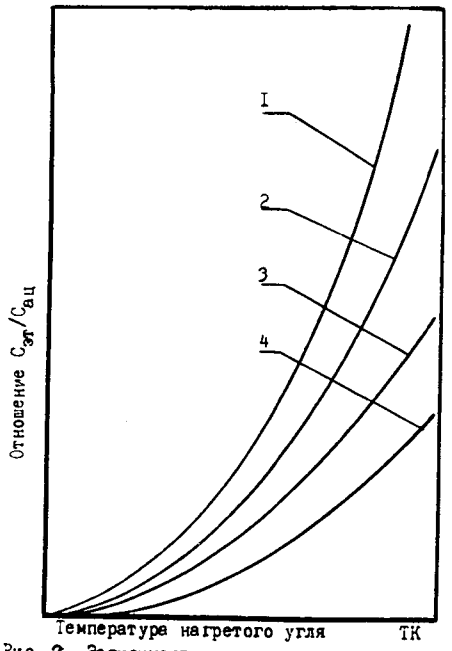


Рис. 2. Зависимость отношения концентрации этилена ($C_{эт}$) и ацетилена ($C_{ац}$) от температуры нагретого угля для марок: I- длиннопламенный; 2- коксующийся; 3- тощий; 4- антрацит

Отбор проб для анализа производится путем прокачивания не менее 1 литра рудничного воздуха через металлическую U-образную трубку - концентратор, заполненную специально подготовленным сорбентом. При прокачивании происходит насыщение сорбента микропримесями целевых компонентов до установления термодинамического равновесия.

Срок хранения пробы при обычных температурах окружающего воздуха практически не ограничен. Анализ шахтных проб на содержание этилена и ацетилена производится на газовых хроматографах с пламенно-ионизационным детектором, снабженных специальной приставкой, которая позволяет отделить компоненты пробы, мешающие анализу целевых продуктов. Продолжительность анализа одной пробы зависит от количества мешающих примесей и составляет 15...40 мин.

В ряде бассейнов, в частности в Кузбассе, используется метод обнаружения эндогенных пожаров в стадии самонагревания по соотношению оксида углерода и водорода. По этому методу предварительно оценивается

фон обоих индикаторных газов в контролируемом выемочном поле. Фоновая концентрация водорода в рудничном воздухе близка к атмосферной (10^{-5}%). Поэтому перед анализом на хроматографах производится его обогащение фронтально - адсорбционным способом на вакуумированных колонках с последующим разделением обогащенной примеси проявительным методом.

В соответствии с классификацией стадий пожара по CO и H_2 признаком самонагревания угля является превышение концентраций фоновых значений индикаторных газов с тенденцией к их росту при одновременном увеличении показателя CO/H_2. Лабораторными исследованиями установлено, что в интервале температур 293-683 К этот показатель увеличивается для каменного угля с 2,0 до 420,0. Аналогичный характер изменения этого показателя имеет место и для бурого угля. Однако его пиковое значение, соответствующее переходу в стадию горения, значительно выше (равно 578) и достигается на 40° ниже, чем у каменного угля.

В последние годы увеличивается нагрузка на очистные забои, оснащенные дорогостоящими комплексами, и соответственно растут размеры выемочных полей. Экономический ущерб от изоляции такого поля при эндогенном пожаре достигает значительной величины. Поэтому для всех бассейнов СССР задача защиты выемочных полей, оборудованных высокопроизводительными комплексами, приобрела особую актуальность. Поиски ее решения привели к созданию нового направления в профилактике эндогенных пожаров, основанного на создании в выработанном пространстве воздухонепроницаемой "пенной завесы", обладающей следующими основными свойствами:
плотность пены в К раз меньше плотности воды (К - кратность);
вязкость пены на порядок больше вязкости воды (μ воды = $1{,}0 \times 10^{-2}$ Н·с/м^2; μ пены кратностью 80=4.2 Н·с/м^2);
пены средней кратности являются хорошими теплоизоляторами, так как их теплопроводность близка к теплопроводности воздуха;
теплоемкость пены пропорциональна содержанию жидкой фазы, при попадании 1 м^3 пены кратностью 160 на очаг самовозгорания охлаждающий эффект составляет 14650 кДж;
химическая активность углей, обработанных пенообразующей жидкостью, в 5...7 раз ниже, чем естественного угля; за счет лучшей смачиваемости углей эффект дезактивации на 20...30% выше, чем при применении других антипирогенов на жидкой основе;
"пенная завеса" перекрывает зону фильтрации воздуха в обрушенных породах отработанной части пласта и изолирует очистной забой от проникновения в него ядовитых газов и тепла, если последние имеют место, герметизирует полости выработанного пространства, эффективно охлаждает нагретые массы угля и породы и не нарушает рабочего цикла очистного забоя.

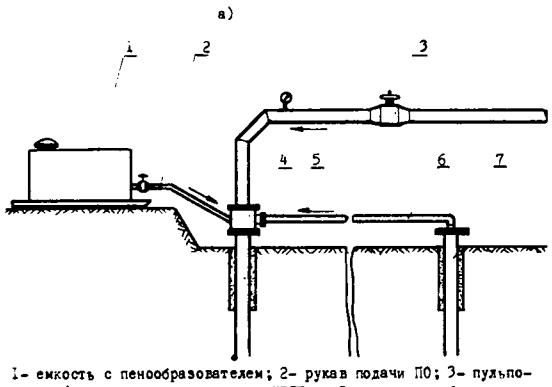
Изолирующий эффект от применения пен на крутопадающих пластах оценивается снижением подсосов воздуха через щитовое перекрытие. Экспериментальные наблюдения на шахте - "Центральная" (Кузбасс) показали, что непрерывная шестичасовая работа пеногенератора производительностью 2 м^3/мин позволяет создать пенную завесу над действующим щитом. При этом подсосы воздуха сокращаются с 53 до 15 м^3/мин при одновременном снижении температуры на 7 °С. Аналогичными замерами при локализации пожара на пологопадающем пласте шахты "Инномината" (Испания) зафиксировано увеличение в 100 раз аэродинамического сопротивления выработанного пространства после нагнетания пены.

Первое практическое применение водовоздушной пены (ВВП) для целей профилактики имело место в Кузбассе в 1975 году. Развитием идеи использования пен в практике борьбы с подземными пожарами явилось предложение по стабилизации их глиной. Вспененная глинистая пульпа (ВГП) имеет высокую стойкость по сравнению с водовоздушной пеной. Последняя, например, разрушается над действующим щитом за шесть часов, в то время как ВГП полностью распадается через 3...8 суток.

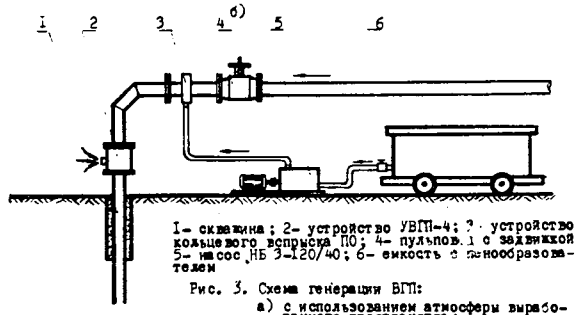
Технологическая схема приготовления ВГП представлена на рис.3.

Глинистая пульпа в этой схеме готовится по общепринятой напорной технологии и подается к скважине, пробуренной в выработанное пространство с поверхности. На обсадной трубе скважины закреплено специальное устройство, соединенное с емкостью для пенообразователя. Устройство

для вспенивания глинистой пульпы работает на эжекционном принципе. Генерируемая ВГП имеет кратность 10.



1- емкость с пенообразователем; 2- рукав подачи ПО; 3- пульповод с задвижкой; 4- устройство УВГП-4; 5- скважина; 6- рукав резиновый; 7- скважина



1- скважина; 2- устройство УВГП-4; 3- устройство кольцевого вспрыска ПО; 4- пульпов. с задвижкой; 5- насос НБ 3-120/40; 6- емкость с пенообразователем

Рис. 3. Схема генерации ВГП:
а) с использованием атмосферы выработанного пространства;
б) с использованием атмосферного воздуха.

Простота приготовления (традиционная технология практически не меняется), повышенная стойкость, отказ от энергии сжатого воздуха и высокий охлаждающий эффект – все эти качества способствовали широкому внедрению ВГП в практику. Ограничением применения ВГП является необходимость групповых или отдельных скважин с перепадом высот не менее 50 м. При меньших перепадах высот коэффициент связывания эжектируемого воздуха меньше 1.

С 1980 года на шахтах СССР для борьбы с эндогенными пожарами применяется инертная пена (ИП) и инертная вспененная глинистая пульпа (ИВГП). В качестве инертного газа используется азот. Применение азота для борьбы с эндогенными пожарами имеет ряд преимуществ по сравнению с другими газами, не поддерживающими горения (водяной пар, диоксид углерода и др.):

плотность азота близка к плотности воздуха (0,97), что обеспечивает его равномерное распространение в атмосфере выработанного пространства;

незначительная сорбция углем и породами (в 20...40 раз меньше, чем CO_2) и малая растворимость в воде (почти в 60 раз меньше, чем CO_2) способствуют равномерному перемешиванию и малым потерям на пути движения в выработанном пространстве;

при испарении 1 т жидкого азота образуется 854 $м^3$ газа, что в 1,8 раза больше, чем из 1 т жидкого диоксида углерода;

недефицитен (побочный продукт кислородных цехов металлургических заводов).

Для перевозки, хранения и газификации азота в СССР разработано и серийно выпускается криогенное оборудование, технические характеристики которого (емкость, подача) варьируются в широких пределах. Существуют два способа получения инертной пены – с использованием газообразного и жидкого азота. При получении инертной пены без предварительной газификации жидкого азота определяется соотношение расходов раствора пенообразователя и жидкого азота, исключающее замерзание жидкости:

$$q_a = q_ж \cdot \frac{T_ж}{1465} \qquad (4)$$

где q_a – расход жидкого азота, $м^3/с$;
$q_ж$ – расход раствора пенообразователя, $м^3/с$;
$T_ж$ – температура раствора пенообразователя, К.

При газификации жидкого азота и генерации инертной пены и инертной вспененной глинистой пульпы используются три основные технологические схемы. Первая схема (рис.4) рекомендуется для шахт, на которых профилактическая обработка выработанных пространств ведется из подземных выработок с использованием шахтной сети трубопроводов, соединенных с групповой скважиной. В этом случае ИП и ИВГП генерируется в стационарном поверхностном комплексе, оборудованном газификационной установкой, резервной емкостью для хранения жидкого азота, емкостью с пенообразователем и насосом для подачи пенообразователя в поток воды или пульпы. Первый комплекс для приготовления ИП и ИВГП был построен в 1981 году. В настоящее время в Кузбассе эксплуатируются 20 таких комплексов.

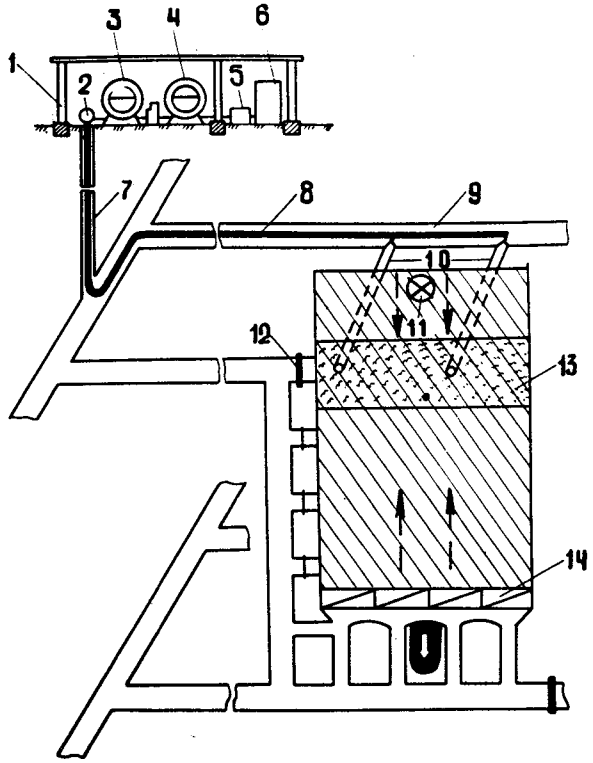


Рис. 4. Технологическая схема профилактики и локализации эндогенных пожаров на пластах крутого падения: 1–стационарный поверхностный комплекс, 2–пеногенератор, 3–цистерна, 4–газификатор, 5–насос, 6–емкость с пенообразующей жидкостью, 7–групповая скважина, 8–трубопровод, 9–вентиляционный штрек сближенного пласта, 10–скважины, 11–очаг потушенного пожара, 12–перемычка, 13–завеса из инертной пены, 14–щитовой агрегат.

Генерируемая на поверхностном комплексе пена под давлением не более 2,0 МПа с расходом 6–7 м³/мин транспортируется по шахтным трубопроводам к скважинам, пробуренным в выработанное пространство с вентиляционного штрека сближенного пласта.

Профилактическая обработка потерь угля инертной пеной кратностью 60–150 ведется не менее двух раз в месяц из расчета заполнения всех пустот в объеме выработанного пространства. Объем пустот (V), время нагнетания (τ), расход жидкого азота (Р) и пенообразователя (М) определяется по зависимостям:

$$V=(1-\beta)\cdot m\cdot \ell\cdot h \qquad (5)$$

$$\tau=C_v\cdot \ell n\frac{Q\cdot C_v}{Q\cdot C_v-V} \qquad (6)$$

$$P=Q\cdot\tau\cdot a^{-1} \qquad (7)$$

$$M=0{,}01\cdot C\cdot Q\cdot\tau\cdot K^{-1}\cdot a^{-1} \qquad (8)$$

где: β – коэффициент перепуска пород,
m,ℓ,h – соответственно мощность, длина и высота отработанной части пласта, м,
C_v – стойкость пены, ч,
Q – производительность пеногенератора, м³/ч,
a – коэффициент газификации, м³/т,
C – концентрация пенообразователя, %,
K – кратность пены.

Средний месячный расход жидкого азота на профилактические цели в Кузбассе колеблется в пределах 5–13 т на одно выемочное поле. При локализации активизировавшихся очагов пожаров при наличии утечек воздуха расход жидкого азота увеличивается в 1,5–2,0 раза. Это необходимо для постоянного восстановления "пенной завесы", формируемой в обрушенных породах на пути утечек воздуха к очагу самонагревания угля.

Вторая технологическая схема (рис.5) реализуется с применением автономных передвижных средств. В схеме задействованы автомобильная газификационная установка с запасом жидкого азота 4,2 т и пожарная цистерна на автоходу с пенообразующей жидкостью. Схема используется при возможности точного бурения скважины с поверхности в зону потерь угля в выработанном пространстве.

В шахтах Донбасса из-за большой глубины (900...1000 м) узел приготовления инертной пены располагается в горных выработках. При этом газообразный азот может подаваться как с поверхности (рис.6), так и из горных выработок (рис.7).

Для осуществления подачи инертной пены из горных выработок разработана и серийно выпускается специальная газификационная установка, которая предназначена для хранения, транспортирования и холодной газификации жидкого азота в подземных условиях при выполнении работ по профилактике, локализации и тушению пожаров. Она состоит из цистерны с запасом жидкого азота 730 кг и блока-испарителя с принудительным обдувом. Установка позволяет получать 15 м³/мин газообразного азота при давлении 1,6 МПа.

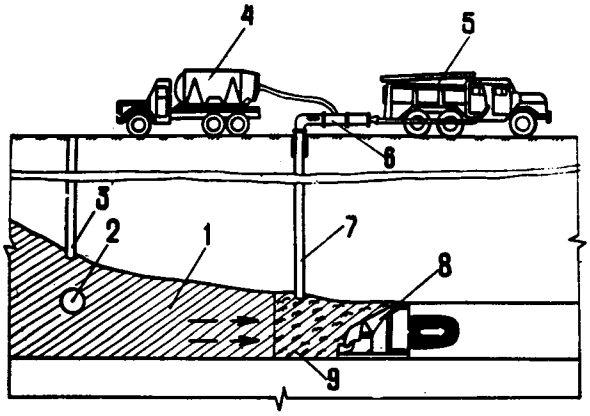
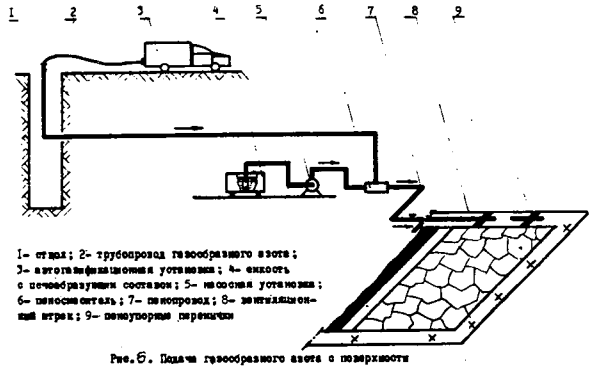
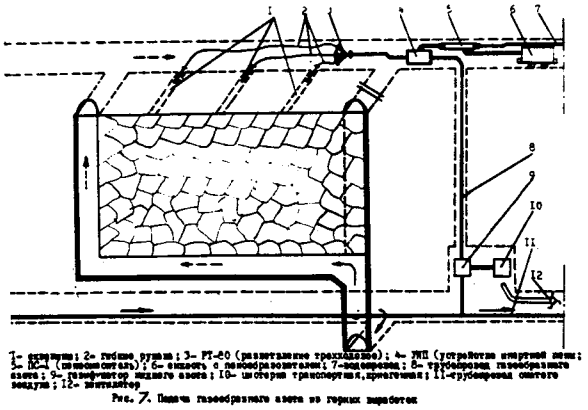


Рис.5. Мобильная технологическая схема инертизации выработанных пространств по скважинам с поверхности: 1–выработанное пространство; 2–очаг самовозгорания угля; 3,7–скважины; 4–газификатор; 5–цистерна; 6–генератор; 8–механизированный комплекс; 9–пенная завеса



1– отвал; 2– трубопровод газообразного азота; 3– автотранспортная установка; 4– емкость с пенообразующим составом; 5– насосная установка; 6– пеносмеситель; 7– пенопровод; 8– вентиляционный штрек; 9– пенные перемычки

Рис.6. Подача газообразного азота с поверхности



1– скважины; 2– гибкие рукава; 3– РТ-80 (разветвление трехходовое); 4– УКП (устройство инертной пены); 5– ПС-1 (пеносмеситель); 6– емкость с пенообразователем; 7–задвижка; 8– трубопровод газообразного азота; 9– газификатор жидкого азота; 10– цистерна транспортная, прицепная; 11–трубопровод сжатого воздуха; 12– вентилятор

Рис. 7. Подача газообразного азота по горным выработкам

Прогрессивная технология предупреждения эндогенных пожаров с применением инертных пен и вспененных суспензий находит все более широкое применение на шахтах СССР. Только на шахтах Кузбасса ежегодно отрабатывается более 40 выемочных участков с профилактической обработкой выработанных пространств инертными пенами и вспененными суспензиями, что обеспечивает добычу угля более 15 млн.т в год из наиболее пожароопасных угольных пластов. В 1987 году расход жидкого азота и пенообразователя на эти цели составил в Кузбассе соответственно 6455 и 1959 т. Внедрение новых способов и средств профилактики в наиболее пожароопасных бассейнах позволило снизить число ежегодно возникающих эндогенных пожаров на шахтах отраслей с 1983 по 1987 гг. в 1,7 раза.

SCIENTIFIC PRINCIPLES AND INTRODUCTION OF METHODS AND MEANS FOR PREVENTING SPONTANEOUS FIRES

M.N. Sudilovsky, V.I. Lagutin, P.S. Pashkovsky

Conventional methods of spontaneous fire control do not provide for the required level of fire prevention. An integrated approach to spontaneous fire prevention must involve methods of local and current forecasting of the tendency of panels and sections to spontaneous fires, identifying initial stages of low-temperature coal oxidation, foam techniques, and facilities for preventing spontaneous coal combustions, etc. Methods of prediction are based on the mutual impact of geological and mining factors, including chemical coal activity. A novel method has been developed to identify initial stages of spontaneous coal combustion based on the unsaturated hydrocarbon content, namely, the emission of microconcentrations of ethylene and acetylene in the mine air during spontaneous combustion; the ethylene-acetylene ratio depends on the coal temperature. Foam treatment of the goaf result in decreased oxygen inflow. A film that forms on the coal surface reduces the sorptive activity of the coal; nitrogen present in the foam gas phase inerts the goaf. Cryogen and foam generating technology has been developed and is used in USSR coal mines. Annually, over 40 working sections treat the goafs with inert foams and foamed suspensions, allowing a total of 15 million tons of coal to be extracted from the most combustible coal seams.

PRINCIPES SCIENTIFIQUES ET PRESENTATION DES METHODES ET MOYENS DE PREVENTION DES INCENDIES SPONTANES

M.N. Sudilovsky, V.I. Lagutin, P.S. Pashkovsky

Les méthodes conventionnelles de contrôle des incendies spontanés n'apportent pas le niveau de prévention anti-incendie nécessaire. Un moyen d'approche intégrée de la prévention des incendies spontanés doit inclure des méthodes de détection locales et modernes des chambres et des sections sujettes aux combustions spontanées, des méthodes de détection des stades initiaux de l'oxydation du charbon à faible température, des techniques utilisant des mousses, des équipements pour la prévention des combustions spontanées de charbon, etc... Les méthodes de détection sont basées sur l'impact réciproque des facteurs géologiques et miniers, ainsi que sur l'activité chimique du charbon. On a développé une nouvelle méthode permettant de détecter les stades initiaux de la combustion spontanée du charbon; cette méthode est basée sur la teneur en hydrates de carbone non saturés et, en particulier, sur l'émission de microconcentrations d'éthylène et d'acétylène dans l'air minier au cours d'une combustion spontanée; le pourcentage d'éthylène-acétylène est fonction de la température du charbon. Le traitement du remblai par la mousse réduit l'arrivée d'oxygène. Un film qui se forme à la surface du charbon diminue la capacité de sorption de celui-ci; l'azote dans la phase gazeuse de la mousse rend le remblai inerte. La technologie de la production de cryogène et de mousse s'est développée, et on l'utilise à l'heure actuelle dans les mines de charbon de l'URSS. Plus de 40 sections d'exploitation traitent chaque année les remblais avec des mousses inertes et des mousses en suspension, ce qui permet d'extraire 15 millions de tonnes de charbon des couches de charbon les plus combustibles.

WISSENSCHAFTLICHE PRINZIPIEN UND DIE EINFÜHRUNG VON METHODEN UND MITTEL ZUR VERHÜTUNG VON SELBSTENTZÜNDUNGSBRÄNDEN

M. N. Sudilovsky, V. I. Lagutin, P. S. Pashkovsky

Herkömmliche Methoden zur Beherrschung von Selbstentzündungsbränden berücksichtigen nicht die Brandverhütung. Eine allumfassende Behandlung des Problems der Verhütung von Selbstentzündungsbränden muß Methoden der lokalisierten und aktuellen Vorhersage der Selbstentzündungsneigung von Bauhöhen und Gebieten, sowie die Identifizierung von Anfangsstadien der Niedertemperatur-Kohleoxidation, Schaumtechniken und Einrichtungen zur Verhütung von Selbstentzündungen usw. einschließen. Vorhersagemethoden basieren auf dem gemeinsamen Einfluß der geologischen und bergbaulichen Faktoren, darunter auch die chemische Aktivität der Kohle. Es wurde eine ungewöhnliche Methode zur Identifizierung der Anfangsstadien der Selbstentzündung entwickelt, welche auf dem Anteil der ungesättigten Kohlenwasserstoffe beruht, nämlich der Freisetzung von Mikrokonzentrationen von Ethylen und Azethylen in die Grubenwetter während der Selbstentzündung. Das Verhältnis zwischen Ethylen und Azethylen hängt von der Kohletemperatur ab. Die Einbringung von Schaum in den Alten Mann resultiert in einer Verringerung der Sauerstoffzufuhr. Ein Film, der sich auf der Kohleoberfläche bildet, reduziert die Aufnahmebereitschaft der Kohle; als Schaumgas verwandter Stickstoff inertisiert den Alten Mann. Techniken der Kryogen- und Schaumerzeugung wurden in den sowjetischen Gruben entwickelt und angewandt. Jährlich werden in über 40 Abbaubetrieben die Bruchfelder mit inertem Schaum und Schaumlösungen behandelt und ermöglichen so den Abbau von insgesamt 15 Mio t Kohle aus den feuergefährlichsten Flözen.

防止自发火灾的科学原理及措施方法

M。N。苏的罗夫士基

V。I。拉古丁， P。S。帕希可夫士基

普通的自燃火灾控制办法不足以达到需要的防火标准．一个综合的自燃火灾防止办法必须包括现时现地的仪表板和型材自燃的趋势，断定低温度煤碳氧化的初期，泡沫技术及防止煤碳自动燃烧的设备．预测方法系根据地质和煤矿因素相互作用的关系，包括煤的化学性能在内．我们发展了一种新的煤碳自燃初期的断定，是根据未饱和碳氢化合物的含量，亦即燃烧时矿气内乙烯及乙炔微量的放射性；乙烯乙炔的比例与煤的温度有关．空岩区的泡沫处理可以减少氧气的内溢．煤碳表面生成的薄膜可以减低煤的吸咐作用．在空岩区里泡沫气体里的氮气会产生惰性作用．冷冻剂和泡沫制造技术在苏联煤矿中已在发展应用．每年四十多矿场用惰性泡沫及泡沫悬液处理空岩区，使易燃煤层里可得一千五百万顿的产量．

PREVENTION POSSIBILITIES AGAINST MINE FIRES

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ABSTRACT

The authors examine the possibilities of occurrence of endogenetic mine fires, then analyze the circumstances of fire events, that is the conditions of development of fires upon the data of Borsod coal-basin.

22 % of the endogenetic fires was developed in drifts, 78 % was in workings. This ratio characterizes the main area of protection against fires, the workings.

At fires in drifts in all cases there was broken coal in environment, the consequences of slipshod work could be established.

In the case of fires in workings, in all cases, it can be established, that in the consequence of deposit development and the working system significant quantity of coal got into caving. The appropriate air closing was not created. Fires were developed in workings where the advancing rate of working was little, 0,6-0,8 m/day, and in stopped workings. The analyses of ventilation parameters pointed out on the one hand, that in all cases an attempt was made to improve the generally very weak ventilation by using air tube, on the other hand significant air leakage happened over the goaf.

The main field of protection against fire is the improvement of the ventilation circumstances, because the measure of coal waste and the rate of working can not be selected freely.

The authors gives solutions for optimal ventilation.

Amongst the elementar dangers threatening the mining activity the most significant on is the occurence of endogenetic mine fires and the hazard caused by them. Breakout of fires theatens the safety of working, and in the presence of methane there is a danger of explosion.

Spontaneous ignition needs three basic condition.

a) Material tending to oxidation in an increased degree and having appropriate quantity and specific surface; in the mines it is generally coal;

b) appropriate quantity of oxygen at different phases of oxidation;

c) increasing quantity of heat continously raises the temperature of the inflammeble material and of its environment.

Primary site of endogenetic fire is the goaf where the material left behind is crushed and there is an oxygen-supply behind the working-face - as the result of air-stream. Besides the general conditions or fire can occur in the drifts, too.

Fire hazard is present in the Hungarian coal-mining at working of the brown coal, especially in thick beds. During exploitation of thick beds - if thickness changes irregularly and significantly - in the goafs is left behind coal in such quantities which always means potential hazard.

ANALYSIS OF THE FIRE CASES

In our examination had been elaborated the data of the fire cases occured in the works of the Borsod coal-basin.

Amongst the 15 cases 6 fires were of exogenetic origin. The external effects were welding (omission of cooling) in three cases, non-determined flame effect (probably smoking) in two cases, in the sixth case the cause of the fire was the overheating of a conveyor roller.

Amongst the endogenetic fires two cases (22 %) accored in drift, seven others in workings or in worked areas. This ratio shows that at the prevention of fires the greater attention most be paid to he later sites.

Fires around drifts occured mainly in the roofs. During the driving of the drifts - in the absence of quickly applicable support and because of shootings - the coal in the roof becomes loose, sometimes caves are formed. In the loosened layers fires develop at 1-1,5 m distance from the surface, as the circumstances are the most favourable there. In the case of slow airflow or air stagnation warming up occurs near the surface, as the radius of the oxygen-supply is smaller and there is minimal cooling at the surface.

Fires occur most often near the layer and spalling of the coal occurs in an increased degree. At such sites it is very important to avoid bursting, to pack the caves with not-inflammeble, air-proof material.

ANALYSES OF THE FIRES IN WORKINGS

On the analysis of the cirsumstances of fires in workings and the conditions of warming up, our statements are the following. In all the cases considerable part of the layer - 0,5-1,5 m in thickness - was left behind and it got into the debris after the abandonment of the working. It is characteristis in all the ases that spelled coal was left behind as a result of the transmission of stresses from the working-face and the breaking--off. Sometimes pyrite layers were revealed in the immediate roof.

During the examination of the investigation materials the aeration conditions and their special circumstances were analysed. It was established that in two cases (28 %) the aeration of the working was not sufficient at all. In six cases the goaf was unsealed after the abandonment of the working-face and there was a significant air-flow in the abandoned area. Air traps were installed after 15-20 days.

Upon examining the circumstances of the ventilation it was established that in three cases is the surroundings of the working the desired air supply could becured only by means of ventilating ducts. Around the workings there were 5-6 ventilating ducts in operation. On the evidence of the examination in every cases some deficiency of the ventila-

ting system was detectable:
a) generally insufficient ventilation which was improved by means of ventilating tubes;
b) sealing of the goafs was neglected for a long time, a significant air-flow existed in the hewed area.

The third examined parameter was the velocity of working, the advancement of the working-face. It was extablished that in five cases fires occured in abandoned workings - mining was finished or the working-face was not worked for a long time - and in two cases advancement of the working-face was very low - 0,6-0,8 m/d. Consequently it was demonstrated in every case that the material left behind in the goaf was exposed to the effect of oxydation, and the air-flow had no cooling effect because of the low speed.

It was also examined how long after the caving occured warming-up, i.e. how long is the incubation time. In five cases this period could not be determined exactly. Periods between caving and the occurence of fire were: 17-26; 14-15; 26-30; 45-50; 42 days. It can be said that the minimal period is 14--15 days, approximately two weeks, during which warming up or rather the fire developed.

On the basis of our examination it was estabiiched that the conditions and circumstances of the endogenetic mine fires in the Borsod coal basin decisively confirm the observations of other surveys (1, 2, 3), and the theoretical considerations. It could be demonstrated in all cases that warming-up and fire occured at such sites where inflammable material - having great specific surface - was present in large quantity (mainly coal in the cases examined). The ventilation of the area was not sufficient, it was improved by building in some ventilation ducts. Considerable air passing was detectable through the caving. Fires occured without exception in workings where working-face had been stopped for a long time (71 %) or where advancing was very slow (29 %).

THE TASKS OF THE FIRE HAZARD'S PREVENTION

Determining the tasks of the prevention we must start from the fact that coal-losses can not be regulated exactly - because of the irregularly and in great extent variable layer thickness, the necessary residue in the roof in thin layer, when the workabla thickness is less than the actual thickness of the layer, in the case of tearing-down technology the tearing of the roof (exploitation) is not complete, therefore the presence of inflammable material behind the working can not be excluded. So the prevention of self-ignition can be solved by suitable changing of the working parameters.

At the ventilation of the workings there are two tasks of fundamental importance:
a) Ventilation of the working must secure the desired quantity of air by steady air-flow at the working-face. Air supply of the working must be solved possibly without auxiliary ventilators (ventilation ducts).
b) The quantity of the air getting into the goaf must be minimized by sealing the inlet side and ripping the sise workings.

The advancement of the working-face must be preferably above 1,5-2,0 m/d. In the case of

abandoned working-face (because of withdrawing) the withdrawing and the joining workings of the abandoned area must be dammed possibly within 15-20 days. In the workings stopped because of technological or other considerations full damming must be secured by means of quickly mountable and dismountable dams.

In continuous live workings the appropriate advancement, ventilation and damming of the goaf side are separate means against self-ignition. Taking into account that advancement can not be assured because of not foreseeable causes (disturbed tectonics, etc.) the appropriate fire control demands the ventilation of the working and damming of the goaf side according to the aspects of fire protection. Measures for the improvement of the ventilation generally are not to be set up quickly, except auxiliary ventilation. However they are possibly to be avoided because according to theoretical considerations and practical observations they intensify the endogenetic fire hazard.

It is to be seen on the basis of the aforementioned that the quantity of the coal left in the goaf and the advancement of the working face can be hardly influenced, the primary possibilities to prevent fire hazard is the optimal solution of the ventilation.

OPTIMAL SOLUTION OF THE VENTILATION

Before examining the effects of the ventilation it must be stated that fire hazard can not be eliminated, it can be only decreased by ventilation. Methods of protection must be applied in combination, they are not effective in themselves.

The goal of fire protection by ventilation is to prevent the air-flow into the goaf or into the coal face. This can be solved only with extraordinary difficulties and expenses, so we must strive to secure the minimal level of air-flow.

In the case of air-flow through the goaf or cracks a linear component must be taken into account in the functionality between the quantity of flow and the pressure drop, this is the laminar flow.

$$h = R_1.Q+R_2.Q^2 \qquad (Pa)$$

where: R_1, R_2 - air resistance
Q - quantity of flowing air

Amount of air leakage may be reduced
- by increasing the resistance between the air path and site of the air-flow;
- by increasing the resistance of the site of air leakage;
- by reducing the pressure difference at the critical sector.

Along the working-faces the damming of the goaf must be the possibly best. In the case of shield sets this can be secured but because of the small sections of the crossings the velocity of the air-flow is relatively high, so the air gets into the goaf without difficulty. Therefore brattice stopping or foil must be applied on the goaf side. In the case of warming-up and powerful CO-generation stopping of this section may be reasonable by means of resinous foams. Application of the foil or brat-

tice stopping is reasonable at the return side, too, because the resistance of the air-flow path is in this case inreased.

Stopping isolation of the workings must be applied primarily in full advances or rather in all cases when the working is abandoned in the goaf. In such cases the best solution is the end--stop is very important also at the roof, because the resistance increases dramatically just before the complete stopping, earlier this increase (till 8-10 cm of gap) is less.

Resistance of the goaf depends on the quality, the screen size of the goaf and the height of it. If ripping occurs in great pieces and occasionally does not fill up completely the caving, then there is a small resistance and a great cross-section. Incomplete ripping occurs mainly in the line of the secondary workings, in the case of well--ripping roof rocks, too. Cause of this is that ripping occurs at sound coal--faces with difficulties and recovery of the frames is not always complete. This means some hazard mainly at the intake side, as great quantity of air can get into the caving. As protection can be recommended careful recovery and provoking of ripping if necessary. At highly hazardous sites flushing can be applied, too.

Decreasing of the pressure difference is an important feature both in drifts and in workings. First of all it is to be achieved of air as the cases this causes no difficulties but the achieving of the minimal necessary air quantity which is secured by auxiliary ventilation.

In workings and drifts the great pressure differences formed within small destances - may be called local pressure differences - are extremely hazardous. Significant local pressure differences develope between the two sides of every local resistance, and the intake and delivery side of the auxiliary ventilation. These pressure differences - also at rather low values - yield the air-flow in the caving or in the cracked coal face, which is necessary to the generating of fire. On the basis of observations and computer modeled experiments it can be said that occasionally 5-6 Pa of pressure difference is sufficient to develop hazardous air-flows.

Secondary drifts of the workings are deteriorated near the workings, their resistance increases, and serious pressure drops are formed out just in the hazardous sections. The best condition of the drifts and crossings must be secured.

Auxiliary ventilators, air doors, air planks, fire barriers, etc. built into the drifts must be mentioned separately. The listed devices cause a pressure drop which results in an air-flow in the cracked coal-face and this phenomenon can often cause fires in drifts. It is very hazardous in that case when drift advences along a caving with some meters of barries establishing, because these

abutments crack and it is not possible to form a suitable barrier to prevent the air-flow.

The best protection is to build in the resistance increasing devices at the definitely necessary sites. If installation is unavoidable, its site must be determined carefully at the less cracked site. If it is not possible, the drift must be shelled in 10-20 m distances on both sides of the device, but complete stopping can not be certain.

The most severe problems are caused by the auxiliary ventilators. At ventilators built into the air gates 100-400 Pa pressure difference can be formed out and as a result of this air-flow can occur in the caving or in the cracked coal-face at a rate of 10-30 m^3/min. Here the only solution is the installation of the devices outside of the territory or the application of another regulation (damping of other territories.).

This is especially important in drifts surrounded by cavings or at small cores.

The computer modell elaborated by the Mining Department of the University in the examined cases demonstrated well the effect of air-flows and their role in the occurence of endogenetic mine fires.

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PREVENTION DES INCENDIES MINIERS

Z. Buocz et F. Kovacs

Les auteurs examinent les circonstances qui provoquent l'apparition des incendies miniers; puis ils analysent les conditions de développement des incendies sur la base de données obtenues dans le bassin minier de Borsod.

22% de ces incendies naissants se sont produits dans les galeries, 78% dans les sites d'extractions. Ces pourcentages démontrent que les principales zones à protéger des incendies sont donc les sites d'extractions. En ce qui concerne les incendies de galeries, dans chacun des cas, du charbon brisé présent dans la vicinité a permis d'établir qu'ils étaient la conséquence de négligences dans les travaux.

En ce qui concerne les incendies des sites d'extractions, l'on a pu démontrer que, dans tous les cas, d'importantes quantités de charbon s'étaient éboulées, par suite du développement des gisements et du système d'exploitation minière, et les remblais n'étaient pas suffisamment étanches. Des incendies se sont produits dans les excavations où le taux d'avancement n'était que de 0,6-0,8 m par jour ainsi que dans les excavations abandonnées. L'analyse des paramètres de ventilation a permis de démontrer d'une part, qu'on avait tenté dans tous les cas d'améliorer la ventilation généralement insuffisante au moyen de tubes d'aérage et d'autre part, que d'importantes fuites d'air se produisaient par les remblais.

Le meilleur moyen de protection contre les incendies consiste à améliorer les conditions de ventilation, car on ne peut sélectionner arbitrairement la quantité de déblais et la cadence d'exploitation.

Les auteurs donnent des recommandations permettant d'obtenir une ventilation optimale.

VERHÜTUNG VON GRUBENBRÄNDEN

Z. Buocz, F. Kovacs

Die Autoren untersuchen die Möglichkeiten des Auftretens von Entstehungsbränden untertage und analysieren dann die Bedingungen für die Brandentwicklung auf Grund von Daten, welche im Borsod-Revier gesammelt wurden.

22% der Entstehungsbrände entwickelten sich in den Strecken, 78% in Abbaubetrieben. Dieses Verhältnis zeigt, daß das Hauptgebiet des Brandschutzes in den Abbaubetrieben liegt. Bei Bränden in Strecken war in allen Fällen als Folge nachlässiger Arbeit Kohleklein in der Umgebung vorhanden.

Bei Bränden in den Abbaubetrieben zeigte sich, daß ein beträchtlicher Anteil an Kohle im Bruchfeld verblieb, was von der Lagerstättenentwicklung und der Abbaumethode abhängt, und daß das Bruchfeld nicht adäquat verdämmt wurde. Es entwickelten sich Brände in Betrieben, in denen der Abbaufortschritt nur 0.6 - 0.8 m/Tag betrug, sowie in abgeworfenen Bauhöhen. Eine Analyse der Wetterdaten zeigt, daß in allen Fällen der Versuch unternommen wurde, die im allgemeinen schwache Bewetterung durch den Einsatz von Luftrohren zu verbessern, aber andererseits ein nicht unerheblicher Verlust durch Schleichwetter durch den Alten Mann vorhanden war.

Die Hauptmethode zum Schutz gegen Feuer ist die Verbesserung der Bewetterung, da der Anteil der Verluste und die Abbaurate nicht willkürlich gewählt werden können.

Die Autoren geben Hinweise zur optimalen Bewetterung.

ПРЕДОТВРАЩЕНИЕ ПОЖАРОВ В ШАХТАХ

З. Буоз и Ф. Ковач

Авторы исследуют причины и условия возникновения пожаров в шахтах используя данные по Борсодскому угольному бассейну. 22% пожаров возникли в штреках, 78% - в выработках. Это соотношение показывает, что главные усилия должны быть направлены на противопожарную защиту выработок. Что касается пожаров в штреках, то во всех случаях отмечалось скопление раздробленного угля и другие следы небрежной работы вблизи очага пожара. Показано, что возникновение пожара в выработках является результатом засорения шахтных проходов, угольных обвалов и недобросовестно проделанной герметизации пустых выработок. Пожары возникали в выработках, где скорость проходки не превышала 0.6 - 0.8 м/день, а также в заброшенных выработках. Проверка вентиляционных параметров показала, что, с одной стороны, были сделаны попытки улучшить слабую вентиляционную систему, но, с другой стороны, происходила значительная утечка воздуха через пустые выработки. Основным способом противопожарной защиты является улучшение вентиляционной системы, так как регулировать количество отходов и скорость проходки довольно трудно.

矿火防止

Z。布屋兹 及 F。柯凡克士

作者在本文里首先讨论起始矿火的可能性，然后根据波尔索矿区的数据分析矿火发展的情形。矿火中百分之二十二发生在横坑，百分之七十八发生在采矿处。这数字指出对于矿火防止主要应在采矿处着手。横坑里的矿火因为附近有碎煤，总是由于疏忽随便所造成的。采矿处的矿火发生的原因总是由于开采计划和采矿设备极多的煤已被掏空，而且废石区的封闭不够妥当。采矿处的矿火可发生在低至每天0。6至0。8米开采率，及遗弃的矿井。从通风分析中发现一方面用通风管来加强微弱的通风现状，一方面在废石区有重大的漏气现象。防止矿火最重要的办法是改善通风，因为开采率和废弃量都不能随意更改。作者对最适当的通风情形作了建议。

A PRACTICAL GUIDE TO THE DETECTION AND CONTROL OF MINE FIRES

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ABSTRACT

The paper takes an essentially practical look at the theory and practice of detecting and controlling mine fires. The paper is in two parts. The first part reviews the Author's experience in the early detection of fires and spontaneous combustion in coal mines and considers the merits of early detection methods such as continuous CO monitoring, CO/O_2 Deficiency Ratio, personal and visual examinations, and pack pipe sampling techniques. Mention is also made of the Multi-Discriminatory Alarm program as a means of applying computer technology to the problem.

The second part of the paper discusses methods of controlling outbreaks of fire and spontaneous combustion and reviews control techniques under the headings of direct attack, ventilation manipulation, inert gas usage, sealing techniques and fire fighting foams.

In conclusion, the paper suggests further lines of thought in tackling this important problem in the future.

INTRODUCTION

From time to time, events occur which concentrate our minds on the enormity of the problems faced when an underground fire or explosion occurs. The events of the explosion in the Stolzenbach lignite mine in Germany and the recent King's Cross disaster in London's Underground network graphically underline the fact that fire in underground tunnels and workings can be extremely costly in terms of human suffering and equipment loss.

DETECTION OF OUTBREAKS OF FIRE

When considering how to fight fires, it is axiomatic that the earlier remedial measures can be applied, the easier the task and the more certain the success. It is therefore useful to consider the methods used to identify fires, both open and concealed.

Monitoring of Carbon Monoxide

Carbon Monoxide detection has been used for more than 20 years in British coal mines. The first installations were based on the German system of tube bundle sampling. Each district has a tube through which air samples were drawn at intervals to an analysis point on the surface. Here analysis takes place and the results are presented on a chart recorder.

The normal CO content or fingerprint of the district under surveillance can be quickly established by reviewing the chart produced and analysing its base level. This can be done easily by eye. Very slow rises may be observed. In the Author's experience, a slow rise of perhaps 1ppm has taken place and been identified three weeks before the heating became noticeable by other methods.

The tube bundle method has some disadvantages. One is the delay which can occur when the sample is being taken some distance away from the analysis point. In some cases this period can be as long as 4 hrs where workings are some distance from the shafts. Such delays may not be important in the day to day monitoring situation but are very irksome and difficult to control in the event of an active heating, when it is vital to have early knowledge of the results of actions taken.

In order to overcome this problem, electrical transducer technology has been introduced with remote sensors transmitting readings to a surface computer. Although this system has the advantage of instantaneous indication, the accuracy of transducer cells has to be held in question in certain circumstances. Some transducers are sensitive to higher hydrocarbon fractions and distillates and may show CO readings two or three times higher than actually is the case. Attempts have been made to overcome this disadvantage by the application of filters and by modification to the cell technology. This has been partially successful but experience needs to be applied.

Experimental infra-red laser analysers have been applied to this problem and the technology seems promising, giving low drift and good selectivity. Work is proceeding to make the instruments intrinsically safe for underground use.

However, current transducers lack discrimination. It is difficult to determine in the initial stages of a rise in CO whether that rise is indeed of product of spontaneous activity or if it is some other factor such as diesel exhaust fume, shotfiring or barometric change.

If these events are treated seriously and examinations put into effect only to discover that the effect is transitory, then very soon the process becomes degraded and it is quite possible that a true warning will be ignored. Some attempt to overcome this problem has been made with the Multi-Discriminating Alarm technique discussed below.

Multi-Discrimatory Alarms

Chart recordings of CO trends are a basic means of identifying heating activity. When automatic alarms are applied to these readings, it is found that many false triggerings occur. The result is that the system is rapidly devalued in the estimation of the operators. McCormick (1986) described an algorithm to deal with this problem.

The Multi-Discriminating Alarm (MDA) project was instituted to develop the idea. The system was applied to a tube bundle sampler utilising a surface analyser. The resulting program will distinguish between transient phenomena and meaningful events.

The algorithm which was proposed analyses historical CO data on a statistical basis. It calculates the norm level as the geometric mean of the historical data and sets five alarm points, MDA1 to 5. MDA1 is set 2ppm (parts per million) above the norm and the others are spaced at intervals of 1 ppm above this. In

addition a duration time is calculated and this has to be exceeded before an alarm signal is generated.

In practice, a further high alarm is generated above MDA5 and this can trigger a change in sampling sequence so that a tube which exhibits high CO readings may be tested more frequently.

Although it is the practise to sample the mine air through the tube bundle system, some work has been done using a COSEC transducer system based on City Technology 2T electrochemical cells.

The routines have been applied to 16 collieries over a two year project from June 1985 to June 1987. At two of these collieries, twelve heatings were successfully detected at a very early stage of development. Some of these were flagged by the MDA system several days before the normal methods were able to confirm the activity. At all the collieries where existing monitoring facilities were run in parallel with MDA, the MDA routines consistently provided earlier warning with fewer false alarms.

CO/O^2 Deficiency Ratio

This quantity has been proposed as being capable of reflecting the activity and stage that a fire has reached. The proponents of the technique argue that it is sensitive and is able to indicate the very early onset of combustion. This is true and there is no doubt that some events have been discovered and managed in this way.

The main disadvantage to its use is that it requires a laboratory process to analyse specific samples. It is not a continuous process and does not readily show trends. Generally, it has been used as an adjunct to fighting active fires but not as a primary detector. British Coal practice is to bring a mobile laboratory to the surface analysis room at a mine and apply the more extensive analysis facilities to the tube bundle samples when required. This additional information is then used as a guide to the progress of measures taken.

The observation of other parameters such as the ethylene and alkane ratio is not used in the U.K. Such factors depend on the composition of the coals being worked and may be more applicable to certain seams.

Pack Pipe Sampling

This technique was developed in the Western Area of British Coal. It relies on the idea that samples aspirated through pipes placed 2 to 3 metres into the goaf at intervals along the return roadway will reflect the activity taking place in the goaf along a line drawn directly between intake and return. It may be induced from this information where the area of air leakage into the waste is located in the intake road. Sealing and injection should then be concentrated in this location.

The system requires that each pipe is regularly sampled and graphs drawn of the CO content. The base figures are naturally large and can be of the order of 2000 - 3000 ppm. The technique does suffer from the disadvantage that as a face advances and the waste consolidates, any particular pipe will pass through an envelope of CO percentages and so the actual reading will have this envelope superimposed on it. This can make analysis difficult. However, high readings do still stand out. The early recognition of a spontaneous combustion problem is made easier by the fact that airflows are very small within the waste and the effect of CO rises are therefore more pronounced with very

little dilution. The technique has been of considerable benefit to experienced practitioners in recognising deep seated heatings before they grew to emergency proportions. The method is labour intensive and requires regular attention.

Physical Examinations

The plethora of technology must not diminish the importance of physical examination of mines using the human senses to detect fires. Sight, touch and smell are very valuable in that they are portable, adaptable and reasonably sensitive. In the Author's experience as many heatings have been detected by this method in the course of regular inspections as by remote sampling. This may be because the automatic sensor is wrongly situated. When designing the environmental layout, the Ventilation Engineer must decide in advance where a fire is likely to occur in order to place the equipment which is to detect it.

Smell, above all, is a good guide to the existence of fire. This is particularly so when wood is involved and higher hydrocarbons are given off. Sight and feel warn of the sweating and haze which can occur where activity is taking place.

Again, experience has a part to play. Warm air issuing from a pack or waste may condense its water vapour on contact with cool intake air without any real problem emerging in the life of the district.

To summarise this part of the discussion, it may be stated that generally within the British mining industry, CO measurement has become the standard tool with which to chart spontaneous heating activity. It is easy to apply and simple to maintain.

Even so, experience is needed in its interpretation. With the introduction of the discriminatory program this aspect is being dealt with and a reliable system is being developed.

CONTROL OF FIRES AND HEATINGS

A number of techniques have been developed to control heatings and fires. By 'heating' is meant a concealed outbreak of spontaneous combustion and by 'fire' is meant an occurrence of open flame. The manner of dealing with each classification is different in terms of strategy and safety of the teams involved.

Ventilation Pressure

The most common place for a heating to occur on an advancing face is on the line of least compaction along the original face start line. It has usually been the case that the heating takes place nearer to the intake side of the old face start line but the actual position is not of great importance when considering the combative measures. The heating is characterised by the issue into the return airway of considerable amounts of Carbon Monoxide. The monitoring arrangements mentioned earlier usually pick up the onset of the heating and for this reason are placed outbye of the old face start line in the return.

Incidents have been handled by the method to be described when the CO concentration has been of the order of 100 ppm.

The requirement for the control of the heating is to prevent air continuing to reach the reaction site. The ventilation pressure difference across the district is the motivating force encouraging the air to leak along the poorly compacted face start line. If this pressure difference could be neutralised, the

problem would be solved. However, in accomplishing this the pressure driving air around the district would be lost. The problem is therefore how to achieve the effect of zero pressure across the old face line whilst maintaining the required ventilation around the district to enable work to continue (Lord 1977).Fig 1. shows diagrammatically how this is achieved.

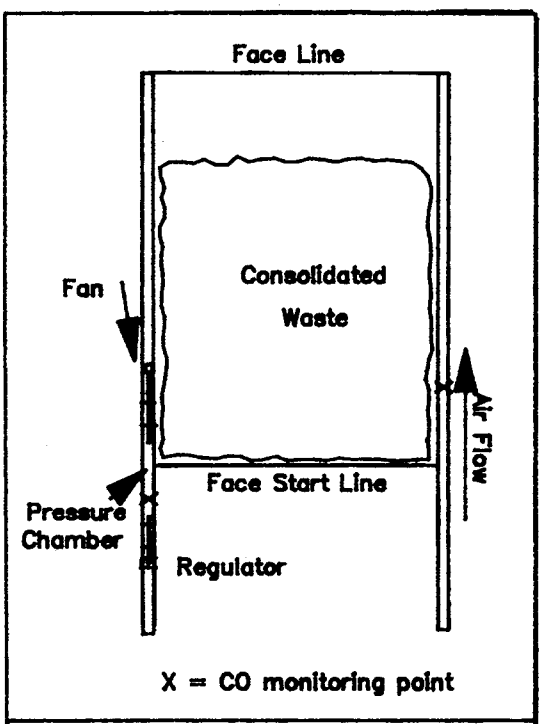


Fig 1 Arrangement of Pressure Chamber

A temporary airlock is erected in the return at a convenient distance inbye the face start line at a site where the roadside is well compacted. Two pairs of airlocks should be constructed of rapidly erected airtight screens. Each pair should have provision for the passage of any haulage vehicles without both doors being open at the same time. There should also be ventilation ducting passing through each pair of screens into what will be the pressure chamber. The screens created on the outbye side of the old face start line should be some 20m away. This distance should be sufficient to take into account any breaks in the ground around the face start line excavation. The ventilation duct passing from the inside of the chamber to the outside should include a regulator section.

Forcing fans should then be placed on the inbye duct capable of passing the required amount of air to ventilate the face more or less normally. When these fans are started they will draw air around the face and into the chamber created by the twin airlocks. The pressure in this chamber can be controlled by the regulator and by slight closure the pressure difference between the intake side of the old face start line and the return side may be removed. This will effectively prevent the leakage and bring the heating under control.

The control of the pressure difference is of course crucial. It is very often not practical to apply instruments across the district to measure the actual value so a more convenient way must be adopted. This can be very neatly achieved by monitoring the CO content in the pressure chamber itself. In practice, it has been found that a value of 5 to 6 ppm is quite sufficient to maintain control over the combustion whilst remedial measures such as pumping and injecting the leakage points around the intake gate is carried out.

The maintenance of this flow of CO into the chamber is also most important from a safety point of view. If the pressure in the chamber is allowed to exceed that which the ventilation circuit produces across the face start line then, of course, the direction of flow will reverse. This will have the effect of forcing the CO along the leakage paths and back into the intake gate from where it will recirculate around the dis-

trict with perhaps unfortunate consequences.

The remedy when applied can bring a situation under control very rapidly and when the fans are started, the results are seen within seconds.

The technique outlined above has proved to be very successful in a number of heatings and has become the standard method of control in such circumstances. It is important to appreciate its limitations. In particular it cannot be used in situations where the waste is "green" or unconsolidated. The fans can cause leakage recirculation through the waste at the side of the road and it has been known for fires to be generated from this.

It should be pointed out that the alteration of pressure distribution within the district does not affect the pressure distribution over the mine. In fact the ventilation pressure across the district outbye of the chamber remains the same.

A number of variations have been used in the application of the method. If for some reason it is not practical to install the forcing fans in the return airway, a similar result may be obtained by installing exhausting fans in the intake and so effectively reducing the intake pressure to a value approaching that at the return end. All the caveats equally apply and a small flow of CO into the return gate should be maintained.

One instance occurred in a mine where superjacent seams had been worked 10 to 15m above the production district (Fig 2). The upper workings were old and were sealed off at the shaft sides in inaccessible places. A heating took place in the seams above and the working face was polluted by the products of combustion. The remedy was to install forcing fans blowing through an airlock screen in the intake gate and regulate the pressure at an

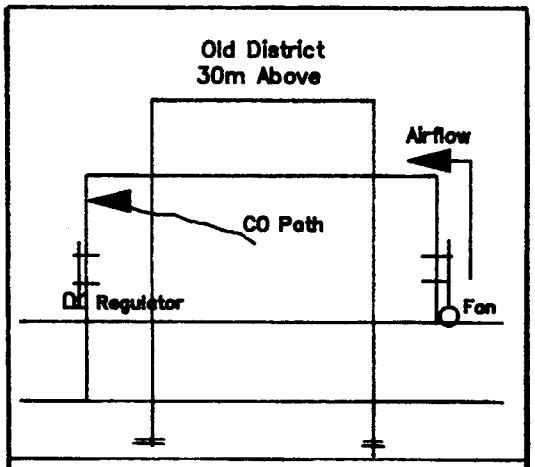


Fig 2: CO from seam above

airlock screen in the outbye end of the return gate. This effectively lifted the pressure in the whole of the district and prevented the leakage flow of contaminants. The production district operated in this manner for the whole of its life - some 18 months.

Nitrogen inertisation

Nitrogen inertisation has been used successfully in Britain for 8 years. The technology has progressed rapidly over that time. The initial equipment was "borrowed" from the oil industry and it had a capacity far in excess of that which British Coal required for the application. The hire of this equipment was also very costly, although when considered against the loss of a face with its capital equipment and reserves, it was viable. However, a drive was instigated to gain the advantages of the method at a lower cost. The aim is to make the technology cheap enough to allow routine inertisation of faces rather than to have to react to a heating situation which had become out of hand.

The track record has been outstanding. In a six year period from 1980 twenty-one serious heatings took place in a British Coal Area involving 8 of the 11 mines. Eighteen of the incidents were successfully controlled. Of the remainder, in the first case the technique was not appropriate as the fire was located in a overlying seam (it was in fact the case mentioned above in the previous section), in the second case, the fire was located immediately behind the chocks on a salvaging face and the nitrogen could not be realistically applied to the site, and in the third case, which was the second actual incident to apply the technique, the leakage paths for the air feeding the heating were not properly identified due to extensive faulting and the district was finally lost.

With such a record the application of Nitrogen soon gained favour and ways were sought to reduce the cost. The original equipment first used in 1980 vapourised liquid nitrogen to normal temperature and a pressure of about 7bar prior to delivery by pipeline into the mine. The heat for vapourisation was supplied by diesel oil burners. The whole equipment was manually controlled 24 hrs per day. The cost was in the region of £3000 per day for a flow rate of 5 to 10 m^3/min.

In association with the Service Company providing the equipment, a specification for an evaporator was drawn up which would achieve two objectives,

a) be tailored to deliver gas in a quantity and pressure more suited to the application

b) to be considerably cheaper to run.

Out of these ideas was born Mobile Electric Vapouriser Mk1. MEV1 as it came to be known, utilised the ambient air temperature to heat the liquid Nitrogen in the first place, through the use of a heat exchanger not unlike a huge car radiator. The temperature and delivery pressure were then trimmed to final acceptable limits by electric heaters and pumps. The whole operation was computer controlled and removed the necessity for human supervision. The required flow rate was set on a dial and the machine ensured that this was delivered.

Initial studies had shown that this equipment, rented over a five year period for British Coal's exclusive use, would break even on a 100 day per year usage basis compared with the Diesel rig. In the event, the first year of operation saw its use for 353 days as a result of the inertisation performance and halving of costs of operation compared to similar flow rates with the Diesel rig.

Development has proceeded and in 1985 another rig, MEV2, was ordered and delivered. This again had its specification reviewed and the duty cycle was designed as a maximum flow rate of 40 m3/min at 9 bar. The resulting economies meant a reduced capital cost and the rig is operating at about £1000 per day at the same criteria as above.

However the goal of cheap routine inertisation is still to be achieved, although it must be said that the present equipment is being used in this mode at two mines in Warwickshire which have an extremely high spontaneous combustion risk and which are also two of the most profitable mines in the country.

To further reduce cost, an investigation has been commissioned into Pressure Swing Absorption (PSA) technology to generate gaseous Nitrogen on site by extraction from

the atmosphere. With this technique there is a tradeoff between the quantity of the nitrogen delivered and its purity for a given size of plant and the investigations have to take account of this. It is generally accepted in the UK that a waste inertised by 97% pure nitrogen is unlikely to exhibit spontaneous combustion. Initial figures show that PSA technology can produce 5 to 10 m3/min of nitrogen at a cost of £200 per day at 97% purity. This would represent a negligible "insurance" premium.

Nitrogen use at the point of injection has also progressed. Nitrogen is very good at controlling heatings in unconsolidated wastes in the circumstances in which pressure chambers do not perform at all well. On Advancing faces, the region in the waste about 100-150m back from the face line is in the process of becoming consolidated and is therefore porous to air leakage.

Nitrogen injection into this area through standpipes placed along the side of the intake road tend to form a triangular inertisation pattern. A rule of thumb states that the length of face inertised will be similar to the length back along the roadside which is being injected. If a heating is generated a long way down the faceline by air sweeping off the face , it is unlikely to be within the zone of control.

However, a technique is being developed involving the anchorage of 25mm diameter steel pipes to each fifth chock through the face. These pipes extend 10 to 15m into the waste and are dragged through the waste as the chocks are moved forward. The fines left under the conveyor protect them from the falling roof rock.

The pipes are connected through valves to a conventional armoured hose carrying Nitrogen through the face. Such an arrangement allows the gas to be injected into the waste and form an inert curtain against air leakage. See Fig.3

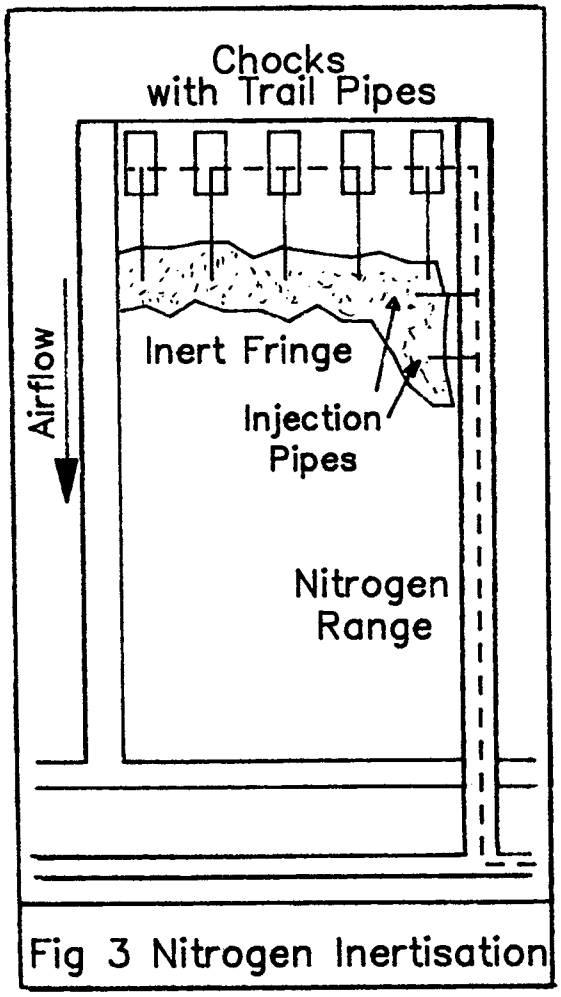


Fig 3 Nitrogen Inertisation

Pump Packing

One method which has been extensively used in the Staffordshire and Lancashire coalfields to prevent and reduce the risk of spontaneous combustion is pump packing. This involves the use of materials which have a capacity for absorbing many times their own volume of water and then setting hard. Most of the materials are cementitious and the method is favoured because of its

speed.

The material is mixed with water on site and pumped into reinforced plastic bags placed in the roadside pack hole but great care has to be exercised to seal the corners of the pack and roof.

If leakage does occur, the combustion problem can be made worse as the packs present a very strong line and prevent the waste fully collapsing. Thus a leakage path is created in parallel to the roadway and the conditions are ripe for spontaneous combustion to occur.

Open Fires

The method of dealing with open fires is very often the ultimate one of sealing the roadway off. Even this operation is fraught with difficulties and it is important that air access is maintained around the affected circuit until such time as both intake and return seals are in a position to be closed simultaneously.

When a district has to be so sealed, it is a very good practice to leave remote sensing instruments, Methane and Carbon monoxide, behind the stopping to give an indication of the conditions there. Sampling tubes should most certainly be left through the stopping walls and should terminate some 20 to 30m inbye the stopping site to be out of the immediate area affected by the seals "breathing" under the influence of barometric changes. Tubes installed in this manner allow air samples to be drawn out and analysed after sealing.

The most difficult stopping to accomplish is that in the return airstream. Very often conditions are hot, the atmosphere is irrespirable and breathing apparatus may have to be worn. Conditions can be made more tolerable if the site is chosen in the vicinity of a unpolluted air supply. Two actions need to be accomplished,

i) The hot gases may be ducted past the stopping work site by erecting a screen across the road and incorporating a ventilation duct in it. The fumes may then be channelled in the duct past the workers.

ii) A fresh air supply may be established by using an auxiliary fan . See Fig.4.

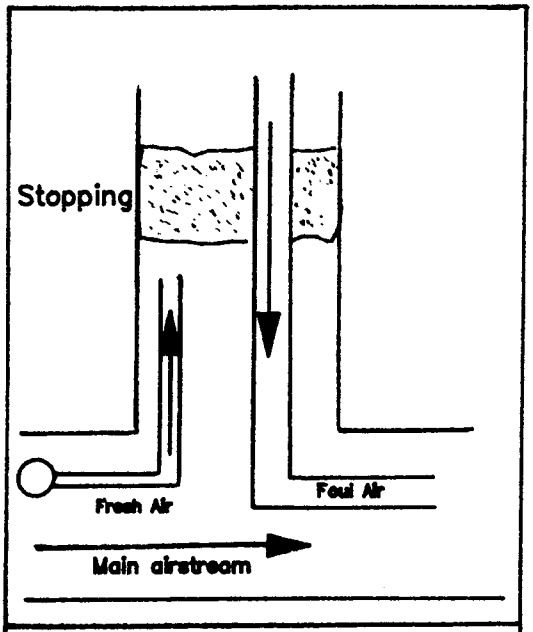


Fig 4: Stopping Construction

Open fire should be attacked as early as possible to ensure the greatest chance of success. Fire ranges and hydrants should be positioned at regular intervals through the mine and portable extinguishers placed on the intake side of all danger spots such as conveyor drives, switches and haulage motors. Conveyor belts represent one of the greatest hazards underground and account for half of the reported fires in British Coal Corporation mines. The best safeguard is adequate

maintenance, and cleansing. All defective rollers should be replaced as soon as possible.

Foam Firefighting Techniques

Research is being undertaken within the British Coal Corporation to evaluate foams in the control of serious mine fires.

The subject was first approached in the late 1960's. In these trials, a foam solution was sprayed over a fine mesh net stretched across the roadway so that the ventilation current would form bubbles as it passed through. The technique did not prove viable due to low air speeds and the fact that where foam was produced in significant quantity, the airstream was impeded and in some cases stopped completely.

A new development has sought to overcome this problem by using a water hydrant powered fan directing its airflow through a net attached to the fan and onto which is a foam solution is sprayed. This arrangement produces copious amounts of foam independently of the roadway airspeed. It was tested in airspeeds of 4m/sec and 2 m/sec. At 4m/sec the foam never completely filled the roadway and therefore could not attack roof fires.

The tests carried out on wood, oil, conveyor belt and coal and roof fires were so contrived to give a 4 MegaWatt output. In all cases the fire was brought under control and its severity greatly reduced. The time taken varied between 1 and 3 minutes but none of the fires were completely extinguished. However, conventional hoses could have been used in safety to damp down the remaining smouldering embers.

Conclusion

The paper has attempted to outline in a practical manner, the subject of underground fire fighting. The best means of firefighting is prevention and it is an important function of all those who bear responsibility for underground workings to ensure that adequate preventative measures are in force.

Should a fire occur, then due regard must be given to the provision of training and equipment to regain the situation as soon as possible. The paper has covered the techniques currently found of use to aid the process of safer coal mining.

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GUIDE PRATIQUE DE DETECTION ET DE CONTROLE DES INCENDIES MINIERS

M.J. Herbert

Ce rapport présente une vue essentiellement pragmatique de la théorie et de la pratique de la détection et du contrôle des incendies miniers. Il se compose de deux parties: la première, qui passe en revuel'expérience de l'auteur concernant la détection à l'avance des incendies et des combustions spontanées dans les mines de charbon, examine les mérites des méthodes de détection suivantes: surveillance continue du CO, taux de détection CO/O_2, examens visuels et personnels, ainsi que des techniques de prélèvement de remblais de tuyaux. Elle mentionne également la technique d'alarme à multi-sélection, qui permet d'appliquer la technologie de l'informatique à ce problème.

La seconde partie du rapport examine les méthodes de contrôle des incendies et passe en revue les techniques de contrôle suivantes: attache directe, méthodes de manipulation de la ventilation, utilisation de gaz inertes, techniques d'obturation et mousses anti-incendie.

En conclusion, le rapport suggère des directives permettant de traiter cet important problème à l'avenir.

EINE PRAKTISCHE ANLEITUNG ZUR ENTDECKUNG UND BEHERRSCHUNG VON GRUBENBRÄNDEN.

M. J. Herbert

Der Vortrag gibt einen praxisbezogenen Überblick über die Theorie der Entdeckung und Bekämpfung von Grubenbränden. Der Vortrag besteht aus zwei Teilen. Der erste Teil berichtet über die mit Frühentdeckungen von Grubenbränden und Selbstentzündungsbränden gesammelten Erfahrungen und betrachtet die Vorzüge einer Früherkennungsmethode, wie z. B.: die kontinuierliche CO Überwachung, die CO/CO_2 Relationsmethode, visuelle Untersuchungen und die Entnahme von Gasproben aus Rohren in Bruchzonen. Der Einsatz eines Computers für die "Multi-Discriminating Alarm" Methode wird vorgestellt.

Der zweite Teil des Vortrages betrachtet Methoden zur Kontrolle von Brandausbrüchen und Selbstentzündungen und berichtet über Bekämpfungsmethoden in Vortriebsstrecken, Manipulation der Bewetterung, den Einsatz von inerten Gasen, Abdichtungsmethoden und die Brandbekämpfung mit Löschschäumen.

Als Schlußfolgerung werden weitere Gedankenwege zur zukünftigen Beherrschung dieses wichtigen Problems vorgeschlagen.

ПРАКТИЧЕСКОЕ РУКОВОДСТВО ПО ОБНАРУЖЕНИЮ И ЛИКВИДАЦИИ ПОЖАРОВ В ШАХТАХ

М.Дж. Герберт

Настоящий доклад рассматривает теорию и практику обнаружения и ликвидации пожаров в шахтах. Доклад состоит из двух частей. В первой части описывается опыт автора по раннему обнаружению пожаров и внезапных возгораний в угольных шахтах, и рассматриваются следующие методы раннего обнаружения: постоянное наблюдение за содержанием CO, соотношение CO/O , инструментальная и личная проверка, и отбор проб воздуха. Упоминается также использование многоканальной избирательной аварийной системы как одного из путей применения вычислительной техники к решению этой проблемы.

Во второй части доклада обсуждаются методы подавления очагов пожара и внезапного возгорания, а также дается обзор методов контроля в следующих разделах: непосредственная связь, методы использования потоков воздуха, использование инертных газов, техника герметизации и тушение пожара пеной. В заключение предлагаются некоторые идеи по борьбе с этой серьезной проблемой в будущем.

矿火测定和控制的实用指南

M．J．赫勃特

本文叙述矿火测定和控制的实况．一共分成两部份．第一部份回顾作者关于在煤矿中自燃和矿火及早测定的经验，并且列述一氧化碳侦测器，一氧化碳/氧气比值测定，目视和亲自的侦察，和充填管件取样这些及早测定方法的优点．此外应用计算机技术的多种识别警报器也在此提到．

第二部份讨论起火和自燃的控制方法，并综述下列控制的技术：直接救火，通风调配，惰性气体的使用，封闭的办法，和泡末救火法．

结论部分本文对未来应如何对付这个重要问题做了一些建议性的构想．

Session IV

METHANE

MEASUREMENTS AND SIMULATIONS ON THE RELATION BETWEEN METHANE FLOW RATE INTO HEADINGS AND SURROUNDING GEOLOGICAL CONDITIONS

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ABSTRACT

In Japanese coal mines, drivage works of in-seam gate road headings are mainly carried out by blasting methods.
A large number of change of methane concentration in headings after blastings have been recorded by centralizing monitoring systems. From the results of analyses of the collected data, the changes of methane concentration with time could be divided into the five different patterns. It is conceivable that the appearance of the each pattern depends upon the geological conditions surrounding the headings. For the discussion, simulative numerical calculations were performed by the mathematical models including geological conditions like faults, gas pressure and gas content in coal seams, gas permeability of coal seams and so on.
From the results of calculations it is confirmed that the methane flow rates into headings after blastings can be strongly affected by surrounding geological conditions.

INTRODUCTION

Japan produced about 12 million tons of coal last year. About half of the production comes from six coal mines in Hokkaido, northern island of Japan.
Most of them are gassy coal mines and the average mining depth of these coal mines has attained about 600m from the surface.
These coal mines have been suffered from occurrences of coal and gas outbursts.
To prevent outbursts some countermeasures are taken and studies on methane flow rate from the coal seams have been carried out.
We attempted to analyze the changes of the methane flow rate from coal seams into in-seam gate roads after blastings.

METHANE MONITORING SYSTEM

As shown in Fig.1, a methane monitoring system consists of two methane sensors and an air flow sensor. A methane sensor is located near a fan in intake air and the other is located about 20m from a face in return air. An air flow sensor is set in a tubing. These sensors are connected to a digital transmission control unit and signals gained are transmitted to a CPU at the surface through fiber cables.
Fig.2 shows an example of out put data.

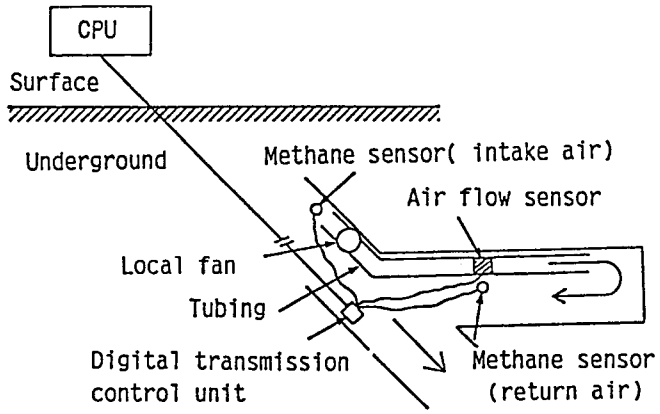


Fig.1 Methane monitoring system

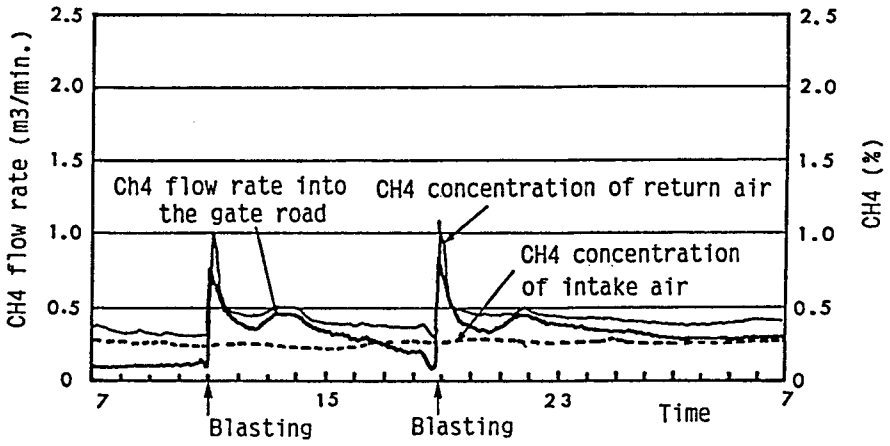


Fig.2 An example of out put data

METHANE FLOW RATE INTO A GATE ROAD AFTER BLASTINGS

Methane flow rate is calculated as follows;

$$\text{Methane flow rate} = \frac{(B - A) \times F}{100} \quad (1)$$

Where B = Methane concentration of outlet air;[%]
A = Methane concentration of intake air;[%]
F = Air flow rate in the tubing;[m3/min.]

The change of methane flow rate into the gate road with time is shown in Fig.2,too.

CLASSIFICATION INTO FIVE PATTERNS

After the collection and the observation of a number of the changes of methane flow rate into gate roads with time, we classified them into five patterns as shown in Fig.3. However,there are some curves that are not similar to the five patterns. We treat them as the others and put aside as Pattern VI.

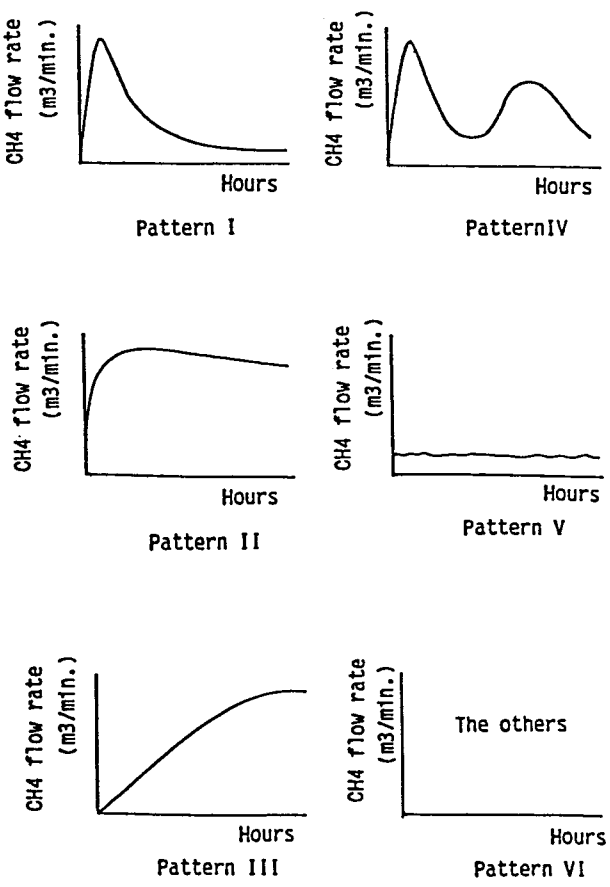


Fig.3 Characteristic curves of methane concentration changes in gate roads after blastings.

TESTING FIELD CONDITIONS

The collection of the data was carried out at Akabira and Minami Oyubari Coal Mine in Hokkaido. The data were collected from five driving gate roads in two coal seams which were called the 6th and the 7th seam at Akabira Coal Mine and from one driving gate road in the lower seam at Minami Oyubari Coal Mine.

The depth of these gate roads are about 700m and 920m from the surface respectively.
The data collection works were restricted within usual blasting shifts in drilling pattern and number of explosives.

RELATIVE FREQUENCY OF APPEARANCE OF EACH PATTERN

Fig.4 shows the relative frequency of appearance of each pattern for each coal mines.
Akabira Coal Mine produces steaming coal and Minami Oyubari Coal Mine produces coking coal.
And there is rather big difference in strength of coals. For example, friability of 7th ,6th and the lower seam is 45-45,30-35 and 35-40 respectively. But there is no big difference in the relative frequency of appearance of each pattern.

Fig.5 shows the difference of the appearance of each pattern between the 7th and the 6th coal seam. The relative frequencies of pattern I and IV are higher in the 6th coal seam than in the 7th coal seam. In addition to this,the relative frequencies of pattern II ,III and V are very low in the 6th coal seam.

Fig.6 shows the difference of relative frequencies of appearance of each pattern in the case where there were faults on the face and not. It can be seen that in the case of existing faults on the face of the frequency of pattern IV is the highest and the second highest is pattern I . However, in the case where there is no faults on the face of the same gate road the highest frequency is pattern I.

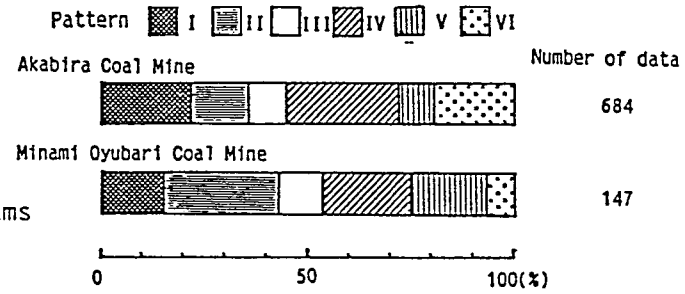


Fig.4 Relative frequency of appearance of each pattern

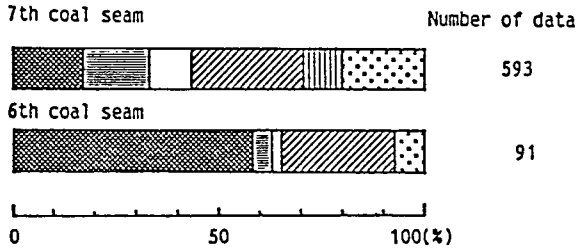


Fig.5 Difference between coal seams

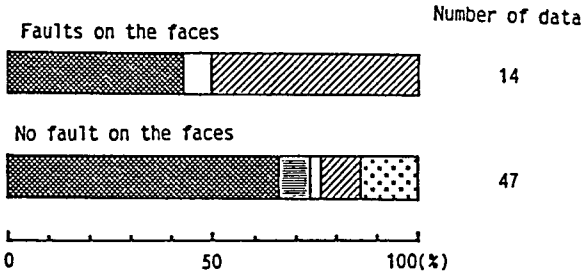


Fig.6 Effect of faults

From the before mentioned observations of the classification,it is considered that the following items may affect the shape of patterns.

1) Difference of coal seam
2) Gas drainage methods applied
3) Countermeasures taken for gas outbursts
4) Existence of faults
5) etc.

To confirm the effects of these items,theoretical and numerical calculations were conducted by using mathematical models.

SIMULATION BY MATHEMATICAL MODELS

Methane flow into a heading can be divided into two categories. They are 1) methane flow from broken coal lumps by a blasting (Q_1) and 2) methane flow from a coal seam surrounding a gate road(Q_s). Total methane flow (Q) into a heading should be evaluated by $Q_1 + Q_s$.

Methane flow from broken coal lumps (Q_1)

Under the assumptions of 1) each coal lump has spherical shape, 2) methane movement in a coal lump is governed by Darcy's law ,3) unit volume of coal has an apparent porosity of q_n , change of methane pressure in a lump coal shown in Fig.7 is described by the equation (2).

$$\frac{\partial P}{\partial t} = \frac{K}{2\mu q_n}\left(\frac{2}{r}\frac{\partial P^2}{\partial r} + \frac{\partial P^2}{\partial r^2}\right) \qquad (2)$$

Where, P; gas pressure
t; time
K; gas permeability of coal
μ; viscosity of methane.

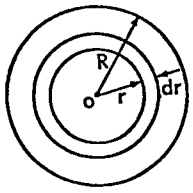


Fig.7 Spherical coordinates

Basing upon the equation (2),methane flow from a lump coal of radius R can be described by the equation (3)

$$Qi = \frac{4\Pi}{P} q_n \int_0^R r^2 (P - Pn)dr \qquad (3)$$

Where,Pn means barometric pressure in a gate road. Total methane flow from a heap of broken coal lumps (Q_1) will be evaluated by the equation (4) under the consideration of size distribution or numbers (i) of broken coal lumps as a function of R.

$$Q_1 = \sum_i Qi \qquad (4)$$

Fig.8 shows the results of calculations for a drivage of $5.12m^3$ of coal by a blasting.
For the simplicity,the additional assumption that the radius of broken coal lumps are same was applied in these calculations.
From the figures,it is anticipated that methane flows into headings from broken coal lumps are strongly affected by initial gas pressure in coal lumps, gas permeability of coal and size distribution of broken coal. But the declining tendencies of gas flow rate curves have a monotonous shape and it is considered that methane flow from broken coal lumps will not give strong changes to total methane flow patterns.

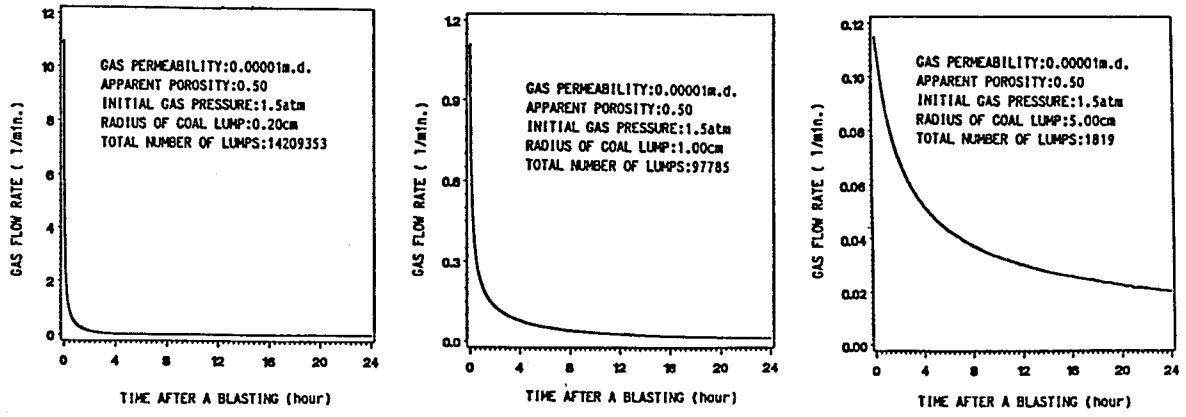


Fig.8 Calculated methane flow from broken coal lumps

Methane flow from a coal seam surrounding a gate road (Qs)

To calculate the methane flow rate from the coal seam surrounding a gate road by a computer, following assumptions were made.
Methane movement in coal seams is governed by Darcy's law. Coal seams contain only methane . If adsorption of methane in the coal seam is neglected, change of methane pressure in the coal seam can be represented by the following basic partial differential equation (5).

$$\frac{\partial P}{\partial t} = \frac{K}{2\mu\phi}\left(\frac{\partial^2 P^2}{\partial X^2} + \frac{\partial^2 P^2}{\partial Y^2}\right) \quad (5)$$

Where P: Methane gas pressure in the coal seam
t: Time
K: Gas permeability
ϕ: porosity
μ: Viscosity of methane
X: Distance to the direction of the X axis
Y: Distance to the direction of the Y axis.

It was assumed that the coal seam is a board of the thickness of 2m and gas does not move to the vertical direction.

As shown in Fig.9, the plane section of the calculated region is symmetry with respect to the center line of the gate road. X axis is taken on the center line of gate roads and Y axis is taken in the direction as shown in the same figure.
As the coal seam surrounding a gate road is thought to be disturbed, a higher permeability zone is established in the surrounding coal seam as shown in Fig.10. Approximate solutions of the equation (5) were obtained by the numerical integration of the derived difference equation.
From the obtained distribution of the methane gas pressure in the coal seam, methane flow rate Qs from the coal seam into the gate road can be calculated as follows.

$$Qs = \int_0^{Y1}\int_0^{X1} (Pi - P)\, dx\, dy \quad (6)$$

Where, Pi: Initial methane pressure in the coal seam
P : Methane pressure in coal seam at arbitrary time
X1: The range in the direction of X axis
Y1: The range in the direction of Y axis.

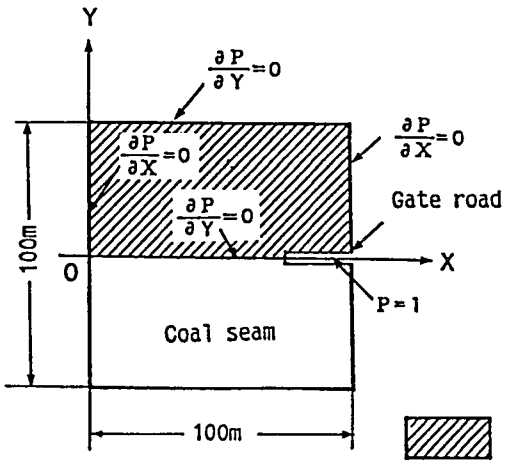


Fig.9 Region for calculation and boundary conditions

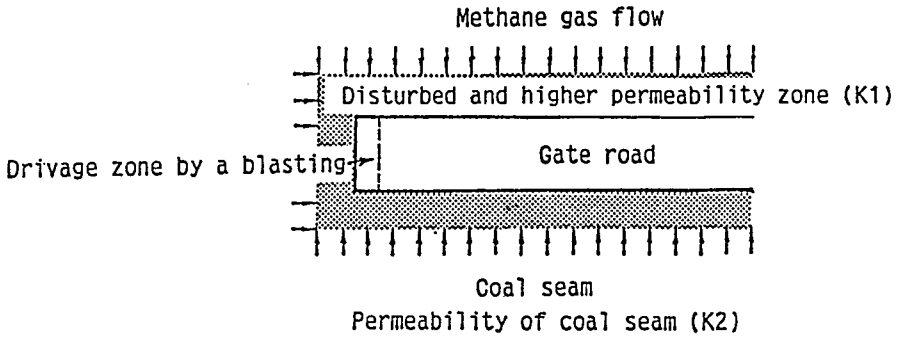


Fig.10 Disturbed zone surrounding a gate road

Calculation is performed in the following steps. At first, methane pressure in the coal seam surrounding the gate road before the blasting is calculated by using initial and boundary conditions shown in Fig.9.
From the calculated pressure distribution, methane flow rate Qs is obtainable using equation (6) as a function of time.
This calculated methane flow rate decreases with time and after the calculations for a period of the few days, the flow rate approaches to a constant value.
We considered that the distribution of the methane pressure in the coal seam at the above mentioned time is the initial condition of the coal seam before the blasting.
As the second step, the methane flow rate from the coal seam surrounding the gate road after it was driven 1.4m by the blasting is calculated for three models.
Table 1 shows the values used for this calculation .

Table 1 Values used for the calculations

Length of the gate road	25m
Width of the gate road	3.2m
Width of disturbed zone	3.2m
Mesh-width X of a net	0.4m
Mesh-width Y of a net	0.4m
Time interval of calculation	3.0sec.
Porosity of the coal seam	0.1
Viscosity of methane gas	0.011c.p.

Model A
As shown in Fig.11, the gas permeability of the large part of the coal seam is K2 and that of the disturbed zone surrounding the gate road is K1. The value of K1 is thought to be higher than K2.
Fig.12 shows some results of simulations. These curves show the change of the methane flow rate for one day after the blasting. It was assumed that seven days have passed before the blasting.

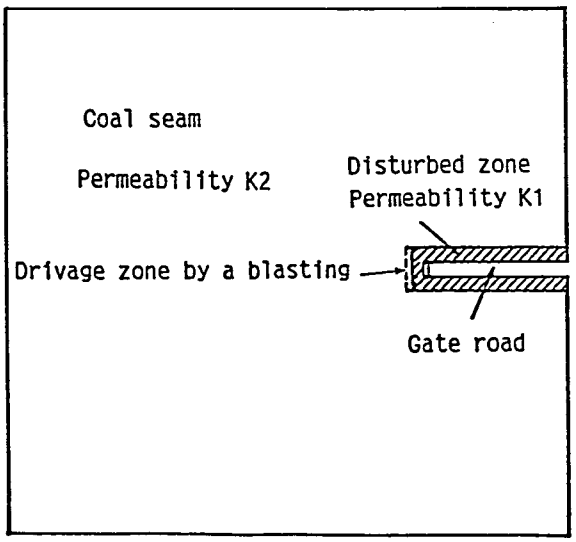


Fig.11 Model A

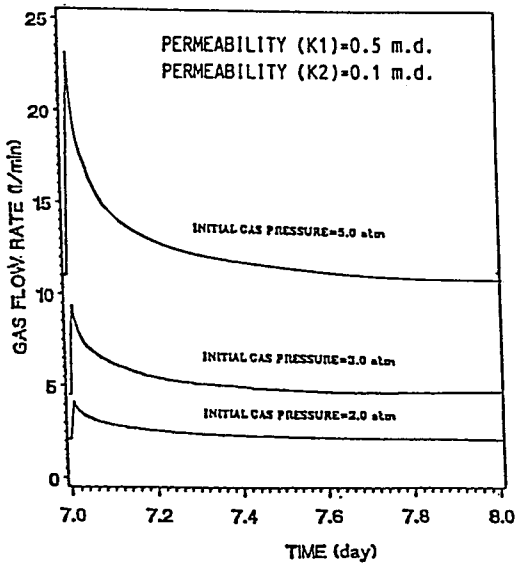


Fig.12 Calculated methane flow rate curves (Model A)

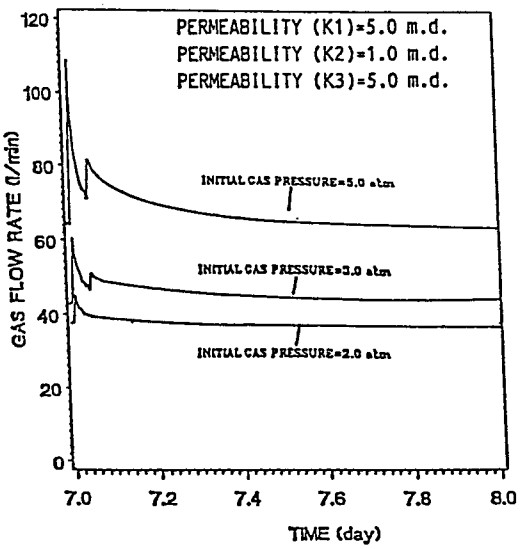


Fig.14 Calculated methane flow rate curves (Model B)

Model B

Fig.13 shows simulation model B. In this model,there is a fault near the face and the fault is gradually connected with the disturbed zone surrounding the gate road by a fissure zone an hour after the blasting. The fault is 3.2m wide and the distance from the headings is 6.4m.
It is assumed that the fissure zone is 0.8m wide and the permeabilities of this fissure zone (K3) and the fault (K1) are the same value .
In these conditions the results of simulation are shown in Fig.14 .

Model C

Fig.15 shows simulation model C . In this model, there is a fault and a degassed area in an inner part of the coal seam . It is considered that the degassed area has been drained methane by methane drainage methods and so it contained little methane.
It is assumed that this zone is Lm long and Wm wide. The results of simulation are shown in Fig.16.
When the size of the degased zone was changed to be wider and longer, we can get the methane flow rate curves as shown in Fig.17.

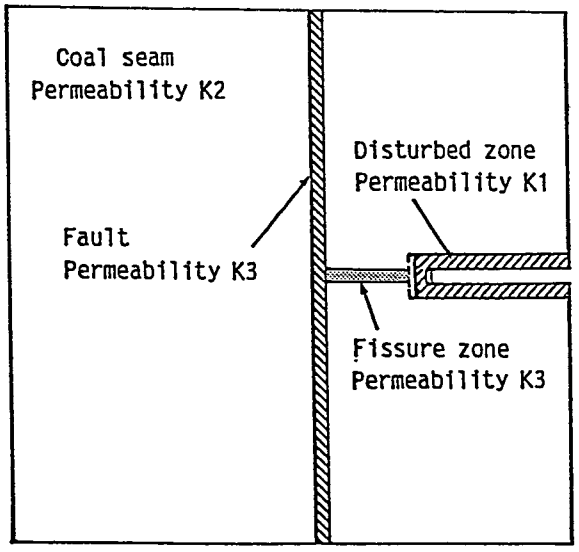


Fig.13 Model B

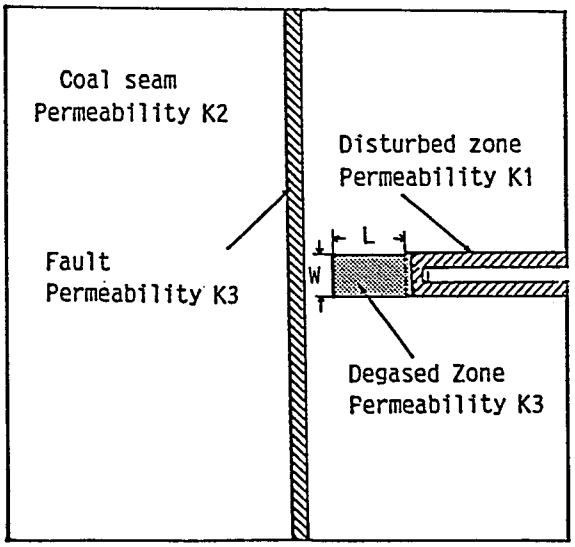


Fig.15 Model C

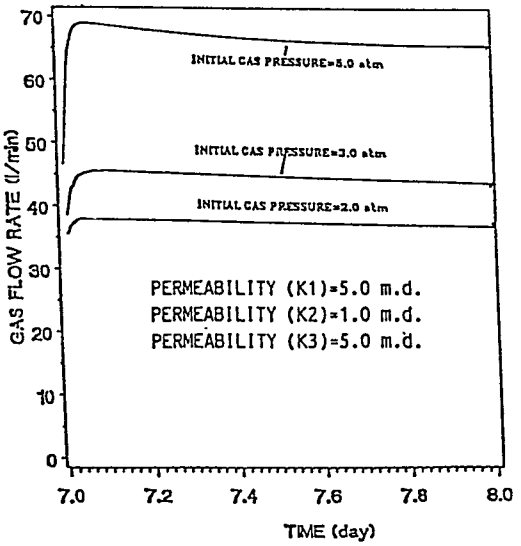


Fig.16 Calculated methane flow rate curves
(Model C , Degased area L=9.6m, W=3.2m)

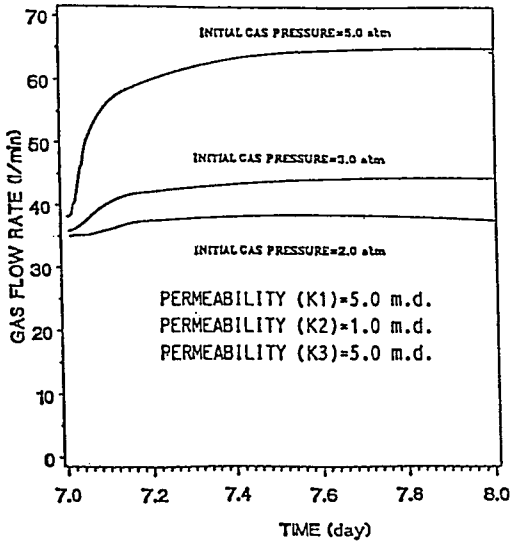


Fig.17 Calculated methane flow rate curves
(Model C , Degased area L=9.6m, W= 12.8m)

DISCUSSIONS

Under the assumption that the methane flow from broken coal lumps (Q_1) does not give remarkable change to the pattern of total methane flow (Q) into a gate road, we can consider as follows.

As shown in Fig.11 and 12, usual driving works in uniform coal seams are thought to make methane flow patterns similar to Pattern I in Fig.3. In this case the coal seam surrounding the heading contains some methane before the blasting. After the blasting,methane is emitted to the gate road from the broken coal, the face and the surrounding coal seam.
Pattern IV in Fig.3 and the curves in Fig.14 have similar two characteristic peaks. It can be considered that the first peak is resulted from the methane emission from the blasted coal lumps ,the face and the surrounding coal seam. The second peak can be explained by the additional gas flow through the fissured zone as shown in Model B or Fig.13. The relation between the existence of faults on a face and the appearance of pattern IV is observed in Fig.6.
From the simulation shown in Fig.15, 16 and 17, the existence of degased and higher permeability zones may make gas flow patterns similar to Pattern II or III.
When the coal seam surrounding the heading contains little methane, any increase in methane flow into the heading cannot be seen like Pattern V.

CONCLUSIONS

(1) From the results of analyses of curves of methane flow rate change with time after blastings in in-seam gate roads, we recognized that some different shapes of curves may appear in spite of the same coal seam and the same drivage method.

(2) The change of methane flow rate with time could be divided into the five different patterns. It is conceivable that the appearance of the each pattern depends upon the geological conditions like the difference of coal seams, existence of faults and so on.

(3) Simulative numerical calculations were performed by the mathematical models which were including the degased zone and/or faults in the deeper part of the coal seam surrounding the gate road. From the results of simulations, we could obtain some shapes of methane flow rate into the gate roads which were similar to the patterns shown in Fig.3.

(4) Therefore, we consider that the monitoring of the patterns of methane flow rate into the gate road after the blastings enables to predict or evaluate the existence of faults, the effect of the methane drainage and the possibility of the occurrence of outbursts.

(5) Additional considerations in the simulation models like the loading work of the broken coal and the absorption of methane by coal are should be done in future.

MESURES ET SIMULATIONS CONCERNANT LE RAPPORT ENTRE LE TAUX DE DEGAGEMENT DU METHANE DANS LES AVANCEES ET LES CONDITIONS GEOLOGIQUES ENVIRONNANTES

Kiyoshi Higuchi, Kotaro Ohga

Dans les mines de charbon japonaises, les travaux de creusement des avancées de voies à l'intérieur des filons sont principalement effectués par tirages.

Après ces tirages, le système de surveillance central a relevé de nombreuses modifications de la concentration du méthane dans les avancées. Sur la base des résultats des analyses des données recueillies, on a pu diviser en cinq catégories différentes les modifications de la concentration du méthane qui se produisent au bout d'un certain temps. Chacune d'entre elles dépend sans doute des conditions géologiques autour des avancées. Des calculs numériques de simulation ont été effectués au moyen de modèles mathématiques tenant compte des conditions géologiques suivantes: fissures, pression et teneur en gaz des couches de charbon, perméabilité des couches de charbon au gaz, etc...

Les résultats de ces calculs ont confirmé que le taux de dégagement du méthane dans les avancées après les tirages peut être fortement influencé par les conditions géologiques environnantes.

MESSUNGEN UND SIMULATIONEN DER BEZIEHUNG ZWISCHEN METHANGASFLUßRATEN IN STRECKENVORTRIEBEN UND DEN UMGEBENDEN GEOLOGISCHEN BEDINGUNGEN.

Kiyoshi Higuchi, Kotaro Ohga

Japanische Gruben führen Flöz - Hauptstreckenvortrieb hauptsächlich nach der Schießarbeitsmethode durch.

Zahlreiche Änderungen in der Methangaskonzentration im Streckenvortrieb nach dem Schießen wurden durch das zentrale Erfasssungssystem aufgezeichnet. Die Analyse der gesammelten Daten ermöglichte die Einteilung der Änderungen der Methangaskonzentration in einem bestimmten Zeitraum in fünf verschiedene Muster. Es ist denkbar, daß das Auftreten eines jeden Musters von der jeweiligen geologischen Struktur der Umgebung abhängig ist. Für die Diskussion wurden simulative, numerische Rechnungen nach einem mathematischen Modell durchgeführt, wobei geologische Bedingungen wie Störungen, Gasdruck und Gasgehalt im Flöz, Gasdurchlässigkeit des Flöz, usw. berücksichtigt wurden.

Die Ergebnisse der Rechnungen bestätigen, daß die Geologie der Umgebung den Methanfluß in die Vortriebe nach dem Schiessen stark beeinflußt.

ВЫЧИСЛЕНИЯ И МОДЕЛИРОВАНИЕ НА ОСНОВЕ СВЯЗИ МЕЖДУ КОЛИЧЕСТВОМ ВЫДЕЛЕНИЯ МЕТАНА В ЗАБОЕ И ОКРУЖАЮЩИМИ ГЕОЛОГИЧЕСКИМИ УСЛОВИЯМИ

Кийоши Хигучи и Котаро Ога

В японских угольных шахтах горизонтальная проходка нарезного штрека внутри пласта осуществляется в основном взрывными методами. Большое количество изменений в концентрации метана после взрывов было зарегистрировано централизованной системой наблюдения. В результате анализа собранных данных было установлено пять типов распределения метана. Можно представить, что характеристики каждого типа зависят от геологических условий вокруг штрека. Вычисления для машинного моделирования были выполнены с помощью математических моделей, которые включают количественные измерения таких геологических факторов, как разломы, давление газа и содержание газа в угольных пластах, газопроницаемость угольных пластов и т.д.

Результаты моделирования подтвердили, что количество выделения метана в забое после взрывов в большой мере зависит от окружающих геологических условий.

周围地质条件和甲烷进入巷道时流速之间关系的测定与模拟

日口清，大画小大郎 （均译音）

在日本，内层采区顺槽掘进多半是采用爆破方式。

爆破后巷道中发生大幅度的甲烷气浓度变化，这已由中央监测设备的记录证实了。根据所搜集资料的分析结果，甲烷浓度变化情况可分为五种模式。模式的状态大致与巷道周围的地质情况有关。为了便于讨论起见，模拟数值计算用的数学模型包括了断层，气压，煤层气体含量，煤层气体渗透率等因素。

从计算结果，我们证实了爆破后甲烷流入巷道的速率极受周围地质情况的影响。

TOWARDS A METHOD OF METHANE PREDICTION IN SOUTH AFRICAN COLLIERIES

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Chamber of Mines of South Africa Research Organization, South Africa

ABSTRACT

Methane is an unavoidable feature of coal mining and represents a potential explosion and fire hazard if not controlled. Normally, safe working levels can be maintained by conventional ventilation methods. To establish ventilation requirements for the dilution of methane, it is essential to determine how much gas is present in the seam and adjacent strata (gas content) and how easily it can enter the workings (permeability).

This paper presents methods for determining gas pressures in coal seams and adjacent strata (using inflatable borehole packers), assessing the sorption potentials of various South African coals and evaluating in situ permeabilities.

These parameters can then be used to calculate the seam gas content and also model methane emission, thus assisting effective ventilation planning.

INTRODUCTION

Methane is everpresent in coal mining and is a potential explosion and fire hazard. Safe working methane levels are normally maintained by conventional ventilation methods.

Ventilation requirements for adequate methane dilution require knowledge of the volume of methane present in the coal seam and surrounding strata (gas content), and how easily it will enter the workings (permeability).

The Coal Mining Laboratory (CML) of Chamber of Mines Research Organization (COMRO) is conducting ongoing research into these objectives, and is working towards the development of a 'user friendly' computer model aimed at assisting mine environmental officials with ventilation requirements.

Presented here are procedures and results in the identification of the seam gas content, seam gas pressure, and the seam permeability, using laboratory and in situ techniques.

SEAM GAS CONTENT

The identification of the seam gas content involved firstly the laboratory determination of the adsorption potential of coal and secondly, measurement of in situ seam gas pressures.

Adsorption Potential

Laboratory procedure

A detailed description of the procedure is given by Billenkamp (1988).

Crushed coal samples are placed in special containers, immersed in water, and all air evacuated from them. They are pressurised with methane for sufficient time to permit complete adsorption of the gas. The gas is then desorbed, the volume given off being recorded. All samples are evacuated again, and the procedure repeated using helium. Helium is not adsorbed by coal, hence the difference in volume of desorbed methane and desorbed helium represents the volume of adsorbed methane.

The methane adsorbed relates to the methane gas pressure by Langmuir's equation.

$$Q = \frac{K_1 \cdot K_1 \cdot P}{1 + [K_2 \cdot P]} \qquad (1)$$

where Q = volume of methane adsorbed (m^3t^{-1})

K_1 = constant, methane adsorbed at infinite pressure (m^3t^{-1})

K_2 = constant, (kPa^{-1})

P = gas pressure (kPa)

Twelve coal samples collected from the Gus Seam at a deep colliery in the South Eastern Transvaal were tested at a constant temperature and the results averaged to calculate the Langmuir constants.

Laboratory results

The results of the adsorption potential tests are given in Table 1. Each sample number represents an average of three identical tests.

Table 1: Langmuir constants for Gus Seam coal

Sample	Constant K_1 (m^3t^{-1})	Constant K_2 (kPa^{-1})
1	12,37	0,00077
2	14,75	0,00058
3	11,75	0,00075
4	11,02	0,00086
Mean	12,47	0,00070

Substitution of these constants into Langmuir's equation permits plotting of methane content against gas pressure for any constant temperature. The sorption potential of coal is temperature dependent.

The gas content at any pressure is then read from the resulting isotherm. Figure 1 gives the isotherm for 20 °C calculated from the mean values in Table 1. The 20 °C isotherm correlates well to the in situ temperature at the colliery of 23 °C.

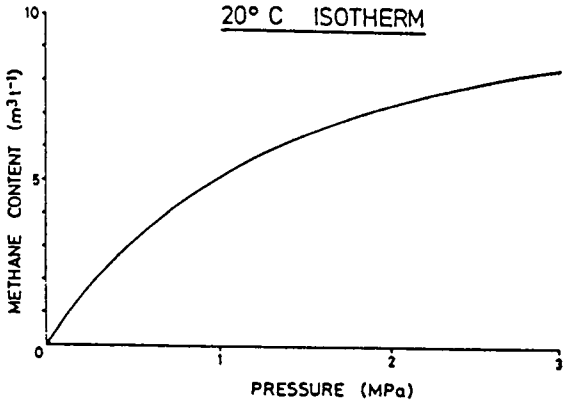


Figure 1 20 °C ISOTHERM

Seam Gas Pressure

Direct measurement of seam gas pressure requires boreholes drilled into the coal. The gas pressure is then allowed to build up in the borehole until it equals that of the gas within the seam. This involves isolating a section of the borehole, and for this, inflatable borehole packers were developed by CML. Figures 2 and 3 show sketches of single and double packers installed in 80 mm diameter boreholes. Extension rods are used to position the packers which are then inflated with inert nitrogen gas.

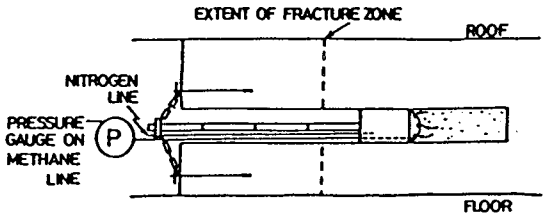


Figure 2 SINGLE PACKER

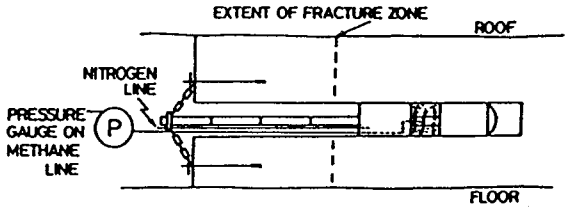


Figure 3 DOUBLE PACKER

Single packers permit pressure build up over a length of borehole. Double packers record pressure at a known position in a borehole. To avoid excessive leakage of methane around the boreholes, the packers are positioned beyond the coal fracture zone, as indicated in the previous Figures 2 and 3.

In situ procedure

Two 80 mm boreholes were drilled 20 metres apart, to a depth of 30 metres into virgin coal. A single packer was installed in one borehole and a double in the second.

The purpose of the double packer was to determine the extent of the fracture zone, and to this end, it was

advanced one extension rod length further after each pressure reading.

The single packer was positioned further into the borehole, beyond the estimated extent of the fracture zone, to determine the virgin seam gas pressure.

In situ results

Results obtained from the seam gas pressure tests were limited by the time required for the methane pressure to equalise. Approximately 5 days were needed for each test before stable methane pressures were noted. This was considerably longer than the estimated 1 - 2 days, and was an indication of the low in situ permeability results to be obtained later.

The high virgin methane pressures resulted in packers being ejected from the boreholes and damaged. This was potentially dangerous and was resolved by increasing the inflation pressure from 2 MPa to 3 MPa, and securing the packer rods to the sidewall by roofbolts, clamps and chains, as seen in Figures 2 and 3.

Results from the single packer are given in Table 2, and for the double packer in Table 3.

Table 2: Seam gas pressures

Packer Depth (m)	Methane Pressure (kPa)
8	1 600
10	1 200
12	1 250

Table 3: Fracture zone gas pressures

Packer Depth (m)	Methane Pressure (kPa)
2	650
4	900

The initial high methane pressure, 1 600 kPa, was never reproduced, possibly due to degassing of the seam with time, or to localised borehole fracturing, in the vicinity of the packer, thus permitting leakage. Calculation of the in-seam gas content however was made using 1 600 kPa, being the virgin pressure.

The true extent of the fracture zone was not determined due to the delay previously mentioned. However after the initial high virgin pressure of 1 600 kPa, those recorded beyond 8 m depth were reasonably constant at around 1 200 kPa. This indicated that the extent of the fracture zone had been passed, and its limit was therefore considered to be 8 m.

In Situ Seam Gas Content

Substituting the virgin gas pressure, 1 600 kPa, into the Langmuir equation,

with the obtained values for K_1 and K_2 gives the results in Table 4.

Table 4: Seam gas content

Sample No.	Constant K_1 (m^3t^{-1})	Constant K_2 (kPa^{-1})	Gas Content (m^3t^{-1})
1	12,37	0,00077	6,8
2	14,75	0,00058	7,1
3	11,75	0,00075	6,4
4	11,02	0,00086	6,4
Mean	12,47	0,00070	6,6

It is seen therefore that the in situ methane content ranges between 6,4 m^3t^{-1} and 7,1 m^3t^{-1} with the mean result of 6,6 m^3t^{-1}.
This is also shown on the 20 °C isotherm in Figure 4.

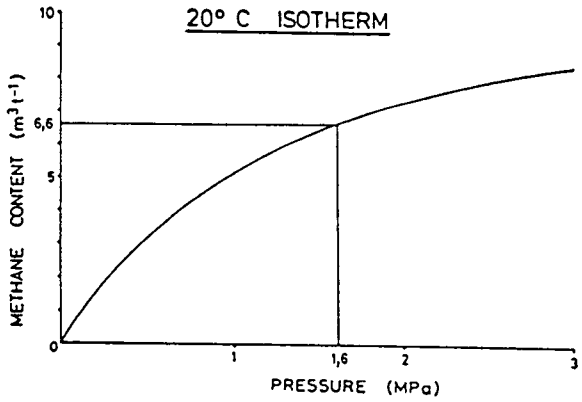


Figure 4. 20 °C ISOTHERM INDICATING SEAM GAS CONTENT

COAL SEAM PERMEABILITY

Laboratory Permeability

Laboratory procedure

A detailed description of the procedure is given by Billenkamp (1988).

Core samples are taken from the coal, both parallel and perpendicular to the bedding plane. A core is placed in a triaxial loading cell and gas passed through it. A range of horizontal and vertical stresses are applied, as well as various supply gas pressures, and the gas flow rate measured during loading and unloading cycles.

In all, 14 samples were tested and the results averaged in three categories:

(i) constant vertical and horizontal loading, varied gas pressure

(ii) constant horizontal loading and gas pressure, varied vertical loading

(iii) constant vertical loading and gas pressure, varied horizontal loading.

Laboratory results

Laboratory permeability results are presented in Table 5, where it is seen they range from 0,974 μDarcy to 9,301 μDarcy. Individual results ranged from 0,5 μDarcy to 25 μDarcy.

In Situ Permeability

Laboratory permeability experiments cannot accurately simulate in situ conditions. This is due partly to the effects of destressing and stressing and the difficulty in accurately measuring the horizontal stress component of the strata. The

Table 5: Laboratory permeability

Sample	Test Conditions					Permeability
	Horizontal	Vertical	Gas Pressure	Bed Plane	Cycle	(μDarcy)
1	C	C	V	Per	L	2,694
2	C	V	C	Per	L	4,113
3	C	V	C	Per	UL	2,218
4	C	V	C	Par	L	1,831
5	C	V	C	Par	UL	0,974
6	V	C	C	Per	L	1,540
7	V	C	C	Par	L	9,301

laboratory results indicate expected permeabilities within the range 1-10 μDarcy, but possibly extending from 0,5-25 μDarcy. To determine a precise permeability value in situ measurements are required.

A computer model developed by CML calculates theoretical pressure gradients for conditions corresponding to those recorded in situ. Actual pressure gradients are compared against theoretical predictions, calculated from varied permeabilities, to determine which permeability best describes the rate of gas migration through the coal.

The model considers radial gas flow towards an axial sink, which is used as the basis for three different computer programs.

(i) A simple solution of the above system with all variables (including permeability) input by the operator. Graph plotting facility to see pressure decay as a function of radius for each time period.

(ii) An iterative permeability search. Permeability values are input by the operator and pressure gradients calculated and stored for each value. The theoretical pressure gradients are compared to the experimentally determined gradients (which are required as input for the program) and a percentage error is calculated. It is possible to plot any experimental and theoretical gradients on the same set of axes for comparative purposes.

(iii) An automatic permeability search. Two permeability values are input. The program automatically searches between these values for the permeability value which gives pressure gradients of a specified percentage accuracy compared to the input experimental gradients. It is possible to plot any experimental and theoretical gradients on the same set of axes for comparative purposes.

In situ procedure

Seven 80 mm diameter boreholes were drilled, and inflatable packers installed, as indicated in Figure 5. A packer initially installed to plug the bleeder hole or sink, at Borehole No. 6, was removed once the methane pressure had stabilized.

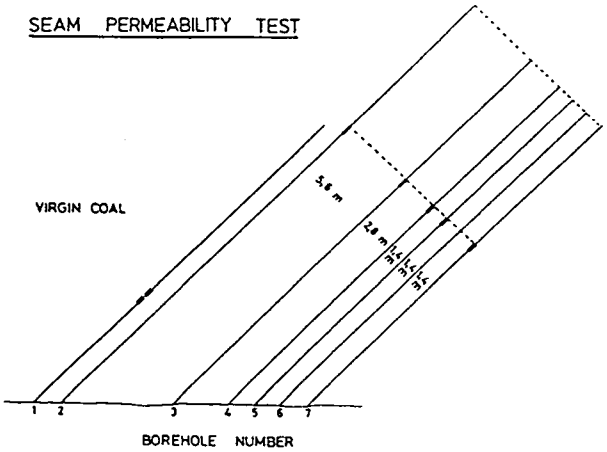


Figure 5 POSITIONS OF PERMEABILITY TEST BOREHOLES

After removal of the bleeder hole packer, pressure decay in the remaining boreholes was recorded with time.

Borehole No 7 served only to confirm radial symmetry and was not used in the calculations.

In situ permeability results

Pressure gradients recorded over an 18 day period are given in Figure 6. Borehole No 2, at 16 metres from the sink recorded 250 kPa throughout the test and has therefore not been considered in the calculations. However, it may have had an effect on the results, particularly Borehole No 3, at 5,6 m from the sink, causing methane flow away from the sink, as well as towards it. This explains the apparant drop in pressure from the expected curve at 11 and 18 days for this borehole, also due to the effect of the sidewall acting as a sink, increasing with both distance from the bleeder hole and time.

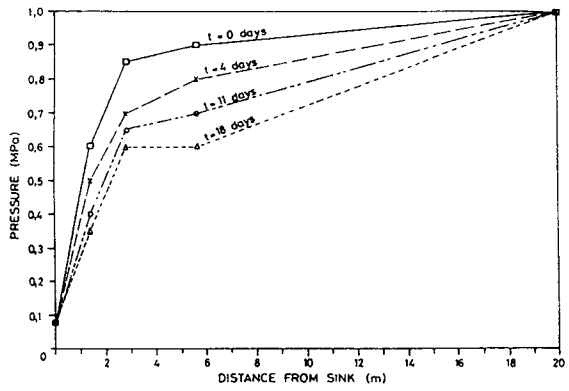


Figure 6 EXPERIMENTAL PRESSURE GRADIENTS

Because of this effect, only pressures at Boreholes 4,5 and 6 were considered for the calculations.

Figure 7 shows results for time periods 100 and 260 hours, with the calculated optimum superimposed to 5,6 m. The calculated curves correspond to an optimum permeability of 175 μDarcy at 3,8 % error to Borehole No 4. It is seen that the error increases greatly beyond Borehole No. 4, due to the sidewall sink, and possible low pressure in Borehole No 2, causing the drop in recorded experimental pressures below the calculated curves.

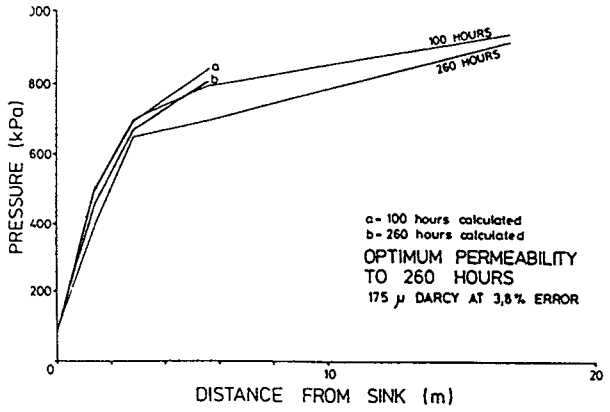


Figure 7 **OPTIMUM PERMEABILITY TO 260 HOURS**

Figure 8 shows results for the time period up to 400 hours, with the calculated optimum superimposed to 5,6 m. The calculated curve correspond to an optimum permeability of 250 μDarcy, with an error of 7 % to Borehole No 4. It is seen that by 400 hours the calculated curve is further removed from the experimental curve. This indicates the effect of increased time on both the sidewall sink and leakages.

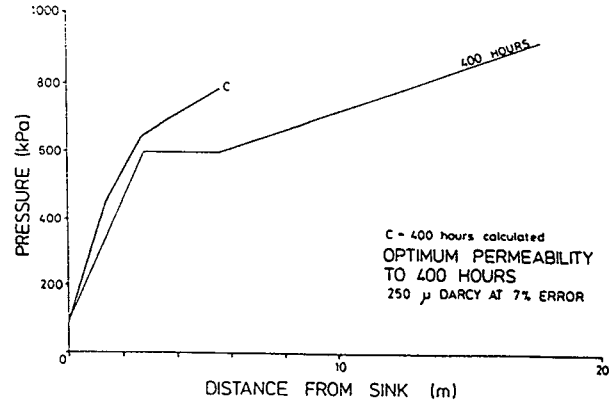


Figure 8 **OPTIMUM PERMEABILITY TO 400 HOURS**

These in situ permeability results are a factor of 10 to 10^2 greater than laboratory values. This difference appears very significant until compared with permeability values of approximately 100 Darcy, Lama (1983), or 10^6 greater than those calculated for this colliery. The tested Gus seam is therefore a very low permeability coal, with the difference in the results due to uncontrolled in situ factors. Several of these combine to reduce the recorded borehole pressures, viz, fracturing within the coal, leaking around the packers and the sink effect of the sidewall. This is supported by the increase in calculation error over the greater time period, as these leakages have become a more significant proportion of the total.

CONCLUSION

The in situ methane pressure for the Gus seam in the South Eastern Transvaal is 1 600 KPa. At this pressure the coal contains approximately 7 m^3t^{-1} of methane.

Permeability of the coal is low, laboratory results ranging from 0,9 to 9,3 μDarcy and the in situ measurement being 10-100 times this, calculated at 175 μDarcy and 250 μDarcy.

The error in the optimum in situ permeability calculations increased with time. To reduce the effect of

leakages and the sidewall sink future work will be restricted to a minimum time, which depends on the accuracy of the pressure recording equipment, and the rate of pressure decay.

The identification of the seam gas pressure, gas content, and seam permeability is required for the further prediction of methane emission rates and the potential for methane drainage. As results are limited to one area of the Gus seam, the work must now be extended within this seam, as permeability may not be consistent throughout it. Adjacent strata and coal seams are also potential methane sources and must be considered. Combined with monitoring of methane build up rates these results are used to calculate ventilation requirements.

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ACKNOWLEDGEMENT

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VERS UNE METHODE DE CALCUL DU METHANE DANS LES HOUILLERES SUD-AFRICAINES

P. Sullivan, A. Cook, G. Stripp

Le méthane est un élément inévitable de l'exploitation du charbon. Non contrôlé, il constitue un risque d'explosion et d'incendie potentiel. Les méthodes de ventilation conventionnelles permettent habituellement de maintenir des conditions de travail sûres. Pour déterminer la ventilation nécessaire à la dilution du méthane, il est crucial d'établir la quantité de gaz présente dans le filon et les strates voisines (teneur en gaz), et de déterminer si ces gaz peuvent pénétrer facilement les chantiers (perméabilité).

Ce rapport présente des méthodes permettant de calculer les pressions de gaz dans les couches de charbon et les strates avoisinantes (au moyen de remblais de trous de mines gonflables), d'évaluer le potentiel de sorption de divers charbons sud-africains ainsi que les perméabilités in situ.

Ces paramètres servent ensuite à calculer la teneur en gaz des filons et à effectuer des simulations d'émissions de méthane; ceci aide à une planification effective de la ventilation.

METHODE ZUR VORHERSAGE VON METHANGASVORKOMMEN IN SÜDAFRIKANISCHEN GRUBEN.

P. Sullivan, A. Cook

Methangas ist eine unvermeidliche Gegebenheit im Bergbau und stellt eine potentielle Gefahr für Explosion und Feuer dar, wenn es nicht unter Kontrolle gehalten wird. Im Normalfall können sichere Arbeitsbedingungen durch konventionelle Bewetterung aufrechterhalten werden. Der Wetterbedarf zur Verdünnung von Methangas ist abhängig vom Gasgehalt im Flöz und im umgebenden Gebirge (Gasgehalt) und von der Fähigkeit, in den Vortrieb einzudringen (Permeabilität).

Der Vortrag behandelt Methoden zur Bestimmung des Gasdruckes in Kohleflözen und dem umgebenden Gebirge durch den Gebrauch von aufblasbaren Bohrloch-Verschlüssen, zur Bestimmung der Aufnahmefähigkeit von verschiedenen südafrikanischen Kohlesorten, sowie zur Bewertung der Durchlässigkeit vor Ort. Diese Parameter werden dann in der Berechnung des Gasgehaltes in Kohle-Flözen und in der Ausgasung von Methan angewendet, wodurch eine effektive Bewetterungsplanung ermöglicht wird.

О РАЗРАБОТКЕ МЕТОДА ПРЕДСКАЗАНИЯ УРОВНЯ МЕТАНА В ЮЖНО-АФРИКАНСКИХ УГОЛЬНЫХ ШАХТАХ

П. Салливан, А. Кук и Ж. Стрип

Метан неизбежно присутствует в угольных шахтах и, если его не контролировать, может послужить причиной взрывов и пожаров. В большинстве случаев допустимый уровень содержания метана может поддерживаться традиционными вентиляционными методами. Для установления норм безопасной концентрации метана необходимо определить количество газа в угольном пласте и в прилегающих структурах (содержание метана), а также легкость, с которой газ может проникнуть в выработку (проницаемость). Настоящий доклад предлагает методы определения давления газа в угольных пластах и прилегающих структурах (используя надувные пакеры в скважинах), методы оценки сорбционной емкости различных видов южно-африканского угля и методы оценки проницаемостив коренном залегании. Эти параметры могут быть использованы для вычисления содержания газа в пласте, а также для моделирования метановой эмиссии, таким образом содействуя эффективному планированию вентиляционных систем.

南非煤矿用的一种甲烷预测方法

P．沙利文 A． 柯克 G． 史特立普

甲烷是煤矿开采中不可避免的产物，若无控制措施其可导致爆炸和火灾事故．在正常情况下，普通通风方法就能将甲烷含量维持在安全工作量之下．为了制定冲淡甲烷通风需求计划，首先必须测定煤层和邻近层段内的煤气含量以及其能渗入工作地段的可能性（渗透率）．

本文叙述了测定煤层及邻近层段气压的方法（用膨胀的钻孔式充填机），估评南非煤的吸附能力，以估计它就地的渗透率．

这些参数可以用来计算煤层的含气量及标准甲烷散发量，以便用于辅助有效的通风设计．

PREDICTION AND VERIFICATION OF METHANE EMISSION IN COAL MINES

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Huainan Coal Mining Admisistration, PRC

Abstract

Different methods of prediction of methane emission are analysed and verified on the basis of various factors affecting CH_4 emission, such as CH_4 content in coal seams, geological conditions, distribution of work faces and mining technology. This article also describes the applicable conditions for statistics method and separate calculation method and the ways in which reliability of prediction is expected to be raised.

1. Introduction

Prediction of methane emission in coal mines and its distribution provides a basis for design of ventilation and prevention of CH_4 — related disasters. Its reliability has a direct bearing on the rationality of mine layout and ventilation design, especially in high methane mines, where reliability of prediction is of vital importance.

At present, a number of methods for prediction of CH_4 emission have been put forward in many countries, but most of them are more or less limited to the local experiences. So far as Huainan mine area is concerned, statistics method was adopted in the past. With the popularization of direct determination of CH_4 content in geological exploration, several methods of calculation have been introduced in design of new mines. As prediction methods are different, the results sometimes vary greatly. Therefore, it is necessary to verify the prediction methods in accordance with productive conditions so as to select the appropriate method for Huainan area.

Verification means a comparison between actual emission Qs and predicted emission Qy.

If the relative error $\frac{Qs - Qy}{Qy} \leqslant \pm 20\%$, the prediction method proves to be feasible.

2. Prediction Method

Following methods have been employed in Huainan area, mine statistics method, content coefficient method, integrated calculation method and separate calculation method Here a brief account will be given to mine statistics method and separate calculation method which are more commonly used.

2.1 Mine Statistics Method

This method requires to make mathematical figures or to use mathematical statistics on the basis of measured specific methane emission from different levels of the mine, then to seek out functional relations between the mining depth and the emission quantities of the mine and various mining districts and work faces respectively. Normally this can be expressed by the following linear equation,

$$Q_H = aH + b \quad (m^3/t)$$

where, Q_H - specific methane emission (m^3/t);
H - average mining depth of mine or work faces (m);
a, b, - constant

2.2 Separate Calculation Method

1. Methane content in coal seams measured in geo-exploration and production is to be corrected and screened. Correction coefficients are 1.2 - 1.3 for collection method, 1.1-1.2 for analytic method and 1.0 for indirect measurement.

2. Isopleth maps are to be drawn for CH_4 content of separate seams.

3. The methane content of work faces which are being worked at the same time is to be calculated out on the isopleth map by insertion method according to distrtibution layouts and succession charts of mining districts and work faces in the design. Then, considering mining method and production technology, the methane emission quantity of each work facem of coal mining as well as heading, is to be predicted respectively. On this basis, the total CH_4 emission of each mining district and the whole mine is readily worked out. The formula for calculation and its procedure are shown in Fig. 1.

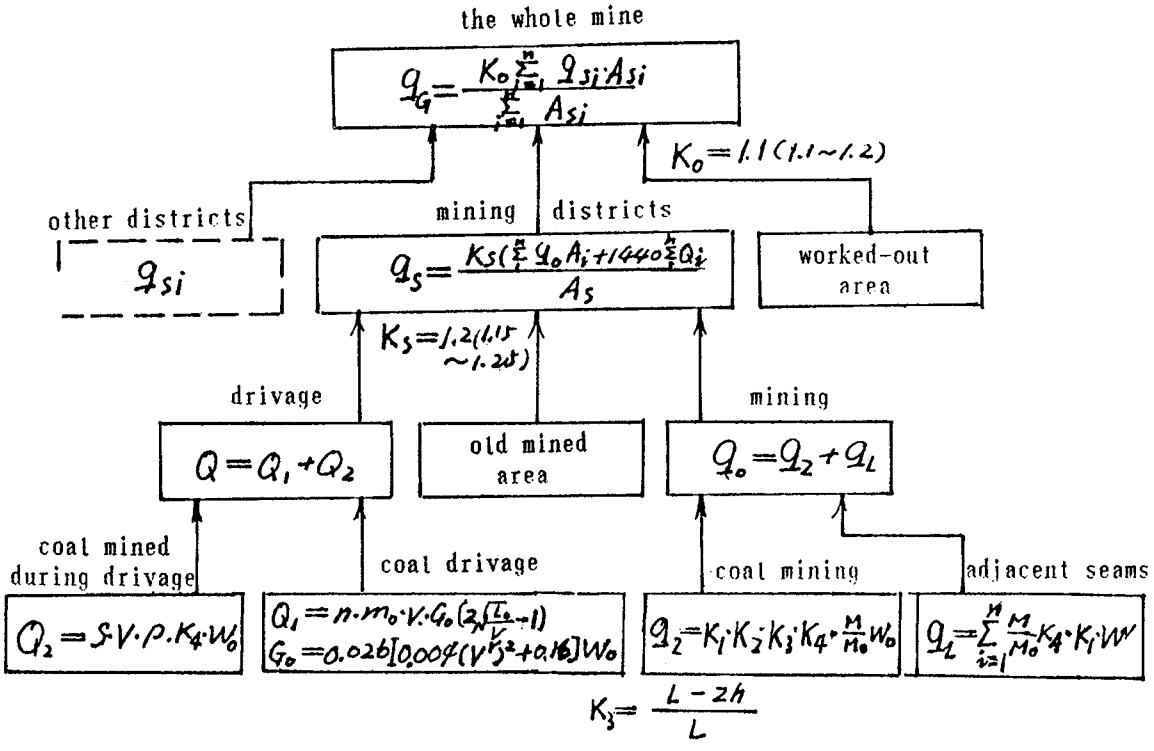


Fig. 1 Block Diagram for Calculation of Mine Methane Emission

Legent

q_G, specific CH4 emission of the mine (m3/t)

q_S, specific CH4 emission of the mining districts (m3/t)

q_o, specific CH4 emission of the work faces (m3/t)

q_Z, specific CH4 emission for coat mining (m3/t)

q_L, specific CH4 emission of adjacent seams (m3/t)

Q , CH4 emission in drivage (m3/min)

Q1, CH4 emission in coal drivage (m3/min)

Q2, CH4 emission of coal mined during drivage (m3/min)

G2, CH4 emission intensity (m3/m2.min)

Ko, CH4 emission coefficient for worked-out area of the whole mine

Ks, CH4 emission coefficient for old mined area of mining districts

K1, CH4 emission coefficient for rock K1 = 1.2

K2, coefficient of mining loss at work-face K2 = 1.11

K3, pre-drainage coefficient in drivage

K4, analytical CH4 coefficient K4 = 0.8

Ki, CH4 drainage coefficient in adjacent seams, taken from the curve.

M , coal seam thickness (m)

Mo, mining height (m)

As, average daily output of mining district (t)

Ai, average daily output of workface (t)

S , cross-section of coal drivage (m2)

V , drivage advance rate (m/min)

P , coal density (t/m3)

n , number of exposed coal walls

LO, roalway length (m)

WO, CH4 content in coal seam (m3/t)

W1, CH4 content in adjacent seam (m3/t)

V7, volatile-matter content in coal (%)

L , length of work face (m)

h: width of drainage zone in roadway h = 18

3. Prediction and Verification

(1) As for mine statistics method, let us take Xie'er Mine in Huainan as an example.

[a] General situatiion. There are 15 workable coal seams in Xie'er Mine, with a total thickness of 32.2m and a dip of 23°. The main workable seams, in descending order, are No. 13, No.11, No.9, No.8, No7, No.4, six seams in all. These seams constitute 65% of the total thickness. All fifteen seams are divided into two groups, the first group being No.15 - No.10; the second group being No.9 - No.1. Before 1970, only seams above No.10 were worked.

[b] Method. Methane content varies with each seam. No.13 and No.11 seams contain highest methane content with shallow gas weathered zones, while seams below No.9 contain relatively low CH4 content with deep gas weathered zones. Along the strike, CH4 contents between various faults vary, and along the dip, CH4 contents increase with the depth. The structure of No.13 and No.11 seams and CH4 con tent distribution are shown in Fig. 2.

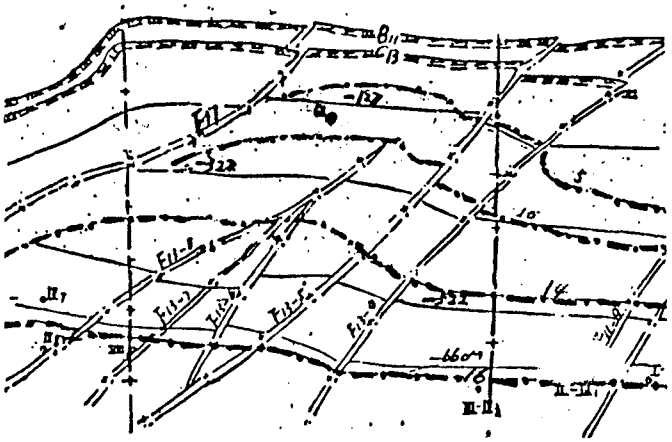


Fig. 2 The Structure of No. 13 and No. 11 seams and CH4 Content Distribution

coal seam contour ————
CH4 content isopleth —·—·—
boundary between mines +—+—

[c] The relationship between mine CH4 emission and mining depth in top slice work faces of No. 13 seam and high-gas districts is shown in Table 1, Table 2, and Fig. 3 . No. 13 seam is intercepted by faults along strike into three sections with different gas emission, F17 upper Q_M<5m3/t; F17 lower — F13-8, $Q_M = 0.055\ H + 0.32$, coefficient of correlation, $r = 0.985$; F13-8 — F13-4, $Q_M = 0.052\ H + 2.40$, $r = 0.973$.

For high gas mining districts, $Q_C = 0.026\ H + 16.5$, $r = 0.963$

For the mine, 1966-1969 $Q_G = 0.069\ H + 1.79$, $r = 0.977$

1970-1987 $Q_C = 0.029\ H + 9.52$, $r = 0.976$

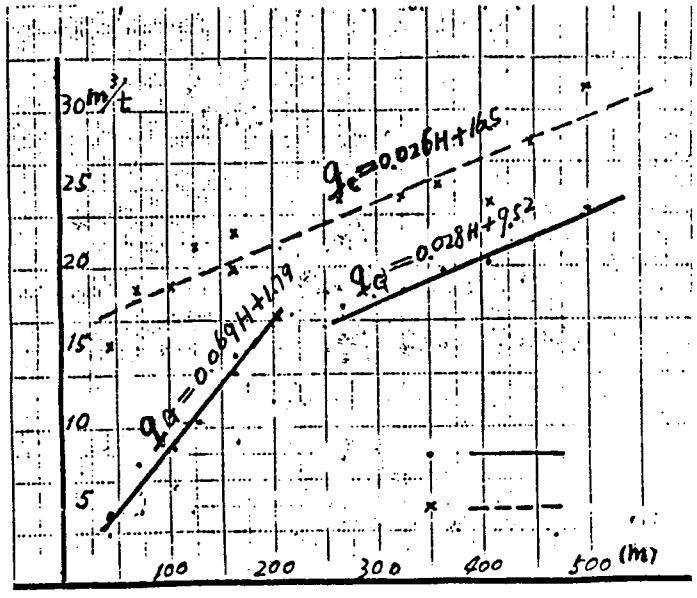
Table 1 CH4 Emission of Different Mining Depth for Work Faces at Top Slice of No. 13 Seam of Xie'er Mine

F17lower — F13-8				F13-8 — F13-4			
elevation of sub-level (m)	average elevat. of faces (m)	CH4 content (m3/t)	CH4 emission (m3/t)	elevation of sub-level (m)	average elevat. of faces (m)	CH4 content (m3/t)	CH4 emission (m3/t)
-62	-44	1.21	2.4	-62	-44	2.05	4.77
-88	-75	2.31	4.78	-127	-105	2.86	8.94
-127	-105	3.14	6.25	-162	-144	4.39	10.68
-162	-145	4.16	8.84	-202	-175	5.60	11.87
-200	-178	6.12	12.7	-242	-230	7.87	15.50
-252	-236	7.31	17.33	-282	-260	9.12	19.04
-322	-296	8.31	18.37	-322	-296	10.15	22.08
-365	-336	8.89	19.43	-365	-336	10.72	19.42
-400	-377	9.43	22.87	-400	-386	11.23	22.18
-440	-418	9.92	23.37	-450	-432	11.67	24.89
-470	-452	10.49	24.88	-500	-472	12.16	28.36

Table 2 Statistics of Mining Depth and CH_4 Emission in Xie'er Mine and High CH_4 Districts

year	No.13&No.11 average depth of faces (m)	NO.13&NO.11 CH_4 content related to mining depth(m^3/t)	average daily output (t/d)	mine	high methane mining districts		
				CH_4 emission (m^3/t)	output percentage(%)	percentage in mine CH_4 emission %	CH_4 emis. of mining districts (%)
1957~1958	45	1.56	2185	4.29	35.58	85.00	15.17
1959~1960	74	2.19	4262	7.49	31.86	85.30	18.81
1961~1962	105	3.44	3883	9.21	32.50	75.80	19.13
1963~1964	135	4.66	3666	10.31	44.30	72.97	21.93
1965~1966	167	5.42	3246	14.44	47.23	54.60	22.40
1967~1969	208	6.46	3074	15.44	76.80	50.70	22.74
1970~1972	266	7.69	2883	17.13	64.00	82.20	24.76
1973~1975	326	8.73	2957	18.66	60.10	81.90	24.80
1976~1978	366	9.33	3166	19.73	57.10	80.90	25.04
1979~1981	407	9.87	3234	20.13	60.60	80.60	24.11
1982~1984	438	10.36	3462	22.07	64.70	84.55	28.07
1985~1987	475	10.91	3587	23.78	49.50	80.10	31.46

Fig. 3



[d] The Source of Mine Methan

Statistics over past years from four high methane mines in Huainan show,

Coal mining accounts for 35.4% to 45.1% of the total emission, with an average of 40%.

Drivage of 10000 tons coal roadway (106-114m) accounts for 39.2% to 19.4%, with an average of 29%.

Roadways in operation and worked-out area make up 39.2% to 19.4%, with an average of 31%, of which worked-out areas make up 20% on the average.

[e] Statistics show that methan emission of work faces and high-gas districts has good linear functional relation with mining depth, but the rate of progressive increase of CH_4 emission in work faces at different foult-divided sections

varies. Before 1969, CH_4 emission increased rapidly with the increase of the coal output and mining depth for high methane districts. After the readjustment of mine boundaries in 1970, with the increase of coal output from the low methane seams below NO.9 seam, the rate of progressive increase apparently declined and has kept pace with the rate of progressive increase of high methane mining districts. Fig. 3 shows that after the re-distribution of work faces, the relationship of progressive increase $Q_G = 0.069\ H + 1.19$ is no longer appropriate to predict the methane emission of the extended level. Otherwise, extending from -200m to -340m level, the relative error would be, $\frac{19.04 - 25.25}{25.25} = -0.25 > -20\ \%$.

(2) As for separete calculation method, let us take No.1 Panji Mine for example.

[a] General Situation. No.1 Panji Mine is a newly-built mine which went into operation a few years ago. The length of the mine field along strike is 6.8 km. The number and total thickness of workable coal seams are the same with Xie'er Mine, with a dip of 9° on the average. On both wings of the mine, there is a diagonal fault with a throw of 55 - 110m, dividing No.13 and No.11 seams into two sections with different CH_4 content. The east wing contains higher CH_4 and involves the risk of coal and methane outburst. The relation between CH_4 content and the bedrock thickness of coal-bearing strata (Hj)is,

$$Wo = 0.07\ Hj \div (1 + 0.0015\ Hj)\ [\ m^3/t\ (CM)\ ]$$

Whereas the west wing contains lower CH_4,

$$Wo = 0.039\ Hj - 379\ [\ m^3/t\ (CM)\ ]$$

[b] Prediction of Methane Emission

Three methods yield three different results. Integrated calculation method was adopted for mine design. On the basis of directly measured maximum CH_4 content for No.13 and No.11 seams, the maximum methane emission at the first level (-530m) was predicted at 10.59 m^3/t. Using content coefficient method, mine methane emission was predicted at 31.18 m^3/t. Separate calculation method produced the predicted emission at 24.79 m^3/t, with 31.17 m^3/t at three mining districts east of the fault and 10.31 m^3/t at two districts west of the fault.

Table 3 Comparison between Actual CH_4 Emission and Predicted Emission by Separate Calculation Method in No.1 Panji Mine

year	mine CH_4 emission (m^3/t)				east of the fault				west of the fault			
	avera. elevation(m)	actual (q_s)	calculated (q_y)	$\frac{q_s}{q_y}$	avera. elevation(m)	actual (q_s)	calculated (q_y)	$\frac{q_s}{q_y}$	avera. elevation(m)	actual (q_s)	calculated (q_y)	$\frac{q_s}{q_y}$
1986	-428.3	12.73	12.57	1.01	-442.3	17.72	20.68	0.86	-410	5.63	2.57	2.19
1987	-448.3	12.61	13.37	0.94	-464.7	18.86	22.66	0.84	-431	4.85	3.86	1.26

[c] Verification. Comparison was made between actual emission and predicted emission by separate calculation method.

(1) A comparison between actual mine CH_4 emission and predicted emission by separate calculation is shown in Table 3.
(2) A comparison between actual emission at work faces east of the fault and calculated emission is shown in Table 4.

Table 4 Comparison between Actual CH_4 Emission and Calculated Emission at Work Faces of No.1 Panji Mine

work face of mining				work face of drivage			
location	emission m^3		q_s/q_y	location	emission m^3/min		q_s/q_y
	calculated	actual			calculated	actual	
1111(3)	8.92	9.27	1.04	1211(3) lower	1.67	1.47	0.88
1112(3)	8.77	10.42	1.19	1221(3) lower	1.73	2.81	1.62
1122(3)	15.26	16.81	1.10	12 raise	2.63	2.77	1.05
1211(3)	6.01	5.34	0.89	1112(3) lower	1.69	2.14	1.27
1212(3)	7.02	7.18	1.02	1122(3) upper	1.95	1.7	0.87
1221(3)	9.57	9.03	0.94	1111(3) lower	1.88	1.68	0.89
1132(3)	15.43	16.55	0.93	1121(3) lower	2.31	2.38	1.03
1211(1)	3.55	4.33	1.22	11 raise	3.01	3.63	1.21
1361(3)	15.55	14.86	0.96	1361(3) lower	4.52	5.06	1.20
				1361(3) upper	4.35	4.12	0.95
average			1.03	average			1.10

(3) A comparison between actual emission of the mine and mining districts and calculated emission is shown in Table 5.

Table 5 Comparison between Actual CH_4 Emission and Calculated Emission of Mining District and The Whole Mine

mining district	calculated emission (m^3/min)			actual emission	
	total at faces of mining & drivage	mining district		actual emission (m^3/min)	emission coefficient
		emission coefficient	emission		
16.11采区	14.81	1.2	17.771	17.11	1.15
16.12采区	5.01	1.2	6.01	7.14	1.18
16.13采区	17.61	1.2	21.13	17.91	1.02
16.14采区	4.16	1.2	4.99	5.63	1.25
whole mine	41.58	1.10	54.89	45.47	1.00

(4) Sourse composition of mine methane.

Coal mining accounts for 47.0-55.9% of the total emission with an average of 52.6%. Drivage for 10000 tons coal roadway (45-55m) accounts for 34.1-37% with an average of 35.6%. Others account for 18.9-7.1% with an average of 11.8%.

It can be seen from above verification that the predicted emission by integrated calculation method is on the low side; by content coefficient method, the calculated CH_4 emission east of the fault on the basis of content is near the actual emission, whereas the prediction for the whole mine is on the high side; predicted emission by separate calculation method for the mine, mining districts and work faces east of the fault is within the limits of allowable errors compared with actual emission, except that the work faces west of the fault are near the CH_4 weathered zone and therefore their prediction error is comparatively greater. With the expansion of worked-out area and the rise of drivage rate to 80m as required by calculation, the errors east of the fault will be further reduced.

4. Conclusion

It follows from the above verification that, by any prediction method, various factors must be taken into consideration, namely, methane content in coal seams, the occurrences, the geological structure, distribution of work faces and productive technology. Of the above two methods, statistics method has the advantage of synthesizing various factors and is easy to take sample values over a wide range. Furthermore, it makes analysis and screening of geological, mining and ventilation factors and then makes out a general plan of mine methane geology. On this plan, the lower limit line of the methane weathered zone of the mine and the isopleth of methane emission for the mine, the mining districts and the work faces of mining and heading are to be drawn respectively. This not only reflects the variation of parameters for CH_4 emission, compensates for the Q - H relationship, but also helps to find out practical coefficient required by other prediction methods. This method still has considerable value for prediction in operational mines, especially during the stage of underground development and extension.

As for separate calculation method, it takes various factors affecting methane emission into consideration and therefore, this method can be utilized and popularized as the main prediction method for design of new mines. However, in order to raise the reliability of prediction of separate calculation, two aspects should be pointed out. First, a certain number of CH_4 content measurement points should be set up in main coal seams along strike and dip in accordance with different geological structure during geo-exploration preiod so as to grasp the variation of CH_4 content in detail. Secondly, some coefficients need to be corrected in the light of statistics of adjacent mines.

Reference ,

[1] К.З. Ушаков: справочник по рудничной вентиляции. Недра. 1977.

矿井沼气涌出量的予测及其验证

中国淮南矿务局通风处　　刘秉钧

内容提要：本文根据影响沼气涌出量的煤层沼气含量、地质条件、采场分布、生产工艺等因素，对不同的方法进行分析验证，阐述了矿山统计法和分源计算法的使用条件和提高予测工作可靠性的途径。

一、引言

矿井沼气涌出量及其分布的予测是通风设计，制订防止沼气灾害事故的基础。其可靠性直接关系到开采布局，通风设计是否合理；关系到矿井的安全可靠程度和经济效益，尤其是高沼气矿井，在某些程度上决定着设计的成败。

目前，各国提出的矿井沼气涌出量予测方法已有多种，但多属于半经验性质，通用性差，就淮南矿区而言，过去是采用统计法，随着地质勘探中直接测定沼气含量方法的推广，在新井设计中曾采用多种计算法，但由于予测方法不同，予测结果有时相差很大，因此就有必要根据生产条件对予测方法进行验证，以决定采用适合淮南矿区的予测方法。

验证，是以实际涌出量q_{sc}与予测涌出量q_{yc}进行比较，若其相对误差

$$\frac{q_{sc}-q_{yc}}{q_{yc}}\leqslant\pm 20\%$$

，就可以满足工程要求，则这种方法便可采用。

二、予测方法

淮南矿区曾采用予测方法有：矿山统计法、含量系数法、整体计算法、分源计算法等。现就使用比较普遍的矿山统计法和分源计算法介绍于下：

㈠矿山统计法：是根据矿山不同生产水平的实测相对沼气涌出量，用作图或数理统计，分别找出矿井及采区、回采工作面的涌出量与开采深度的函数关系。通常可用线性方程表示：

即　　$q_H=aH+b$　（m^3/t）；

式中：　q_H——相对沼气涌出量　（m^3/t）；

H——矿井或采区的回采工作面的平均采深　　（m）；

a，b——常数。

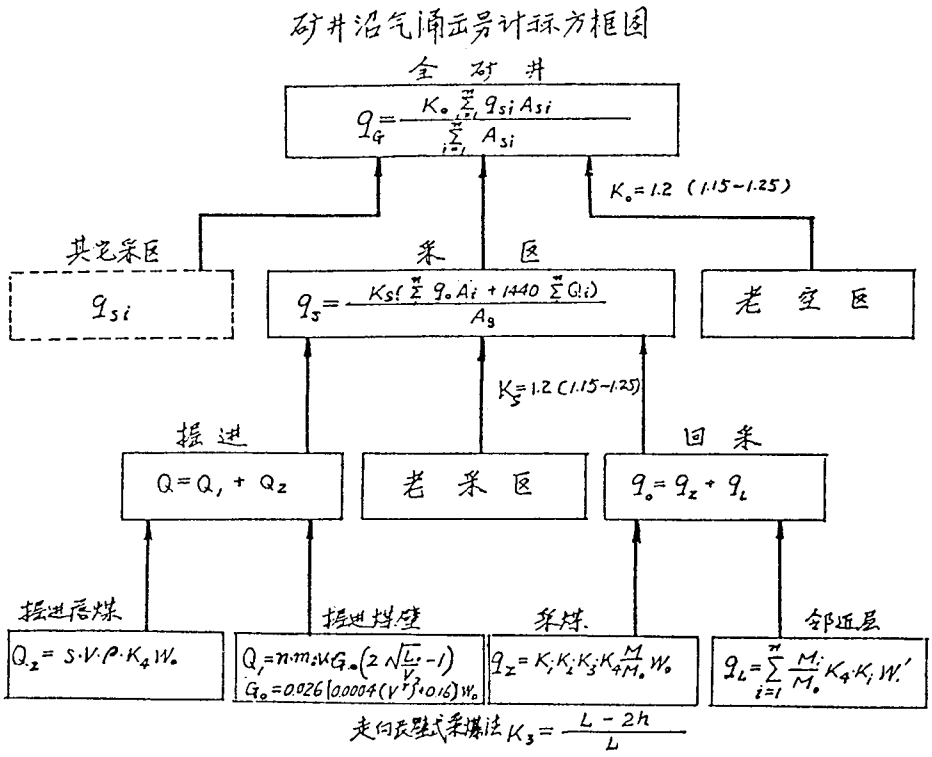
㈡分源计算法

1．根据地质勘探和生产中测得的煤层沼气含量进行校正、筛选。其校正系数：集气式为1．2～1．3，解析法为1．1～1．2，间接测定法为1．0。

2．编制分层沼气含量等值线图。

3．根据设计的采区，采掘工作面分布图和接替表，在含量等值线图上用插值法求出同时开采的各采掘工作的沼气含量，结合采煤方法，生产工艺等分别计算各回采工作面，掘进工作面的沼气涌出量，汇总为采区和矿井沼气涌出量，其计算公

式和过程见图一。(1)



图一

符号意义：

q_G：矿井相对沼气涌出量（m³／t）

qo：回采面相对沼气涌出量（m³／t）

Q：掘进巷道沼气涌出量（m³／min）

Go：煤壁沼气涌出强度（m³／m²·min）

Ks：采区采空区沼气涌出系数

K2：工作面采煤损失系数，取K2＝1.11

K4：可解析沼气系数，K4＝0.8

M：煤层厚度（m）

As：采区平均日产量（t）

S：掘进煤断面（m²）

P：煤的密度（t／m³）

Lo：巷道长度（m）

W'：邻近层沼气含量（m³／t）

L：工作面长度（m）

qs：采区相对沼气涌出量（m³／t）

qz：采煤相对沼气涌出量（m³／t）

Q1：巷道煤壁沼气涌出量（m³/min）

Ko：全矿老空区沼气涌出系数

K1：岩石沼气涌出系数，取K1＝1.2

K3：掘进预排系数

Ki：邻近层沼气排出系数（按曲线选取）

Mo：采高（m）

Ai：回采面平均日产量（m）

V：掘进速度（m／min）

n：暴露煤面个数

Wo：煤层沼气含量（m³／t）

Vr：煤的挥发份含量（%）

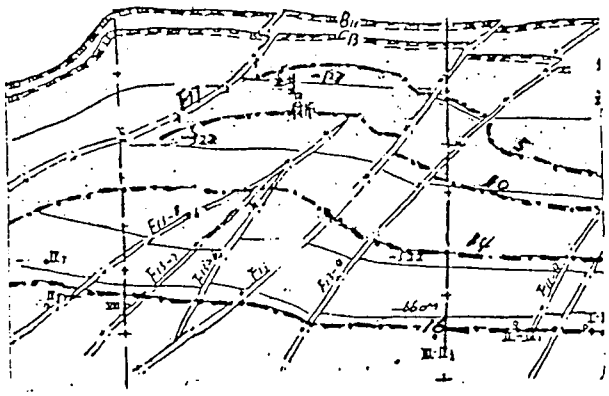
h：巷道排放带宽度，取h＝18m

三、予测及验证

㈠矿山统计法，以淮南谢二矿为例。

1．概况：谢二矿有可采煤层15层，总厚32．2m，倾角23°，其主采煤层自上而下有NO13，NO11，NO9，NO8，NO7，NO4六层，占总可采厚度的65%。15个采煤层分两组开采，其顶部的NO15～NO10为一组，NO9～NO1煤层为一组。在1970年前该矿只开采NO10～NO15煤层组。

2．煤层沼气：各煤层的沼气含量不一，其中以NO13，NO11的沼气含量的最高，瓦斯风化带浅；NO9以下各层的沼气含量低，瓦斯风化带深，在走向上各断层之间的沼气含量也不相等，沿倾斜则随深度的增加而增大，其中NO13，NO11煤层构造及沼气含量分布见图二。



图二．
NO.13．NO.11煤层构造及沼气含量分布图．
图例
煤层等高线 ———
含量等值线 —·—·—
石 门 界 +——·+

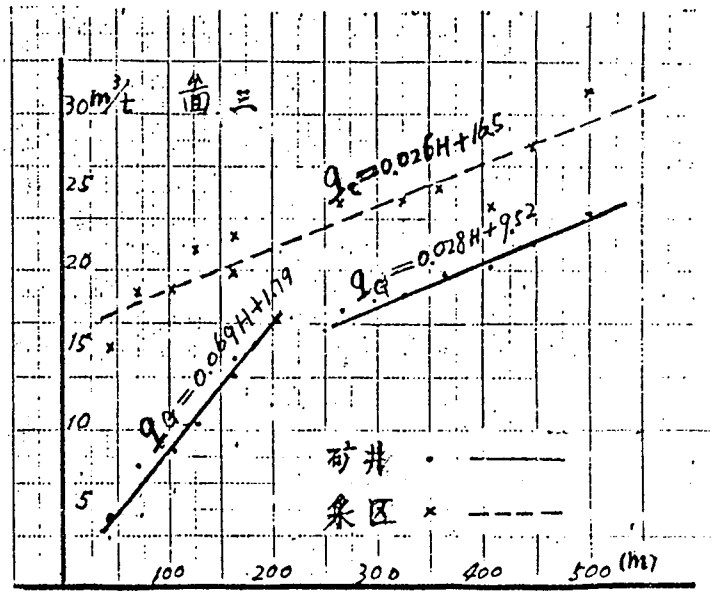
3．NO13顶分层工作面，高沼气煤层采区及矿井沼气涌出量与采深的关系统计于表一，表二及图三。

谢二矿13枝顶分层采面不同采深CH_4涌出量　表一

F17下盘——F13-8				F13-8——F13-4			
阶段标高（m）	采面平均标高（m）	CH_4含量（m^3/t）	CH_4涌出量（m^3/t）	阶段标高（m）	采面平均标高（m）	CH_4含量（m^3/t）	CH_4涌出量（m^3/t）
-62	-44	1.21	2.4	-62	-44	2.05	4.77
-88	-75	2.31	4.78	-127	-105	2.86	8.94
-127	-105	3.14	6.25	-162	-144	4.39	10.68
-162	-145	4.16	8.84	-202	-175	5.60	11.87
-200	-178	6.12	12.7	-242	-230	7.87	15.50
-252	-236	7.31	17.33	-282	-260	9.12	19.03
-322	-296	8.31	18.31	-322	-296	10.15	22.08
-365	-336	8.89	19.43	-365	-336	10.72	19.42
-400	-377	9.43	22.87	-400	-386	11.23	22.18
-440	-418	9.92	23.37	-450	-432	11.67	24.89
-490	-452	10.49	24.88	-500	-472	12.16	28.36

谢二矿井及高沼气采区采深与沼气涌出量统计 （表二）

年度	NO.13.No.11 工作面平均深度 (m)	No.13.No.11 对应采深瓦斯含量 (m^3/t)	他层对平均产量 (t/d)	矿井 CH4涌出量 (m^3/t)	高沼气采区		
					产量比重 (%)	占矿井瓦斯量 (%)	采区 CH4涌出量 (m^3/t)
1957～1958	45	1.56	2185	4.29	35.58	85.00	15.17
1959～1960	74	2.19	4262	7.49	31.86	85.30	18.81
1961～1962	105	3.44	3883	9.21	32.50	75.80	19.13
1963～1964	135	4.66	3666	10.31	44.30	72.97	21.93
1965～1966	167	5.42	3246	14.44	47.23	54.60	22.40
1967～1969	208	6.46	3074	15.44	76.80	50.70	22.74
1970～1972	266	7.69	2883	17.13	64.00	82.20	24.76
1973～1975	326	8.73	2957	18.66	60.10	81.90	24.80
1976～1978	366	9.33	3166	19.73	57.10	80.90	25.04
1979～1981	407	9.87	3234	20.13	60.60	80.60	24.11
1982～1984	438	10.36	3462	22.91	64.70	84.55	28.07
1985～1987	475	10.71	3587	23.78	49.50	80.10	31.46



统计资料表明：NO13煤层沿走向被断层切割成三个涌出量不同的区段：F17上盘q_M≤5m^3／t；F17下盘－F13－8之间；qM＝0．055H＋0．32，相关系数r＝0．985；F13－8－F13－4之间，qM＝0．052H＋2．40，r＝0．973，高沼气采区：qc＝0．026H＋16.5；γ＝0．963，矿井1966～1969年，qG＝0．069H＋1．79，　r＝0．977，1970～1987年，q_G＝0．029H＋9．52，r＝0．976．

4．矿井沼气来源：根据淮南老区四个高沼气矿历年资料统计，采煤占35．4～45．1%，平均为40%；　万吨煤煤巷道掘进量106～114m，掘进占39．2～19．4%，

平均为29%；其它，包括在用巷道及已采区　占39．2～19．4%，平均为31%。其中已采区平均占20%。

5．统计结果表明，回采工作面与高沼气采区的沼气涌出量与采深都有良好的一次函数关系，但回采工作面在各个断层块段的递增速度不一，矿井沼气涌出量与采深的关系，1969年以前随着高沼气采区产量和采深的增加，涌出量迅速增长。1970年矿界调整后，随着NO9以下低沼气煤层产量的增加，递增速度明显下降，当其产量保持在40%左右，矿井递增速度与高沼气采区的递增速度也保持一致；图三还表明，在采场分布发生变化以后，就不能继续用qG＝0．069H＋1．19递增关系来予测延深水平的沼气量，否则，由当时的－200m下延到－340m，其相对误差：

$$\frac{19.04-25.25}{25.25}=-0.25>-20\%$$

因此，在采用统计法予算时，必须考虑被予算的条件和范围。

㈡分源计算法：以潘一矿为例。

1．矿井概况：潘一矿是投产不久的新矿，井田走向长6．8km，其可采煤层的数量和厚度与谢二矿相同，倾角平均9°，井田西翼有一落差55～110m的斜切断层，将NO13、NO11等煤层分割成沼气含量不等的两个块段，东翼大，且有煤与沼气突出危险。其含量与煤系地层岩厚度(Hj)的关系为Wo＝0．07Hj÷(1＋0．0015Hj)［m³／t(C1M)］；西翼小，Wo＝0．039Hj－379［m³／t(C1M)］。

2．沼气予测情况：采用三种方法有三个结果。在矿井设计中，采用整体计算法。根据最先开采的NO13，NO11两个煤层的直接测定的最大沼气含量，予测一水平(－530m)最大沼气涌出量为10．59m³／t；用含量系数法，予测矿井沼气涌出量为31．18m³／t。用分源计算法予测结果为24．79m³／t；其断层以东三个采区平均为31．17m³／t；以西两个采区平均为10．31m³／t。

3．验证：以实际涌出量与分源计算法予测量进行比较：

(1) 1986、1987年矿井涌出量与分源计算涌出量比较见表三：

潘一矿矿井沼气涌出量与分源法计算量比较　(表三)

年度	矿井沼气涌出量 (m^3/t)				其中：断层以东				断层以西			
	平均标高 (m)	实际 (q_s)	计算 (q_y)	$\frac{q_s}{q_y}$	平均标高 (m)	实际 (q_s)	计算 (q_y)	$\frac{q_s}{q_y}$	平均标高 (m)	实际 (q_s)	计算 (q_y)	$\frac{q_s}{q_y}$
1986	-428.3	12.73	12.57	1.01	-442.3	17.72	20.68	0.86	-410	5.63	2.57	2.19
1987	-448.3	12.61	13.37	0.94	-464.7	18.96	22.66	0.84	-431	4.85	3.86	1.26

⑵断层以东采、掘工作面沼气涌出量与计算涌出量比较见表四。

潘一矿采掘工作面沼气涌出比较　表四

回采工作面				掘进工作面			
地点	涌出量 m^3		$\frac{q_s}{q_y}$	地点	涌出量 m^3/min		$\frac{q_s}{q_y}$
	计算	实际			计算	实际	
1111(3)	8.92	9.27	1.04	1211(3)下	1.67	1.47	0.88
1112(3)	8.77	10.42	1.19	1221(3)下	1.73	2.81	1.62
1122(3)	15.26	16.81	1.10	12 上山	2.63	2.77	1.05
1211(3)	6.01	5.34	0.89	1112(3)下	1.69	2.14	1.27
1212(3)	7.02	7.18	1.02	1122(3)上	1.95	1.7	0.87
1221(3)	9.57	9.03	0.94	1111(3)下	1.88	1.68	0.89
1132(3)	15.43	16.55	0.93	1121(3)下	2.31	2.38	1.03
1211(1)	3.55	4.33	1.22	11 上山	3.01	3.63	1.21
1361(3)	15.55	14.86	0.96	1361(3)下	4.52	5.06	1.20
				1361(3)上	4.35	4.12	0.95
平均			1.03	平均			1.10

⑶采区及矿井的沼气涌出系数见表五。

采区及矿井计算与实际沼气涌出比较表　(表五)

采区名称	计算涌出量 (m^3/min)			实际涌出量	
	采掘工作面总量	采区		实测量 (m^3/min)	涌出系数
		涌出系数	涌出量		
11采区	14.81	1.2	17.77	17.11	1.15
12采区	5.01	1.2	6.01	7.14	1.18
13采区	17.61	1.2	21.13	17.91	1.02
14采区	4.16	1.2	4.99	5.63	1.25
全矿	41.58	1.10	54.89	45.47	1.00

⑷矿井沼气来源；回采占47.0～55.9%，平均52.6%万吨煤煤巷掘进45～55m，掘进占34.1～37%，平均为35.6%；其它占18.9～7.1%，平均为11.8%。

从验证结果表明：整体计算予测量偏小；含量系数法，以断层以东的含量值为计算基础，只反映出断层以东的量，对于全矿其值偏大；分源计算法予测的矿井、采区（断层以西因采掘工作面处于瓦斯风化带附近，测定误差较大）以及断层以东的各采、掘工作面的沼气涌出量与实际对比，均在允许误差范围之内。随着矿井已采区范围扩大，掘进率提高，达到计算中所依据的80m／万吨时，其误差将进的步缩小。

四、结论

从验证中表明，对于任何一种预测方法都必须考虑煤层的沼气含量，赋存条件，地质构造，采场分布，生产工艺等影响因素及其变化。在上述两种方法中：统计法，其值是各种因素的综合结果，而且取值容易，量多面广，切实可靠。但早期的统计法，只单一的反映矿井沼气涌出量与开采深度的关系，予测范围受限，不能满足实际计算风量的需要。现已在原方法的基础上，结合地质、开采、通风因素对其原始数据进行分析筛选，找出规律。再据以编绘矿井瓦斯地质平面图，在图上分别描绘出矿井的瓦斯风化带的下限、矿井、采区、采煤、掘进工作面的沼气涌出量等值线，这样，不仅能反映沼气涌出参数的多变性，弥补了单一的q、H的关系的不足，缩小了予测的误差，还可得到其它予测方法所需要的实际涌出系数。这个方法仍广泛应用于生产矿井的予测。

分源计算法，已较全面的考虑了影响沼气涌出量的多种因素，其定量具体，分布清楚。可作为新井设计的主要予测方法推广应用。但为了提高予测的可靠程度，是要在地质勘探阶段根据不同的构造类型，在各主采煤层沿走向和倾斜布置一定数量的含量测定点，以较详细的掌握沼气含量的变化；二是有些参数要参照邻近矿的统计资料进行修正。

主要参考文献：

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METHODES DE CALCUL ET DE VERIFICATION DES EMISSIONS DE METHANE MINIER

Liu Binjun

Cet article considère les facteurs majeurs des émissions dangereuses de méthane, tels que la teneur en méthane des couches de charbon, la structure géologique et la distribution des fronts de taille pour le processus d'extraction, de creusement et de production. Nous suggérons et analysons trois méthodes de calcul : celle des statistiques, celle du coefficient de la teneur et celle du calcul séparé. Nous examinons leurs conditions d'application et leurs corrélations, dans le but d'améliorer l'exactitude des calculs d'une émission spécifique de méthane.

La vérification des résultats calculés a pour objectif non seulement d'établir la fiabilité des méthodes de calcul, mais aussi de connaître de manière adéquate les conditions géologiques du méthane minier. Cette vérification sert de base pour le contrôle de la sécurité, l'adaptation rationnelle des aménagements miniers et l'amélioration de la ventilation dans les mines à taux élevé de méthane.

METHODEN ZUR VORHERSAGE UND BESTÄTIGUNG VON GRUBENGASAUSSTRÖMUNGEN.

Liu Binjun

Die Abhandlung befaßt sich mit den Hauptfaktoren, die im Zusammenhang mit Methangasausströmungen bestehen, wie z. B. der Gasgehalt im Flöz, die geologische Struktur, die Verteilung von gebauten und erschlossenen Flözen und die Abbaumethode. Drei Methoden der Voraussage, nämlich die statistische Methode, die Gehalt - Koeffizient - Methode und die separate Berechnungs-Methode werden vorgeschlagen und analysiert. Die Anwendungsbedingungen und ihre Wechselbeziehungen werden im Hinblick auf eine Steigerung der Vorhersage-Zuverlässigkeit von spezifischen Methangasausströmungen diskutiert.

Die Bestätigung von vorausgesagten Ergebnissen zielt nicht nur auf eine Untersuchung der Zuverlässigkeit der Voraussagemethoden ab, sondern soll zur genauen Beherrschung der geologischen Bedingungen in Bezug auf Methangas-Ausströmung dienen. Damit entsteht eine Grundlage für eine sichere Gasbeherrschung, eine rationelle Feldauslegung und Verbesserung in der Bewetterung von Gruben mit hohem Gasgehalt.

МЕТОДЫ ПРЕДСКАЗАНИЯ ЭМИССИИ МЕТАНА В ШАХТАХ И ИХ ПРОВЕРКА

Лю Винджун

Настоящий доклад рассматривает следующие основные факторы, связанные с опасной эмиссией метана: содержание метана в угольных пластах, геологическая структура, распределение участков забоя для проходки и технологический процесс. Предлагаются и анализируются три метода: статистический, коэффициентов содержания и отдельный вычислительный метод. Обсуждаются условия их применения и улучшения надежности предсказания специфических эмиссий метана.

Подтверждение результатов доказывает не только надежность методов предсказания, но также помогает точно определить геологические условия для присутствия метана в шахтах. Точные методы предсказания составляют основу для контроля над безопасностью, приспособления добычи к существующим условиям и улучшения вентиляционной системы в шахтах с высоким содержанием метана.

FIREDAMP CONTROL AT THE FACE/RETURN AIRWAY INTERSECTION IN A RETREATING FACE

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SUMMARY

The present technical methods of production, concentrated on selected veins in the coal deposit, make it possible to reach very high daily outputs over the same face (5 to 10,000 tonnes net). Even if the seam is not very gassy, these high outputs result in a considerable emission of firedamp.

Increase in the air supplies and improved methane drainage make it possible, in the return airway, to maintain concentrations below the limiting values. Nevertheless, the dilution, in the ventilation air, of thin air streams with a high content of methane, coming from the caving zone, remains an important problem in the matter of preventing local heterogeneities in the concentration.

The paper starts by presenting the results of measurement programmes carried out in different faces, for the purpose of estimating the flow rates of firedamp and determining where the firedamp occurs. It appears that, on an average, in a retreating face, $\frac{2}{3}$ of the firedamp emission occur at the intersection between face and return airway.

The paper analyses different solutions applied in France and in other countries, in order to deal with the intersection of the retreating faces; a retreating system of ventilation is, in effect, necessary when there is a risk of fire in the goaf.

A model of air-firedamp exchanges is proposed; it makes it possible to judge the efficacy of the devices used.

A suction ventilation device, operated by a horn-effect nozzle and by means of compressed air ejectors is proposed. The paper describes the tests carried out on a model and then in an actual mine.

INTRODUCTION

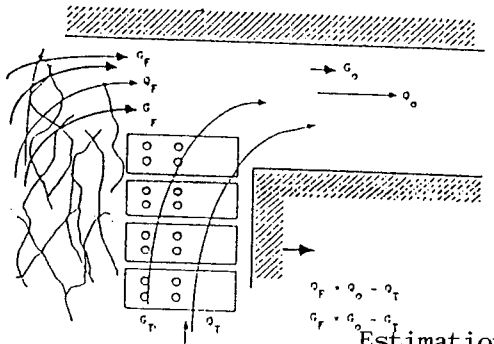
Very high daily outputs may be achieved in long faces by the present technical means; by selection of the deposit and by the layout (retreating working with slightly dipped face). Average daily productions of between 5 and 10,000 tonnes net are frequent, with peak outputs of as much as 14,000 tonnes (Houve and Reumaux workings).

Large air flows (30 to 40 m^3/s) and good methane drainage (rates of 70%) may, on an average, control the firedamp emission at the face and in the return airway. But, without special treatment, thin streams of highly concentrated firedamp, coming from the caving, cause local heterogeneity.

The paper describes a treatment based on a suction ventilation device, with a horn effect and operated by compressed air ejectors.

LOCATION AND MEASUREMENT OF FIREDAMP EMISSION IN A RETREATING FACE

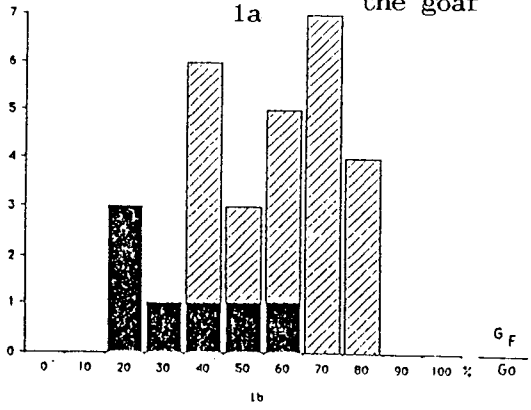
A large part of the firedamp emitted from a retreating face rejoins the ventilation circuit at the intersection of the face and return airway. The methane is mixed with air leakages which, deflected from the current of air circulating in the face, have passed into the goaf behind the supports (fig. 1a). This mixture produces very highly-concentrated air streams which rejoin the return airway and/or the end of the face, according to the methods of treatment.



Ventilation (Q)
Firedamp (G)

Estimation of volumes of leakages in the goaf

1a



1b

Distribution in accordance with
29 measurements in 11 faces

G_F braked: fabric or Isofoam barrier

figure 1

Homogeneity is not immediate, especially at the wall opposite the face in the return way (concentration >2% over several metres, and even over some tens of metres if special measures are not taken (Remillieux 72, Cwiklinski 84).

In order to get a better estimation of the volume of this firedamp emission, we measured the flow rates and the concentrations of the air at the end of the face and in the return airway. Fig. 1b shows the distribution of the percentage volume of firedamp coming from the face - return airway intersection; 29 measurements were made in 11 different faces in the Lorraine Coal Mines (Houilleres de Lorraine, H.B.L.). The distribution shows the large amount of firedamp emitted at this point; the volume may exceed 80% of the firedamp produced by the face.

On an average, 60% of the firedamp released into the ventilation comes from the face/return airway intersection. In the case of the 7 measurements made in faces, where the dead end of the return gate was sealed, this ratio is only 39 %.

TREATMENT OF THE FACE-RETURN AIRWAY INTERSECTION

This problem has long been recognised in retreating faces in gassy seams (Thorp 70, Dieltiens 80). With the great increase in the outputs per unit, the problem has also appeared in seams that are only slightly gassy (5 to 15 m^3/t total firedamp emission).

Now, the retreating working system presents a great number of advantages, both as regards the technical results and collective and individual safety (Masrey 77, Chironis 80):

- preliminary reconnaissance of the deposit,

- easy access to the working, better control of rock pressure, better air supply,

- and, especially, in a deposit subject to heating, control of the risks related to the phenomena of spontaneous combustion by limiting the circulation of air leakages in the caving zone.

In the H.B.L. (Lorraine) mines in France, control of the hazards related to fires in the goaf demand a ventilation system which must be retreating and often necessitate the use of nitrogen to neutralise the atmosphere in the goaf.

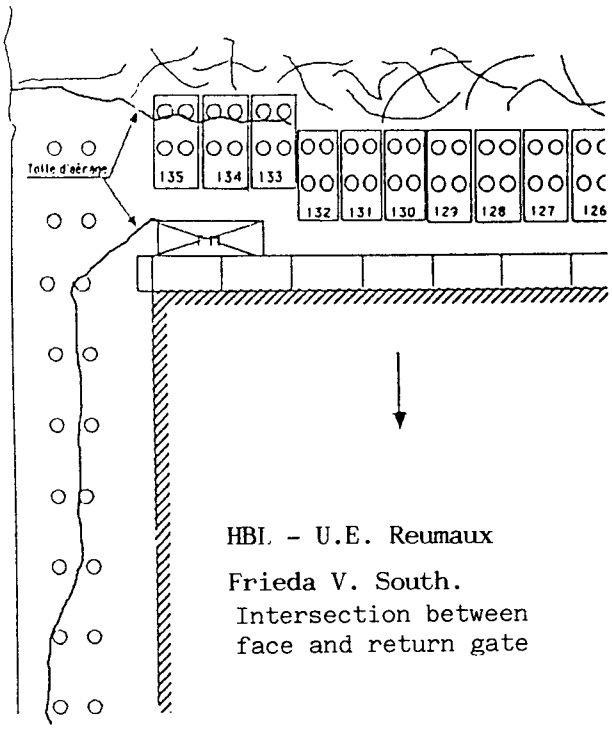
Volume of nitrogen used in the Lorraine Coal Field (in thousands of Nm^3)

1979	561
1980	4085
1981	5862
1982	6259
1983	15693
1984	13967
1985	2837
1986	5606
1987	26448
1988	25215

But as well as the preliminary investment for heading operations, retreating faces present the problem of purifying the intersection of the face - return way. Numerous solutions exist for dealing with this problem in France and abroad. Without claiming to be exhaustive, let us recall the principles of these different solutions.

Accelerated dilution (increase in the air speed)

It is a question of increasing the speed of the air current, at the points where the thin streams of air emerge, by the use of air deflectors, barriers and/or compressed air ejectors. (France, mainly, fig. 2). These devices demand frequent and precise adjustment, with possible risks of recirculation.



HBL - U.E. Reumaux

Frieda V. South.
Intersection between face and return gate

Figure 2

Another solution consists in ventilating this zone by auxiliary ventilation, the fan being installed in the face in the return airway or outside the panel with, in this last case, an extra amount of air supplied (FRG, Belgium).

Diversion of air ....

It is a question of spreading out the zone where firedamp emerges by arranging, locally, at the end of the face, an auxiliary blower fan. Either by driving, in advance, a double return airway connected by cross-cuts to the main gate (this has already been done in a large number of countries: FRG., GB., China, France .....); but this method is costly; or driving the second road as the face advances, leaving a panel of coal between the two roads (GB., Poland ....).

Exhaust ventilation (suction fan)

This system of suction ventilation has been used mainly in France (Remilleux 72) and is still being used at present in Belgium (Berigen).

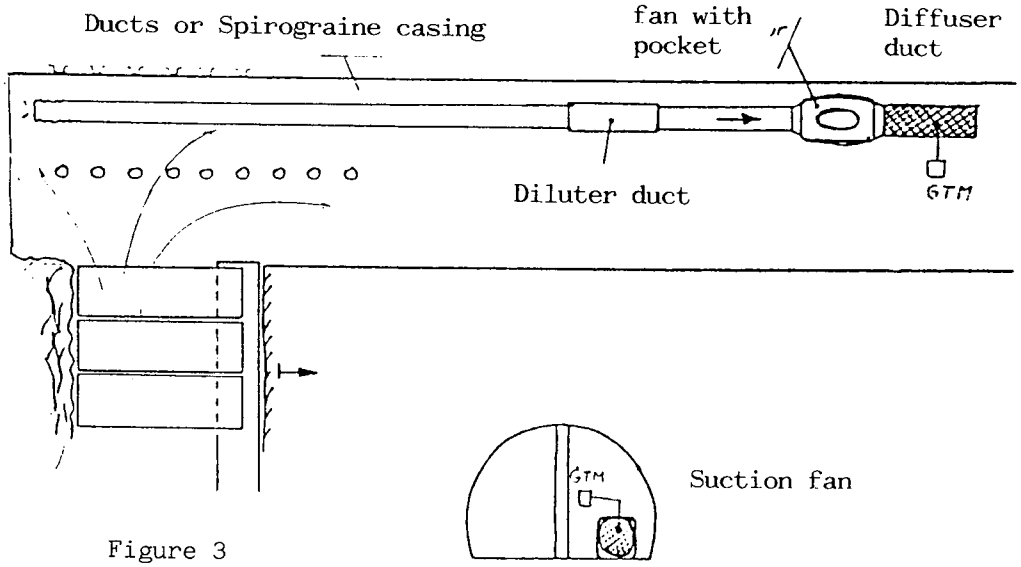


Figure 3

It requires the installation of a power source in the return airway and there is the risk of ignition in the suction line (heating of rotating parts, bearings ....).

Also the flow of this fan must keep pace with the increase in the production and thus in the supply of primary air.

In our most recent operations, we have retained the same concept of firedamp control by exhaust ventilation, but in order to improve the safety, a suction,device driven by a compressed air ejector has been used, instead of an electric fan.

PRINCIPLE OF CLEANSING THE ATMOSPHERE AT THE TOP OF THE FACE BY MEANS OF A EXHAUST VENTILATION DEVICE

The purpose of the suction device is to collect the highly-concentrated air leakages from the cul-de-sac of the return way, diluting them with an extra supply of air from the face. The flow of this air supply from the face towards the suction opening situated in the cul-de-sac sweeps over a route which, in the absence of special devices, would be taken only by the gases from the goaf.

A suction device thus prevents gas accumulation in the critical zone of the cul-de-sac and expels, into the return way, beyond the equipment, a large part of the methane released.

Basic hypothesis

In a permanent system (stability of production, of barometric pressure, of methane drainage .....), the distribution of the firedamp between the face and the intersection is constant.

The flow lines are shown as a diagram in fig. 4, which gives the reference markings.

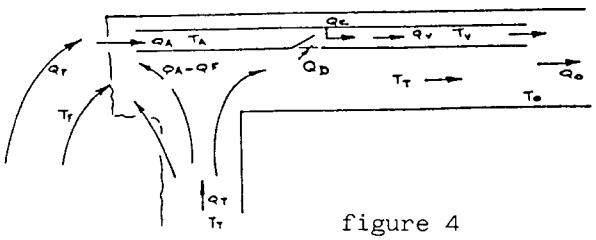


figure 4

In order to collect all the thin, highly-concentrated air streams, the volume aspired in Q_A must be greater than the flow in the thin highly-concentrated air streams Q_F :

$$Q_A > Q_F$$

In this case, the highly-concentrated air streams mingle with the flow $Q_A - Q_F$ of air coming from the face, where the concentration is low.

Schematic constitution of the suction device

A column, rigid or consisting of a reinforced air duct, installed in the return airway has its suction end situated in the cul-de-sac of the return gate.

The motive organ for the suction is a horn-effect, compressed-air nozzle, the flow, Q_c, of which, comes to mingle with the displaced volume Q_v.

Upwind of the nozzle, a diluter provides for the admission of a diluted volume Q_D taken from the air in the return airway frim the face.

The suction column, at its outlet, rejects a mixture the concentration of which may reach the limit L (for example 3%); measures must be taken to ensure that the expelled air mixes with the air in the gallery.

The final content, in the return airway of the face T_o, depends, in a permanent system, only on the flow of firedamp G_o and on the volume of air Q_o, which represent basic data independent of the functioning of the suction device.

Functioning conditions

The functioning conditions are evaluated by a modelling method, consisting of writing the equations expressing the conservation of the flows of air and of firedamp at the different mixing points.

- Principal condition

From the condition to be adhered to:

$$T_v \leqslant L$$

(concentration T at the outlet of the suction column less than the limit L),

we deduce:

$$Q_v \geqslant Q_o \frac{T_o - T_T}{L - T_T}$$

The flow rates and concentrations used in this expression are observable values, when the suction device is in use; if one wishes to evaluate, provisionally, the same expression, as a function of the observable values in the working, without a suction device, we must, strictly speaking, allow for the fact that the introduction of the compressed air flow Q_c will modify the initial flows Q_{oi} and Q_{Ti}. In practice, taking into account the relative values of the flows, for example:

$Q_c = 0.4\ m^3/s$ and $Q_o = 30\ m^3/s$,

the influence of this correction is negligible.

The existence of a minimum limit for the flow rate of the suction device Q_V corresponds to the fact that if we wish to dilute all the highly-concentrated thin streams of air (Q_F T_F) in the suction column, we must mix them with a sufficient flow of air of lower concentration ($Q_A - Q_F$, T_T).

If it is not possible to have a suction volume greater than the limit calculated above, we shall not be able to collect, in the suction column, all the high-concentration air streams; part of these will flow away outside the suction column. We must dilute these high-concentration air streams, as in the case of the face end without suction device.

- Choice of the limiting concentration L.

The limiting concentration L comes into the calculation of the value of the minimum flow rate in the suction device:

$$Q_{v\ min} = Q_o \quad \frac{T_o - T_T}{L - T_T}$$

If one chose, for L, a low value, not much greater than T_o, the result would be a very high value for $Q_{v\ min}$. It would not be possible to develop a suction device providing such a high rate of flow.

It is, however, necessary, that the limit L should be less than 5% with an adequate safety margin.

The choice of the value of L must thus be made between these limits:

- 5% less a safety margin

- 2%,

taking into account analysis of the safety of the system. A value of L = 3%, as already quoted, (Remillieux 72) seems reasonable.

- Influence of the dilution rate Q_D

The content T in the first part of the suction column is an increasing function of Q_D; it is minimum for $Q_D = 0$. It will therefore be advantageous to operate with a permanent system and with the diluter closed.

The diluter is, nevertheless, necessary, for it makes it possible to operate the system without risks of being troubled by the high concentrations possibly present in the cul-de-sac; it also makes it possible to make the installation safe even if a highly-concentrated puff of air should pass ;
the automatic system controls the opening of the diluter, bringing air, of concentration T_T, from the face, into the installation.

Apart from these two circumstances, it is preferable to keep the diluter closed.

Influence of the suction device on safety at the end of the face

The installation of a suction device in the return airway from a retreating face modifies out analysis of the risks in the zone at the end of the face.

A comparative examination of the two situations was carried out and enabled us to conclude that the suction system, applied judiciously, reduces the risks.

TESTS ON A SUCTION DEVICE

Preliminary measurements were carried out on a mine model in order to measure the performances of a suction device operating with a horn effect.

Measurements of flow rates from different models of nozzles as a function of the setting of the ejector and of the compressed air feed pressure.

Measurements of the total flow rates (ejector plus induced flow) in accordance with different

configurations of a column of air ducts of 600 mm Ø.

- Influence of the opening of the diluter.

- Tests on the reliability of the control provided by the GTM (firedamp transducer) in the installation.

In conclusion, with a JET FLOW type 200 nozzle, a column of ducts of 600 mm Ø and a compressed air pressure of 4 bar, flow rates of 6 to 7 m^3/s can be obtained.

Test of the suction device in a mine

The test was carried out in the Georgette II North face of the Reumaux Mine Unit in the Lorraine Coal Mines, in the months of August and September 1988. This is a mechanised longwall face on the dip (thickness 3 m, length 270 m and inclination of about 30 degrees), with caving and mining by shearer

During the period of the test, the average daily output was 4,048 tonnes in August and 3,320 tonnes in September (end of panel).

The average firedamp emission and the ventilation conditions are shown by the following characteristics with the reference markings used in fig. 4.

Q_T = 20 m^3/s G_T = 0.05 m^3/s

Q_o = 26 m^3/s G_o = 0.12 m^3/s

Q_F = 6 m^3/s G_F = 0.07 m^3/s

For this face, with its average firedamp emission, T_o = 0.5%, the only firedamp control problem arose at the intersection of the face and the return airway, where local heterogeneities were measured when the shearer was at the top of the face or when irregular influxes of firedamp occurred (roof falls, barometric depression .....), before the installation of the suction device.

The model for calculation, developed in Paragraph 4, shows that the suction device, tested in a mine model, may dilute thin streams of firedamp with a safety coefficient of 2, if L = 2%.

- Layout and operational principle (fig. 5).

One or two ejectors placed at the top of the face, in the current flow of air from the face, blow into the cul-de-sac, making it possible:

- to mix the air at the suction point

- to reduce the firedamp content of the mixture aspired.

The concentration of the mixture which passes into the column is monitored by the GTM.

If the concentration monitored by the GTM exceeds 2% CH_4, the compressed air supplying the controls of the flap valve is automatically cut off and the flap drops; suction ceases in the cul-de-sac and the ejector JA 200 draws air from the top road.

A klaxon sounds to indicate this trip action and the sound signal can be stopped only by intervention of the personnel.

If the GTM breaks down, this causes the same effects as those that occur if the fixed limit is exceeded.

If the compressed air supply to the flap controls fails, the flap drops.

The suction can be restored only by hand, from an AC (slide) valve in the controls of the flap and when concentration in the column is less than 2%.

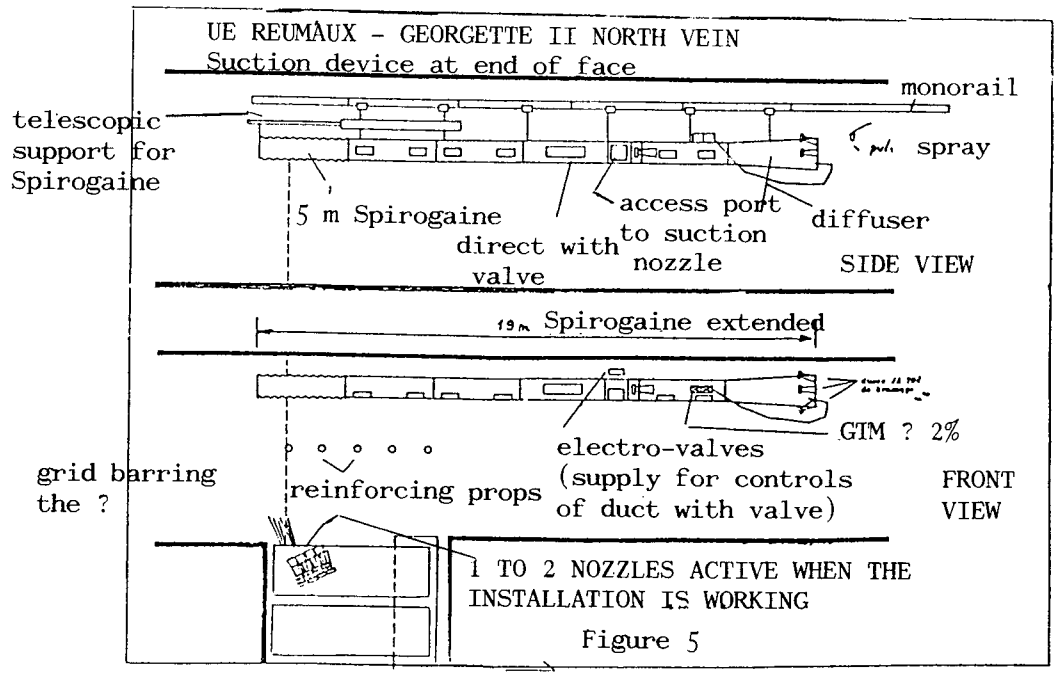
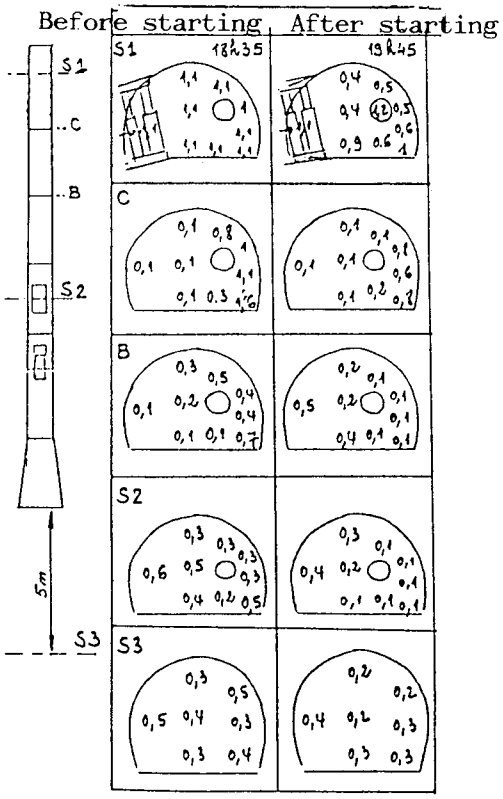


Figure 5

Before starting to operate the installation, the person in charge measures the concentrations in the VM1 methanometer, in five different sections (fig. 6).



Q (rate) : 5,3 m³/s
CH_4 inlet 1,2%
CH_4 outlet 0,9%

Figure 6

After the installation has been started a fresh series of concentration measurements is carried out.

The person in charge of the installation then measures the flow rate sucked in by means of a Pitot tube, then reads off the compressed air pressure on the manometer, before noting the concentration at the inlet and at the outlet of the suction device.

- Results of test.

The test carried out has demonstrated the advantages anticipated by such a method.

After the device has been working for 5 minutes we observe, as is shown by the concentration charts in fig. 7:

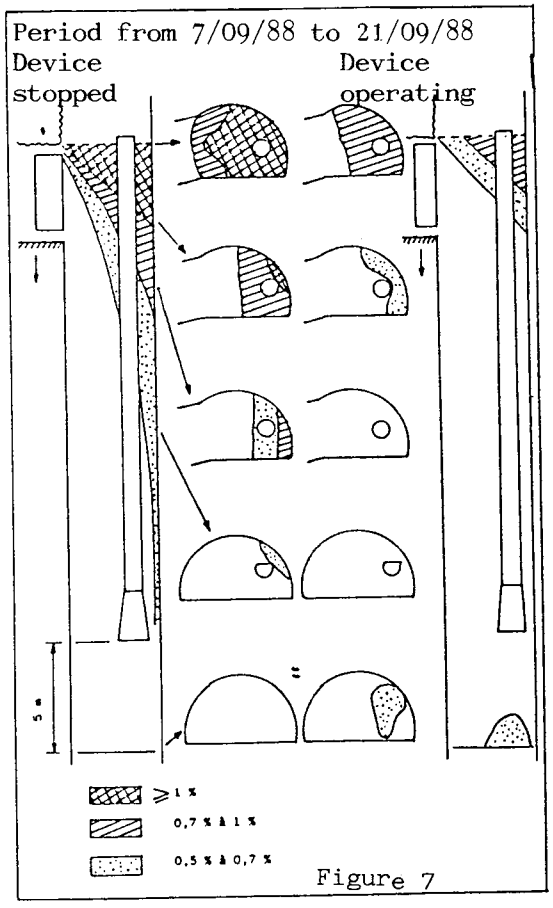


Figure 7

- a more homogeneous atmosphere at the face/roadway intersection,

- elimination of recirculation,

- reduction of the great irregularities in the concentrations, especially at the wall of the gallery opposite the face.

Examination has made it possible, too, to confirm the qualities of this device:

- no tricky, unstable or improvised adjustments,

- facilities for putting into operation and moving it (suspension on mono-rail),

- device operates silently,

- great reliability.

CONCLUSIONS

In a retreating face a large part of the firedamp gets into the ventilation in the vicinity of the intersection between face and return airway. A simple calculation model is proposed for dimensioning a suction ventilation device, making it possible to dilute thin streams of air carrying a high concentration of methane. Tests on a new prototype of aspirator have shown its importance when its performance is adequate. From the point of view of safety, the solution proposed is that which provides the least probability of igniting fire-damp, both inside the device (no electrical energy, no rotating parts) and outside the working, when a strictly retreating system of ventilation is necessary.

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MAITRISE DU GRISOU AU CARREFOUR TAILLE-VOIE DE RETOUR D'AIR EN TAILLE RABATTANTE

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RESUME

Les moyens techniques actuels de production, concentrés sur les veines sélectionnées du gisement, permettent d'atteindre des productions journalières très élevées sur une même taille (5 à 10 000 tonnes nettes). Même en gisement moyennement grisouteux, ces productions élevées se traduisent par un dégagement important de grisou.

L'augmentation des débits d'air et l'amélioration du captage permettent de maintenir, dans le retour d'air, des teneurs en deçà des valeurs limites. Cependant la dilution dans l'aérage des filets à haute teneur, en provenance du foudroyage, reste un problème important pour éviter les hétérogénéités locales de teneur.

L'article présente d'abord les résultats de campagnes de mesures effectuées dans différentes tailles pour estimer les débits de grisou et leur localisation. Il apparaît qu'en moyenne, en taille rabattante, les 2/3 du dégagement grisouteux se produisent au carrefour de la taille et de la voie de retour.

Il est présenté une analyse de différentes solutions utilisées en France et à l'étranger pour traiter le carrefour des tailles rabattantes ; l'aérage rabattant est en effet nécessaire lorsque le gisement présente des risques de feux dans l'arrière-taille.

Une modélisation des échanges air-grisou est proposée pour représenter le carrefour taille-voie ; elle permet de juger de l'efficacité des dispositifs utilisés.

Un dispositif d'aérage aspirant par effet de trompe grâce à des éjecteurs à air comprimé est proposé. L'article décrit les tests effectués en mine image puis en exploitation.

INTRODUCTION

Des productions journalières très élevées peuvent être atteintes dans les longues tailles au pendage, grâce aux moyens techniques actuels, à la sélections du gisement et au mode de découpage (taille rabattante avec front légèrement descendant). Des moyennes de production journalière entre 5 et 10 000 tonnes nettes sont fréquentes, avec des pointes à 14 000 tonnes (siège de la Houve et Reumaux).

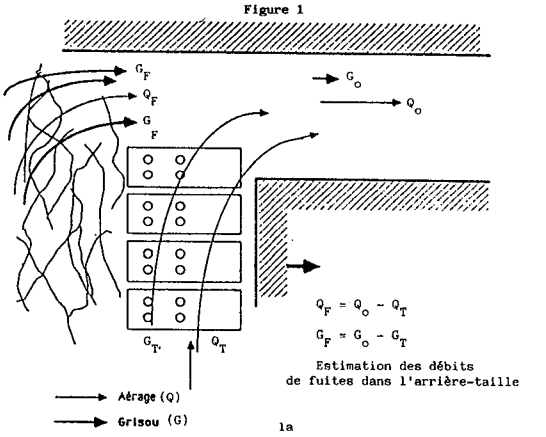
Des débits d'air importants (30 à 40 m^3/s) et un bon captage (taux de 50 à 70 %) peuvent maîtriser en moyenne le dégagement de grisou en taille et dans le retour d'air. Mais, sans traitement particulier, des filets de grisou à hautes teneurs, provenant du foudroyage, provoquent localement des hétérogénéités.

L'article décrit un traitement basé sur un dispositif d'aérage aspirant par effet de trompe, grâce à des éjecteurs d'air comprimé.

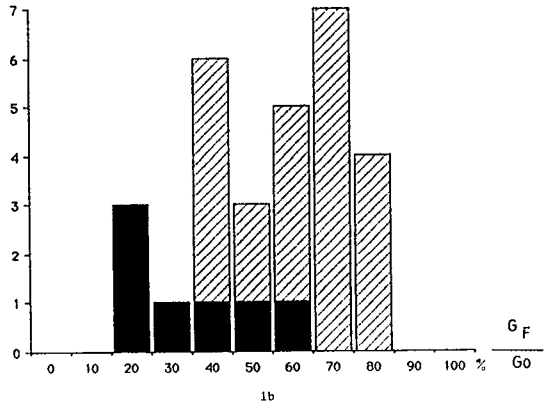
LOCALISATION ET MESURE DU DEGAGEMENT DE GRISOU EN TAILLE RABATTANTE

Une partie importante du grisou dégagé par une taille rabattante rejoint le circuit d'aérage au carrefour taille-voie de retour d'air. Le méthane est mélangé aux fuites d'air qui, dérivées du flux d'air circulant en taille, sont passées dans l'arrière-taille derrière les piles de soutènement (figure 1a). Ce mélange donne des filets à teneur élevée en rejoignant la voie de retour d'air et/ou la tête de taille

Figure 1



1a



1b

Répartition d'après 29 mesures dans 11 tailles

G_F freiné : barrage toile ou isomousse

selon les modes de traitement. L'homogénéité n'est pas immédiate, surtout au parement opposé à la taille dans la voie de retour (teneur $>$ 2 % sur quelques mètres, voire dizaines de mètres si on ne prend pas de dispositions spéciales (Remillieux 72, Cwiklinski 84).

Pour mieux connaître l'importance de ce dégagement de grisou, nous avons mesuré les débits et teneurs de l'air en extrémité de taille et dans la voie de retour. La figure 1b montre la répartition du pourcentage du débit de grisou provenant du carrefour taille-voie pour 29 mesures prises dans 11 tailles différentes des Houillères de Lorraine (H.B.L.). Elle met en évidence l'importance du dégagement de grisou en ce point qui peut dépasser 80 % du grisou produit par la taille.

En moyenne, 60 % du dégagement de grisou dans l'aérage proviennent du carrefour taille-voie. Pour les 7 mesures effectuées dans les tailles où le cul-de-sac de la voie était barré, ce rapport n'est que de 39 %.

TRAITEMENT DU CARREFOUR TAILLE-VOIE DE RETOUR

Ce problème est connu depuis longtemps dans les tailles rabattantes en gisement grisouteux (Thorp 70, Dieltiens 80). Avec l'augmentation importante des productions unitaires, il est apparu également en gisement peu grisouteux (5 à 15 m^3/t de dégagement total de grisou).

Or l'exploitation rabattante présente de nombreux avantages, tant sur le plan des résultats techniques que sur le plan de la sécurité collective et individuelle (Masrey 77, Chironis 80) :

- reconnaissance préalable du gisement,

- accès aisés au chantier, meilleure maîtrise des pressions de terrains, meilleur débit d'aérage,

- et surtout, en gisement susceptible d'échauffement, maîtrise des risques liés aux phénomènes de combustion spontanée en limitant les circulations parasites d'air dans le foudroyage.

Dans les exploitations des H.B.L. en France, la maîtrise des risques liés aux feux d'arrière-taille nécessite un aérage strictement rabattant et souvent l'utilisation d'azote pour inertiser l'atmosphère de l'arrière-taille.

Volume d'azote consommé pour le Bassin de Lorraine (en milliers de Nm^3)	
1979	561
1980	4085
1981	5862
1982	6259
1983	15693
1984	13967
1985	2837
1986	5606
1987	26448
1988	25215

Mais en plus d'un investissement préalable pour le creusement de voies, les tailles rabattantes posent le problème de l'assainissement du carrefour taille-voie de retour. De nombreuses solutions existent pour traiter ce problème, en France et à l'étranger. Sans avoir la prétention d'être exhaustif, rappelons les principes de ces différentes solutions.

Dilution accélérée (augmentation de la vitesse d'air).

Il s'agit d'augmenter la vitesse du courant d'air au niveau des zones de sortie des filets de grisou par l'utilisation de déflecteurs d'air, de barrages et/ou d'éjecteurs à air comprimé (France, principalement, figure 2). Ces dispositifs demandent des réglages fréquents et précis avec risques éventuels de rebrassage.

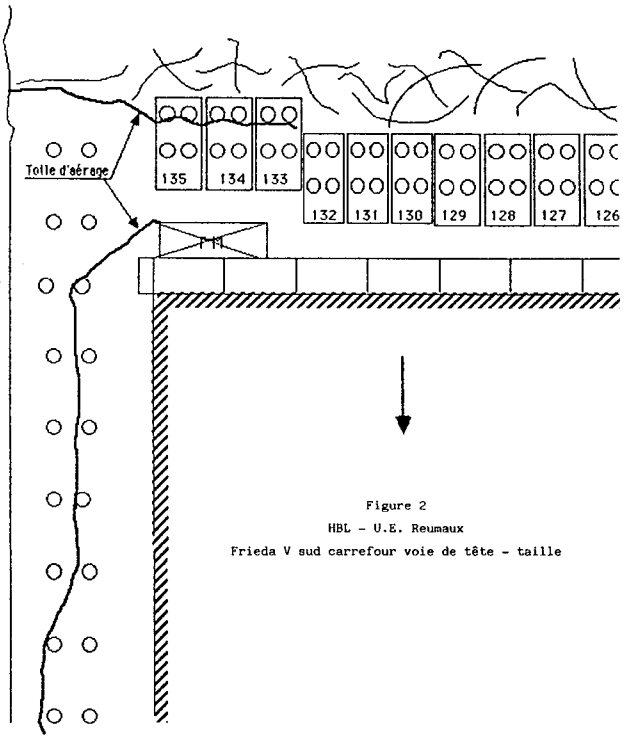


Figure 2
HBL - U.E. Reumaux
Frieda V sud carrefour voie de tête - taille

Une autre solution consiste à aérer cette zone par un aérage secondaire, le ventilateur étant installé en taille, dans la voie de retour ou hors du quartier, avec appoint d'air dans ce dernier cas (RFA, Belgique).

Dérivation de l'aérage..

Il s'agit d'étaler dans l'espace la zone des sorties de grisou en pratiquant localement en tête de taille l'aérage chassant. Soit en creusant à l'avance une double voie de retour d'air reliée par des recoupes à une première voie (a déjà été pratiquée dans de nombreux pays: RFA, G-B, Chine, France...) mais cette méthode est coûteuse ; soit en creusant la seconde voie avec l'avancement de la taille, en laissant une planche de charbon entre les deux voies (G-B, Pologne...).

Lorsque la tenue des terrains et la susceptibilité aux échauffements le permettent, cette dérivation de l'aérage peut être définitive sur toute la longueur du panneau, c'est l'aérage en Z (G-B, RFA, Nord de la France...).

Ventilation aspirante (ventilateur aspirant)

Ce système de ventilation aspirante a été utilisé surtout en France (Remilleux 72) et encore actuellement en Belgique (Berigen).

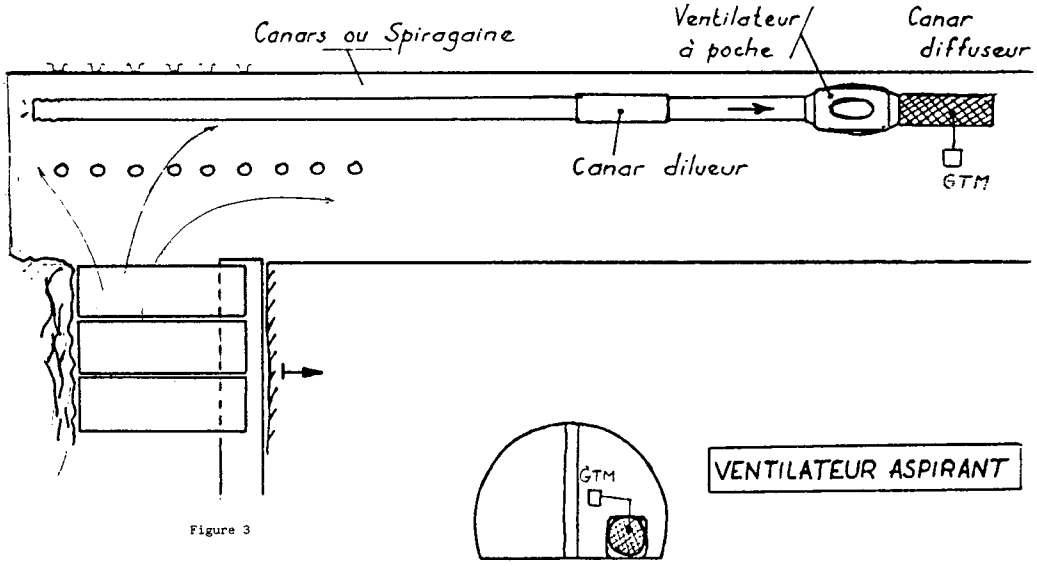


Figure 3

Il nécessite l'installation d'une source d'énergie dans la voie de retour et le risque d'inflammation dans la ligne aspirante existe (échauffement de pièces tournantes, de paliers...).

De plus, la puissance de ce ventilateur doit suivre l'augmentation de la production, donc du débit d'air primaire.

Dans nos travaux les plus récents le même concept de contrôle de grisou par une ventilation aspirante a été gardé, mais pour améliorer la sécurité, un dispositif aspirant par effet de trompe à air comprimé a été employé au lieu d'un ventilateur électrique.

PRINCIPE DE L'ASSAINISSEMENT DE LA TÊTE DE TAILLE AU MOYEN D'UN DISPOSITIF ASPIRANT

Le but du dispositif aspirant est de collecter les fuites à teneur élevée du cul-de-sac de la voie de retour en les diluant par un appoint d'air venant de la taille. L'écoulement de cet appoint d'air de la taille vers l'orifice d'aspiration situé dans le cul-de-sac balaie une zone qui, en l'absence de dispositifs spéciaux, ne serait parcourue que par les gaz sortant de l'arrière-taille.

Un dispositif aspirant évite ainsi l'accumulation de gaz dans la zone critique du cul-de-sac et rejette dans la voie de retour, au-delà des matériels et équipements, une partie importante du grisou dégagé.

Hypothèse de base.

En régime permanent (stabilité de la production, de la pression barométrique, du captage...) la répartition du grisou entre la taille et le carrefour est constante.

Les lignes d'écoulement sont représentées schématiquement sur la figure 4 qui précise les notations.

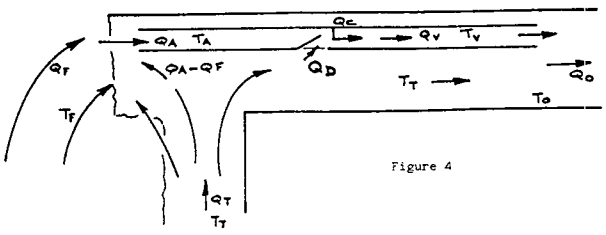


Figure 4

Pour collecter la totalité des filets à forte teneur, il faut que le débit aspiré Q_A soit supérieur au débit des filets à forte teneur Q_F :

$$Q_A > Q_F$$

Dans ce cas les filets à forte teneur se mélangent au débit $Q_A - Q_F$ d'air venant de la taille dont la teneur est faible.

Constitution schématique du dispositif aspirant

Une colonne, rigide ou en ventube armé, installée dans la voie de retour d'air a son extrémité aspirante dans le cul-de-sac de la voie de tête.

L'organe moteur de l'aspiration est une trompe à air comprimé dont le débit Q_c vient se mélanger au débit déplacé Q_v.

En amont de la trompe un dilueur permet de faire entrer une débit de dilution Q_D prélevé dans l'aérage de la voie de retour de la taille.

La colonne aspirante rejette à sa sortie un mélange dont la teneur peut atteindre la limite L (par exemple 3 %), il faut prendre des dispositions pour assurer le mélange de ce rejet avec l'air de la galerie.

La teneur finale dans le retour d'air de la taille T_o, dépend seulement, en régime permanent, du flux de grisou G_o et du débit d'air Q_o qui sont des données de bases indépendantes de la marche du dispositif aspirant.

Conditions de fonctionnement

Les conditions de fonctionnement sont évaluées au moyen d'une modélisation consistant à écrire les équations de conservation des débits d'air et de grisou aux différents points de mélange.

- Condition principale.

De la condition à respecter :

$$T_v \leq L$$

(teneur T_v à la sortie de la colonne aspirante inférieure à la limite L), on déduit :

$$Q_v \geqslant Q_o \frac{T_o - T_T}{L - T_T}$$

Les débits et teneurs utilisés dans cette expression sont les valeurs observables quand le dispositif aspirant est en service, si on veut évaluer à titre prévisionnel la même expression en fonction des valeurs observables dans le chantier sans dispositif aspirant, il faut en toute rigueur tenir compte du fait que l'introduction du débit d'air comprimé Q_c va modifier les débits initiaux Q_{oi} et Q_{Ti}. En pratique, compte tenu des valeurs relatives des débits, par exemple : $Q_c = 0{,}4\ m^3/s$ et $Q_o = 30\ m^3/s$, l'influence de cette correction est négligeable.

L'existence d'un seuil minimum pour le débit du dispositif aspirant Q_v correspond au fait que si l'on veut diluer tous les filets à forte teneur $(Q_F\ T_F)$ dans la colonne aspirante, il faut les mélanger à un débit suffisant d'air à plus faible teneur $(Q_A - Q_F,\ T_T)$.

Si on ne peut pas disposer d'un débit aspirant supérieur à la limite calculée ci-dessus, on ne pourra pas collecter dans la colonne aspirante tous les filets à forte teneur, une partie de ceux-ci s'écoulera en dehors de la colonne aspirante. Il faut diluer ces filets à forte teneur comme dans le cas des extrémités de taille sans dispositif aspirant.

- Choix de la teneur limite L.

La teneur limite L intervient dans le calcul de la valeur du débit minimum du dispositif aspirant :

$$Q_{v\ min} = Q_o \frac{T_o - T_T}{L - T_T}$$

Si on choisissait pour L une valeur faible, peu supérieure à T_o, il en résulterait une valeur très élevée pour $Q_{v\ min}$ on ne pourrait pas réaliser un dispositif aspirant fournissant un débit assez élevé.

Il faut d'autre part que la limite L soit inférieure à 5 % avec une marge de sécurité suffisante.

Le choix de la valeur de L doit donc être fait entre ces limites :

- 5 % moins une marge de sécurité,
- 2 %,

en tenant compte de l'analyse de la sécurité du système. Une valeur L = 3 % déjà citée (Remillieux 72) semble raisonnable.

- Influence du débit de dilution Q_D.

La teneur T_A dans la première partie de la colonne aspirante est une fonction croissante de Q_D ; elle est minimum pour $Q_D = 0$. on aura donc avantage à fonctionner en régime permanent avec le dilueur fermé.

Le dilueur est cependant nécessaire car il permet d'effectuer la mise en marche du système sans riques d'être gêné par les teneurs élevées éventuellement présentes dans le cul-de-sac ; il permet également la mise en sécurité de l'installation en cas de passage d'une bouffée à teneur élevée, l'automatisme commande alors l'ouverture du dilueur faisant ainsi passer, dans l'installation, de l'air venant de la taille à la teneur T_T.

En dehors de ces deux circonstances, il est préférable de conserver le dilueur fermé.

Influence du dispositif aspirant sur la sécurité de la tête de taille

L'installation d'un dispositif aspirant en voie de retour d'une taille rabattante modifie l'analyse des risques dans la zone d'extrémité de taille.

L'examen comparatif des deux situations a été effectué et a permis de conclure que le système aspirant appliqué judicieusement diminue les risques.

ESSAI D'UN DISPOSITIF ASPIRANT PAR EFFET DE TROMPE

Essai au jour

Des mesures préliminaires sont effectuées en mine image pour mesurer les performances d'un dispositif aspirant par effet de trompe.

Mesures des débits de différents modèles de trompes, en fonction de réglage de l'éjecteur et de la pression d'alimentation en air comprimé.

Mesures des débits totaux (éjecteur plus débit induit) selon différentes configuration d'une colonne de canars en Ø 600 mm.

- Influence de l'ouverture du dileur.
- Tests sur la fiabilité du pilotage par le GTM (capteur grisou) de l'installation.

En conclusion, avec une trompe Jet Flow type 200, une colonne de canars de Ø 600 mm et une pression d'air comprimé de 4 bars, des débits de 6 à 7 m^3/s peuvent être atteints.

Essai du dispositif aspirant en mine

Il a été effectué en taille de Georgette II Nord de l'Unité d'Exploitation de Reumaux aux HBL en août et septembre 1988. C'est une longue taille mécanisée au pendage (ouverture 3 m, longueur 270 m et environ 30 degrés de pente), foudroyée, abattage par haveuse.

Durant la période de l'essai, la production moyenne journalière est de 4 048 tonnes en août et 3 320 tonnes en septembre (fin de panneau).

Le dégagement de grisou moyen et les conditions d'aérage se traduisent par les caractéristiques suivantes avec les notations de la figure 4 :

$Q_T = 20\ m^3/s$ $\quad G_T = 0,05\ m^3/s$

$Q_o = 26\ m^3/s$ $\quad G_o = 0,12\ m^3/s$

$Q_F = 6\ m^3/s$ $\quad G_F = 0,07\ m^3/s$

Pour cette taille moyennement grisouteuse, $T_Q = 0,5$ %, le seul problème de maîtrise du dégagement de grisou se posait au carrefour taille voie de retour où des hétérogénéités locales étaient mesurées lorsque la haveuse était en tête de taille ou lors de venues irrégulières de grisou (chutes de toit, dépression barométrique...), avant l'installation du dispositif aspirant.

Le modèle de calcul, developpé au paragraphe 4, montre que le dispositif aspirant, essayé en mine image peut diluer ces filets de grisou avec un coefficient de sécurité de 2, si L = 2 %.

- Implantation et principe de fonctionnement (figure 5).

Un ou deux éjecteurs placés en tête de taille dans le flux d'air de la taille soufflent dans le cul-de-sac et permettent :

. de réaliser un brassage de l'air à l'aspiration,

. de diminuer la teneur en grisou du mélange aspiré.

La teneur du mélange qui passe dans la colonne est contrôlée par le GTM.

Trois éjecteurs soufflants montés sur le diffuseur diluent la sortie d'air du dispositif.

Si la teneur contrôlée par le GTM dépasse 2 % CH_4, l'air comprimé alimentant les vérins de manoeuvre du clapet est automatiquement coupé et ce dernier tombe ; l'aspiration cesse dans le cul-de-sac et l'éjecteur JA 200 aspire l'air de la voie de tête.

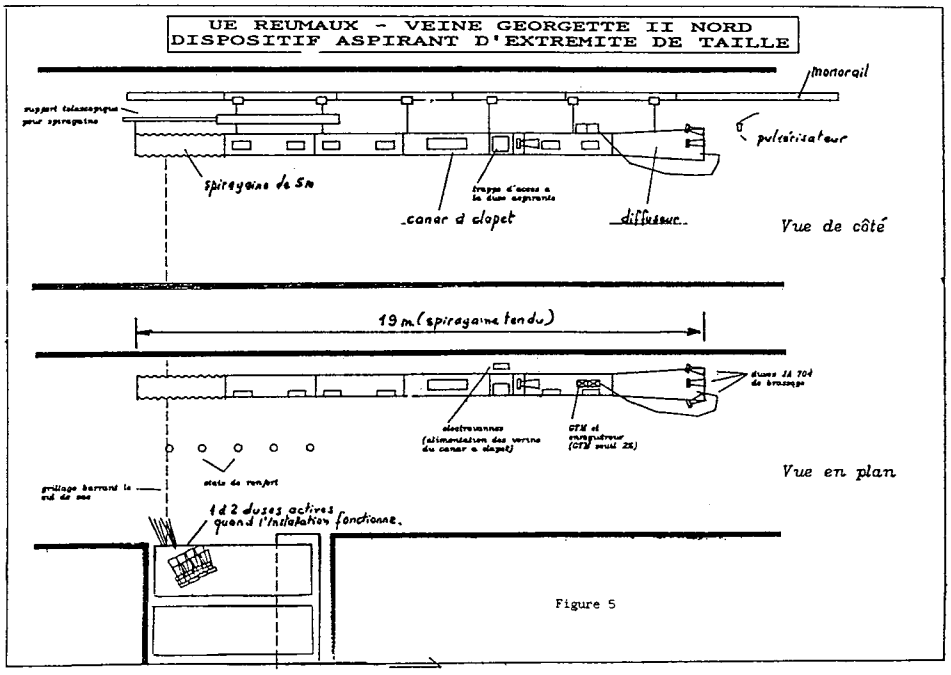


Figure 5

Un klaxon signale le déclenchement et ne peut être interrompu que sur intervention du personnel.

Une panne du GTM entraîne les mêmes effets qu'un dépassement du seuil fixé.

Une coupure de l'alimentation en air comprimé sur les vérins du clapet entraîne la chute de celui-ci.

La remise en marche de l'aspiration ne peut s'effectuer que manuellement, à partir d'une vanne d'alimentation en AC des vérins du volet, et avec une teneur dans la colonne inférieure à 2 %.

Avant démarrage de l'installation, le responsable mesure les teneurs au grisoumètre VM1 dans cinq sections différentes (figure 6) :

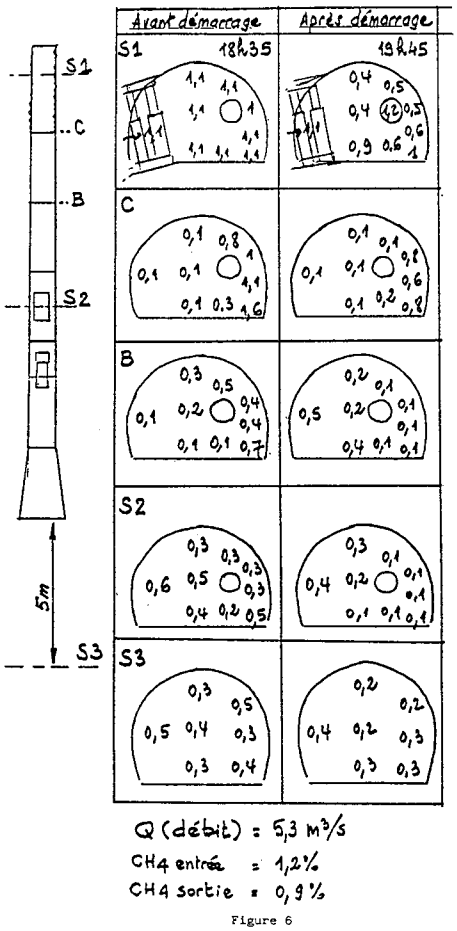


Figure 6

Après démarrage de l'installation, une nouvelle série de mesures de teneurs est réalisée.

Le responsable de l'installation mesure ensuite le débit aspiré au moyen d'un tube de Pitot, puis relève la pression d'air comprimé au manomètre de contrôle avant de noter la teneur à l'entrée et à la sortie du dispositif aspirant.

- Résultats de l'essai.

L'essai effectué a mis en évidence les avantages escomptés d'une telle méthode.

Au bout de 5 minutes de fonctionnement du dispositif aspirant, il est observé, comme le montre les cartographies de teneurs de la figure 7 :

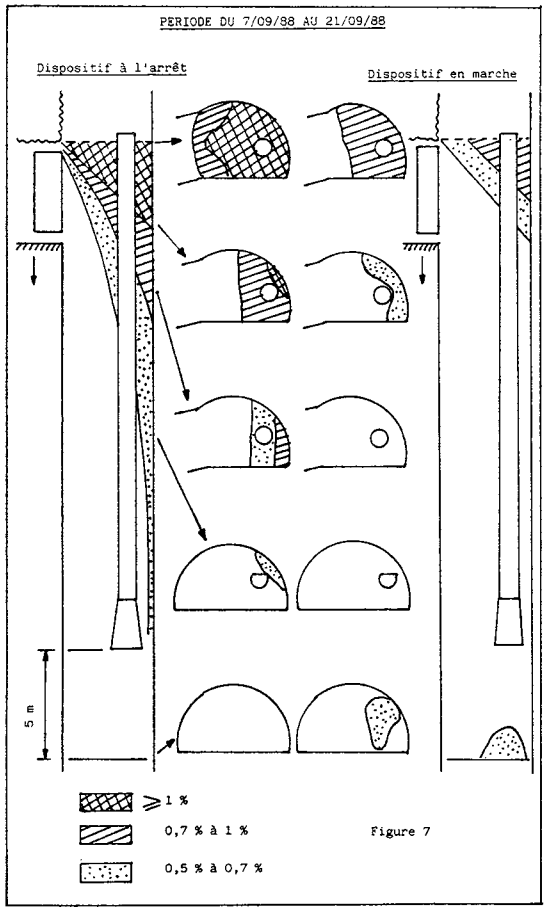


Figure 7

- une meilleure homogénéité de l'atmosphère au carrefour taille-voie,

- la suppression du rebrassage,

- la diminution des grandes irrégularités des teneurs, surtout au parement de la galerie opposé à la taille.

L'examen a permis de confirmer, de plus, les qualités de ce dispositif :

. pas de réglages délicats, instables ou improvisés,

. facilités de mise en oeuvre et de déplacement (suspension au monorail),

. dispositif silencieux,

. grande fiabilité.

CONCLUSIONS

En taille rabattante, une part importante du grisou pénètre dans l'aérage à proximité du carrefour entre la taille et la voie de retour. Un modèle de calcul simple est proposé pour dimensionner un dispositif de ventilation aspirante permettant de diluer les filets d'air à forte teneur de méthane. Les essais d'un nouveau prototype de dispositif aspirant ont montré l'intérêt de son emploi lorsque ses performances sont suffisantes. Du point de vue de la sécurité, la solution proposée est celle qui présente la plus faible probabilité d'inflammation du grisou, aussi bien à l'intérieur du dispositif (pas d'énergie électrique, pas de pièces en rotation) qu'à l'extérieur dans le chantier, lorsque l'aérage strictement rabattante est nécessaire.

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METHANGASBEHERRSCHUNG AM STREB-STRECKENÜBERGANG BEI STREBEN IM RÜCKBAU

C. Froger, D. Cwiklinski, J. P. Josien et al.

Heutzutage können unter guten Bedingungen hohe Fördermengen erzielt werden (5 bis 10 000 Tonnen sind normale Zahlen). Doch auch in nicht sehr gashaltigen Flözen erzeugen diese hohen Fördermengen eine erhebliche Ausgasung.

Verstärkte Bewetterung und Verbesserungen in der Effizienz der Gasvorabsaugung ermöglichten es, den Methananteil unterhalb der gesetzlich vorgeschriebenen Grenzwerte zu halten. Trotzdem bleibt die Verdünnung von Wetterströmen, welche mit einer hohen Methanbelastung aus dem Alten Mann kommen, ein Problem im Hinblick auf die Verhütung von lokalen Konzentrationsspitzen.

Die Studie stellt die Ergebnisse von Messungen vor, welche in verschiedenen Streben durchgeführt wurden, um den Methangasfluß und seine Verteilung zu erfassen. Es scheint, daß in Rückbaubetrieben zwei Drittel des Methangas-Flusses am Streb-Streckenübergang der Abwetterstrecke auftritt.

Die verschiedenen Maßnahmen, welche in Frankreich und in anderen Ländern angewandt werden, um dieses Problem zu lösen, werden am Beispiel eines Rückbaubetriebes beschrieben.

Ein auf dem Unterdruckprinzip beruhender 'Air-mover' wurde vorgeschlagen. Tests mit diesem Gerät in einer Übertageeinrichtung und untertage werden beschrieben.

РЕГУЛИРОВАНИЕ МЕТАНА НА ПЕРЕСЕЧЕНИИ ОЧИСТНОГО ЗАБОЯ С ВЫХОДНЫМ ШТРЕКОМ В ОБРАТНОЙ ВЫРАБОТКЕ

К. Крогер, Д. Цвиклинский, Дж.П. Джозиен и др.

В настоящее время при хороших условиях может быть достигнута высокая продуктивность в длинном забое (обычно от 5 до 10 тысяч тонн). Эта высокая добыча приводит к значительному выходу метана даже не в газоопасных угольных пластах.

Повышенная вентиляция и улучшение эффективности дренажа позволяют удерживать количество метана в дозволенных пределах. Несмотря на это, засорение потока воздуха высоким содержанием метана, поступающего из зоны обрушения, все еще представляет собой важную проблему, особенно при высоких локальных концентрациях этой смеси.

Настоящий доклад представляет результаты измерений в различных забоях с целью оценки потока метана и его перераспределения. Очевидно, что в отступающем длинном забое, две трети метанового потока обнаруживается на пересечении очистного и выходного забоев. Рассмотрены различные методы регулирования метана в отступающем длинном забое, которые используются во Франции и других странах для разрешения этой проблемы. Предлагается устройство для всасывания метана и описывается испытание этого устройства на поверхности и в подземной шахте.

交叉工作面——后退式长壁回风巷道中甲烷的控制

C． 弗滔格，D．切克灵斯基，J．P． 爵希恩，等

目前，在好的条件下长壁工作面采矿法可达很高产量（5 至 10．000 吨是常见的数字）。 如此高的产量，即使在非气性煤层中也可导致显著的甲烷外溢。

增进通风和改善排水能将回风巷中的甲烷浓度控制在允许浓度之下。但是用空气流来冲淡垮落带冒出的高浓度甲烷以避免局部气体浓度分布的不均匀还是个大问题。

本文提供不同工作面的检测结果，以估计溢流及其重新分布情形。在后退式长壁工作面，三分之二的甲烷似乎外溢在工作面与回路交截处。

本文在后退式长壁例子中叙述了法国及其他国家所用的解决此问题的方案。

本文建议使用一种空气流动式的吸气装置，并说明了这种装置的地面及井下测试。

CENTRALIZED PUMPING OF GAS FROM NEIGHBOURING MINING FIELDS BY STATIONARY DRILLING SITE

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ABSTRACT

It is a successful practice to pump gas from gobs by holes drilled in neighbouring workings at gob boundary when main part of gas comes from neighbouring coal seams. Gas has been successfully pumped from near faces infiltrated from neighbouring coal seams at Mines "Cheng Zi-he" where pillarless mining is operated. Pumped ratio of gas from mining fields --76%; productivity of pumped gas --17m^3/min.; density of pumped gas -- more than 60%. This practice has decreased work projects and overcome difficulty of hole-layout with pillarless mining.
The gas pumping principle and its technology are delt with on the basis of dynamics of fluids in porous media in the paper.

GENERAL SITUATION

At Cheng Zi-he Coal-Mine low, and medium-high seams have been mined with annual output 2.000.000 t of coal. Number order of these seams goes up down. There are 9 coal-seams (42, 37...4, 3...)devided into three groups (the upper, the middle and the lower ones). The main characteristics of these minable seams are shown in Table 1. the seams no. 1, 2and the seams situated between the minable seams are unminable or partionally unminable.
The Cheng Zi-he coal-mine has been mined for 50 years. Except for western part (west inclined shaft) the eastern part had reconstructed with vertical shafts for development.
The group of eastern inclined shafts had been changed into the regional airshafts. At the coal-mine there are five mining areas (i.e. West inclined shaft area, eastern 1st, eastern 2d, western 1st, western 2d areas beloging to the vertical shaft) The elevation of mining work by the vertical shaft is -400 m; the slope and plaine development system has been adopted, and twostaged winding system is used in the west inclined shaft. The development system is of stope-inclined shaft and main cross-cut. The elevation of mining work is -320 m. The elevation of the surface is 210 m in average. With increasing depth of mining work the absolute discharge of gas has reached 106m/min.

Table 1. Main characteristics of minable coal-seams

name of seam-group	no. of minable seams	Height of seams (m)	Inclination grd. (°)	distance between seams (m)
upper group	42	1.40	20°	96
	37	1.15	20°	20
middle group	36	1.60	20°	34
	29	2.15	20°	41
	25	1.40	20°	20
	24	1.50	20°	67
lower group	8	1.60	20°	50
	4	1.00	20°	30
	3	1.60	20°	

There are 8 mining faces, in it there are three complex-mechanized (with ranging drums-miners) .In order to match the needs of mining these three faces must be working along a certain coal-seam, which would lead to discharge of gas from neighbouring seams into the complex-mechanized faces. According to statistical data the absolute gas-discharge has reached 17 m^3/min. --43 m^3/min. The high gas-discharge has limitted average annual output--capacity of these complex-mechanized faces (annual output is only 370.000 t) . Thus the gas-pumping system has been set up at the mine to discharge the gas from these faces.

At the mine the long wall system along strike is adopted and the layout of workings consists in driving the drifts along the goaves with coal pillarless system of mining, i.e. the air drift of coal-faces is driven along the goaf of upper-neighbouring areas. Obviously, the utilization coefficient of drill-holes would be too low if gas was pumped from the neighbouring seams through the holes drilled in the air-drift along the goaf. The practice of neighbouring mines has shown the gas-discharge can reach 30-40% through the holes drilled from rock drifts driven along seam' roof as shown in Fig 1.

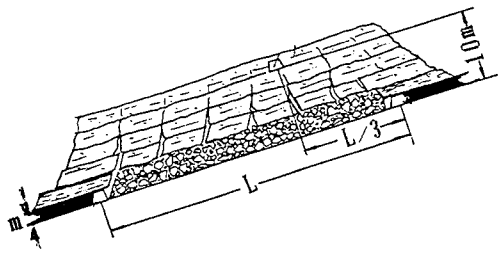


Fig.1 Gas-pumping from roof rock drift

But this pumping system has some shortcomings, difficult to determine the site of pumping drift (if it is located too low, the extra amount of air would be pumped in; if too high, the expected pumped output would be short) ;too much work is needed for driving of drifts in rock with high cost and low speed to drive rock drifts. So it can not meet the needs

of mining work. In order to get rid of these shortcomings and to keep the high recovery coefficient of the coal-pillarless system, the gas-pumping drift and pumping holes of most-upper coal-faces must be kept to pump out gas from the adjacent to each other seams, serving two or more faces at the same time, in place of the system of pumping gas from the roofrock drift.

PRINCIPLE OF MULTI-AREA GAS-PUMPING BY STATIONAL PUMPING DRIFT

The gas-pumping out of the adjucent coal-seams of several faces requires proper control of gas-flow in the goaf, lessening of dispersion of gas flow covering a long way, coming from adjucent seams and prevention from its dilution by leakaged air from the goaf. Therefore, it is needed to know if these conditions can be satisfied.

1. the direction of air-flow can be controlled in the goaf.

In the goaf filled with fallen stones there are a lot of pores among these waste-stones and the air flows in these pores following the vetilation current in the coal-faces. This flow can be dealt with on the basis of dynamics of fluids in porous media, i,e the air-flow would satisfy the conditions of conservation equation in a porous media and Darcy's Law. Introduction of Darcy's Law into the concervation equation gives us equation of motion of air in the goaf. If the air-motion in the goaf is simplified into a stable plane motion, the equation can be written,

$$Kx\frac{\partial^2 H}{\partial X^2} + Ky\frac{\partial^2 H}{\partial Y^2} + \varepsilon = 0 \quad \cdots\cdots(1)$$

Where Kx, Ky-x, y conductivity of directions or permeability

H - distribution function of total air pressure

ε - sources, sinks.

The differential equation (1) describes a common law of air-flow in the goaf. As the partial-differential equation has countless possible solutions, additional conditions must be given for solution to the equation in order to obtain the only solution from these countless possible ones that could describe the air-flow in a certain goaf. These conditions are called boundary conditions and the problem is called boundary value problem. In these additional conditions, first of all, the geometric configuration of a studied goaf is known, and the other parameters (for example, air conductivity Kx, Ky) are known. At the same time it is required to know the air-pressure distribution function at the boundary of goaf, i, e.,

$$H(x,y)\big|_r = H_0 \quad (x,y) \in r \quad \cdots(2)$$

This condition is also called boundary value problem of first kind or when the air-flow per unit length of the boundary of goaf is known it is called flow boundary condition or boundary value problem of second kind.

$$T\frac{\partial H}{\partial n}\Big|_r = q(x,y), \quad (x,y) \in r \quad \cdots(3)$$

When air-pressure distribution function is known of partional boundary of the goaf and the flow distribution is also known at other parts of the boundary, these conditions are called boundary value problem of mixed kind. In solving practical problems, only after determination of boundary conditions in the goaf, would the flow situation be the only one with

either boundary condition selected. On the contrary, the air-flow situation could be changed in the goaf according to the changed boundary conditions in the goaves to meet the practical needs of mining work.
As shown in Fig.2, when the 1st area was finished, the coal-pillarless system is used in the 2d area. Therefore the goaves of upper and lower areas have been united as one. As the air-pressure condition is known both in the face and along the goaf and there is no air-leakage in the goaf at other boundary, the air-pressure in the goaf appears ordinal distribution. $\cdots\cdots H_5 > H_6 > H_7 \cdots\cdots$ the air-flow direction is shown with ψ_0 ψ_1 、ψ_2 …… . If two air-leakage sinks are formed (the air leaks out of the goaf), the air-leakage amount $Q_1+Q_2=\psi_1-\psi_0$, the airflow situation would be changed (i.e.Fig.2 would be changed into Fig.3) .The gas will be drawn out through the air-leakage sinks Q_1, Q_2, leaked out from roof and floor and mixed with the gas in flow. Apparently, the gas-pumping is able to alter the air-flow amount at the boundary of the goaf, thus to alter the gasflow direction in the goaf only by means of pumping drilled holes. The direction and speed of gas-flow can be controlled.

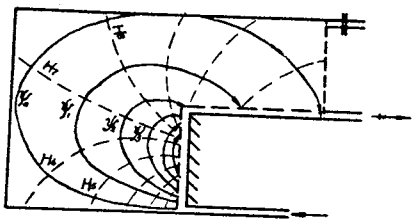


Fig.2 non-air-leakage at the upper boundary of the goaf

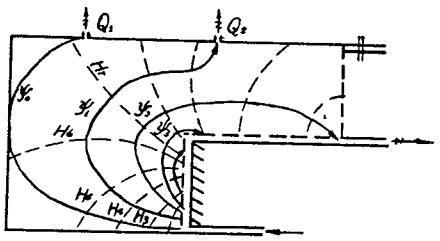


Fig.3 Air-leakage at the upper boundary of the goaf

2. the fissures for gas-flow

In order to study the law of roof-rock movement in faces a television device was used in the drill-holes at Da-Tun Coal-mining Bureau and a lot of observations were made. Observations show that in the process of face-advance the roof-rock had been caved slice by slice and block by block with very low expansion coefficient (1.06--1.10), so the broken stones caved from the roof could not fill the goaf completely, with its upper part in hanging state. The caving height at the first stage is seven times the excavation height. In the so called caved zone the arrangement of caved stones was apparently disorderly. The observations made at Da—Tun Mining Bureau show that the large area caving of roof at late stage happens only after the area of the goaf reaches 70.000—90.000 m^2 with increasing breakage upwards in the roof rock and fissures in rock layers. The summary height of caved stones can reach 18—20 times of the excavation height. The caved rock at late stage is so called broken zone with fissures. The parameters of caving observed by means of television device are shown in table 2.

Table 2. Parameters of roof-caving

site of observation	Height of mining m	no. of of holes	caving height at 1st stage m	unsupported height m	expansion coefficient	caving height at late stage m	summary caved height m
face no. 8305 of Yun Gang Mine	2.7	1	24.4	0.75	1.06	23.7	48.1
		2	19.7	0.43	1.10	30.3	51.0
		3	13.3	1.23	1.10	35.0	51.9
		average	19.1	0.83	1.09	29.7	50.3

The experiment by means of a model similar to rock movement in faces shows that in the zone disturbed with excavation and over it, the roof is being caved slice by slice and stage by stage, and there is always a hanging state in the upper part. The horizontal fissures in the broken fallen stones are greater than the vertical ones, i. e. the conductivity Kx, Ky in horizontal direction is much more than vertical one Kz. That is, the gas coming from the upper unloaded neighbouring seams can flow along the interlayer's fissures and the flow direction can be controlled by gas-pumping drill-holes. (fig 4)

The observations and experiments show a thriangle ABO of excaved zone (fig.5). After the neighbouring pillarless faces are excavated, the thriangle ABO of disturbed excavation zone would be violated and transformed into a new thriangle A'B O' of a new disturbed excavation zone.

When the seam is deposited shallow the faces are long, the top of the disturbed zone could reach the earth' surface forming a pan cousing surface movement. But once there is no air-leakage canals connecting the surface with the ground, the interlayers' fissures in lower broken area, forming a good gas-flowing canals.

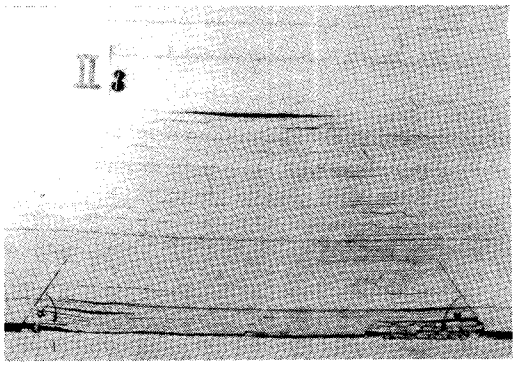


Fig. 4 Experiment with falling process in goaf.

3. The influence of places of gas-sources on pumping effectiveness

In mining of a group of coal-seams the gas of the first disturbed seam comes mainly from the neighbouring seams near to the floor and roof of the coal-seams being mined. After

unloading by excavation the neighbouring coal-seam would become the gas-sources for the goaf. The distance between the gas-sources and the seams being mined has great influence on the pumping jobs. The upper neighbouring coal-seams would be elapsed with the caving of the working faces. These gas-sources would lose their value to pump out, being deluted by the air-leakage flow from the goaf.

When the gas-sources are in the depths with advancing of coal-faces, the density of gas could be raised. But as the intensity of gas sources apparently decreases (gas-discharge per unit time) to lose its value of pumping out. Only when the gas sources of the goaf are lying in broken fissures or over the interlayers' fissures of disturbed zone offering good canals for gas discharge can the stational

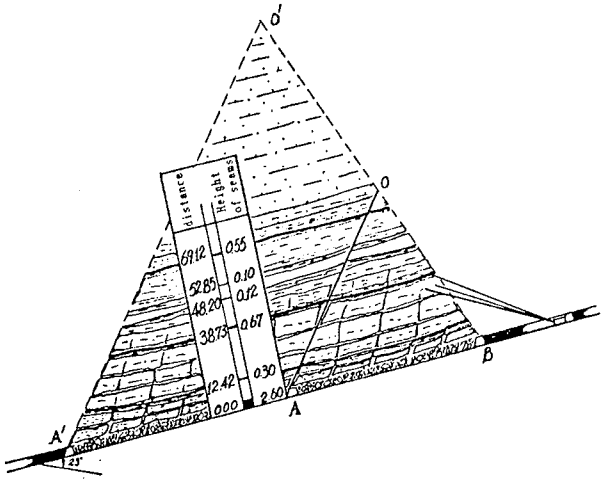


Fig.5 Forming of interlayers' fissures

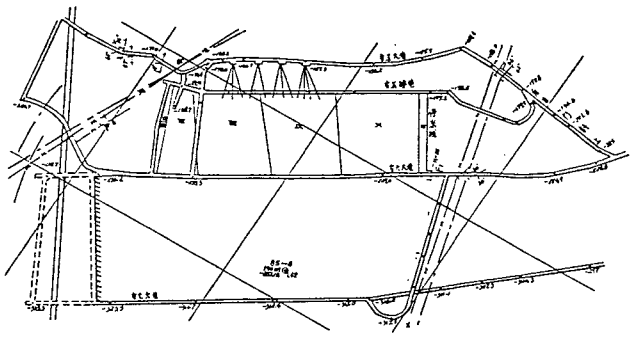


Fig.6 Plan of face Right 7

drilling site be used of upper area's boundary to pump out the gas from the neighbouring seams of lower areas. These favourable conditions exist at Chen Zi-he Mine everywhere.

CHOICE OF TEST-PLACES AND DRILL-HOLE ARRANGEMENT

In order to obtain the results of industrial test as fast as possibly, according to the idea of gas-pumping stated above, it is decided to carry out the test at an old goaf of the face Right 6 in the seam 36 A and faces Right 7. The face Right 6, long 110 m, was being mined from June to November 1985. Air-intaking drift was at elevation -245.7 m. with absolute gas discharge 13m^3/min. The face Right 7 to start is the neighbour to the goaf of the face Right 6, but is lower than it and coal-pillarless system is used. (fig.6) .The face is increased to 150 m and its depth is more by 67.4 m. The expected gas-discharge from the face Right 7 can reach 18m^3/min.or more in the whole process of excavation. It is the complex-mechanized face that works with the lowest gas-discharge of all mine. The reason this face has been chosen for test is that there is a dirft at the upper boundary of the goaves related to the faces Right 6 and Right 7 and the drift is communicated with a main air drift at -160 m. Once the gas-pumping test failed the communication drift could be used to discharge the gas from these goaves without interruption of mining work. To pump the gas from the seams near to the goaves of the face Right 7, holes must be drilled previously into the goaves of face Right 6, penetrating through the break line Bo (fig.5) of disturbed zone of the face Right 6 and ending to the distance 35—45m from the floor of the seams to be mined. There are 12 holes, their distance is 10m along strike with each hole communicated with interlayers' fissures.

REGULATION OF GAS-FLOW IN GOAF AND RAISING THE COEFFICIENT OF GAS-PUMPING

The face Right 7 has been excavated since. Mart 18. 1988. Before initial caving there was only about 4m^3/min. of gas in the air-flow. With advancing of the faces the amount of gas increases. As the open-off cut in the face Right 7 lags behind the one in the face Right 6 and these two goaves have not been communicated the pumping system can't work tempolarily. The gas is discharged from the goaves only by connecting drift at-240m. After communication of these two neighbouring goaves the amount of gas being pumped increases (the weekly gas-amount pumped out increases from 36% to 40%) .The ventilation is good and the air-flow in the goaf is shown in fig.7. As the communication is widened between goaves and activities of roof rock increase the gas-discharge is raised. The amount of gas exceeded the gas-limits in the air-flow April 23. Three measures were taken, (1) to increase the air-supply amount to 915m^3/min. (2) to raise the pumping depression for increasing of pumped out gas. (3) to lead the gas from the goaf into the main air-flow along the communication drift at -160m. Although the amount of gas pumped out increases and the problem has been solved of

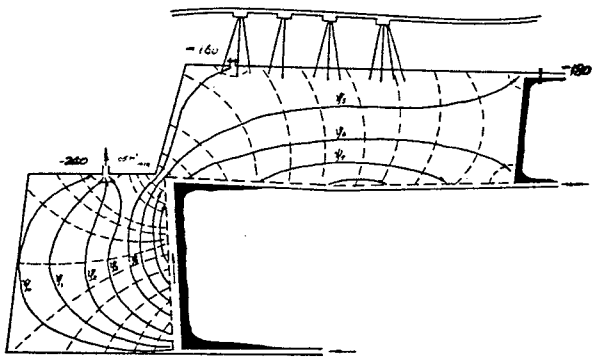


Fig.7 Initional situation of communication in goaf

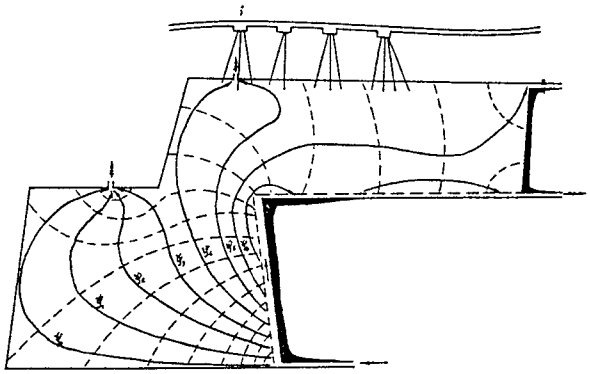


Fig.8 Situation in goaf after communication

exceeding gas-limits, the coefficient of pumping of gas falls from 41% to 31% (ventilation and pumping parameters are shown in table 3) .
At that time the ventilation and the pumping system met the needs of mining work, but raising of the coefficient of pumping is the key problem to be solved in this test. So the following measures have been taken, (1) increasing of pumping depression to raise the amount of pumped gas; (2) To bulkhead the air drift of face Right 6 and to connect it with pumping pipe; (3) to close the communication drift at -160m and -240m in order to decrease gas-leakage from the goaf and increase the density of gas with raised amount of pumped gas; (4) in the air-drift there is only 0.3% of gas density (there is large room in the limits of Safety Regulations) .
It ought to lower the amount of air in faces and to lessen the air-leakage from goaves in order to raise the gas density pumped out.
All of these measures have carried out except for the bulkhead in the drift -240 because of the needs for material haulage. According to the results monitored. April 29, intaking air in faces falls from 708-530m^3/min. the coefficient of pumping in goaves raises from 29.5% to 61%. The air flow is shown in fig.8 after regulation. Practice shows that the proper regulation of the boundary conditions in goaves is necessary and effective to raise the coefficient of pumping.

Monitored data of ventilation and gas-pumping

Tab.3.

data			4.13	4.20	4.23	4.29	5.1	5.17	6.4	7.1	7.22
air-intake		m^3/min	780	780	919	660	660	720	660	660	790
outgoing	Q	m^3/min	648	642	708	530	540	587	535	530	660
	CH_4	%	0.5	0.5	0.4	0.4	0.5	0.5	0.5	0.4	0.3
-240	Q	m^3/min	132	138	139	130	120	133	125	130	130
	CH_4	%	2.7	2.6	2.6	2.9	2.9	2.9	2.6	2.9	2.7
-160	Q	m^3/min			72						
	CH_4	%			11						
gas-discharge		m^3/min	10.58	11.42	20.32	13.85	24.19	20.59	20.44	19.43	18.63
amount of pumped gas at drilling site	I	1	0.64	0.71	1.05	1.17	2.80	2.17	3.09	2.42	3.51
		2	0.44	0.65	1.85	0.87	1.95	2.17	2.54	2.32	2.53
		3	0.74	0.41	0.61	0.97	2.25	1.6	1.77	1.61	2.36
	II	1	0.41	0.52	0.78	0.98	1.58	0.77	0.51	1.09	0.84
		2	0.26	0.47	0.35	0.40	1.42	0.71	0.91	0.52	0.76
	III	1	0.41	0.69	0.84	0.69	2.01	1.66	1.89	1.35	0.60
		2	0.52	0.77	0.42	0.92	2.15	1.25	1.30	1.82	0.84
		3	0.36	0.41	0.40	0.76	0.49	0.94	1.01	0.85	0.88
	IV	1						1.25	0.24	0.39	0.29
		2									
		3							1.03	0.87	0.32
m^3/min		4						0.20	0.23	0.30	0.30
	-180	1				1.30	1.67	0.49			
		2				0.40	1.69	0.58			
(1)		m^3/min	3.78	4.63	6.30	7.96	18.01	13.79	14.52	13.54	13.23
(2)		%	36	41	31	58	75	68	71	70	71

note: (1) Q of pumped gas from arear
(2) pumping coefficent

CONCLUSION

In the face Right 7 in coal-seam 36 A at Chen Zhi-he Mine, the excavation height is 2.6m, face length is 150m, the strike length is 375m with gas-discharge 24.19m /min, in it 80% coming from the neighbouring seams. The gas-pumping work is carried out through the holes at the upper boundary of goaf of face Right 6. There are 12 holes. After regulation the coefficient of pumping is max. 76% and min. 61%. The time (day) weighed average coefficient is 72%. In this period the gas limit exceeding did not occur. Owing to the success in the test the recorvery of the face is raised by 6%. With elimination of gas-discharge drift the drift-driving work is less by 20%. Up to now the former holes have still been used for gas-pumping from the face Right-8 of lower goaf of face Right 7. With the new success the technological and economical indexes would be improved in the future. On the basis of the experience available so far, this gas-pumping system can only be applied to the conditions, With coal-pillarless faces the location of the upper neighbouring coal-seams is 10 times higher than its excavation height.

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POMPAGE CENTRALISE DU GAZ PAR DES SITES DE FORAGE FIXES SITUES DANS LES CHAMPS D'EXTRACTION AVOISINANTS

Huan Bo-Xiang

La méthode consistant à pomper du gaz par des trous forés dans les chantiers avoisinants, aux limites des remblais, lorsque la majeure partie de ce gaz provient de couches de charbon voisines, donne de bons résultats. Dans les mines "Chen Zhi-he", où l'exploitation s'effectue sans piliers, on a pompé du gaz des fronts de taille qui s'était infiltré par des couches de charbon avoisinantes. Taux de gaz pompé des exploitattions minières -- 76%; productivité du gaz pompé -- $17m^3/mn$; densité du gaz pompé -- supérieure à 60%. Cette méthode réduit l'envergure des travaux et permet de surmonter les difficultés d'aménagement des orifices dans les exploitations sans piliers.

Notre rapport examine le principe de pompage du gaz et sa technologie, sur la base de la dynamique des fluides en milieux poreux.

ZENTRALE GASABSAUGUNG VON BENACHBARTEN GRUBENFELDERN DURCH EINE STATIONÄRE BOHRSTELLE.

Huan Bo - Xiang

Die Methode, Gas aus Bruchfeldern durch in benachbarte Baufelder gebohrte Baulöcher abzupumpen, ist erfolgreich, wenn der Hauptanteil des Gases aus benachbarten Flözen kommt. Aus benachbarten Gruben eingedrungenes Gas in der 'Chen Zhie - he' Grube, wo ohne Pfeiler gearbeitet wird, wurde bereits erfolgreich aus Streben abgesaugt. Das Pumpverhältnis des Gases aus den Baufeldern beträgt 76%, die Ergiebigkeit beträgt $17m^3$/Min., die Reinheit des gepumpten Gases beträgt mehr als 60%. Diese Methode hat das Arbeits-Aufkommen reduziert und Probleme der Bohrlochanordnung in pfeilerlosen Sektionen gelöst.

Der Vortrag behandelt das Pumpenprinzip und seine Technologie auf der Basis der Dynamik von Flüssigkeiten in porösen Medien.

ЦЕНТРАЛИЗОВАННАЯ ОТКАЧКА ГАЗА ИЗ СОСЕДНИХ ВЫРАБОТОК ПОСРЕДСТВОМ СТАЦИОНАРНОГО БУРЕНИЯ

Хуан Бо-Зянг, КНР

Практика откачки газа через скважины, пробуренные в соседних пустых выработках, успешно зарекомендовала себя в случаях, когда основная часть газа поступает из соседних угольных пластов. Газ, проникший из соседних угольных пластов в шахте "Чен-Жи-Хе", где производится безопорная добыча, был успешно откачен из близлежащих забоев. Процентное содержание газа из пустых выработок - 76%; производительность откаченного газа - 17 куб.м\мин; плотность откаченного газа - более 60%. Этот метод уменьшает затраты труда и облегчает планирование буровых скважин в безопорных шахтах. Настоящий доклад рассматривает теорию и практику откачки газа, основанные на динамике жидкости в пористой среде.

中心化抽吸用于排除周围矿井渗入固定钻矿场的煤气

宦博祥 （译音）

实践证明当煤气主要来源于邻近煤层情况下，从邻近废石边缘打的孔里抽出煤气的方法行之十分有效。在"陈址河"无柱式煤矿，曾用此法将从邻近煤层中渗进的煤气由近工作面泵出。煤区渗入气体泵出率为 76%，排气速率为每分钟 17 立方米，排出气体的浓度高于 60%。此法减轻了工作规划，并克服了孔眼排列的困难。

本文根据多孔介质内流体动力学，阐述了抽吸排气的原理及技术。

DEPENDANCE OF METHANE CONTENTS IN BROWN COAL AND LIGNITE SEAMS ON DEPTH OF OCCURRENCE AND NATURAL CONDITIONS

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ABSTRACT

This paper presents the results of research on influence of depth of occurrence and physical-chemical properties of brown coals and lignites on methane contents in coal seams. The forecast on increase in methane content with depth over the period of next ten years is given.

INTRODUCTION

Investigation of the mine gas geology, dynamics and relations in coal seams is subject to numerous studies and analyses. In countries with a developed mining industry, these investigations are proportional to dangers and catastrophes due to presence of explosive gases in underground mines.

The need for a timely recognition of gas influence on future exploitation effects in the conditions of Yugoslav deposits is also imposed by the fact that the problems regarding the gas contents in conditions of Miocene and Pliocene brown coal and lignite deposits are investigated in the world to a very small extent.

The numerous site and laboratory tests on methane contents and other factors of influence were made on the selected specific and significant locations in brown coal and lignite deposits. An analysis was made of the influence of seam depth, (H), gas pressure (p), temperature of rock massif, thickness of coalseams (m), ash contents (A), humidity (w) and pure coal substance (pcs).

BASIC DATA ON MIOCENE AND PLIOCENE COAL DEPOSITS

The Middle Bosnia Basin features Upper Miocene deposits and a large number of coal seams. The basin area is app. 900 km^2. The productive coal zone has 9 seams with thicknesses ranging from 2 to 26 m (Figure 1). The coal seams occur at depths over 1,000 m, whereas their exploitation takes place at a depth of app. 650 m from the surface. The whole basin is divided in four regions with 10 underground and 6 open-pit mines. With its carbonization-degree values (from SC=63 in Bila Mine to SC=74 in Kakanj Mine), and according to the current world standards, the coal falls into a group of hard, bright brown coals (Kakanj and Zenica regions) or hard, matt brown coals (Breza region), with a transition to stone coal (Modrinje region).

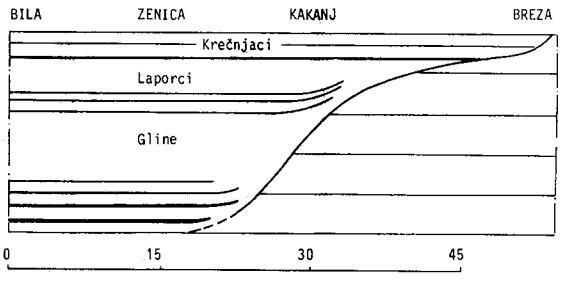


Fig. 1. Cross section through coal series

The quality of coal varies in both the horizontal and the vertical spread (Table 1).

The Kreka basin is among the largest Pliocene coal deposits. The coal is of good quality, with a relatively low carbonization degree - 50 to 61 at the depth of 500 m (Figure 2).

The lignite deposit features a uniform coal quality by seams, as shown in Table 2.

The northern part of the coal basin is app. 600 m deep. Working of the seams takes place at a depth of 230 m from the surface.

DEPENDANCE OF METHANE CONTENT ON DEPTH OF COALSEAMS

The geological structure of the Middle-Bosnian Pliocene brown-coal seam frequently necessitates a simultaneous working of several seams. A common trend for a general assessment of increase in methane with depth has been determined in the form of equation:

$$MN_m = f(H) \qquad \text{(Figure 3)}$$

$$MN_m = 0.76056 \cdot 1.38935"$$

The lignite deposit is characterized by simultaneous working of several coalseams. The research established a functional dependance of methane content on depth of the coalseams in the following form (Figure 4):

$$MN_p = 58.11875/H + 0.76551$$

DEPENDANCE OF METHANE CONTENT ON OTHER FACTORS OF INFLUENCE

A single-factor analysis covered a research into dependance of methane content on other factors of influence made "in situ" and laboratory conditions expressed in the form:

$$y = f(x)$$

where y and x are the parameters of depth (H), gas pressure (p) massif temperature (T), thickness (m), content of inorganic matter in the coal (A), humidity of the coal (w), and pure coal substance (pcs).

The influence of individual factors on methane content value is determined on the basis of regression coefficient.

The following laws were established by investigating mutual influence of the parameters for individual seams and basin as a whole:

(a) The least dependance is found in the function MN = f (p). For all seams of the miocene basin, and for the basin as a whole, the pressure comes last among the factors of influence. For the lignite basin, the pressure (p) comes in the fourth place. A general estimate for both basins shows that the methane content is least dependant on the gas pressure.

(b) The highest dependance of the methane content is observed in the function MN = f(T) and MN = f (pcs) for all tested coalseams and basins as a whole.

The multi-factor analysis took into account several independently variable quantities-factors of influence. A multiple effect of certain factors on methane content was determined in the analyses.

The rating of factors of influence on the methane content in coal seams and basins is determined on through Ki -square, on the "weight account" basis. According to the 2-factor analysis, the methane content is most dependant on the pressure, thickness, temperature and ash content, and least dependant on depth of a coalseam (Table 3).

The results of a mathematical analysis of the investigated parameters shows that the length, which is equivalent to most factors of influence (e.g. pressure, temperature, carbonization degree, etc.) comes last in the rating and cannot provide a reliability of theoretically obtained parameters

Table 1: Quality of coal in the Miocene basin, by seams

Coalseam	Humidity	Ash	Combustible	Volatile
Roof seam	13.12	20.26	67.67	33.29
Main coalseam	9.46	22.30	67.94	34.97
I floorseam	13.53	20.14	69.43	32.55
II floorseam	11.97	24.16	53.92	32.84
III floorseam	16.20	29.31	58.59	33.45
Orasi seam	2.32	41.39	56.29	27.17
IV floorseam	10.54	20.20	69.26	31.23
V floorseam	14.11	19.45	66.43	30.30
VI floorseam	16.58	31.25	51.99	26.94
VII floorseam	12.19	36.57	63.11	26.80
Average in basin	12.00	26.39	62.36	30.96

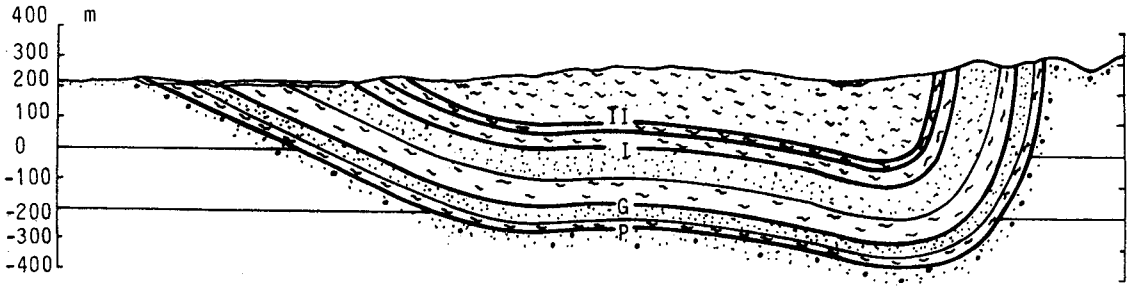


Fig. 2. Cross section through lignite deposit

Table 2: Quality of coal in the Pliocene basin

Quality parameters	II roof seam	I roof seam	Main coal seam	Floor seam	Average
Humiditiy	42.45	42.08	37.28	23.86	39.14
Ash	7.08	7.66	8.23	9.83	8.12
Volatile	27.72	26.93	28.12	29.11	27.91
C-fix	22.37	13.16	26.28	26.17	24.34
Min. heat value, in kJ	11,715	12,343	13,390	12,970	12,760

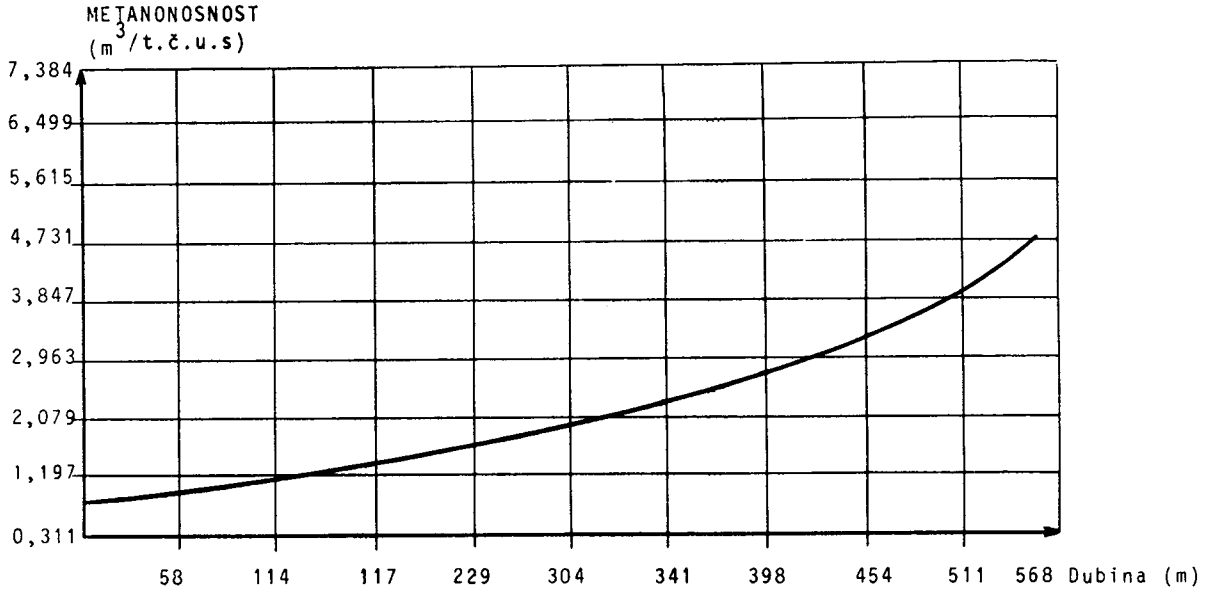


Fig. 3. General trend of dependance of methane content on depth for The Middle Bosnia miocene coal deposit

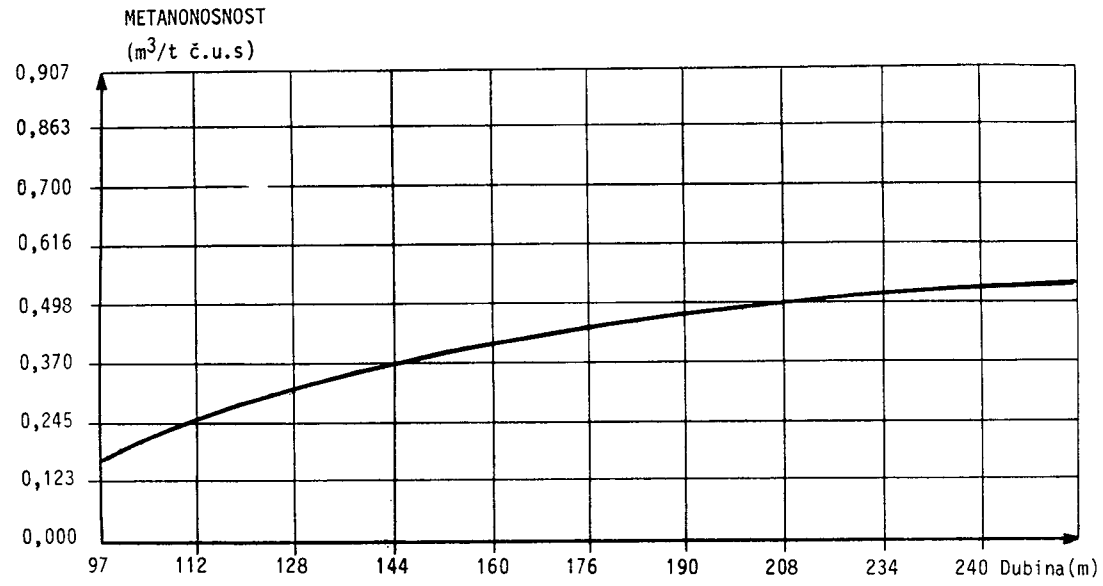


Fig. 4. General trend of dependance of methane content on depth of lignite seams

Table 3: Rating of the factors of influence in the 1- and 2- factor analysis

Influencing rating	BASIN				Both Basins	
	Kreka		Middle Bosnia			
	f(x)	f(x, y)	f(x)	f(x, y)	f(x)	f(x, y)
1	T	P	T	P	T	P
2	pcs	A	m	m	pcs	m
3	W	T	pcs	T	m	T
4	p	H	H	A	V	A
5	m	pcs	A	W	H	W
6	A	W	W	pcs	A	pcs
7	H	m	p	H	p	H

with empirical values of methane content.

High in the rating is the influence of inorganic matter (A) in coal. All multifactor analyses of pairs with ash point to the high probability of a reliable theoretical model with the empirical values of methane content. The average regression coefficient values for ash in both investigated basins are rated as fourth.

A particularly indicative data is the result of the 2-factor regressive dependance analysis:

$$MN = F(p, A)$$

The comparison of methane content values obtained through design of the dependance graphs to the empirical data showed that the difference between the theoretical and empirical data is less than $\pm$ 10% only for the function $\overline{MN} = f(p, A)$.

Comparing the results of the 1- and 2-factor analyses of methane dependance on ash, or pressure, one comes to the conclusion that ash acts as a catalyst to pressure, i.e. to methane content of the coal. This means that mineral inclusions may encourage or impede the specific reactions, i.e. act as natural catalysts.

Starting from the hypothesis on catalytic features of ash in natural conditions, the ash in the coal-humidity-gas system may affect the gas dynamics of a coal. An uneven distribution of ash in a seam may result in various gas capacities and pressures in different locations in the seam.

The explanation of the gas pressure and capacity variation is realistic and logical in respect to the catalytic action of ash in natural conditions, and this has been confirmed by our research.

The 3-factor analysis takes into account the influence of several independently variable quantities on methane content. The analysis results confirmed the assumption the interrelations of pressure (p), temperature (T) and ash (A) are substantial for the methane content in coalseams.

The 3-factor analysis has been made for all possible combinations of the factors.

POSSIBILITIES TO FORECAST THE METHANE CONTENT IN BROWN COAL AND LIGNITE SEAMS

The methane forecast charts for both deposits have been made by using the theoretical functions of dependance $MN = f(H, p)$ and $MN = f(p, A)$. The underlying assumption was that in the next 8 to 10 years the production will reach the depth of 800 m in the miocene, and 250 m in the pliocene deposit.

The desorption gas pressure increases with depth.

The Figures 5, 6, 7, and 8 give forecast curves for methane content in dependance of depth and pressure,

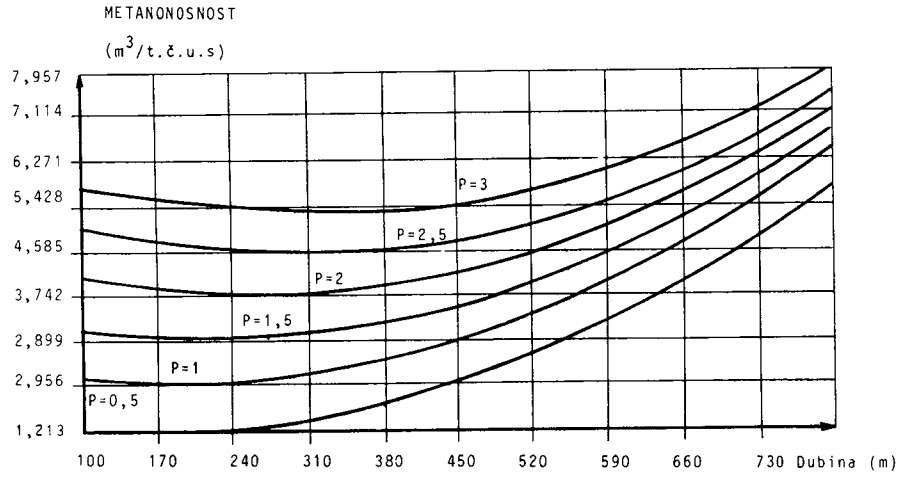


Fig. 5. 2-factor regressive analysis of dependance of methane content in coalseams = f(depth, pressure) Middle Bosnia Basin

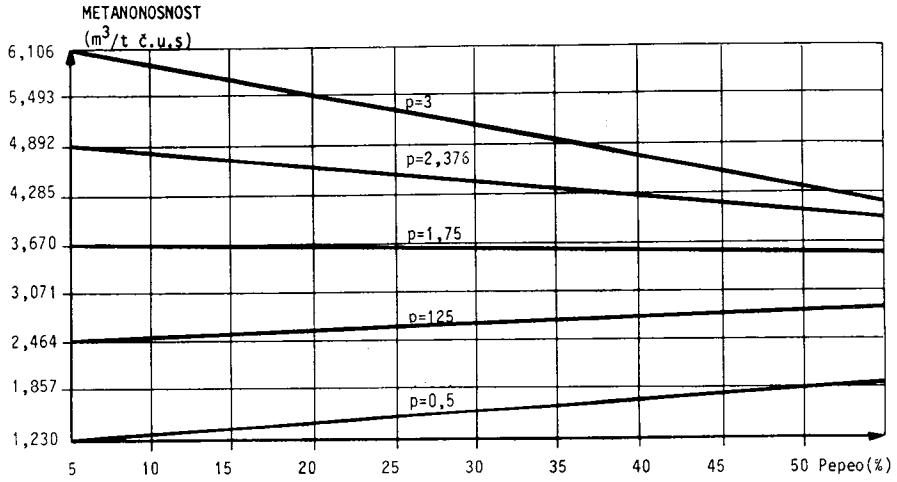


Fig. 6. 2-factor regressive analysis of dependance of methane content in coalseams = f(ash, pressure) Middle Bosnia Basin

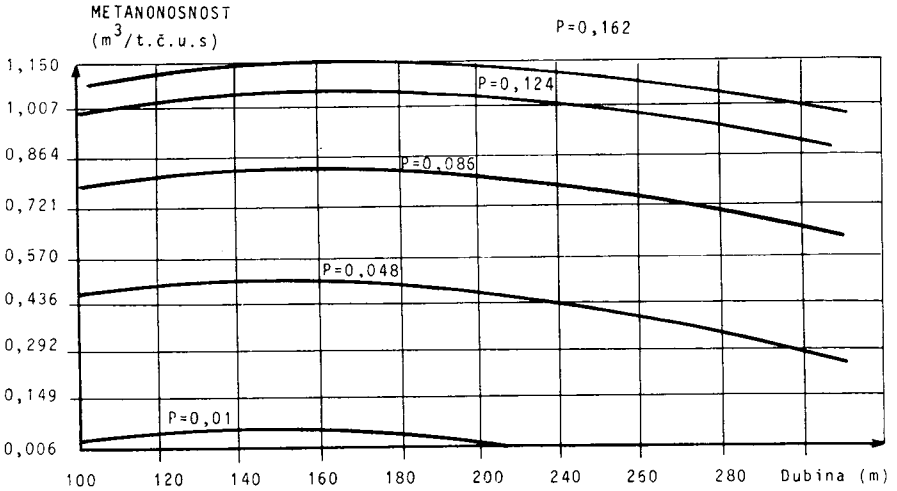


Fig. 7. 2-factor regressive analysis of dependance of methane content in coalseams = f(depth, pressure) Kreka Basin

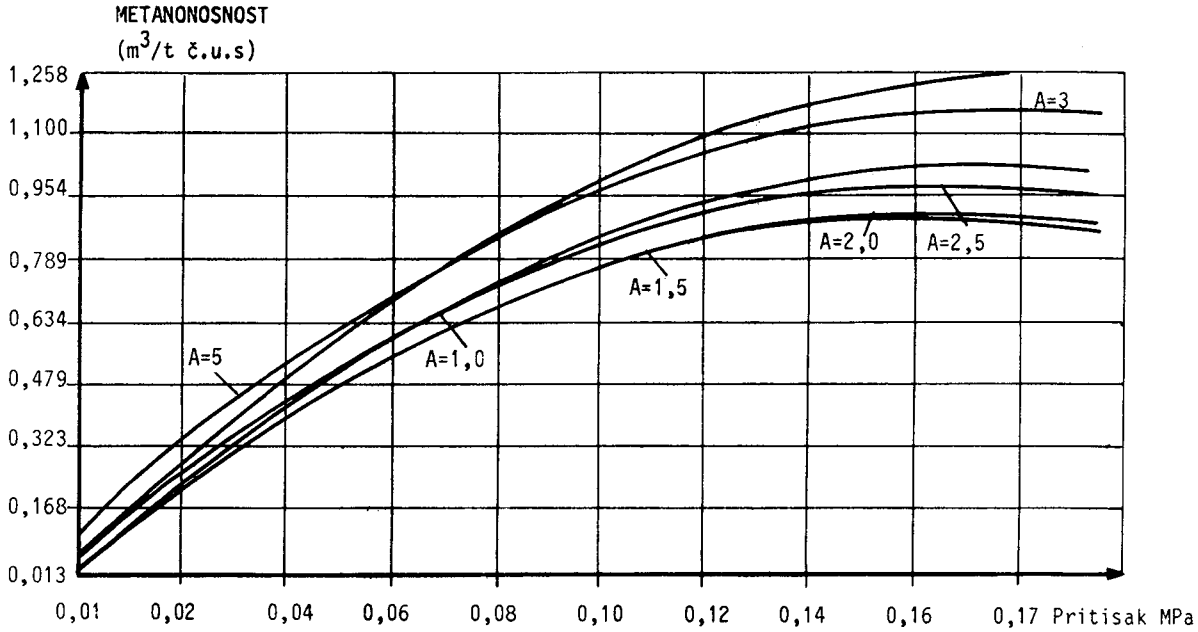


Fig. 8. 2-factor regressive analysis of dependance of methane content in coalseams = f(pressure, ask) Kreka Basin

or pressure and content of inorganic matter.

As may be seen in the Figure 5 above, the forecast methane content would be app. 8 (m^3CH_4/t pcs) with the increase in depth, i.e. pressure, to 800 m in the Middle Bosnia brown coal basin.

The gas pressure with depth of the miocene basin, whereas the carbonization degree rises with reduction of the inorganic matter, the Figure 6 shows that the methane content in a coalseam rises with a reduction of ash content and increase in pressure.

The methane forecast chart for future production depths in the Pliocene lignite deposit (Figure 7) shows that the maximum methane content is present at the depths of 200 to 220 m. The content drops with an increase in depth. The dependance curves for pressure and ash show almost the identical trend.

CONCLUSION

The presented material analyses and explanations of research in brown coal and lignite deposits point to the following conclusions:

- Regardless of the development degree of mining works, each coal basin must make a regional forecast of dangers from methane.
- When real natural properties of factors of influence to the methane content are known, it is necessary to determine a multi-factorial dependance.
- Investigation of influence of mineral inclusions in coal provides a hypothesis on catalytic effect of ash on methane content in a coalseam.
- The methane content curves of sufficient accuracy may be obtained by using theoretical functions of dependance Mn = f(H, p) and MN = f(p, A).

The last research was made at depths over 650 m (brown coal) and 250 m (lignite), and showed that the difference between the theoretical and empirical data was less than $\pm$ 10%.

RAPPORTS ENTRE LA TENEUR EN METHANE DU CHARBON BRUN ET DES COUCHES DE LIGNITE ET LA PROFONDEUR DES GISEMENTS ET AUTRES CONDITIONS NATURELLES

Abdulah Basic, Milutin Vukic

En Yougoslavie, la teneur en gaz du charbon brun et des couches de lignite devient un facteur de plus en plus décisif en matière de sélection d'ouvertures de mines et des systèmes et technologies de développement.

Nous avons examiné, à l'aide de modèles de structures sélectionnées, l'influence de la profondeur des gisements et d'autres conditions naturelles sur la teneur en méthane.

Les résultats de ces études ont fourni des données sur l'influence de plusieurs variables indépendantes, telles que la profondeur du gisement, la pression des gaz, la température à l'intérieur du massif, l'épaisseur, ainsi que la teneur en humidité des cendres et la teneur en charbon pur, sur la teneur en méthane des couches de charbon.

Nous avons obtenu des fonctions appropriées permettant d'évaluer de manière réaliste la teneur en méthane des couches de charbons bruns et de lignite. Une hypothèse de l'effet catalytique des cendres sur la teneur en méthane a pu être ainsi justifiée.

Les travaux de recherches ont également porté sur la nécessité et la possibilité d'introduire, de distribuer et de contrôler l'écoulement d'air par des portes de ventilation automatiques et d'autres structures contrôlées par un ordinateur situé dans un centre d'expédition.

ABHÄNGIGKEIT DES METHANGASGEHALTES IN BRAUNKOHLE UND LIGNITFLÖZEN VON DER TIEFE DER LAGER UND ANDEREN NATÜRLICHEN BEDINGUNGEN.

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Der Gasgehalt in jugoslawischen Braunkohle- und Lignitflözen spielt eine mehr und mehr entscheidende Rolle bei der Auswahl von Grubenentwicklungen und den dabei angewendeten Systemen und Technologien.

Der Einfluß der Tiefe der Lagerstätte und anderer natürlicher Bedingungen auf den Methangehalt sind in ausgewählten Modellstrukturen untersucht worden.

Die Untersuchungsergebnisse zeigen den Einfluß mehrerer unabhängiger Variablen, z. B.: Tiefe der Lagerstätte, Gasdruck, Gebirgstemperatur, Mächtigkeit und Feuchtigskeitsgehalt in der Asche, sowie reine Kohlegehalte, auf den Methangasgehalt der Flöze. Entsprechende Funktionen für die realistische Bewertung des Methangas-Gehaltes in Braunkohle- und Lignitflözen wurden entwickelt, und die Hypothese über den katalytischen Effekt der Asche auf den Methangasgehalt wurde bestätigt.

Die Untersuchungen umfaßten ebenfalls die Notwendigkeit und die Möglichkeit, den Wetterstrom durch automatische Wettertüren und andere Wetterbauwerke, die durch einen Computer in der Grubenwarte gesteuert werden, in die Grube einzuführen, zu kontrollieren und zu verteilen.

ЗАВИСИМОСТЬ СОДЕРЖАНИЯ МЕТАНА В БУРОМ УГЛЕ И ЛИГНИТОВЫХ ПЛАСТАХ ОТ ГЛУБИНЫ НАХОЖДЕНИЯ И ДРУГИХ ПРИРОДНЫХ УСЛОВИЙ

Абдула Башич и Милутин Вушич

Содержание газа в югославских буроугольных и лигнитовых пластах становится все более решающим фактором в выборе места открытия шахт и разработке технологических процессов. Влияние глубины нахождения и других природных условий на содержание метана исследовалось в некоторых выборочных "модельных" структурах.

Результатами исследований являются данные о влиянии таких независимых переменных, как глубина нахождения, давление газа, температура массива, мощность пласта, коэффициент влажности золы и содержание чистого угля на содержание метана в угольных пластах. Обсуждаются методы реалистической оценки содержания метана в буроугольных и лигнитовых пластах, а также гипотеза каталитического воздействия золы на содержание метана.

褐煤及褐煤层中甲烷含量与地层深度和其他自然条件的关系

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在南斯拉夫，褐煤和褐煤层中的甲烷含量已逐渐成为选择矿场，开发系统和技术的决定因素．

本文对几个挑选出的模型结构，探讨了矿床深度和其他自然条件对甲烷含量的影响．

研究结果提供了地层深度，气压，地块温度，地层厚度，灰分湿度，煤层中纯煤含量等独立变量与煤层中甲烷含量之间关系的数据．

本文建立了一个实地评估褐煤和褐煤层中的甲烷含量的函数，并对灰分与甲烷含量的催化作用的假设进行矫正．

本文也论及了使用由调度室内计算机控制的自动风门和其他装置的必要性和可能性．

SIGNIFICANCE OF LOCAL-EXHAUST VENTILATION SYSTEM IN MINES AND A STUDY OF THE WATER TURBINE POWERED NON-PYROPHORIC LIGHT-ALLOY EXHAUST FAN

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ABSTRACT

Following a general discussion of the signigicance of underground local-exhaust ventilation system in coal mines, the paper goes on to elaborate with particular emphases on the newly-developed water turbine driven non-pyrophoric exhaust fan made of cast light-alloy. Further improvement was made on the chemical constitution of the cast light-alloy. Temperature measurement of the thermal field due to collision and friction with foreign particles in operation and, in particular, the running tests of the prototype exhaust fan under high methane concentration(7-10%) conditions well proved the safety and reliability of the fan.

Through trial application for high-concentration(5-10%) gas drainage for a period of 180 days in a coal heading and an upper air-return entry of the coal face at hydromines, an average daily gas drainage of 5400 m^3 was registered. The fan is proved to be a new recruiting device capable of operating safely and reliably in coal mines.

GENERAL

Currently, local ventilation of China's coal mines underground is exclusively of forced ventilation by using the electric fan. Practice shows this method suffers the following drawbacks: 1) The longer the ventilation distance, the longer time required for diluting the methane and blasting fume at the face; 2) Dusting problem will arise at the outtake when the ventilation distance is short; 3) Unhealthy environment due to foul air in the heading; 4) Large volume of air required to dilute the methane to a concentration of 1%.

The concentration of methane in the winning heading is often found to exceed the upper limit because of the large instantaneous gas outburst caused by concentrated hydromining operations carried out at the gassy faces. The driving rates of both the conventional and hydraulic heading faces are also impeded because the gas is incapable of being timely drained after blasting operations. The longwall faces also experience severe accumulation of methane at the corners. The ventilation of these places with local accumulation of high-density methane by using solely the forced ventilation method can not get the result as expected. With an aim to improving the ventilation effect, shortening the time required for methane and blasting fume drainage, avoiding any local accumulation of methane and improving the working conditions in heading operations, a special local-exhaust ventilation system capable of draining high-concentration methane has been developed to effectively tackle the problems encountered in underground coal mines.

THE LOCAL-EXHAUST FAN AND ANALYSIS OF ITS MAIN TECHNICAL ASPECTS

The power source as well as the materials for making the impeller and housing are technically two crucial aspects involved in the study of the local-exhaust fan for high-concentration methane(> 1%) drainage.

Power Source

Currently the local fan commonly used in China's underground coal mines operates with the explosion-proof electric motor as its power source. However the fan of this kind is not allowed to operate in air stream with a gas concentration greater than 0.5%, if a special isolation is not provided, because of its susceptibility to electrical failure and generation of electrical spark. However due to the availability of static water in both concentional and hydraulic coal mines (Otherwise special water pumps can be installed), the pressure water should be used as the power source of local-exhaust fan to avoid the menace imposed by electric spark. Of course, compressed air may be considered as another alternative source.

Materials Used

Of the local electrical fan currently used underground, the housing and impeller are made of steel plate and cast aluminium-silicon(ZL-108). or

aluminium-zinc(ZL-401). These materials will produce spark under the effect of collision and friction with foreign particles(refer to the relevent information as described below). Therefore their use in the case is excluded. Through the study and test over the past 4 years on the performance of light alloy, a non-pyrophoric cast alloy has been developed.

PRINCIPLE AND DESIGN OF WATER TURBINE-DRIVEN LOCAL-EXHAUST FAN AND TECHNICAL PARAMETERS

Working Principle

The power generating machine is in the form of a bucket type water turbine. By applying water jet on the buckets through the nozzle, the water wheel is brought into rotatary motion causing also the impeller at the other end of the shaft to rotate. So a negative pressure is produced, bringing about an air stream.

Design of the Local Water-fan

The construction of the local water-fan is shown in Fig.1. It consists of the control valve 1, nozzle 2, water wheel 3, buckets 4, impeller 5, housing(including the fore section 7, impeller section 8, vane guiding section 9 and tail section 10), air inlet connection 11, air outlet connection 12 and water-drain port 13.

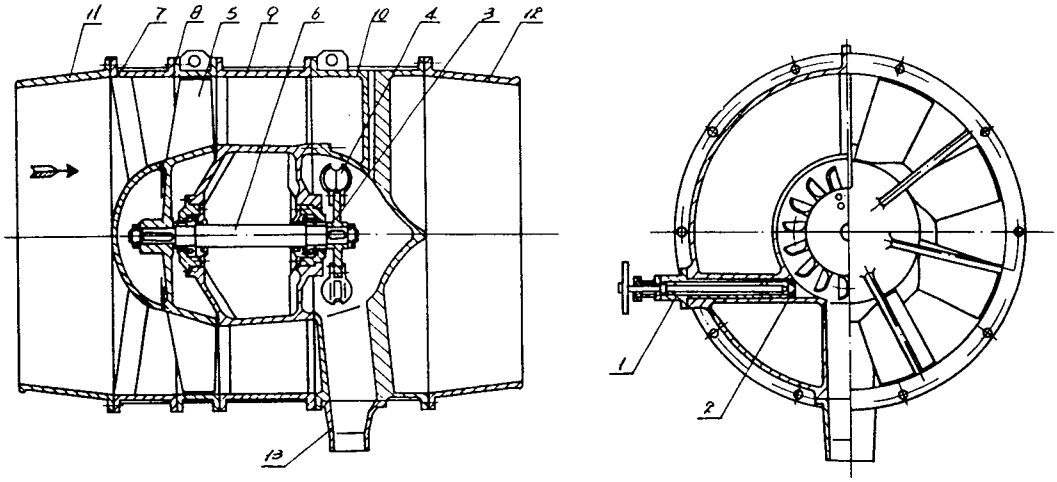


Fig. 1 Construction of the local water-fan. The housing and the impeller are all fabricated with the newly-developed non-pyrophoric cast light alloy.

Main Technical Parameters of the Water-fan

Air quantity:250-300m^3/min.
Total air pressure:981-392Pa.
Speed:2300 rpm.
Efficiency at total pressure:45-25%.
Working pressure:294 x 10^4Pa.
Water consumption:13 m^3/h.
Overall dimension:670 x 1000mm.
Weight:120kg.

TESTS ON THE SAFETY PERFORMANCE OF THE LOCAL FAN MADE OF CAST LIGHT ALLOY

The safety performance constitutes the key factor governing the successful operation of the fan under high-concentration gas conditions. Whereas the safety performance hinges, on the one hand, on the thermal field temperature produced by collision and friction between the housing and impeller and the coal, rock particles or even other foreign matters like metals sucked into the fan under the effect of airflow during normal operation, and, on the other hand, on the time of duration of the prevailing conditions with firedamp explosion hazards, such as the firedamp concentration, temperature of the fire source and induction period as tabulated in Table 1.

Table 1. Induction Period of Firedamp Explosion

Fredamp concentration %	Temperature of fire source, C^0						
	700	725	750	775	825	925	1025
	Induction Period, sec.						
4	8.2	3.6	2.4	1.4			
6	10.2	4.3	2.6	1.5	0.62	0.21	0.07
8	14.0	5.2	3.0	1.6	0.67	0.25	0.08
10		6.3	3.5	1.75	0.72	0.26	0.09
12		7.9	4.1	1.90	0.77	0.27	0.09

Note: The ignition temperature of firedamp is generally considered at 650^0 - 750^0

The safety performance tests were conducted by connecting a 2-m long iron tubing lined with 1.5mm asbestos plate on the air inlet end, and with coal, shale, sandstone and pyritic ore particles in different sizes. At a speed of 2300 rpm and an air velocity of 16-17m/s in the tubing, the generation of spark caused by collision between the housing and impeller and the sucked-in foreign particles with different sizes is shown in Table 2.

As can be seen from Table 2, the sparkling phenomenon was extraordinarily severe in case when test was made with shale particles 3-6mm in size. The results obtained showed that cast aluminium-silicon (ZL-108) was not the appropriate material suitable for making the fan. A fan of the same design, yet made with cast aluminium-zinc alloy which were usually used for making electrical drilling machine was then tested underground. During the test, spark was

Table 2. Generation of Sparks

Size of Particle	Type of Particle				
	coal	shale	sandstone	pyritic ore	remark
	Generation of sparks				
1mm	no spark	no spark	no spark	no spark	All sucked in
1-3mm	"	weak	weak	weak	"
3-6mm	"	extremely powerful	powerful	powerful	basically all sucked in
6-10mm	"	weak	weak	weak	A minor portion remains unsucked in
10-13mm	"	no spark	no spark	no spark	The majority remains unsucked in

also produced yet with less probability. Therefore its use was also excluded from consideration.

DEVELOPMENT OF THE NON-PYROPHORIC LIGHT ALUMINIUM ALLOY

In the development of a material with favourable safety performance, the factors governing the heneration of spark when metal is subjected to collision and friction are defined as follows: 1)The alloy as a whole or only a certain phase of the alloy is liable to get fractured into fine powder; 2) The powder formed is sensitive to oxidation in air, releasing a large amount of heat; 3) The presence of activation energy for producing spark. The cast aluminium alloy can be made non-pyrophoric if devoid of the spark-generating factors as described above.

The tests on the non-pyrophoric performance were carried out starting with the existing cast aluminium alloys of different grades. Since spark of different levels was observed during the friction test, it was therefore necessary to study the effect of the individual metal elements in the alloy on the spark ignition performance.

Test rods of two-component aluminium-base alloys with 1-15% silicon, copper, zinc and magnesium(purities $>$99.9%) and aluminium(purities $>$99.9%) were prepared. The result of friction tests showed the aluminium-zinc alloy offered the best spark suppression performance. For improving the mechanical and casting properties of the aluminium alloy and for better spark suppression performance, micro-elements, such as chro-

mium, nickel, tiranium, boron, berylium and strontium, etc., were added. Result of tests made with the alloys showed, that under test conditions, no spark was observed and their mechanical and casting properties were remarkably improved.

On the bases of the results obtained through tests with over 50 alloys with different melts and on the overall mechanical and casting properties required, 5 kinds of aluminium-base alloys were selected for making the test impeller in simulation test.

BLASTING TEST AND MEASUREMENT OF THE THERMAL-FIELD TEMPERATURE DUE TO COLLISION & FRICTION

Preliminary friction tests conducted with the 5 selected aluminium-base alloys which possessed the required mechanical properties were insufficient to evaluate their non-pyrophoric performance. It was necessary to carry out further studies and tests.

For simulating the actual working conditions, blasting tests and measurement of the thermal-field temperature produced during collision and friction of solids particles with the test specimen were made. The blasting system and the simulation impeller specimen used are shown respectively in Fig. 2 and Fig. 3.

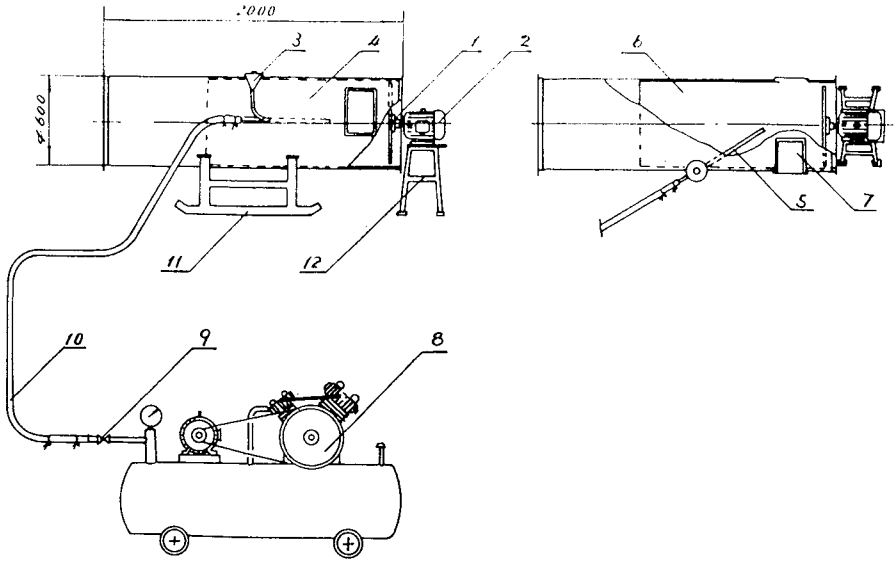


Fig. 2 Blasting systems
1) Aluminum-base specimen; 2)Electrical motor;
2) Feed hopper; 4) Protective cover; 5) Blasting tube;
6) Asbestos liner; 7) Observation glass; 8) Air Compressor;
9) Cutoff door; 10) Rubber hose; 11, 12) Support.

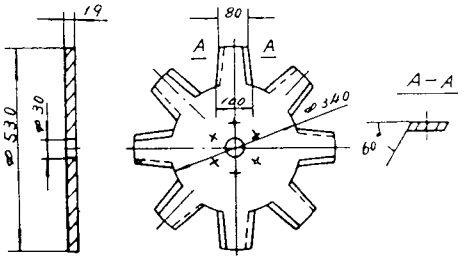
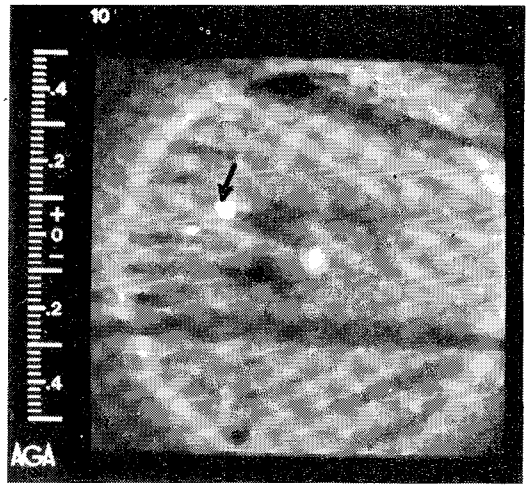


Fig. 3 Impeller Specimen for simulation test.

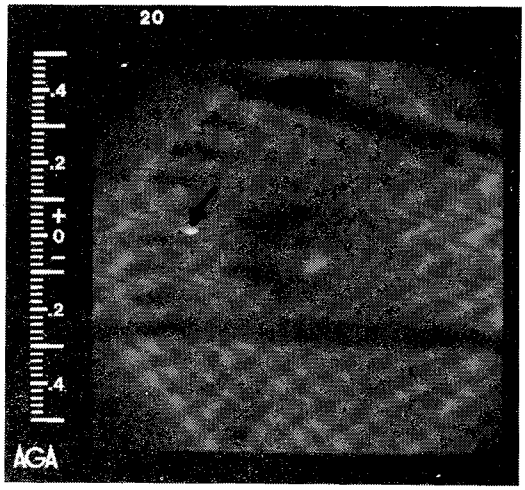
The blasting tests were made by alternately putting into the feed hopper solids particles like shale, sandstone, pyritic ore 3-6mm in size(It had been proved by previous tests that 3-6mm solids particles were most susceptible to produce spark). The particles were blasted through blasting tube 5 under compressed air(50-70 x 10^4Pa) onto the specimen 1 rotating at a speed of 2890 rpm. Then visual observation was made through the observation glass 7 to identify the presence of spark. Meanwhile the thermal-field temperature and its time duration due to collision and friction between the solids particles and the specimen were measured by using the AGA 780 infra-red thermal imaging system(Fig. 4). Results of measurements of three of the five specimens are listed in Table 3. The temperature of the thermal field as measured during the tests was lower than that in the inductive period of firedamp explosion. Based on the test results and in consideration of the overall mechanical and casting properties of the alloys, the No.3 aluminium-base alloy named"non-pyrophoric cast aluminium alloy" was finally selected as the material for making the impeller and housing of the local water fan.

Tale 3. Results of Measurements

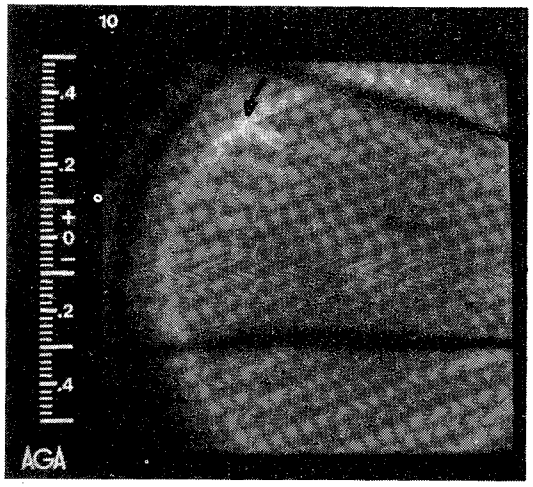
Specimen	Solid Particles	Relative Temp. of High Temp. Spark Tm (°C)	Drop of Temp. During Half of the Period of Observation T_2 (°C)	Maximum Temp. of Spark TM (°C)	Spark Observation Time t_1 (s)	Duration of Sparking Time t (s)	Rate of Decrease of Spark Temp. (°C/0.01s)	No. of Thermal Image
1#	Shale	94.1	76.1	308	0.044	0.084	34.6	9064
	Pyritic Ore	86.6	95.2	320	0.055	0.095	25.4	9089
3#	Shale	77.4	95.2	311	0.055	0.095	22.0	9126
	Pyritic Ore	109.4	114.2	362	0.066	0.106	27.7	9148
4#	Shale	106.6	129.7	374	0.075	0.115	240	9274
	Pyritic Ore	119.5	136.7	394	0.079	0.119	260	9228
	Sandstone	131.0						8161



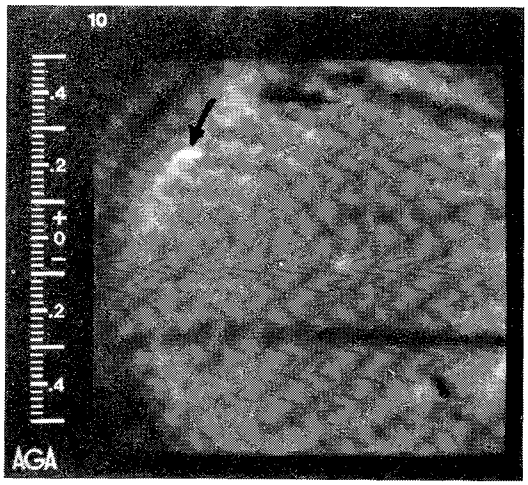
a) No. 9064 (#1 shale)



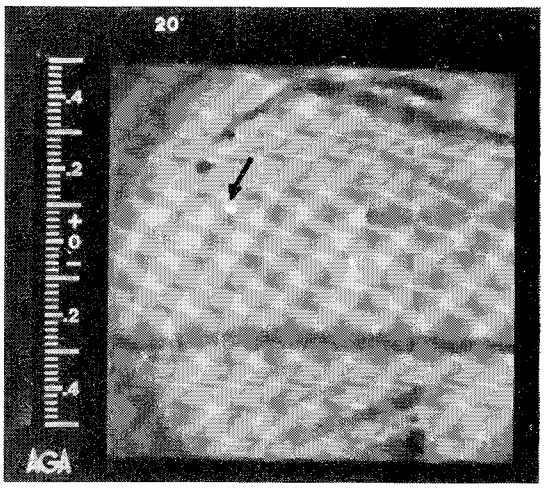
b) No. 9089 (#1 ferrous sulfate)



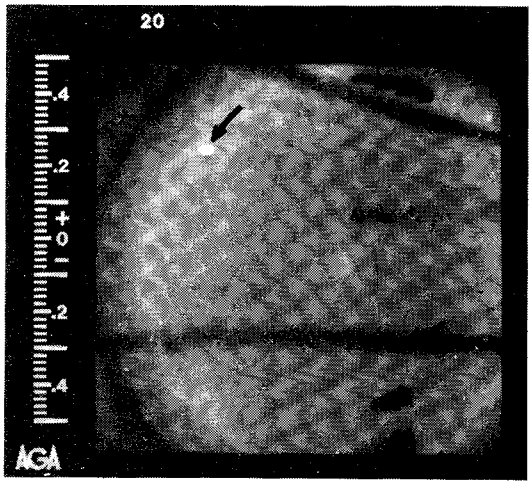
c) No. 9128 (#3 shale)



d) No. 9148 (#3 ferrous sulfate)



e) No. 9274 (#4 shale)



f) No. 9228 (#4 ferrous sulfate)

Fig. 4 Thermal images of relative high temperative spark under various testing conditions.

TEST ON TRICTION-INDUCED IGNITION OF GAS-AIR MIXTURE RESULTED FROM FRICTIONATING THE NON-PYROPHORIC CAST ALUMINIUM-BASE ALLOY

In order to further ascertain the reliability of the safety performance of non-pyrophoric aluminium-base alloy, tests on friction-induced ignition of gas-air mixture were conducted by frictionating the specimen through rotation in a test vessel as shown in Fig.5.

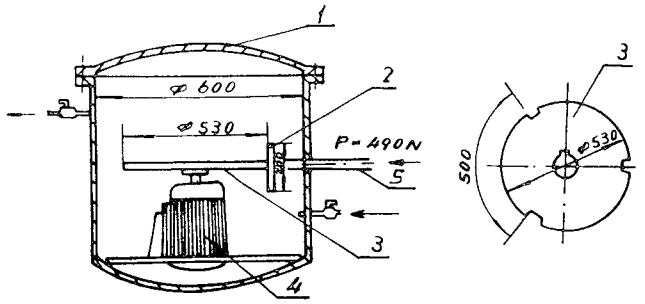


Fig. 5 Test vessel for friction-induced ignition of gas-air mixture

1) Vessel; 2) Fraction disk (simulating the fan housing); 3) Friction wheel (simulating the impeller); 4) Electrical motor; 5) Pressure lever.

The test was made by fixing the friction wheel 3 onto the electrical motor 4 (2890 rpm) and gluing onto the surface of friction disk 2 the mediums like quartz sand, ferric oxide and coal powder, etc. in different mesh sizes. Contained in the vessel was a 6 - 7% gas-air mixture. During each test run, an external force of 490 N was applied to the pressure lever 5, allowing the friction disk 2 and friction wheel 3 to frictionate with each other for a period of 0.5 minute (at each turn of the friction wheel 3, friction happened 3 times). If no explosion or ignition took place at the end of one minute of test, the test would then be stopped and ignition of the gas-air mixture in the vessel was tried with a spark plug. The test proved to be successful if the ignition was observed. The total test runs would last 2 minutes under each testing conditions, with 17200 times of friction. The test results are listed in Table 4.

As can be seen from Table 4, under the testing conditions, no ignition of the gas-air mixture was observed.

Table 4. Results of Test on Friction-induced Ignition of Gas-air Mixture

No.	Testing condition	Result
1	Aluminum disk-aluminum disk (all non-pyrophoric cast aluminium-base alloy specimens. Hereinafter shortened to AD)	No spark No spark-induced ignition observed
2	AD-AD with 20 mesh quartz sand	"
3	AD-AD with 60 mesh coal powder	"
4	AD-AD with 20-35 mesh ferric oxide	Spark observed yet ignition of test mixture unobserved
5	AD-AD with a mixture of 20-35 mesh ferric oxide and 60 mesh coal powder	"
6	AD-AD with a mixture of 20-35 mesh ferric oxide and 20 mesh quartz sand	"
7	AD-AD with a mixture of 20 mesh quartz sand and 60 mesh coal powder	"
8	AD-AD with a mixture of 20 mesh quartz sand and 60 mesh coal powder and 20-35 mesh ferric oxide	"

TEST ON THE SAFETY PERFORMANCE OF PROTOTY FAN UNDER HIGH GAS CONCENTRATION CONDITIONS AND MEASUREMENT OF THE THERMAL-FIELD TEMPERATURE

A prototype fan finally was made by using the newly-developed alloy after test and measurement as described above. Test on its safety performance was also conducted in an explosion test roadway by using the system as shown in Fig. 6.

Test Procedure

Both the two ends of the roadway 2 and the pressure-release pipe No.1 were sealed off with plastic film and kraft paper. Then the roadway was filled with gas which was controlled to maintain a concentration of 7-10% through mixing evenly with air by the fan. Afterwards the solids particles contained in the enclosed hopper were gradually put into the suction end of the fan. The particles would be sucked in under negative pressure and be subjected to collision and friction with the impeller and inner cavity of the fan for a period of over 50 hours of continuous operation. When the concentration of

the gas in the roadway was found to be as low as 7%, gas was timely added to raise it to a level of 10%. The capacity of the hopper was 8 kg and the total amount of particles discharged intermittently over a specified period of 12 - 15 minutes was 85 kg. The solids particles used were shown in Table 5.

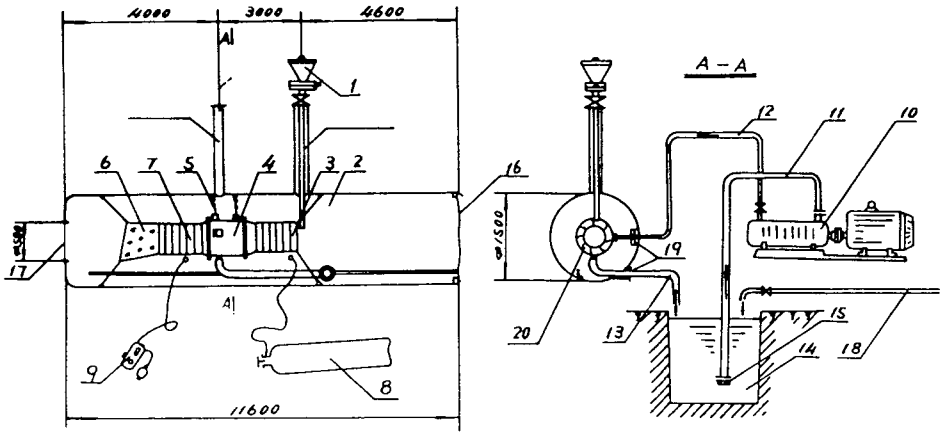


Fig. 6 Safety performance test system

1) Solids particle feed hopper; 2) Test roadway; 3) Telescoping tubing; 4) Local water fan; 5) Hanger; 6) Fender shield; 7) Telescopic tubing; 8) High-concentration methane steel bottle; 9) Gas meter; 10) Water pump; 11) Suction rubber hose; 12) Discharge rubber hose; 13) Drain pipe; 14) Water circulating box; 15) Suction hood; 16) Fore seal membrane; 17) Rear seal membrane; 18) Make-up water (tap water) pipe; 19) Pipe inlet & outlet; 20) Impeller; 21) No. 1 pressure-relief pipe; 22) No. 2 pressure-relief pipe.

Table 5. Solids Particles Used

No. Type	Size (mm)	Amount discharged(kg)	Gas concentration(%)
1 shale	3-6	24	7.8-9
2 sandstone	3-6	32	7.7-10
3 pyritic ore	3-6	24	8.7-9.2
4 iron sand	1-2.5	5	10
Total		85	

Test Result

No gas explosion whatsoever was observed when the fan was operated under the abovementioned conditions.

After the test, the seals at both ends of the roadway were removed and the

fan was operated under the effect of the presence of solids particles. For this a AGA 780 infrared thermal imaging system was used to measure the thermal-field temperature induced by collision and friction between solids particles and impeller and the internal cavity. The temperatures measured as shale particles were used were listed in Table 6.

Table 6. Thermal-field Temperature Induced by Collision and Frictions with Shale Particles

No. of thermal image	3036	3043	3050	3064	3086	3092
Temperature of spark, C°	26	82	47	44	38	50

No. of thermal image	3041	3121	3141	3151	3156	3175
Tempertaure of spark, C°	30	44	62	21	21	38

INDUSTRIAL-SCALE TEST ON THE DRAINAGE OF HIGH-CONCENTRATION GAS AT A HEADING FACE IN A GASSY SEAM

The test was conducted in the No.3119 heading face at the No.3 level(-450m). The seam was 2.4-2.8m thick with a dip angle of 10-15°. The heading was supported by trapezoid-shaped metal support with a net cross section of 5.57m^2. The gas emission was very large because of the blasting operations employed. Formerly forced ventilation with the use of a 28kW electrical local fan was employed. However with the increase of the length of the heading, ventilation became more and more difficult. When the heading advanced to a length of about 400m, it tooks 60-90 minutes or even more to lower the gas concentration at the face to less than 1% after blasting. Since two times of blasting operations were carried out on each shift, the time allocated for ventilation would therefore share 20-40% of the working time.

In order to tackle the abovegoing problems, a compound ventilation system of using a local electrical for-

cing fan and a exhausting water fan was employed in a certain period after the blasting operation. When the gas concentration at the face was lowered to 1%, the water fan was stopped and only forced ventilation was employed by using the electrical fan. The layout of the compound ventilation system is shown in Fig. 7.

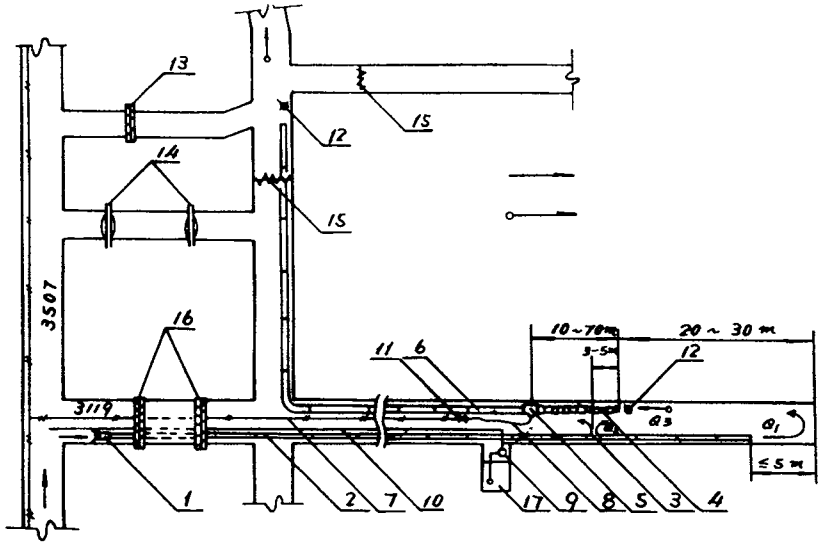


Fig. 7 Layout of the compound ventilation system.

1) 28kW electrical fan; 2) Forced ventilation tubing; 3) Airflow regulating section; 4) Negative pressure ventilation 5) Local water fan; 6) Positive pressure ventilation tubing; 7) Water supply line; 8) High-pressure rubber rose; 9) Sump pump; 10) Water-discharge pipe; 11) Valve; 12) Sensor of the gas telemeter; 13) Air partition; 14) Air door; 15) Railings; 16) Explosion-proof stopping; 17) Water sump.

In order to bring into full play the fume removal and gas clearing performance of the local water fan, an airflow regulating tee joint was installed on the positive pressure ventilation tubing of the local electrical fan at a distance of 3-5m behind the suction inlet of the water fan, allowing the fresh airflow to be split into two streams: one stream Q_1 was directed to the end of the face for short-time mixing with the gas and fume after blasting operation, and then the air mixture was sucked into the inlet of the water fan to be discharged to the air-return way. The other stream Q_2 was bypassed through the tee joint to the heading for diluting the gas released from the surrounding wall of the developed heading. The distribution of the airflow blown in and the airflow exhausted is as follows:

$Q_2 > Q_1$; $Q_3 > Q_1$.

where

Q_1 — the fresh airflow blown to the end of the heading.

Q_2 — the fresh airflow blown to the heading

Q_3 — the mixed air exhausted by the local water fan

The distribution of the airflow at the end of the heading is shown in Fig. 8.

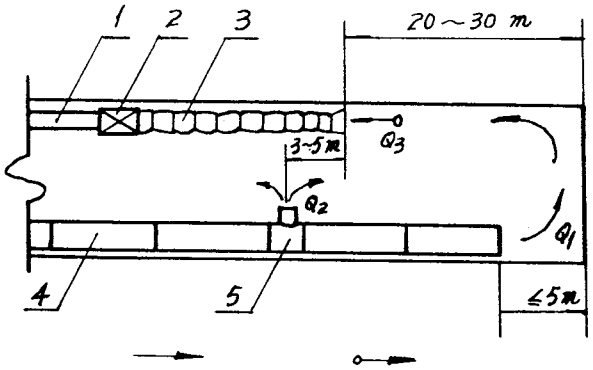


Fig. 8 Airflow distribution.

1) Positive pressure ventilation tubing; 2) Local water fan; 3) Negative pressure ventilation tubing; 4) Forced ventilation tubing; 5) Airflow regulating tee-joint.

The key of the distribution of airflow lay in the control of the airflow Q_1. Through the control of the airflow Q_1, the high-concentration gas caused by blasting operation could be driven into the suction inlet of the local water fan in the quickest way possible. Meanwhile the outlet airflow Q_2 bypassed by the tee joint was regulated for the purpose of diluting the gas released in the heading. Owing to the fact that the airflow Q_1 was below the suction capacity of the local water fan, a little portion of the airflow Q_2 therefore entered the suction inlet of the water fan. As a result, the high-concentration gas at the heading face due to blasting operation could be quickly drawn off with the use of the local water fan.

Test result shows that with the compound ventilation system and the use of the local water fan as described above, the gas concentration at the face after blasting can be reduced to less than 1% in a period of 6-15 minutes. The

maximum gas concentration that can be passed through the water fan is 6.2% and in a range of 2.8-3.2% in general.

DRAINAGE OF HIGH-CONCENTRATION GAS IN THE WORKED-OUT AREA FROM THE AIR-RETURN RETREATING HEADING IN HIGHLY-GASSY COAL SEAM

The newly-developed hydraulic coal mining face of the No.9 mining district, Badaojiang No.3 mine, Tonghua Mining Administration, Jilin Province was selected among others as the site for field trial.

The No.9 district is 100m long along strike with a dip length of 200 m. It is worked in 3 subsections. The only workable seam is seam M3 which has a thickness of 4 - 6 m and a dip angle of 14.5°. The coal is relatively soft with good air permeability. The gas emission is rather large because of the broken roof. The mining method employed is retreating singlewing hydromining system. The layout of the face and the local water fan as well as the ventilation system is shown in Fig.9.

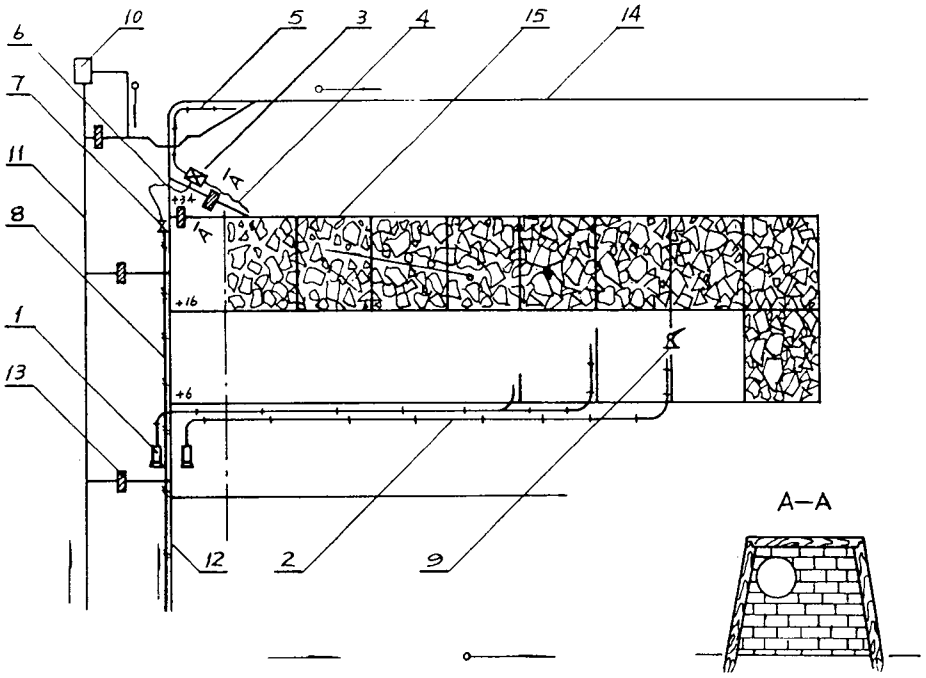


Fig. 9 Layout of the face and the local water fan ventilation system

1) 11kW electrical fan; 2) Forced air duct; 3) Local water fan; 4) Negative pressure ventilation tubing; 5) Positive pressure ventilation tubing; 6) High-pressure rubber rose; 7) Valve; 8) Water-supply pipe; 9) Monitor; 10) Winding rooms; 11) Tracked rise heading; 12) Haulage rise heading; 13) Seal; 14) Air-return way at +50m level; 15) Heading.

Trial Conditions

The local water fan was installed at the mouth of the heading driven at the level of +34m. Erected at the negative pressure end was an airtight partition wall on which an opening was provided for introducing a 10m negative pressure ventilation tubing into the sealed up area behind the partition wall. A 80m positive pressure ventilation tubing was connected to lead to the air-return way at the level of +50m. The cross sections of the retreating heading and the air-return course were $4.4m^2$ and $5.0m^2$ respectively. The local water fan operated at a pressure of 196×10^4Pa with an airflow of $50m^3/m$. Since its installation, the water fan has been working continuously on a 24-hour basis each day. Observations made over a long period of time shows that the concentration of the gas drawn off from the worked-out area by the water fan is generally in the range of 5-10%. The total amount of the gas drawn off each day is up to $5400m^3$ in average. Practice shows that with the stoppage of the local water fan, the concentration of the gas at the face will soon exceed the upper limit. Thanks to the continuous and smooth operation of the water fan, the normal operation of the hydraulic mining face is ensured.

DRAINAGE OF HIGH-CONCENTRATION GAS FROM UPPER CORNERS OF LONGWALL FACE WORKED BY BLASTING METHOD

The Beijing mining district Shanjiaoshu mine, Panjiang Mining Administration, Gueizhou Province was selected as the site for trial application. The water fan was installed at the air-return way of Youba No.12 face in Beijing Mining District. The face is 500m long along the strike with a dip length of 180 m and a dip angle of 3°. The seam is 1.8-2.1m in thickness and is mined by using the longwalling and blasting method together with frictional metal props. The water supplied to the local water fan was closed circulated. At the negative pressure end, a ϕ500mm, 50-m ventilation tubing was connected while at the positive pressure end a ϕ 500mm, 430-m ventilation tubing was provided with its outlet leading into the air-return way of the mining district. The outlet airflow was $50-60m^3/m$. With the use of the water fan, the methane concentration at the upper corners of the face was reduced remar-

kably from 2-3% to 0.7%, while in the air-return way, the methane concentration was lowered from 1.2% to 0.6%. Thus it can be seen that the water fan is remarkably effective in reducing the methane concentration both at the face and in the air-return way.

CONCLUSIONS

1. Through development and repeated tests, it has been ascertained that the cast aluminium-base light alloy material developed is safe and reliable with respect to its non-pyrophoric performance and provides a new material for the equipment used at the highly-gassy mining faces. Moreover, rich experience has also been accumulated in this respect.

2. Field application and measurement underground of the prototype water-turbine powered local-exhaust fan prove that the fan is remarkably effective in exhausting high-concentration gas, and is therefore a new addition of safety equipment suitable for use in local-exhaust ventilation system underground. It may find applications for the ventilation of heading face in hydromines and for ventilation of highly-gassy development face in a compound ventilation system or for draining the gas accumulated at the upper corners at longwall face in conventional mines. Its successful operation shortens consequently the time required for drawing off the fume due to blasting operation at the heading face and ensures thereby a continuous mining operation of the monitor at the highly-gassy hydraulic mining face. Meanwhile a higher drivage rate and mining recovery are ensured.

However for use in conventional coal mines, systemmatic study and improvement should be made on the performance of the fan and problems relating to the appropriate treatment of the waste water be tackled.

ACKNOWLEDGEMENT

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Mine, Fengcheng Mining Administration; Xiaolinghe Mine, Nanpiao Mining Administration; Badaojian Mine, Tonghua Mining Administration; Hunan Province Coal Research Institute; Majiagou Mine, Kailan Mining Administration; Hongshandian Mine, Lianshao Mining Administration.

矿井局部抽出式通风方式的重要意义及"非发火性"轻合金抽出式水力局扇的研究

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摘要

本文论述了煤矿井下局部通风采用抽出式通风方式的重要意义，并着重介绍了一种水轮机驱动，用新研制的“非发火性”铸造轻合金制造的抽出式局扇。通过对“非发火性”铸造轻合金化学配方的研究及其试件与杂质颗粒碰撞摩擦时热场温度的测定，同时对样机进行了在高浓度（7－10%） 沼气条件下的整机运转试验，证实了该局扇的安全可靠性。

通过在井下高沼气煤巷和水采工作面上回风顺槽抽排高浓度瓦斯试用，风机连续运转180天，（瓦斯浓度高达5－10%，平均日抽出瓦斯量为5400立方米）。证明了该机的安全可靠性。 为煤矿井下增添了一种新型局部通风设备。

一、概述

目前，中国煤矿井下的局部通风几乎全是电局扇压入式通风，实践证明压入式局部通风方式存在一定的缺点。1、当通风距离较长时，稀释工作面的瓦斯、炮烟所需的时间长；2、当通风距离短时容易在出风口处扬起粉尘；3、巷道中为浊风流，劳动卫生条件差；4、压入式通风方式必须把瓦斯稀释到1%以下，所需的风量大等等。

高沼气水采工作面水枪落煤量集中，瞬时瓦斯涌出量很大，经常造成回采顺槽瓦斯浓度超限；高沼气水、旱采掘进工作面由于放炮落煤后瓦斯不能及时排除影响掘进速度；长壁工作面上隅角瓦斯积聚问题十分严重；这些局部积聚高沼气的地方，采用单一的压入式局部通风很难奏效。为了提高通风效果，缩短瓦斯，炮烟的排放时间，消除局部瓦斯积聚，改善巷道作业卫生条件，因此研究一种能够抽排含有高浓度瓦斯的局部通风设备和积极采用抽出式局部通风方式，是解决上述煤矿井下局部通风工艺中的重要课题。

二、抽排含有高浓度（1%以上）瓦斯风流风机技术关键的分析。

研究抽排含有高浓度（1%以上）瓦斯风流的抽出式局部扇风机必须解决好动力源和风机叶轮、外壳材质这两个技术关键。

1、动力源

目前，中国煤矿井下常用的局部扇风机，是以防爆电动机为动力源，这种以电动机为动力源的局扇容易出电气故障，并存在产生电火花的危险，因此这种未经专门结构隔离的电动局扇不能安设在瓦斯含量大于0．5%风流中使用。 根据煤矿井下（不论水、旱采矿井）一般都有静压水的情况（无静压水可设置专用水泵）因此选择压力水作为抽出式局扇的动力源，这样就避免了电动力源的火花危险。当然也可以考虑用压风作动力源等。

2、材料

现行煤矿井下应用的电局扇，其外壳是采用钢板制造，叶轮是采用铸造的铝硅（ZL—108） 和铝锌（ZL—401）等制造。 这种铝合金材料遇有岩石等杂质颗粒碰幢和摩擦时有火花产生（详见材料的碰撞、摩擦火花性能考察试验）因此不能采用。我们针对轻合金材料的"非发火性" 问题进行了四年多的试验研究，研制出了一种"非发火性"铸造轻合金材料。

三、水力驱动抽出式局扇的工作原理、结构形式及技术参数。

1、水局扇的工作原理

水局扇的动力机械是一水斗式水轮机，压力水通过喷嘴射到水轮机的水斗上，使水轮盘旋转并带动同轮上另一端的叶轮旋转，产生负压形成风流。

2 水局扇的结构

水局扇的结构如图1所示

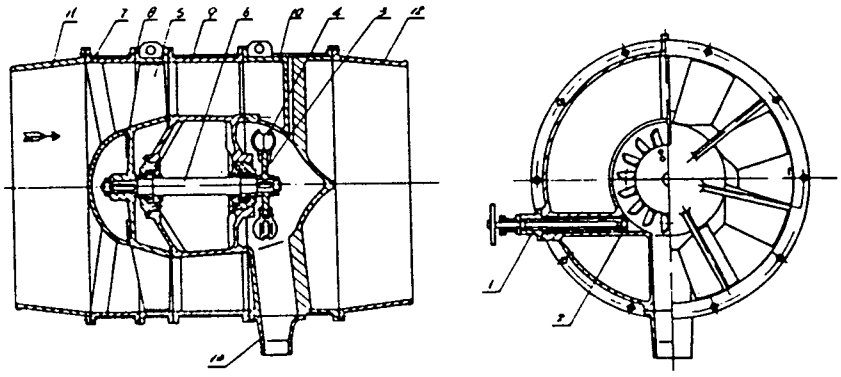


图 1 水局扇结构

由控制阀1，喷嘴2，水轮盘3，水斗4，叶轮5，主轴6，外壳（包括前段7，叶轮段8，导翼段9，尾段10），进气接口11，出气接口12，泄水口13等主要零部件组成。

外壳和叶轮均采用新研制的非发火性铸造轻合金材料制造。

3 水局扇的主要技术参数

风量： 250～300 m'／min

全风压： 981～392 Pa

转速： 2300 r／min

全压效率： 45～25%

工作水压： 294×10' Pa

耗水量： 13m'／h

外行尺寸：ϕ670×1000 mm

重量： 120Kg

四　铸造轻合金的安全性能考察试验

水局扇在高浓度瓦斯条件下运转的关键问题是安全性能，而安全性能是指叶轮与外壳遇有碰幢、摩擦或运转过程中在风流的作用下吸入的煤、岩甚至金属等杂质颗粒与机壳、叶轮碰撞磨擦时所产生的热场温度及其延续时间不应超过瓦斯爆炸的有关条件，如火源温度、感应期等，瓦斯爆炸感应期见表1［1］

表　1　瓦斯暴炸的感应期

瓦斯浓度	火源温度（℃）						
	700	725	750	775	825	925	1025
%	感应期（秒）						
4	8.2	3.6	2.4	1.4			
6	10.2	4.3	2.6	1.5	0.62	0.21	0.07
8	14.0	5.2	3.0	1.6	0.67	0.25	0.08
10		6.3	3.5	1.75	0.72	0.26	0.09
12		7.9	4.1	1.90	0.77	0.27	0.09

注：瓦斯的引火温度一般认为是650°～750℃

考察试验是将局扇负风端接装2m长的铁风筒，内衬1.5mm石棉板。试验时选用不同粒度的煤、页岩、砂岩，黄铁矿石颗粒分别进行，当局扇达到2300r／min时风筒中风速为16～17m／S，吸入不同粒度的杂质颗粒时与叶轮或壳体碰撞而发生的火花情况见表2。

表　2　火花情况

杂质品种 / 火花情况 / 杂质粒度	煤	页岩	砂岩	黄铁矿石	备注
1毫米以下	未见火花	未见火花	未见火花	未见火花	全部被吸入
1～3毫米	未见火花	弱	弱	弱	全部被吸入
3～6毫米	未见火花	极强	强	强	基本全部被吸入
6～10毫米	未见火花	弱	弱	弱	少部分不能被吸入
10～13毫米	未见火花	未见火花	未见火花	未见火花	大部分不能被吸入

从试验结果来看，投放3～6mm粒度的页岩时产生的火花现象最为严重。试验证明该材料（ZL—108）不能用作制造抽排含有高浓度瓦斯风流的水局扇。随后改用了铝锌（ZL—401）合金，即井下电钻外壳常用的材料制造了同一规格局扇，在进行上述试验时也有火花产生，只是机率较少，该材料同样不能采用。

五　非发火性铸造轻合金材料的研制

为了寻求理想的安全性能好的材料，首先分析金属受到撞击，摩擦而发生火花的因素有如下几点：(1)合金材料或合金中的某个物相容易破碎为极细的粉末；(2)这种粉末在空气中极易氧化并大量放热；(3)有发生火花的活化能。铸造铝合金若能避开上述发火因素即可制成非发火性的轻合金［2］。

对铝合金非发火性能的考察试验是从现有的铸造铝合金开始的，现有各种牌号的铸造铝合金，在摩擦试验时均有强、弱、多、少不同程度的火花产生，所以需要对单一金属元素在铝合金中对发火性能的影响作进一步的研究。

用纯度大于99．9%的硅、铜、镁、锌，由1～15%分别和纯度大于99．9% 的铝制备成铝基二元合金试棒，同样进行摩擦试验，结果表明，铝—锌二元合金抑制发火性能最好。为了提高铝合金的机械和铸造性能以及进一步抑制摩擦发火性在铝合金中添加了铬、镍、钛、硼、铍、锶等微量元素，这些微量元素添加到铝合金中后，经摩擦试验表明，在试验条件下并无火花发生，而且大大改善了机械和铸造性能。以上述试验为依据，通过50多种配方的熔炼和试验，并综合考虑到其非发火性和机械性能，最后筛选出五种铝合金作为模拟水局扇叶轮的试件。

六　合金材料的喷砂试验及碰撞摩擦的热场温度测定

筛选出的五种铝合金材料，其机械性能是可以满足制造水局扇要求的，但是对其非发火性能仅做摩擦试验只是初步的，必须对非发火性能作进一步的考察试验研究。

为了模拟制成水局扇后的工作条件，对筛选出的五种铝合金材料进行了物料颗粒喷砂试验和在喷砂试验过程中对颗粒与试件碰撞摩擦时产生的热场温度进行了测定。喷砂试验系统如图2。叶轮模拟试件如图3。

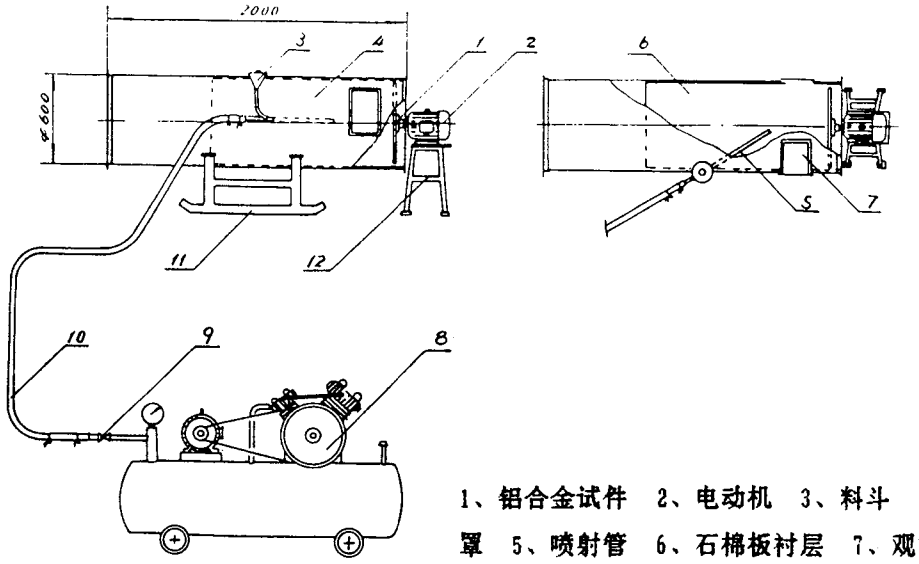


图　2　喷砂试研系统

1、铝合金试件　2、电动机　3、料斗　4、防护罩　5、喷射管　6、石棉板衬层　7、观察窗　8、空压机　9、截门　10、胶管　11、12、支架

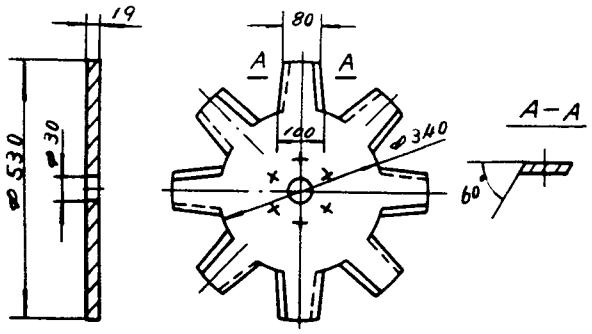


图 3 模拟叶轮试件

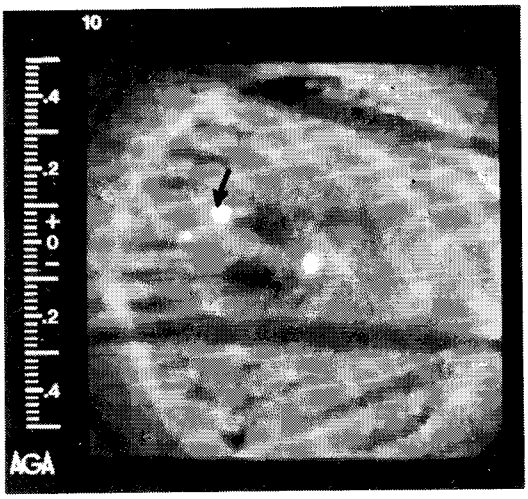
喷砂试验的方法是在料斗中分别放入粒度为3～6mm的页岩、砂岩、黄铁矿石颗粒（前试验已证明3～6mm的物料颗粒最易发火），在50～70×10⁴ Pa的压缩空气作用下经喷射管将颗粒射向转速为2890 r／min 的试件1，通过观察孔7， 以人眼观察喷射的物料颗粒与旋转试件撞击时是否产生火花。在此基础上用AGA780型红外热象仪对上述试验物料颗粒撞击试件时所产生的热场温度及延续时间进行了测定［3］。五个试件中的三件测定结果如表3。喷砂试验时热场温度测定结果低于瓦斯爆炸感应期的数值， 并综合考虑铝合金材料的机械和铸造性能诸因素，最后选定了3号铝合金，定名为"非发火性铸造铝合金"作为制造水局扇的叶轮和外壳的材料。

表 3 喷砂试验时热场温度测定结果

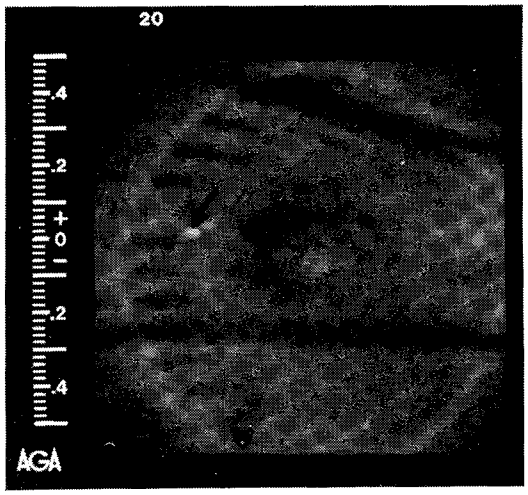
试件	喷射物料	相对高温火花温度 Tm（℃）	1／2观测时间温度降下 △T2（℃）	火花最高温度 TM（℃）	火花观测时间 t1(s)	火花延长时间 t（s）	火花温降速度（℃／0.01s）	热象号
1#	页岩	94.1	76.1	308	0.044	0.084	34.6	9064
	黄铁矿	86.6	95.2	320	0.055	0.095	25.4	9089
3#	页岩	77.4	95.2	311	0.055	0.095	22.0	9128
	黄铁矿	109.4	114.2	362	0.066	0.106	27.7	9148
4#	页岩	106.6	129.7	374	0.075	0.115	24.0	9274
	黄铁矿	119.5	136.7	396	0.079	0.119	26.0	9228
	砂岩	131.0						8161

图 4 热象图 （见下页）

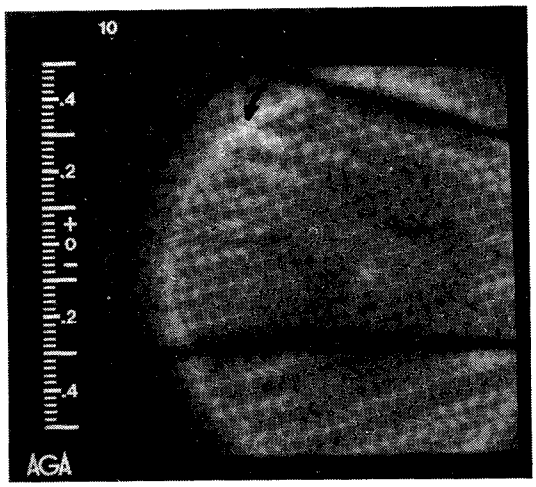
图4中a、b、c、d、e、f所示为各试验条件下相对高温的火花对应的红外热象图象。



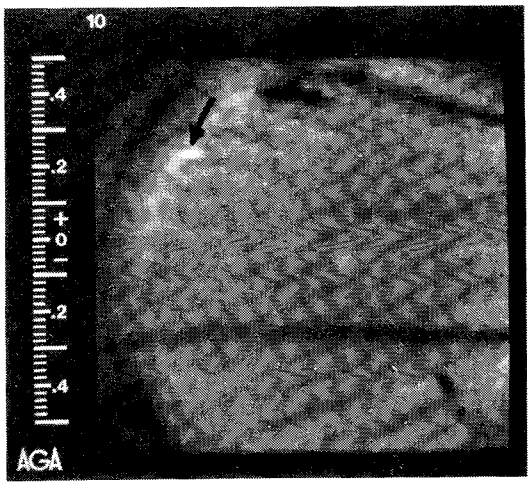
a、热象号9064（#1，页岩）



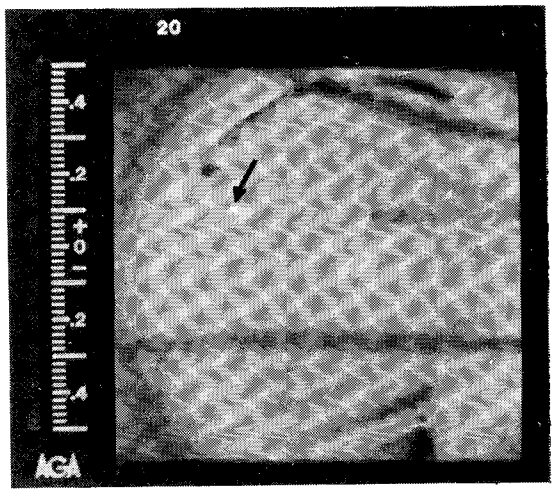
b、热象号9089（#1，硫化铁）



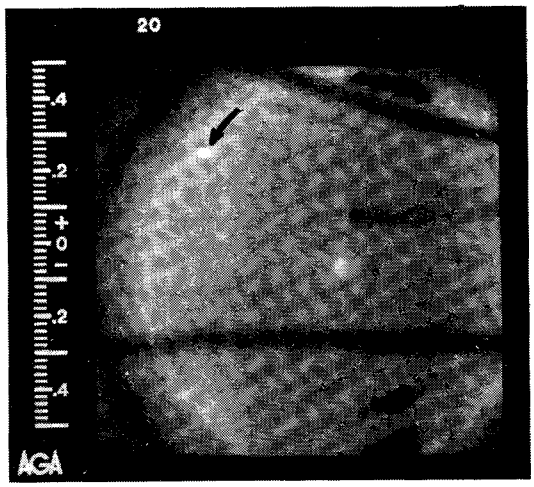
c、热象号9128（#3，页岩）



d、热象号9148（#3，硫化铁）



e、热象号9274（#4，页岩）



f、热象号9228（#4，硫化铁）

图 4 热象图

七　非发火性铸造铝合金材料摩擦点燃沼气—空气混合气体的试验

为了进一步证实非发火性铸造铝合金材料安全性能的可靠性。将试件置於试验槽内进行旋转摩擦点燃沼气—空气混合气体的试验(4)。其试验装置和试件如图5。

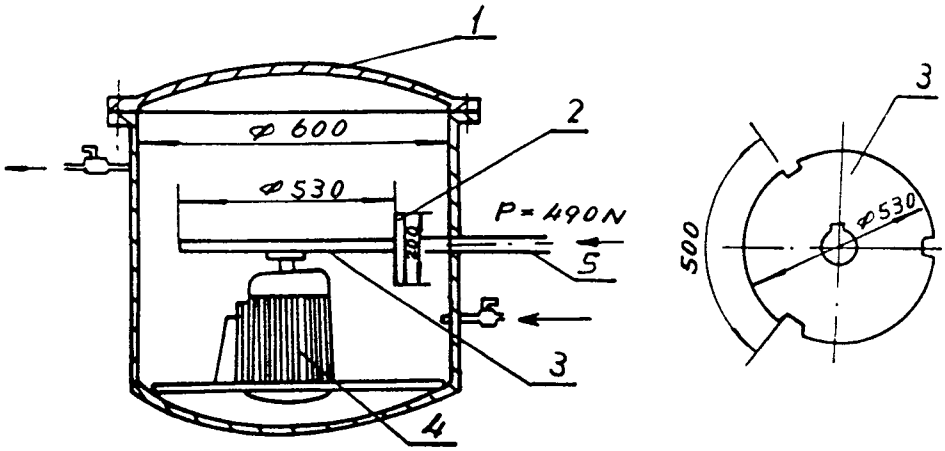


图5　旋转摩擦点燃沼气试验

1—试验槽，2—摩擦试片（模拟局扇外壳），3—大摩擦轮（模拟叶轮）
4—电动机，5—压力杆。

试验方法：将试件3固定在电机上，电机转速为2890r／min，在试件2的摩擦面上粘以不同网目的石英砂、铁锈、煤粉等介质，试验槽内充以含有6～7%沼气—空气混合气体。试验时对压力杆加外力490N，使试件2，3相互摩擦，连续摩擦0．5m为一次试验（试件3每旋转一周摩擦三次），试验每进行一分钟如未发生爆炸或被点燃，停止运转，然后用火花塞对槽内气体点燃，若气体被点燃则试验有效。每种条件试验二分钟，摩擦17200次，试验结果如表4。

表　4　旋转摩擦点燃沼气试验结果

序号	摩　擦　条　件	结　果
1	铝片与铝片。（铝片：非发火性铸造铝合金试片，下同）	未见火花，未发生试验气体被点燃。
2	铝片与粘有20目石英砂的铝片	未见火花，未发生试验气体被点燃。
3	铝片与粘有60目煤粉的铝片	未见火花，未发生试验气体被点燃。
4	铝片与粘有20—35目铁锈的铝片	已见火花，未发生试验气体被点燃。
5	铝片与粘有20—35目铁锈和60目煤粉混合的铝片	已见火花，未发生试验气体被点燃。
6	铝片与粘有20—35目铁锈和20目石英砂混合的铝片	未见火花，未发生试验气体被点燃。
7	铝片与粘有20目石英砂和60目煤粉混合的铝片	未见火花，未发生试验气体被点燃。
8	铝片与粘有20目石英砂和60目煤粉再和20—35目铁锈三种混合的铝片	未见火花，未发生试验气体被点燃。

从表4中可以看出该材料在试验条件下未发生沼气—空气混合气体被点燃。

八　在高浓度瓦斯条件下整机运转的安全性能试验及其热场温度的测定

新研制的非发火性铸造铝合金材料经过上述试验和测定后，用以制成了水局扇，并对水局扇进行了整机安全性能试验，试验是在爆炸试验巷道内进行的。试验系统如图6所示。

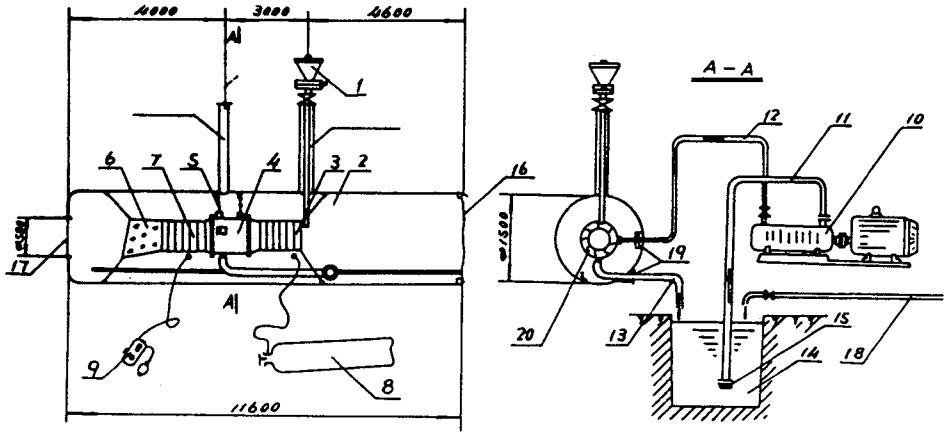


图6　整机安全性能试验系统

1、杂质给料器，2、试验巷道，3、伸缩风筒，4、局扇，5、吊挂铅丝，6、挡沙软帘，7、伸缩风筒，8、高浓度沼气钢瓶，9、沼气检定器，10、水泵，11、吸水胶管，12、排水胶管，13、泄水管，14、循环水箱，15、吸水龙头，16、前密闭膜，17、后密闭膜，18、补给水管（自来水），19、管道进出口，20、叶轮。

试验方法：按图6系统安装后，将试验巷道两端和一号泄压管用塑料膜和牛皮纸密封，而后向巷道内充放瓦斯并起动水局扇使瓦斯与空气混合均匀，瓦斯浓度保持在7～10%，将封闭型给料器中的杂质颗粒按品种先后渐渐投到水局扇负压端，在负压风流作用下杂质颗粒被吸入水局扇，与叶轮、内腔产生撞击和摩擦。水局扇在上述条件下连续运转了50个小时，试验巷道内瓦斯浓度下降到7%，时就及时补充到10%。杂质颗粒间断投入量共85Kg，给料器每斗物料重8Kg，每斗物料投放时间12～15分钟。杂质颗粒投放情况见表5。

表　5　杂质颗粒投放情况

序号	杂质品种	粒度（毫米）	投入量（公斤）	瓦斯浓度（%）
1	页岩	3～6	24	7．8～9
2	砂岩	3～6	32	7．7～10
3	黄铁矿	3～6	24	8．7～9．2
4	钢粒	1～2．5	5	10
	合计		85	

试验结果：该局扇在上述试验条件中未引起瓦斯爆炸。

在完成上述试验后，打开两端密封，开动水局扇， 按上述投料方法并用AGA780型红外热象仪测定杂质颗粒与局扇叶轮、内腔撞击和摩擦时的热场温度。投放页岩颗粒时所测得的温度如表6。投入钢粒时，钢粒在撞击摩擦过程中有卡在叶轮与外壳间隙中的现象，这时温升达到109℃。 这一温度远远低於瓦斯爆炸的引燃温度。

表 6 整机试验时热场温度测试结果

热 象 号	3036	3043	3050	3064	3086	3092	3041	3121	3141	3151	3156	3175
火花温度（℃）	26	82	47	44	38	50	30	44	62	21	21	38

九 在高沼气煤层掘进工作面抽排高浓度瓦斯的工业性试验

该项试验是在江西省丰城矿务局坪湖矿的三水平（－450m）3119煤巷掘进工作面进行。煤层厚度2．4～2．8m，倾角10°～15°，巷道支护使用梯形金属支架，净断面5．57m²，炮掘，瓦斯涌出量很大。工作面原使用一台28KW电局扇压入式通风。随着巷道掘进长度的增加，通风愈加困难，巷道掘至100m左右时，放炮后需要通风60～90分钟甚至更长的时间工作面的瓦斯浓度才能降到1%以下， 每班放炮两次占去工作时间的20～40%。

为解决上述问题，在放炮后的一段时间内，采用了电局扇压入和水局扇抽出的混合通风方式进行通风，待工作面的瓦斯浓度降至1%以下时，停止水局扇运转，但仍保持电局扇压入式供风。 混合通风系统见图7。

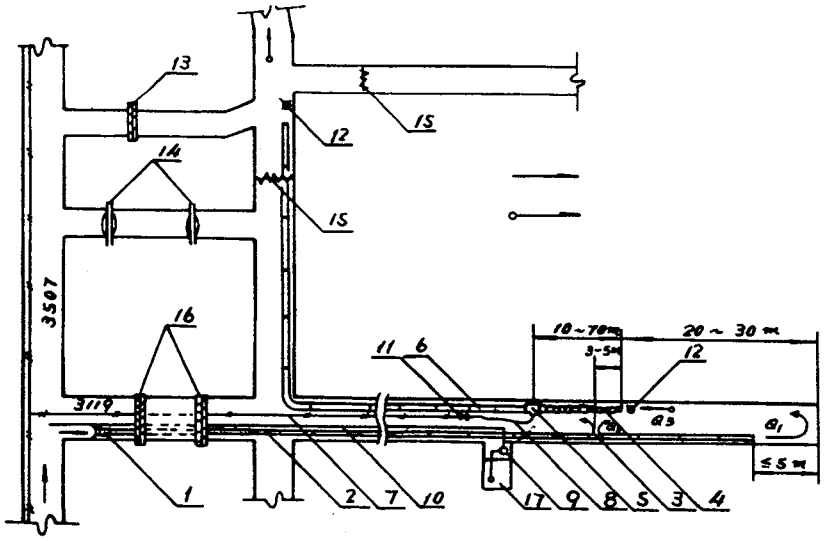


图7，抽、吹混合式通风系统

1 28KW局扇，2 压入式风筒，3 风量调节段，4 负压风筒，5 水风扇，6 正压风筒，7 供水管，8 高压胶管，9 排水泵，10排水管，11阀门，12 瓦斯遥测探头，13 风墙，14 风门，15 栅栏，16隔爆墙，17水窝。

为了充分发挥水局扇抽排炮烟、瓦斯的作用， 在距水局扇吸风口后3～5m处的电局扇正压风筒上接装了一个风量调节三通，使压入的风分成两部分：一部分新风Q_1直接吹向工作面迎头，用作混合放炮后的瓦斯、炮烟，经短时间的混合后进入水风扇吸风口，然后排至回风道。另一部分新风Q_2从三通旁路吹到巷道中，用作稀释已掘巷道周壁释放的瓦斯，压入和抽出风量的分配关系是：

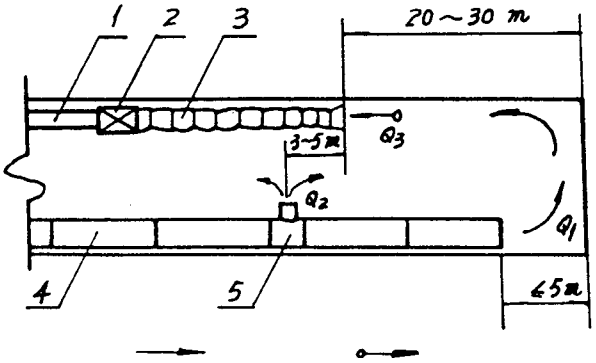
$Q_2>Q_1$；　$Q_2>Q_1$

这里：Q_1—压入到迎头的新风；

Q_2—压入到巷道中的新风；

Q_3—水局扇抽出的混合风。

掘进头风量分配如图8所示



1　正压风筒，2　水局扇，
3　负压风筒，4　压入风筒，
5　风量调节三通。

图　8　风量分配关系

掌握风量分配主要是对Q_1进行控制，驱使放炮后的高浓度瓦斯尽快进入水局扇吸风口。调节三通旁路出口的风量Q_2，主要是稀释巷道壁释放出的瓦斯，同时由於Q_1不能满足水局扇吸风量的要求，故Q_2中的一小部分要进入水局扇吸风口，这样工作面在放炮后的高浓度瓦斯可由水局扇迅速排出去。

试验证明：采用水局扇进行的抽压结合的通风方式，放炮后经6～15分钟的通风，工作面的瓦斯浓度即可降至1%以下，比单一电局扇压入通风缩短排放瓦斯时间30～40分钟。水局扇通过的最高瓦斯浓度为6．2%，一般为2．8～3．2%。

十　在高沼气水采工作面回风顺槽抽排采空区高浓度瓦斯

这次试用地点是在吉林省通化矿务局八道江三井九采区新投产的水采工作面。

九采区走向长100m，倾向长200m，分三个亚阶段开采，只有m₁为可采煤层，煤厚3～6m，倾角14．5°左右，煤层较软，透气性好，顶板破碎，瓦斯涌出量很大， 采煤方法是单面漏斗式水力采煤法。其工作面布置、水局扇安装及通风系统如图9所示。

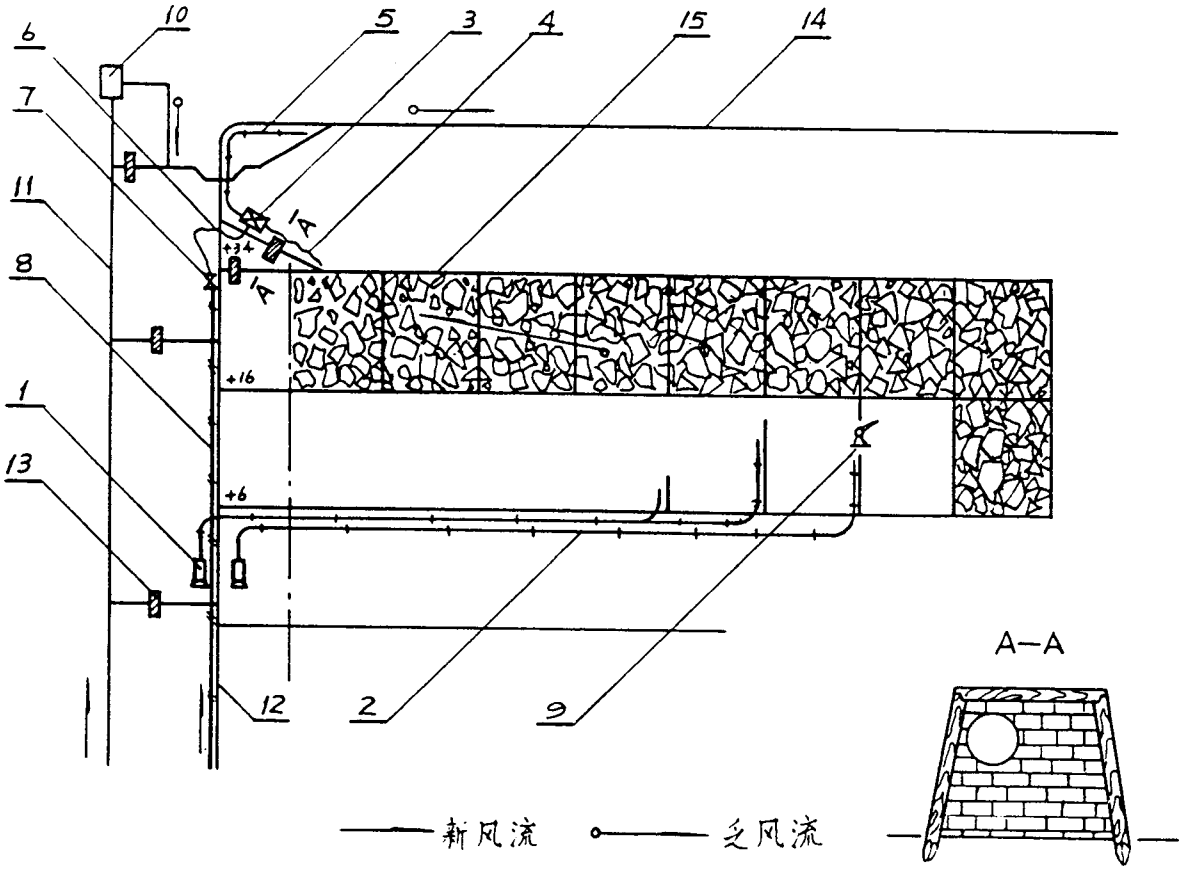


图 9 水采工作面布置及水局扇安装系统

1 11KW电局扇，2 压入式风筒，3水局扇，4 负压风筒，5正压风筒，
6 高压胶管，7 阀门，8 供水管，9水枪，10 绞车房，11轨道上山
12 运输上山，13 密闭，14 +50m回风巷，15 顺槽。

使用情况：将水局扇安设在+34m巷道口，负压端打一个密闭隔墙，墙上留一风口与负压风筒连接，水局扇接装了10m长的负压风筒至密闭墙内，接装了80m长正压风筒至50m采区回风大巷中， 工作面顺槽和回风道断面分别为4．4m²和5．0m²。水局扇工作水压为196×10²Pa风量50m³／min。 水局扇自安装使用后每天24小时运转，经长时间的观测记录，水局扇从采空区内抽出的瓦斯浓度一般都在5～10%， 每天抽出的瓦斯量平均高达5400m³。实践证明如果水局扇停止运转，则工作面瓦斯浓度很快超限，由于水局扇的连续正常运转，保证了该水采工作面正常生产。

十一 在长壁工作面抽排上隅角高浓度瓦斯

这次试用地点是贵州省盘江矿务局山脚树矿北井采区。在北井采区12号右八回采工作面的回风巷道中安设了水局扇。该工作面走向长500m，倾向长180m，倾角3°，煤层厚度1．8～2．1米， 采用炮采长

壁式采煤方法，摩擦金属支柱支护。水局扇的供水采用闭路循环，负压端安装了ϕ500mm风筒50m，正压端安装了ϕ500mm风筒430m，其正压端风筒出口接至采区回风道中。出口风量为50～60m³／min，水局扇开机运转后，上隅角处的沼气浓度由使用前的2～3%降为0．7%，回风道风流中的沼气由使用前的1．2%降至为0．6%，由此可见水局扇的使用，对降低回采工作面上隅角和回风道风流中的沼气浓度效果很好。

十二　结论

1　非发火性铸造铝合金材料

通过一系列的研制试验，证实其非发火性能是安全可靠的，为煤矿井下高沼气工作面使用的设备探索了一条新的材料途径，并积累了经验。

2　水局扇通过整机安全性能和井下工业性应用及测定，证实该局扇具有抽排含有高浓度瓦斯风流的功能，为井下局部抽出式通风增加了一种新型安全设备。它适用於水采矿井采掘工作面通风，也可用於旱采高沼气掘进工作面混合式通风和抽排长壁工作面上隅角积聚的瓦斯。它的使用成功，缩短了掘进工作面放炮后排放瓦斯的时间，保证了高沼气水采工作面水枪能够连续落煤，提高了掘进和回采效率。

今后在旱采矿井使用时对机型的性能和废水处理应作系列化的研究和改进提高。

十三　致谢

非发火性铸造铝合金材料和水局扇的一系列研制试验曾得到，煤炭科学研究院抚顺研究所；北京有色金属加工厂；中国科学院金属研究所；丰城矿务局坪湖矿；南票矿务局小凌河矿；通化矿务局八道江矿；湖南省煤炭科学研究所；开滦矿务局马家沟矿；涟邵矿务局洪山殿矿；盘江矿务局山脚树矿等兄弟单位的大力协助和支持，在此表示衷心感谢。

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IMPORTANCE DES SYSTEMES DE VENTILATION LOCAUX PAR SUCCION DANS LES MINES - ETUDE D'UN VENTILATEUR DE SUCCION EN ALLIAGE LEGER NON PYROPHORIQUE

Yan Peng, Lin Yilin, Yang Guangqi, Wang Wenlie, Lu Yuan

Après une étude générale des systèmes locaux de ventilation souterraine par succion des mines de charbon, notre rapport approfondit le sujet des ventilateurs non pyrophoriques par succion, récemment développés, actionnés par des turbines à eau et fabriqués en alliages légers coulés. L'amélioration de la composition chimique de l'alliage léger coulé, la mesure de la température du champ thermique créé par les chocs et les frictions de particules étrangères et, en particulier, les essais effectués sur des prototypes de ces ventilateurs avec des concentrations élevées de méthane (7-10%), ont permis de prouver leur sûreté et leur fiabilité.

Des essais d'applications pour des concentrations élevées (5-10%) de drainage de gaz effectués pendant une période de 180 jours dans une avancée de charbon et dans une entrée supérieure d'air de retour d'un front de taille aux hydromines ont permis de relever un drainage moyen quotidien de 5 400 m^3 de gaz. Le ventilateur est un dispositif nouveau, capable de fonctionner de manière sûre et fiable dans les mines de charbon.

BEDEUTUNG DER SONDERBEWETTERUNGSSYSTEME IN GRUBEN UND EINE BEGUTACHTUNG "NICHT-PYROPHORER" LEICHTMETALL - LÜFTER.

Yan Peng, Lin Yilin, Yang Guangqi, Wang Wenlie, Lu Yuan

Der Vortrag beginnt mit einer allgemeinen Diskussion der Sonder-Bewetterungssysteme in Kohlegruben untertage und beschäftigt sich dann besonders intensiv mit der Beschreibung des neu entwickelten, nicht brennbaren, von einer Wasserturbine getriebenen Grubenlüfters aus Leichtmetallguß. Die Sicherheit und Zuverlässigkeit des Lüfters ist durch die Verbesserung der chemischen Zusammensetzung, Temperaturfeldmessungen beim Aufprall von Fremdkörpern und besonders durch Versuchseinsätze des Prototyps in hohen Methankonzentrationen (7-10%) bewiesen worden.

Bei Versuchseinsätzen in einer Gasabsaugung mit hoher Konzentration (5-10%) für eine Dauer von 180 Tagen in einem Streckenvortrieb und unter einem Einsatz in eine Wetterabzugstrecke in einer Hydrozeche konnte täglich eine durchschnittliche Gasabsaugung von 5400m^3 erreicht werden. Der Lüfter hat sich als Rekrutierungshilfe für Grubenarbeiter erwiesen, da er sicher und zuverlässig im Untertage-Einsatz ist.

ЗНАЧЕНИЕ ВСАСЫВАЮЩЕЙ ВЕНТИЛЯЦИОННОЙ СИСТЕМЫ В ШАХТАХ НА ПРИМЕРЕ НЕПИРОФОРНОГО ЛЕГКОСПЛАВНОГО ВСАСЫВАЮЩЕГО ВЕНТИЛЯТОРА

Ян Пенг, Лин Йилин, Янг Гуангки, Уанг Уенли и Ли Луан

Дается общий обзор всасывающих вентиляционных систем в подземных угольных шахтах. Описывается недавно сконструированный непирофорный вентилятор, построенный из литого легкого сплава и приводимый в движение водяной турбиной. Безопасность и надежность этого вентилятора были повышены двумя факторами: 1) посредством улучшения химического состава легкосплавного литья и 2) путем измерения температуры поля, которая обусловлена столкновением и трением посторонних частиц. Они были хорошо проверены и подтверждены испытаниями в условиях высокой концетрации метана (7-10%). Описывается экспериментальное применение в течение 180 дней высококонцентрированного (5-10%) газового дренажа в очистном забое и в верхнем отверстии для исходящей вентиляционной струи фронта очистных работ в гидрошахтах, когда был зарегистрирован средний дренаж газа 5400 кубических метров в день.

Session V

VENTILATION

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CONTROL OF METHANE ACCUMULATIONS HAZARD IN THE RETURNS OF LONGWALLS WITH CAVING BY THE USE OF SPECIAL VENTILATION MEANS

P. Krzystolik and Z. Kobiela
Central Mining Institute - Experimental Mine "Barbara", Poland

ABSTRACT

The paper presents investigation results "in situ" on formation methane concentrations in gobs areas of advancing and retreating longwalls with caving, with different ventilation systems.

These investigation results allowed the estimate the formation of hazardous methane accumulations and to introduce in the mining praxis special technical measures and ventilation systems for eliminating the methane hazard in the longwall area.

The paper presents also investigation results received in mines after using the measures of suppression the dangerous methane accumulations in the longwall areas. Some examples of ventilation arrangements, auxiliary devices for decreasing the methane concentrations, principles of control and lay-out of stationary CH_4 monitoring sensors in dependence of the used combating systems are presented as well.

INTRODUCTION

In consequence of strata disturbances resulting from mining the primary gas balance is disturbed and methane emission from goaves increases. The factor, which causes this emission is the difference between the gas pressure in the cracked strata in the destressing zone and the air pressure in the adjacent workings. Investigations on the distribution of aerodynamic potentials in the extraction area have an essential meaning for the safety of work.

Sometimes during the caving process in the extracted area comes periodically to a delayed subsiding of roof rock layers on the debris of the immediate roof. The sudden cracks of those rocks are able to push out methane accumulated in goaves into the longwall area leading to a sudden increase of methane concentration and to a serious methane explosion hazard. The initiation of methane explosion can happen in the goaf or in the adjacent workings.

In the goaves as an initiator can be a mine fire especially in seams liable to spontaneous combustion or shotfiring not properly performed at forcing caving. On the longwall an initiator can be frictional sparking between the mining machines and hard roof or floor rocks or geological disturbances. This is confirmed by ignitions recorded in Polish mines.

In Polish mines as specially disadvantageous mining situations relating to methane explosion hazard according to conducted invetigations should be considered:

- crossings of the longwall upper section with the return, especially retreating longwalls with the "U" ventilation system, where the electric engines of conveyors are installed, transport of materials takes place and miners are present,

- longwalls and longwall headings in thin seams, where the capacity of workings can be not big enough to

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hold the great quantities of methane suddenly pushed out from the goaf into the longwall and adjacent workings area,

- un-profitable location of the longwall face relating to the seam inclination (example methane explosion at "Mysłowice" colliery) or on inclined longwalls where small quantities of air flowing across the longwall allow to form methane cords and local methane accumulations behind the support of longwall headings.

BASIC DIRECTIONS AND METHANE EXPLOSIONS COMBATING MEANS ON LONGWALLS

Investigations carried out on forming methane concentrations in goaves of longwalls allowed to determine methane explosions combating methods and to specify three basic activity directions:

1. Removing explosible methane concentrations of goaves from active workings, by the use of: methane drainage of goaves, regulation aerodynamic potentials, special ventilation systems and auxiliary ventilation devices.

2. Prevention of violent methane pushing out from goaves by early systematical caving and combating rockbursts using special shot-firing techniques.

3. Liquidation of methane ignition sources in goaves on longwalls and longwall roadways by mining with maximum recovery of coal (no coal left in the goaf), goaf sealing by smoke dusts and anhydrite strips, keeping high methane concentrations in goaves and filling the goaves by nitrogen.

But to the technical combating means of methane hazard beyond the longwall front belong:

- advanced methane drainage, current longwall drainage and goaf drainage
- reasonable formation of aerodynamical potentials in the ventilation net,
- application of special ventilation systems with properly selected methods and means of liquidation of local dangerous methane accumulations especially in the upper section of the longwall and on its crossing with the return.

This paper describes combating means presented in the last group, this is the use of special ventilation systems and auxiliary ventilation devices.

METHANE HAZARD SUPPRESSION ON LONGWALLS WITH CAVING BY THE USE OF SPECIAL METHODS AND VENTILATION MEANS

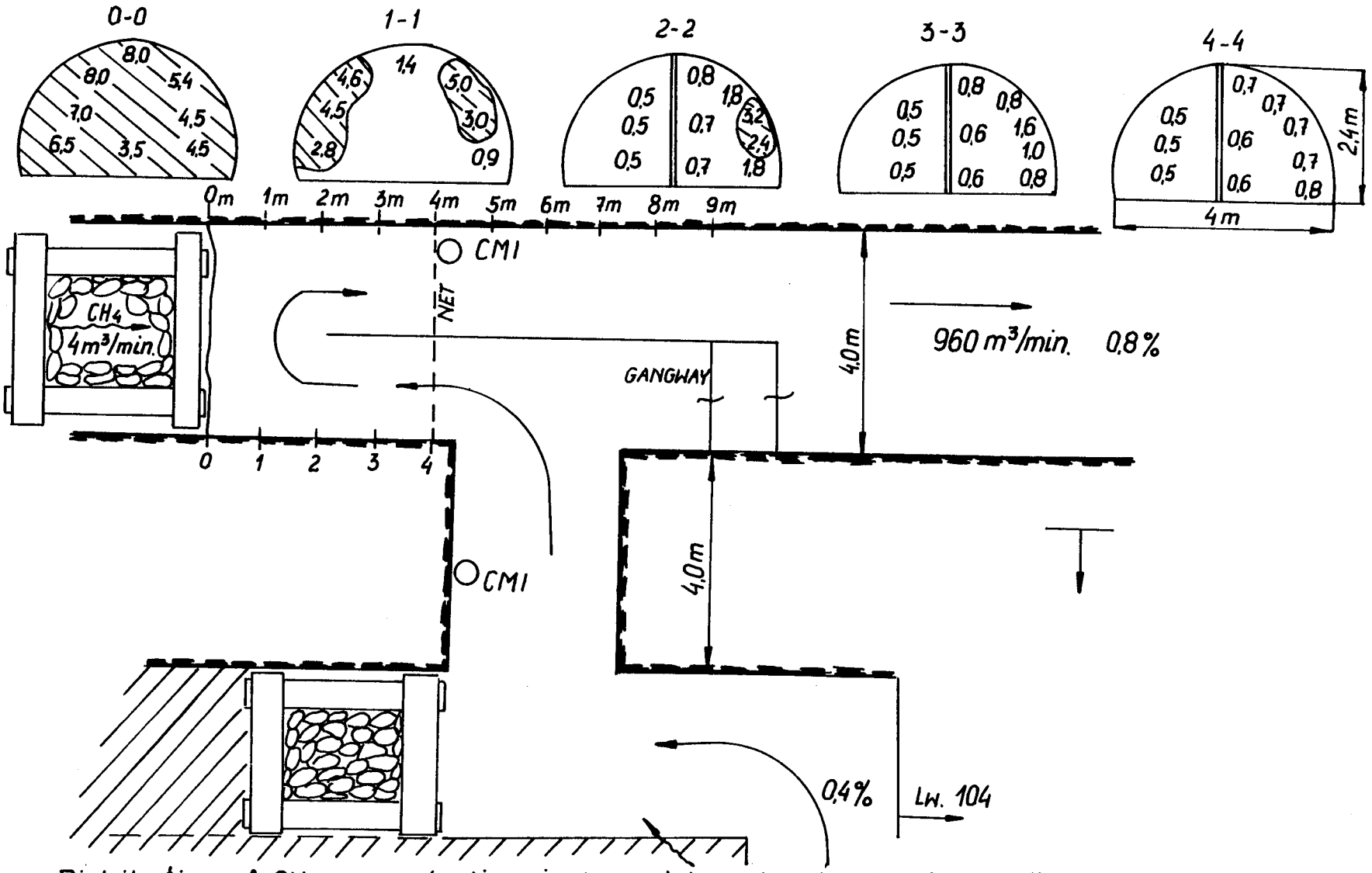
Investigations on methane hazard on longwall corners, where ventilation systems provide for the greatest potential methane explosion hazard, have been carried out on retreating longwalls with caving.

To give a picture of this hazard on the corners comparative investigations have been conducted on longwalls ventilated by the "Z" system, which is most advantageous in high gassy conditions but disadvantageous because of mine fire hazard in goaves.

Investigations have been conducted on longwalls where methane hazard has been combated not only by methane drainage, but also by the use of auxiliary ventilation devices.

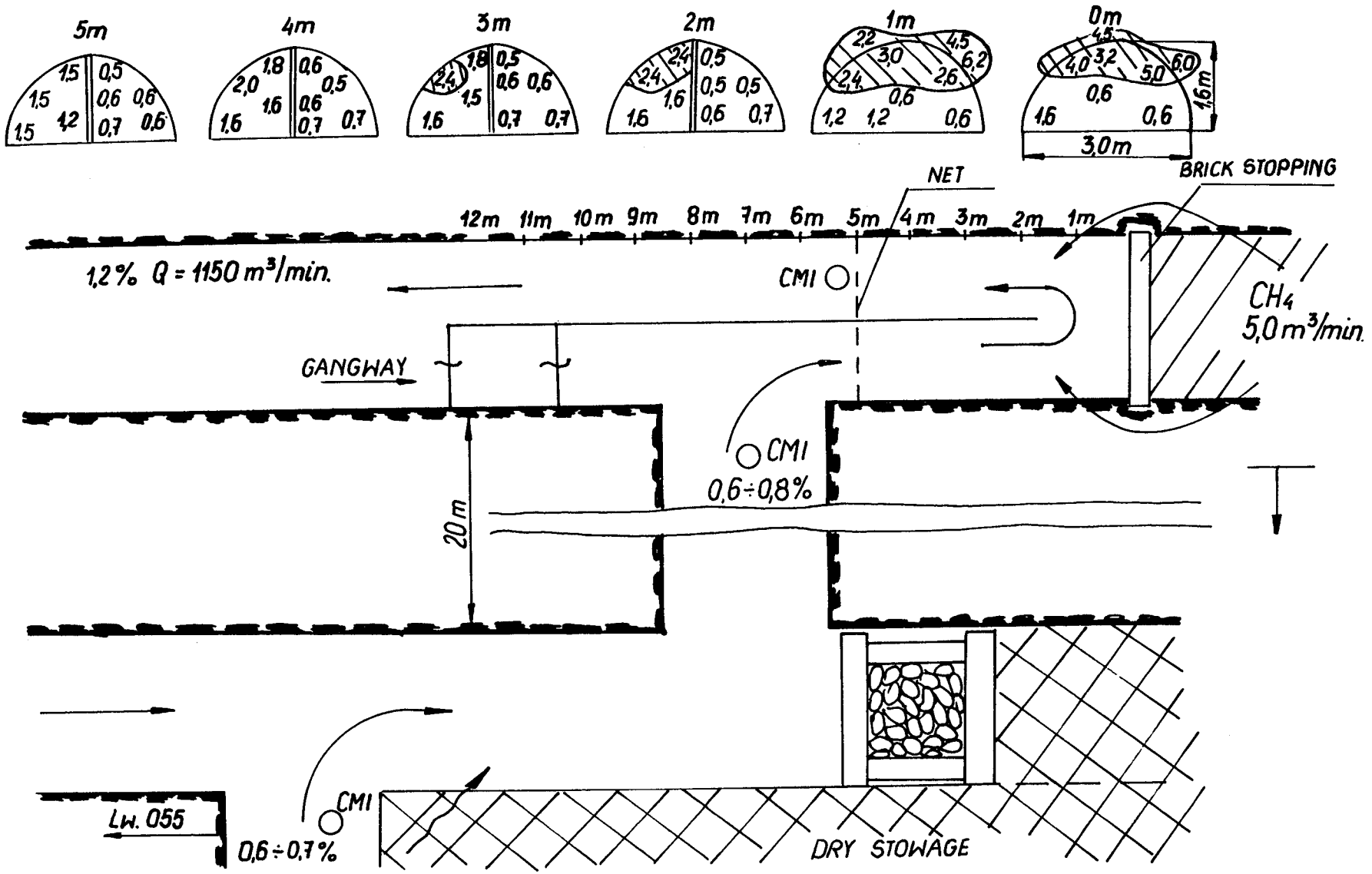
The effectiveness of undertaken means has been documented by measurements of air quantities and methane concentrations.

"Brzeszcze" colliery - longwall system with a 4 m coal strip presented on Fig.1 and longwall with double returns presented on Fig.2. In both cases a system with a so called "mixing chamber" was employed. The ventilation partition is turning the air 4-6 m back behind the longwall in the direction of the chock.



Distribution of CH_4 concentration in the mixing chamber for longwall 104, seam 352, level 360–512 m, „Brzeszcze" colliery.

Fig. 1



Distribution of CH_4 concentration in the mixing chamber for longwall 055, seam 364, level 430÷512 „Brzeszcze" colliery. Fig. 2

This enabled to keep the goaf on the longwall corners under compression and simultaneously to lead methane under control into the returns. The "mixing chamber" where methane concentrations were from 8% in the vicinity of the chock to 1.6% at the end of the chamber (at the net) enabled to use electrical equipment without fear in the separated part of the heading with low methane concentrations as well as safe work for the miners on the crossing of the longwall with the return. As a protection and for control of accessible places beyond the mixing chamber at the net under the roof an alarm methanometer with 2% threshold switching off electricity was installed.

The measurements showed that at the end of the chamber at all points of the perimeter of the heading cross-section methane concentrations have been nearly the same and didn't exceed 2%.

"Manifest Lipcowy" - analized have been two ventilation systems and the best combating methods employed:

- a retreating longwall ventilated by the "U" system with ventilation partition, short forcing airduct and fan WLP-600, and cyclones to liquidate the methane hazard (Fig.3),

- advancing longwall with "Z" ventilation system and return along the goaf with refreshing (Fig.4).

Investigations in this cases indicated, that the employed means for suppressing the methane hazard enabled a continous longwall mining.

"Halemba" colliery - longwalls ventilated by the diagonal "Y" system with refreshing (Fig.5 and 6). Additionally on one longwall the goaf was sealed off by an anhydrite strip, aimed at decreasing methane concentration in the goaf imediately at the longwall and in the return moving higher methane concentrations deeper into the goaf. This enabled to decrease the fire hazard in the goaf and progressive methane drainage from behind the stopping.

In all described and investigated cases, except employed local means of liquidation displacing hazardous methane concentrations, intensive methane drainage was used. Only the combination of both elements of combating the methane hazard enables safe mining of high gassy seams with a retreating longwall system with a greater rate of face advance and a higher production concentration.

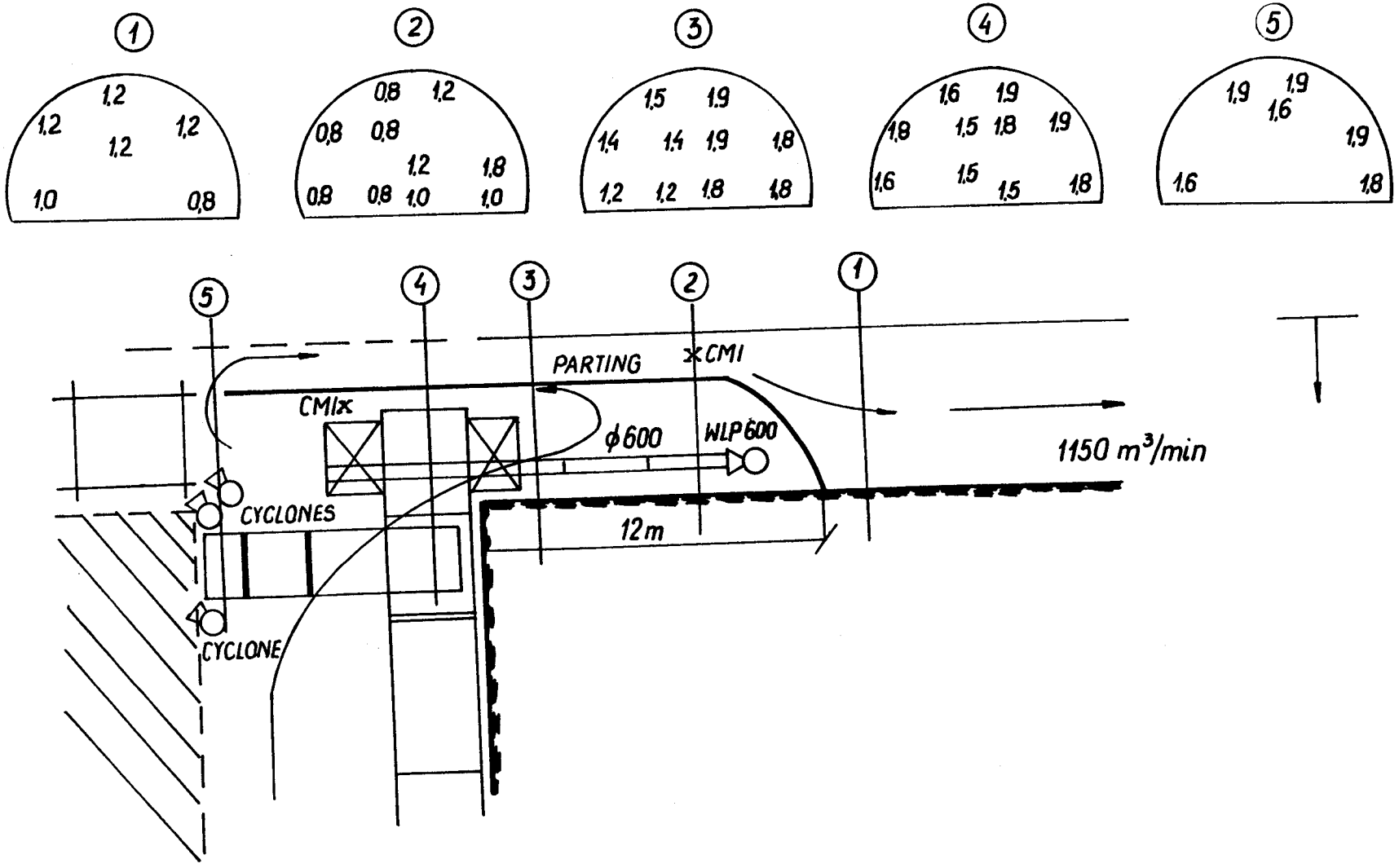
CONCLUSIONS

1. Longwall exploitation in high gassy seams in different mining and geological conditions needs often different technical means for methane hazard suppression. This especially deals with the seam penalling and employing the most advantageous ventilation scheme together with additional technical and organization means aimed at decreasing the methane hazard on longwall corners and in goaves.

2. The use of retreating longwalls with a returnable ventilation system "U" gives reasons for economical grounds, because of lower drivage costs of the single return and avoiding coal pillars, in comparison with the use of double returns.

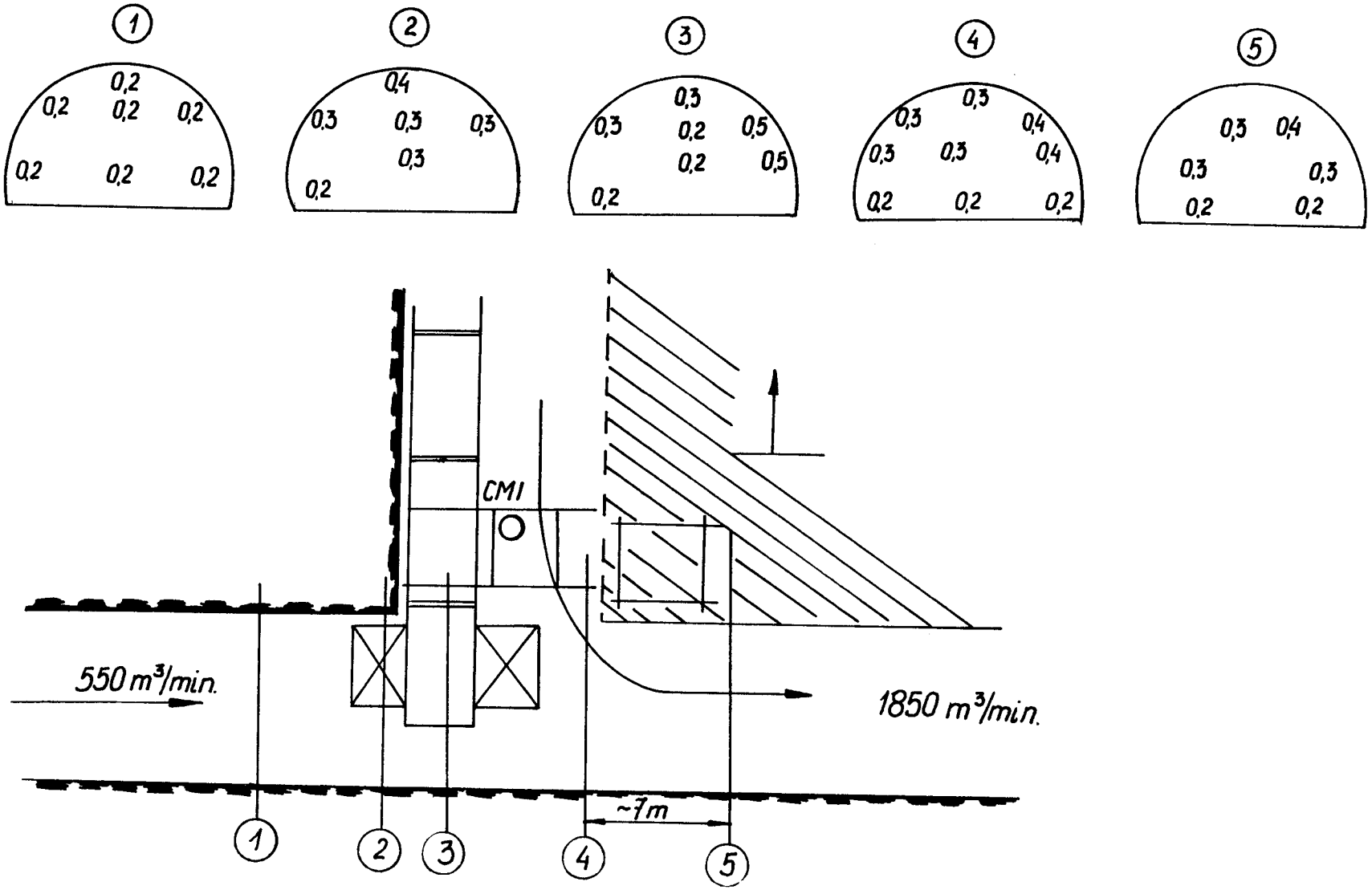
3. In the corner area of the mined longwall should be employed methods aimed at liquidation of zones with dangerous methane concentrations. Their liquidation by the use of different auxiliary ventilation devices needs great discipline in their application, especially a proper control of their operation.

   Particularly should be assured a stable operation form of the whole ventilation system on the longwall corner and a continous control of the methane hazard by the "fast" methanometry.



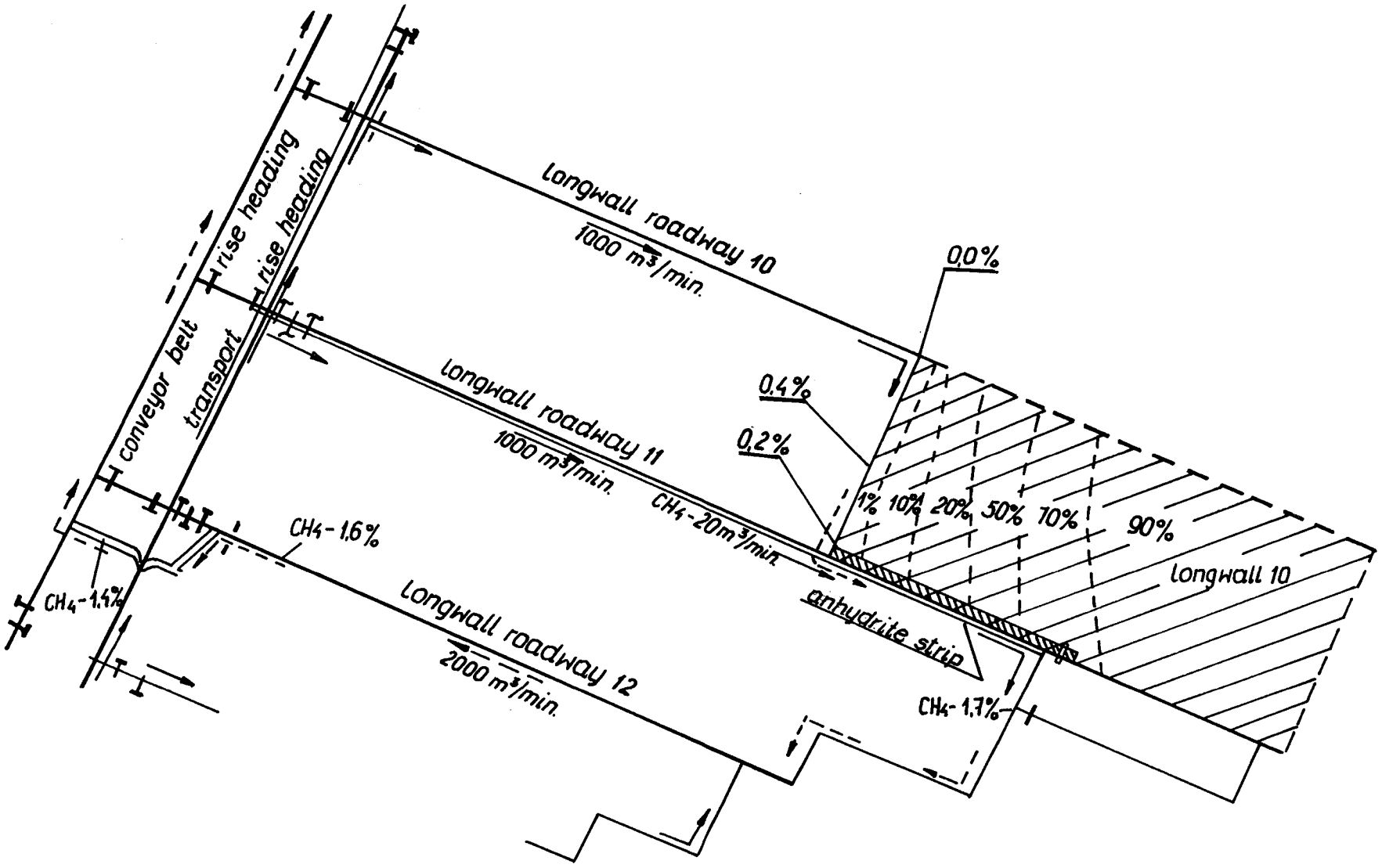
Distribution of CH_4 concentration at the corner of longwall D-5, seam 404/2 „Manifest Lipcowy" colliery.

Fig. 3



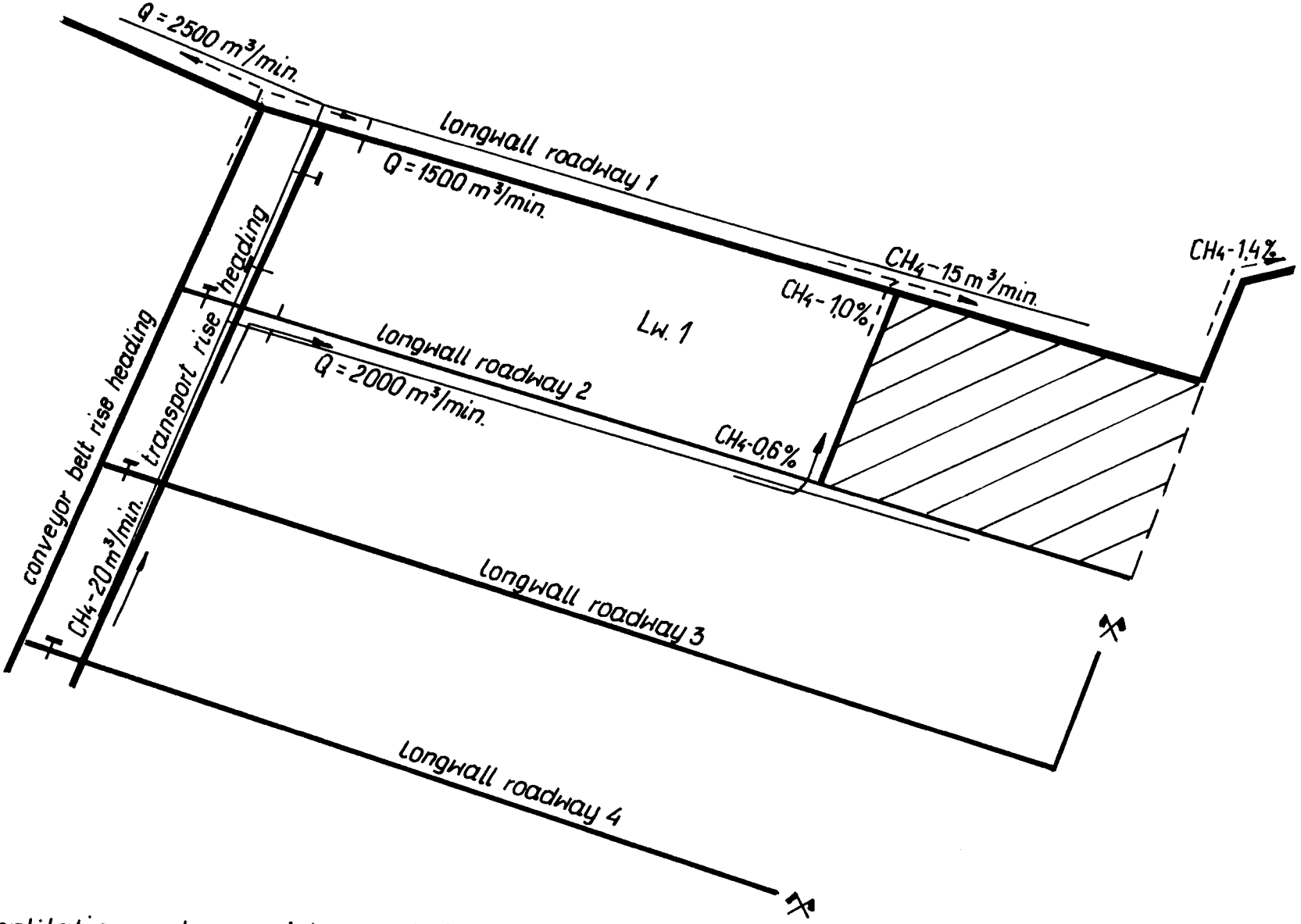
Distribution of CH_4 concentration at the corner of longwall F-17, seam 404/2 „Manifest Lipcowy" colliery.

Fig.4



Ventilation scheme of longwall 10, seam 506 „Halemba" colliery

Fig. 5



Ventilation scheme of longwall 1, seam 506 „Halemba" colliery.

Fig. 6

4. In high gassy seams to keep a high level of safety and to get great longwall advance rates with continous operation of devices (without switching off the eletricity) it is necessary to use complex combating methods - methane drainage of strata, seam, goaves and ventilation means decreasing methane concentrations in the longwall area at simultaneous prevention of fires in goaves.

CONTROLE DU RISQUE D'ACCUMULATION DE METHANE DANS LES FRONTS DE LONGUES TAILLES RABATTANTES A SOUS-CAVAGES PAR DES SYSTEMES DE VENTILATION SPECIAUX

P.A. Krzystolik, Z. Kobiela

Ce rapport présente les résultats d'études "in situ" sur la formation de concentrations de méthane dans les zones de remblais d'avancées et de rabattage de longues tailles à sous-cavage; ces études ont été effectuées avec différents systèmes de ventilation.

Ces résultats ont permis d'évaluer la formation d'accumulations dangereuses de méthane et d'introduire dans le processus habituel des exploitations minières des mesures techniques et des systèmes de ventilation spéciaux pour éliminer le risque causé par le méthane dans les zones de longues tailles.

Notre rapport présente également les résultats d'investigations menées dans les mines après l'utilisation de méthodes de suppression des accumulations dangereuses de méthane dans les zones de longues tailles. Nous donnons des exemples de disposition de systèmes de ventilation, d'installations auxiliaires diminuant la concentration de méthane, de principes de contrôle et de placement de détecteurs de surveillance fixes CH_4.

BEHERRSCHUNG VON GEFÄHRLICHEN METHANANSAMMLUNGEN IN DEN ABWETTERN VON STREBEN MIT BRUCHBAU DURCH DIE ANWENDUNG SPEZIELLER BEWETTERUNGSMETHODEN

P. A. Krzystolik, Z. Kobiela

Der Vortrag stellt Untersuchungsergebnisse zur Bildung von Methangas-Ansammlungen in Bruchgebieten von Streben mit Vor- und Rückbau mit Bruchbau vor, bei denen verschiedene Bewetterungs-Methoden angewandt werden.

Die Untersuchungsergebnisse ermöglichen eine Vorausabschätzung der gefährlichen Methangasansammlungen und die Einführung spezieller technischer Maßnahmen und Bewetterungsmethoden zur Beseitigung der Gefahren im Strebbereich.

Der Vortrag stellt ebenso Ergebnisse von Untersuchungen nach der Durchführung der genannten Gegenmaßnahmen auf Gruben vor. Außerdem werden einige Beispiele zur Bewetterung, Hilfs-Einrichtungen zur Verringerung von Methankonzentrationen und Prinzipien der Kontrolle und Verteilung von stationären CH_4-Monitoren vorgestellt.

МЕТОДЫ БОРЬБЫ С ОПАСНЫМИ СКОПЛЕНИЯМИ МЕТАНА В ВЫРАБОТАННЫХ И ОБРУШЕННЫХ ПРОСТРАНСТВАХ ДЛИННЫХ ЗАБОЕВ ПОСРЕДСТВОМ ИСПОЛЬЗОВАНИЯ СПЕЦИАЛЬНЫХ ВЕНТИЛЯЦИОННЫХ СИСТЕМ

П. Кржыстолик и З. Кобиела

Настоящий доклад представляет результаты полевых исследований образования изначальных концентраций метана в выработанных и обрушенных пространствах прямой и обратной выемки путем применения различных вентиляционных систем.

Результаты этих исследований помогли определить концентрацию метана, превышающую безопасный уровень. Также обоснованы специальные технические меры и вентиляционные системы для предотвращения опасности скоплений метана в длинном забое.

В докладе также описываются результаты исследований, проведенных в шахтах после введения мер, не допускающих опасного скопления метана в длинных забоях. Дается несколько примеров вентиляционных систем и дополнительных приспособлений для снижения метановых концентраций. Также обсуждаются принципы контроля и расположения стационарных датчиков слежения за уровнем метана, которые зависят от вида использованных систем.

用特别通风装置来控制有塌落带的长壁回风巷中有害甲烷的积聚

P．A． 克里席恩多理克，Z．科比拉

本文叙述了在有塌落的前进和后退式长壁中用不同的通风设备时，废石区甲烷浓度的形成现场测试结果．

这些测试结果可用于估计有害的甲烷气体积聚的过程，也可用于引进各种特殊技术措施和通风设备以消除长壁内甲烷的危害．

本文还叙述了在矿井中长壁区域使用了有害甲烷控制设施后的测定结果．同时还举例说明了通风设备的布置，降低甲烷浓度的附属设备，固定式甲烷探测传感器的控制与布置．

THE SUITABILITY OF METHODS USED TO DETERMINE THE SENSITIVITY OF THE AIR VOLUME OUTPUT TO CHANGES IN THE RESISTANCE OF SIDE-BRANCHES IN THE VENTILATION CONTROL OF DEEP COLLIERIES

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Technical University of Silesia, Poland

ABSTRACT

The paper deals with a method of determining the sensitivity of the air volume output to changes in the resistance of side-branches in the ventilation system of a mine. By sensitivity we understand the limit value of the differential quotient $\Delta\dot{V}_i / \Delta R_r$, where $\Delta\dot{V}_i$ is the change of air volume output in the i-th side-branch occurring when the resistance ΔR_r is changed in the r-th side-branch. The sensitivity is determined by means of a set of equations of the network, differentiating these equations and solving the obtained set of linear equations. This method of determining the sensitivity has been interpreted to be a network. Changes in the parameter of the side-branch are modelled as a source of disturbances in the flow of air in the affiliated network having the same structure as the basic one. The fluxes in the affiliated network are the sensitivities which are to be found. In networks with normal structures /parallel in series/ the plus or minus signs of sensitivity are obvious. In networks of diagonal structure ambiguous situations may occur. A change of resistance in any selected side-branch may lead to an increase or drop of the air volume output of air in some other side-branch.

INTRODUCTION

Due to the hazards of mining,the ventilation control in deep collieries demands an adjustment of air supply to the needs of the respective production sections, opening sections and stalls. Changes in the disturbution of the working front make it necessary to change also the parameters and even the distribution of installations which control the circulation of air. Even if the installation fails,it is important that the air is forced to flow in the desired direction and at a proper rate in the respective headings. This task is realised by changing the operating conditions of the gauge doors, main fans and auxiliary fans. Decisions concerning necessary changes in the ventilation system must be based on the knowledge and experience of ventilation experts as well as on computer-aided calculations aiming at the achievement of optimal data for the given conditions, taking into account the circulation of air and pressure in the colliery. As far as computer-aided calculations are concerned, various methods of determining the required parameters of installations controlling the distribution of air in the mine have been applied.

When the properties of the ventilating system of a colliery is being investgated, as well as in the course of carrying out the aforesaid calculations, such methods may come in useful in which the coefficients of the sensitivity of the air volume

output influencing the change of parameters of control systems are utilized, particularly those affecting the resistance of the air-stoppings. Speaking of the air volume output we must understand the following limiting value of the differential quotient:

$$\acute{e}_{ir} = \lim_{\Delta R_r \to 0} \frac{\Delta \dot{V}_i}{\Delta R_r} = \frac{d\dot{V}_i}{dR_r} \qquad (1)$$

where:

$\Delta R_r = R_r - R_{ro}$ - change of the parameter of the r-th element of the system,

R_{ro} - initial value of the parameter of the r-th element of the system,

$\Delta \dot{V}_i = \dot{V}_i - \dot{V}_{io}$ - change in the air volume output in the i-th side-branch of the system, m^3/s,

$\dot{V}_{io}$ - initial value of the air volume output in the i-th side-branch, m^3/s,

$i = 1,2,\ldots,m$

m - number of side-branches in the system.

The sensitivity $\acute{e}_{ir}$ is, therefore, a value of the differential coefficient $d\dot{V}_i/dR_r$ of the air volume output $\dot{V}_i$, depending on the chosen parameter of the r-th side-branch, calculated for the actual state of operation of the system.

The air volume output in each i-th side-branch may react to changes of the parameters of many elements of the system. Thus, we may speak of a matrix of sesitivity:

$$\mathbb{E} = [\acute{e}_{ir}] \qquad (2)$$

$i = 1,2,\ldots,$ m - number of side-branches,

$r = 1,2,\ldots,$ - number of parameters of the elements in the system.

A method of determining any row of the matrix $\mathbb{E}$ in the case of problems connected with the ventilation system of a colliery has been suggested by J.Chojcan (1975). The determination of any column of this same matrix has been dealt with by M.Kolarczyk (1980)

DETERMINATION OF ANY COLUMN OF THE MATRIX OF SENSITIVITY OF THE AIR VOLUME OUTPUT

The sensitivity of the air volume output $\acute{e}_{ir}$ to changes in the resistance R_r of the r-th side-branch can be determined by means of the set of equilibrium of the system.
Having differentiated this set of equations, we obtain:

$$(3)\ \begin{cases} \text{-from the equation for the nodes of the network:} \\ \sum_{i=1}^{m} s_{ji} \cdot \acute{e}_{ir} = 0 \\ j = 1,2,\ldots,n-1 \\ \text{-from the equation for the mesh:} \\ \sum_{i=1}^{m} c_{ki} \cdot \bar{R}_i \cdot \acute{e}_{ir} = \begin{cases} 0 \text{ if the k-th mesh does not comprise the r-th side-branch} \\ -c_{kr} \cdot \dot{V}_r \text{ if the k-th mesh includes the r-th side-branch} \end{cases} \\ k = 1,2,\ldots,\nu \end{cases}$$

where:

s_{ji}, c_{ki} are elements of the matrix of node - side-branch $\mathbb{S}$ and mesh-side-branch $\mathbb{C}$ incidence, assuming the values 1, -1, 0, resulting from the kind of the affiliation of the i-th side-branch to the j-th node or k-th mesh of the network,

n - number of nodes,

$\nu = m - n + 1$ - number of independent meshes in the network,

$\bar{R}_i$ - coefficient assuming the following values (4):

$$(4)\quad \bar{R}_i = \begin{cases} 2 \cdot R_i \cdot \dot{V}_i & \text{for side-branches with resistance} \\ -\dfrac{dH_i}{d\dot{V}_i} & \text{for side-branches with a fan} \end{cases}$$

R_i – resistance of the i-th side-branch, kg/m^7,

$\dot{V}_i$ – expenditure of air in the i-th side-branch at the actual state of operation of the system,

$\dfrac{dH_i}{d\dot{V}_i}$ – value of the differential coefficient of pressure concentration of the fan H_i in relation to the air volume output $\dot{V}_i$, calculated for some given point of operation in the characteristics H(V) of the fan.

It will be assumed in these calculations that the fans always operate in the dropping /lower/ part of the characteristics. Then the following relation comes true:

$$\bar{R}_i = -\frac{dH_i}{d\dot{V}_i} > 0 \qquad (5)$$

In the set of equations (3) the sensitivity $\acute{e}_{ir}$ will be treated as an unknown quantity, the other coefficients being known from the actual state of operation of the system. The number of equations amounts to:

$$n - 1 + y = n - i + m - n + 1 = m$$

and equals the number of unknown quantities. Because of the unknown quantities $\acute{e}_{ir}$, $i = 1,2,\ldots,m$, this set of equations is not a linear one. It can be solved making use of the generally applied methods. In the case of large systems computers should be used.
The determined values of $\acute{e}_{ir}$ may come in useful when evaluating the effect of the change of resistance R_r in the r-th side-branch on the distribution of air in the system A positive value of $\acute{e}_{ir}$ indicates an increase of the air volume output $\dot{V}_i$ accompanied by an increase of the resistance R_r and vice versa. The sensitivity $\acute{e}_{ir}$ may be subjected to further computer-aided analysis and used in subsequent optimising calculations of the system.

NETWORK INTERPRETATION OF THE METHOD OF DETERMINING THE SENSITIVITY OF THE AIR VOLUME OUTPUT

The obtained set of equations (3) for the determination of the sensitivity of the air volume output $\acute{e}_{ir}$ to changes in the r-th resistance may be interpreted to be like a network, as presented also by Frycz (1988) and Kolarczyk (1980). They both describe the flux in the system, further on called the affiliated system. The structure of the affiliated system is identical with that of the basic system, as the coefficients s_{ji} and c_{ki}, and thus also their structural matrices $\mathbb{S}$ and $\mathbb{C}$ of both these systems, are the same. Each group of equations of set (3) may be treated as equations for the nodes and meshes of the affiliated system. The values of $\acute{e}_{ir}$ are the fluxes in this system which are sought for. While investigating an optionally chosen state of operation of the basic system, the values of R_i, $\dot{V}_i$, $dH_i/d\dot{V}_i$ are known and stationary. Thus also the coefficients $\bar{R}_i$ for the side-branch of the affiliated system may be treated as being known and constant. In any other state of operation of the basic system the values of R_i, $\dot{V}_i$, $dH_i/d\dot{V}_i$ are, however, different, so that the coefficients $\bar{R}_i$ in the affiliated system have new values.
The shape of the relation $\bar{R}_i \cdot \acute{e}_{ir}$ occurring in the equations for the mesh (3) justifies the assumption that the affiliated system is a linear network, the coefficients $\bar{R}_i$ expressing the resistance of the side-branches in this system. The free expressions $\dot{V}_r^2$, which are to be

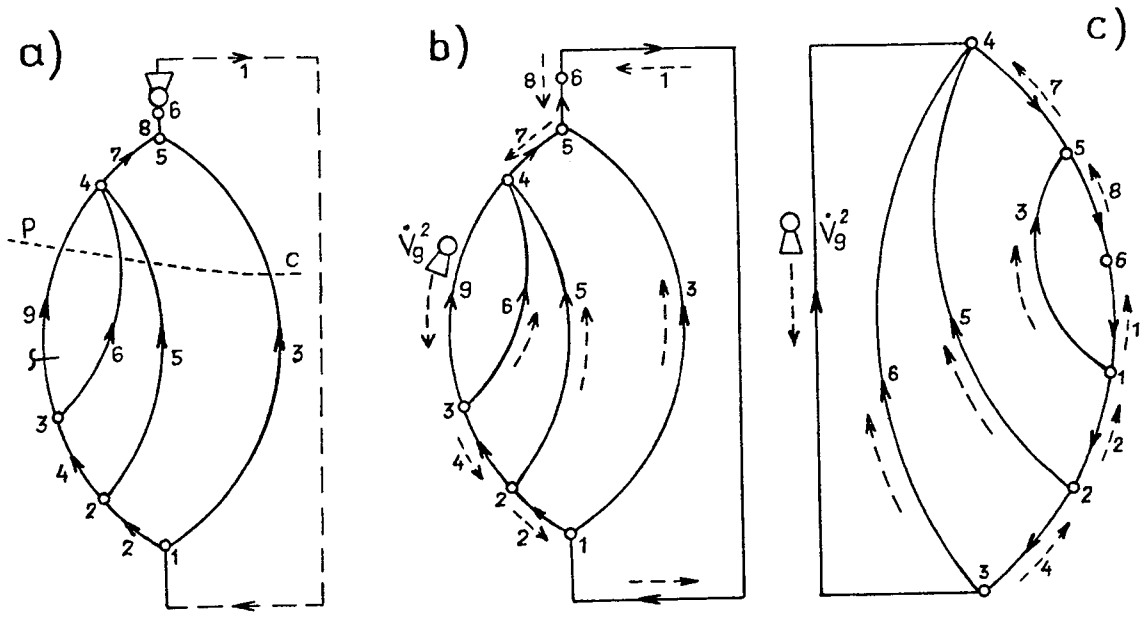


Fig. 1. Basic and affiliated network for the purpose of determining the sensitivity in normal networks.

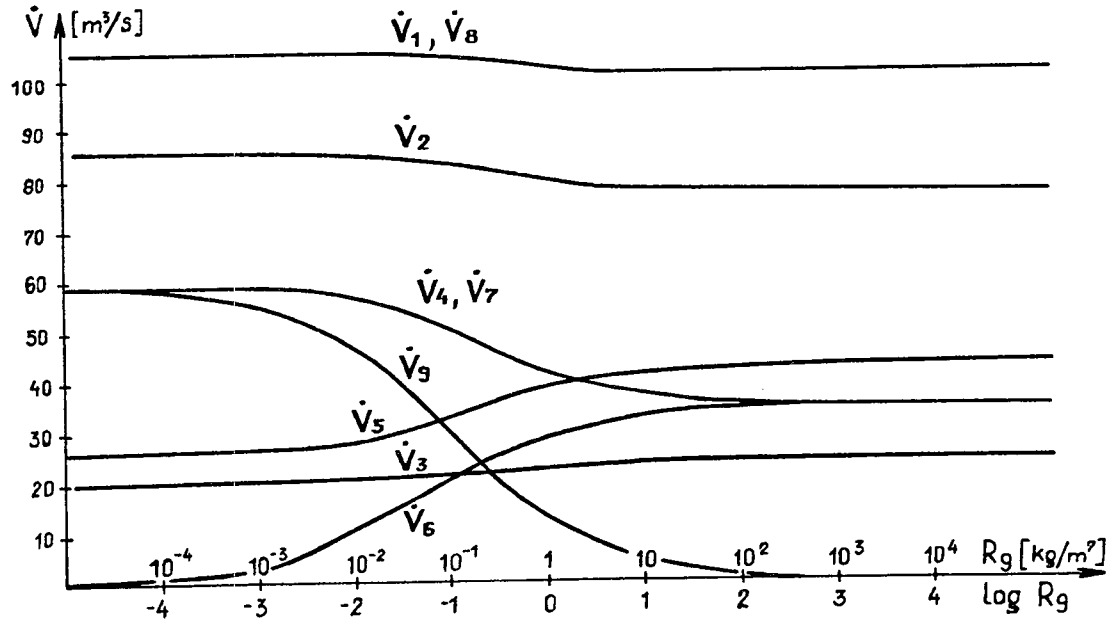


Fig. 2. The effect of changes of the resistance R_g on the air volume output $\dot{V}_t$ in the network presented in Fig. 1.

found in some equations (3) for the mesh of the affiliated system, comprising the r-th side-branch, may be treated as a source situated in the r-th side-branch which forces the flux in the affiliated system. An analysis of the sign at $\dot{V}_r^z$ in the set of equations (3) indicates that this source forces in the affiliated system the flux $\acute{e}_{rr}$ in the opposite direction to the flow of air in the r-th side-branch of the basic system. The sensitivity $\acute{e}_{rr}$ of the air volume output $\dot{V}_r$ to changes of the resistance R_r is a negative one, which is quite obvious, as an increase of R_r in the basic system leads to a drop of the air volume output $\dot{V}_r$. Thus, it is to be supposed that the solution of equations (3) in order to determine the sensitivity $\acute{e}_{ir}$ of the air volume output to changes in the resistance of the r-th side-branch is equivalent with the solution of the affiliated system constructed in the way described above. In the case of computer-aided calculations this fact is inessential. If, however, other new properties of ventilation systems in collieries, resulting from the structure of the system, are to be investigated, it is most interesting to analyse the sensitivity $\acute{e}_{ir}$ basing on the analysis of the flow of air in the affiliated system.

THE SENSITIVITY OF THE AIR VOLUME OUTPUT IN NORMAL NETWORKS

An essential feature of the ventilation network of a coal mine is its structure. The great variety in the occurrence of the mined deposits makes it necessary to apply various ways of opening and extracting them. This, again, is the reason for the formation of different structures of the ventilation system. This concerns both collieries as a whole and the individual production sections. Many papers stress the significance of the structure of this network for the control of the aeration of the mine. The method chosen for the calculation of the respective problems of this kind influences also the regularity of the sensitivity of air volume outputs upon the resistance in lay-bies.

The most regular, and thus also the most obvious, properties are displayed by the so-called normal networks, in which the side branches and sections of the network are conencted in series and parallelly. An example of such a system with one fan has been given in Fig.1a. The influence of disturbances in such networks can be evaluated and determined making use of very simple methods. It can be shown up without much difficulty in which lay-by the expenditure of air will grow or drop. This example has been provided here with the purpose of comparing it with other ventilation systems.
Let the disturbance, in this case the increase of resistance, occur in branch 9, i.e.

$$R_9 \rightarrow R_9 + dR_9$$

The sensitivity of the air volume output

$$\acute{e}_{i9} = \frac{d\dot{V}_i}{dR_9} \qquad i = 1,2,\ldots,9$$

to the change of resistance may be determined by solving the proper set of equations (3). In the network interpretation this can be reduced to the calculation of the flux $\acute{e}_{i9}$ in the affiliated network presented in Fig. 1b. In branch 9 of this system the source forcing the flux $\acute{e}_{i9}$ with a value of $\dot{V}_9^2$ is to be placed. The resistances R_i of the side branches have a value corresponding to that provided by the formulae (4).
In Fig. 1b there have been plotted two double arrows:

- at the side branches, corresponding to the direction of the air volume outputs $\dot{V}_i$ from the basic network;
- beside the side branches /broken arrows/, corresponding to the directon of the fluxes $\acute{e}_{i9}$ in the affiliated network.

The compatibility of both arrows means that:

$$\acute{E}_{ig} = \frac{d\dot{V}_i}{dR_g} > 0$$

The air volume output grows in such a case with the increase of the resistance R_g. In the analysed example this concerns the air volume outputs $\dot{V}_3$, $\dot{V}_5$, $\dot{V}_6$ (Figs. 1b, 1c).

If the arrows point in the opposite direction, it means that:

$$\acute{E}_{ig} = \frac{d\dot{V}_i}{dR_g} < 0$$

The air volume output drops with the increase of the resistance R_g. This concerns the air volume outputs $\dot{V}_9$, $\dot{V}_4$, $\dot{V}_7$, $\dot{V}_2$, $\dot{V}_8$ and $\dot{V}_1$. The fluxes $\acute{E}_{ig}$ in the affiliated network illustrate the propagation of disturbances in the basic network due to the increased resistance R_g to $R_g + dR_g$.

In normal systems with only one fan the directions of the air fluxes are defined unequivocally. The structure of the affiliated network for the determination of $\acute{E}_{ir}$ does not undergo any change. In such a case the affiliated network is also a normal network with explicitly defined directions of $\acute{E}_{ir}$. Thus it is quite easy to divide the whole set of side branches into two subsets:

- side-branches, in which the air volume output has decreased ($\acute{E}_{ir} < 0$) They are situated along the main path from the down-cast shaft via a lay-by with in increased resistance to the outtake shaft (Fig.1a)
- side branches in which the air volume output grows ($\acute{E}_{ir} > 0$). These are the remaining side-branches situated in the entire cross-section "p-c" across the ventilation system (Figs. 1a, 1b).

The analysis of Figs. 1a and 1b, as well as the properties of normal networks indicate that the symbols of sensitivity $\acute{E}_{ir}$ are always explicitly determined (for the operation of the fan in the lower part of its characteristics). The symbol of the derivative $d\dot{V}_i/dR_r$ does not change, which indicates that the relations

$$\dot{V}_i = \dot{V}_i(R_r)$$

are always monotonic in their character. Examples of such relations for a normal network presented in Fig.1a may be seen in Fig.2:

- the air volume outputs $\dot{V}_9$, $\dot{V}_4$, $\dot{V}_7$, $\dot{V}_2$, $\dot{V}_8$ and $\dot{V}_1$ drop with the increase of the resistance R_g;
- the air volume outputs $\dot{V}_3$, $\dot{V}_5$ and $\dot{V}_6$, on the other hand ,increase with the growth of the resistance R_g.

The value of the sensitivity $\acute{E}_{ir}$ is the angle coefficient of the tangent to the given graph and manifests the rate of changes of the air volume output brought about by changes of the selected resistance.

ANALYSIS OF THE SENSITIVITY OF AIR VOLUME OUTPUTS IN DIAGONAL NETWORKS

A similar analysis can be carried out for diagonal networks. As we know, in the branch lines of these systems the direction of the flow of air is not determined unequivocally, being dependent on the actual conditions of depression and resistance. The affiliated network constructed with the purpose of determining the sensitivity of air volume outputs will also be a diagonal network with diagonal side-branches, in which the direction of flow, i.e. the symbol of sensitivity $\acute{E}_{ir}$, will not be defined unequivocally being dependent on the resistances $\bar{R}_i$ of the side-branches of the affiliated network. The diagonal character of the side-branch results also from the position of the source forcing the flux in the given system. In the affiliated network the source $\dot{V}_r^2$ which forces the flow is situated in the r-th side branch. In general this is another side-branch than the one with the fan in the basic network. In the affiliated

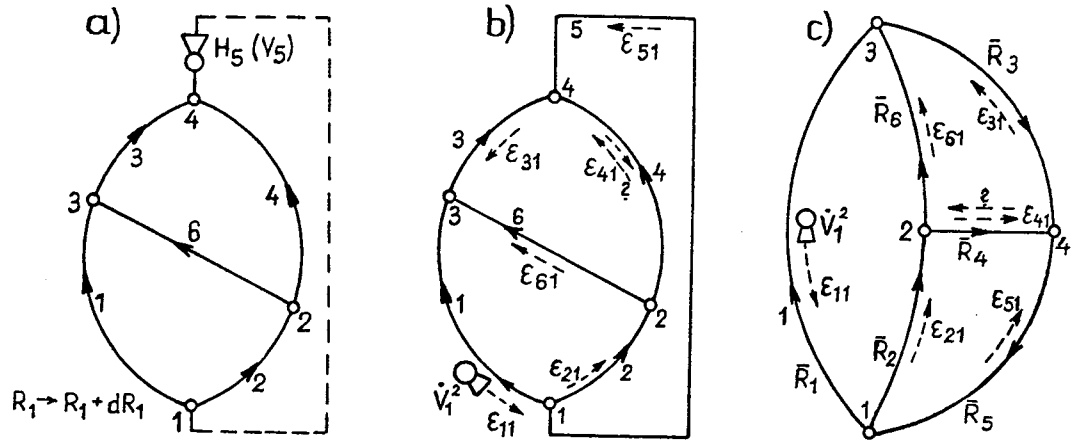


Fig. 3. Simple diagonal basic and affiliated network used to determine the sensitivity of the air volume output.

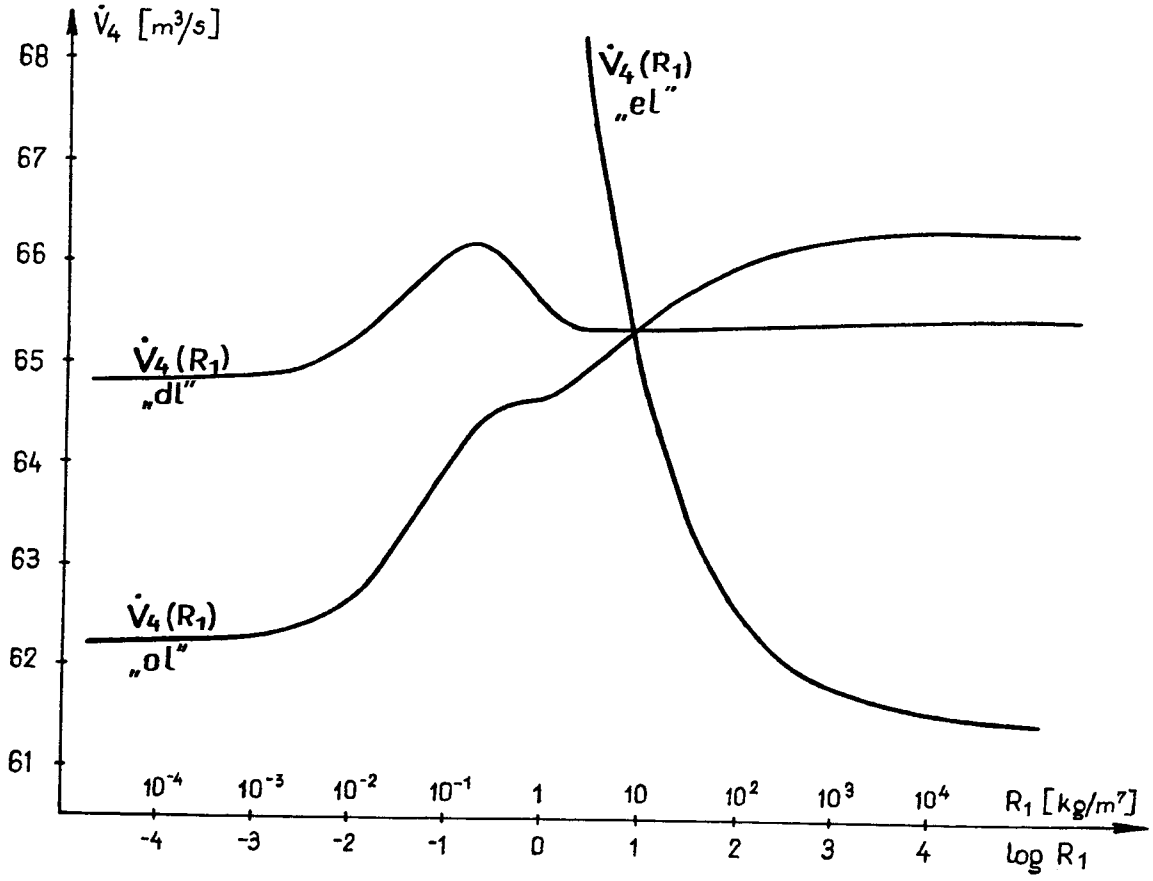


Fig. 4. Examples for the effect of changes of the resistance R_1 upon the air volume output $\dot{V}_4$ in the network shown in Fig. 3.

network other networks may, therefore display diagonal properties. Relying on one's intuition, not supported by calculations, is frequently deceptive in detecting these side-branches.

The simplest example of a diagonal network is one with a single diagonal side-branch (Fig. 3a). The suggested example of analysing the sensitivity of air volume outputs in such a network in the case of the occurrence of disturbances in the form of an increased resistance concerns side-branch 1. The investigated sensitivity $\acute{e}_{i1}$, i = 1,2,3,4,5 may be determined by solving a corespondingly prepared set of equations (3). Fig. 3b represents an affiliated network described by these equations. The source $\dot{V}_1^2$ forcing the fluxes $\acute{e}_{i1}$ operates in side-branch 1, providing positive sensitivities $\acute{e}_{21}$, $\acute{e}_{61}$, as well as negative ones $\acute{e}_{31}$, $\acute{e}_{51}$ (Fig. 3b). The symbol of sensitivity $\acute{e}_{41}$ may be either positive or negative, as side-branch 4 of the affiliated network is a diagonal, which can be easily seen in Fig. 3c.
Thus there may be cases in which the increase of the resistance R_1 is accompanied either by a growth or by a decrease of the air volume output $\dot{V}_4$. This ambiguity does not occur in normal networks.
In Fig. 4 we see three examplary functions expressing the relation between the air volume output $\dot{V}_4$ and the resistance R_1, deduced from the network shown in Fig. 3 a by simulating the distribution of air on a computer. Changing the resistance in side-branch 1 from 0 to ∞, alterations in the air volume output in side-branch 4 could be observed. These three differring functions have been deduced at changed resistances in other side-branches or at another gradient of the ventilation characteristics in side-branch 5. The function "ol" indicates that there may be cases when the air volume output $\dot{V}_4$ increases with the resistance R_1. The sensitivity $\acute{e}_{41}$ is then a positive one. The functon "el" displays a different character. When the resistance R_1 increases, the air volume output $\dot{V}_4$ drops, and then the sensitivity $\acute{e}_{41}$ is negative. The character of the function "dl" is non-monotonic and has a rather distinct maximum point and a faintly visible minimum. In some ranges the sensitivity $\acute{e}_{41}$ is positive, in others negative, at two points being equal to zero. The conditions determining the symbol $\acute{e}_{41}$ in the analysed network have been stated in the already quoted papers by Frycz and Kolarczyk (1980, 1988). A similar phenomenon of the ambiguity of sensitivity concerns $\acute{e}_{14}$ as well $\acute{e}_{23}$ and $\acute{e}_{32}$, because in affiliated networks these pairs of side-branches run diagonally in relation to each other.
The knowledge of the sensitivity facilitates the determination of the character of the relation $\dot{V}_i(R_r)$ as well as the preliminary forecasting of the results of changes of the resistance R_r. The sensitivities may be utilized dealing with some problems of optimizing the control of the distribution of air in the network.
In normal networks with one fan of this kind such peculiarities do not occur. The structure of the network is, therefore, of essential importance also in the case of controlling the distribution of air in the colliery.

Another example of a diagonal network with two diagonal side-branches in the basic network has been presented in Figs. 5. An increase of resistance in side-branch 9 (Fig. 5a) gives rise to disturbances in the air distribution, which can be analysed in the affiliated network (Figs. 5b and 5c). Some sensitivities have been determined unequivocally, viz.:

- positive values appear in $\acute{e}_{29}$ and $\acute{e}_{39}$, which means that the air volume outputs $\dot{V}_2$ and $\dot{V}_9$ do always grow with the increase of the resistance R_9;
- negative values appear in $\acute{e}_{59}$, $\acute{e}_{89}$, $\acute{e}_{19}$ and $\acute{e}_{99}$; this indicates that the air volume outputs $\dot{V}_5$, $\dot{V}_8$, $\dot{V}_1$ and $\dot{V}_9$ become smaller when the resistance R_9 increases.

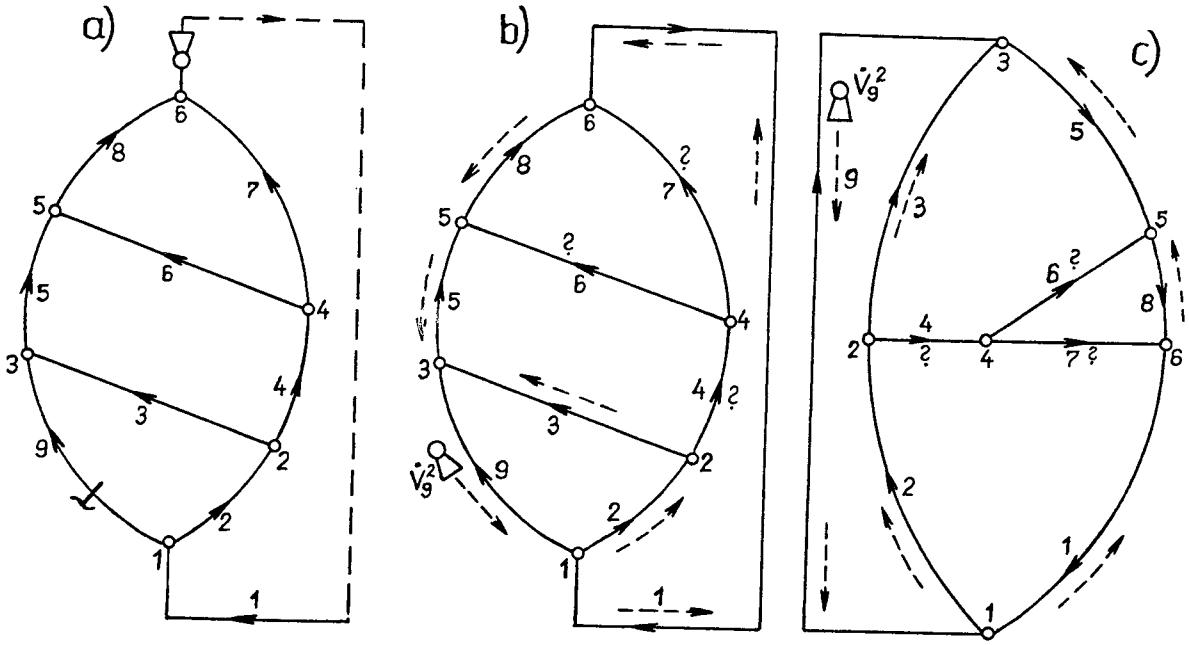


Fig. 5. Another example of a diagonal basic and affiliated network used to determine the sensitivity of the air volume output.

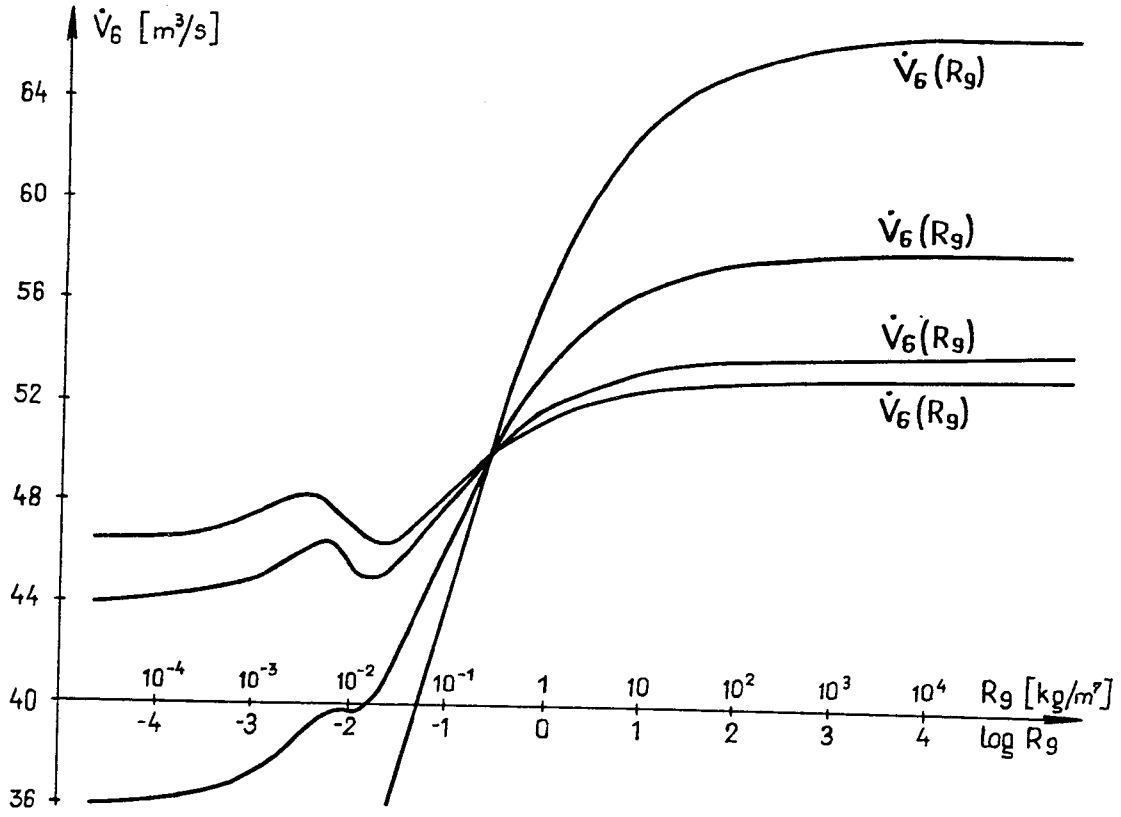


Fig. 6. Examples for the effect of changes of the resistance R_9 on the air volume output $\dot{V}_6$ in the network presented in Fig. 5.

The side-branches 4, 6 and 7 in the affiliated network (Figs. 5b and 5c) are diagonal in their character. As has been said earlier in this paper, the symbols of sensitivity $\acute{e}_{49}$, $\acute{e}_{69}$, $\acute{e}_{79}$ may be determined ambiguously, as shown by the exemplary functions of the relations $\dot{V}_6(\log R_9)$ in Fig. 6, and $\dot{V}_7(\log R_9)$ in Fig. 7. These graphs were plotted analogically to the cases discussed above.
The relation $\dot{V}_6(\log R_9)$ in Fig. 6 is not always monotonic. There may be intervals, in which $\acute{e}_{69}$ takes negative values, which fact confirms the aforesaid remarks concerning the diagonality of side-branch 6 in the affiliated network.
The relation $\dot{V}_7(\log R_9)$ in Fig. 7 can, as we see, either increase or decrease. Attempts to forecast the character of this relation not supported by calculations may be deceptive. If information concerning $\acute{e}_{79}$ is available, it becomes possible to determine the type of this relation and to decide about any planned changes in the network.
The presented examples of the ambiguity of effects brought about by changes in the resistance of side-branches upon the distribution of air in the ventilation system of a colliery do not give a comprehensive picture of this complex and interesting problem. A diagram of the affiliated network needen't be plotted when the respective sensitivities are being determined. The provided graphs are to serve merely the purpose of illustrating the essence of this phenomenon. In practice the suggested calculations may be used to set up a full matrix of the sensitivity of air volume outputs to changes of the parameters of control units adjusting the distribution of air in the network.

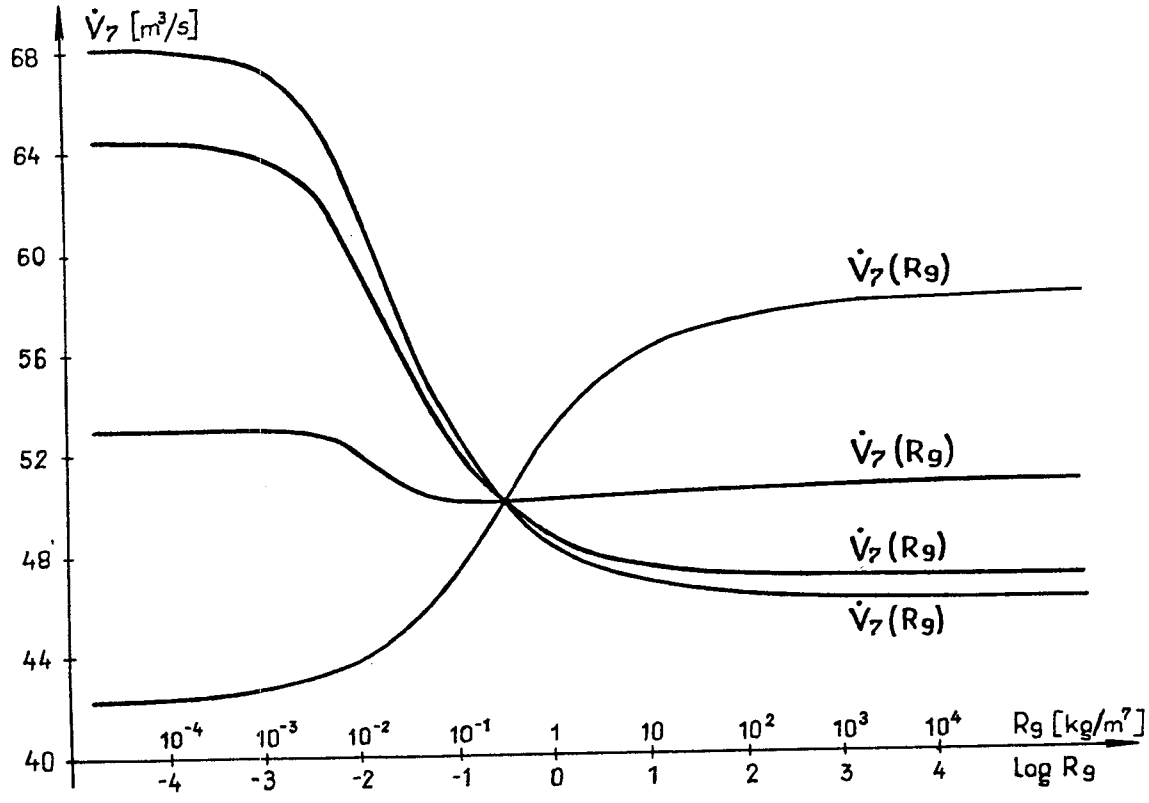


Fig. 7. Examples for the effect of changes of the resistance R_o on the air volume output $\dot{V}_7$ in the network shown in Fig. 5.

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UTILISATION DE METHODES POUR DETERMINER LA SENSIBILITE DE L'ARRIVEE D'AIR AUX MODIFICATIONS DE LA RESISTANCE DES GALERIES LATERALES A LA DIRECTION DE LA VENTILATION DANS LES MINES PROFONDES

A. Frycz et M. Kolarczyk

On modifie la répartition de l'arrivée d'air dans les systèmes de ventilation miniers en régularisant les arrêts et les pressions de sortie des ventilateurs. L'influence de ces modifications sur la quantité d'air de ventilation a donc été déterminée par calculs ou par mesures in situ. Afin de mieux évaluer les modifications qui se produisent, on a établi une méthode pour déterminer la sensibilité de la circulation d'air aux changements de résistance des galeries latérales. La sensibilité se définit comme la limite du quotient différentiel $\Delta\dot{v}_j/\Delta R_i$; $\Delta\dot{v}_j$, c'est à dire la modification de la sortie de l'air dans la galerie côté j-th, provoque une modification ΔR_i de la résistance du côté i-th de la galerie. On détermine ces sensibilités en différenciant un système d'équations d'équilibre du réseau et en résolvant le système des équations linéaires qui en résultent. Les modifications qui se produisent dans les paramètres de la galerie latérale sont représentées comme des sources perturbant la circulation d'un réseau auxiliaire de structure semblable à celle du réseau de base. La sensibilité recherchée est la circulation d'air du réseau auxiliaire. Dans les réseaux à structures normales/en série-parallèles, les manifestations de cette sensibilité sont claires. Dans les réseaux à structures diagonales, des situation ambigues peuvent se produire : par exemple, une modification de la résistance de la galerie latérale en question peut accroître ou réduire l'arrivée d'air d'une autre galerie latérale; sans calculs de réseaux, on ne peut pas déterminer ceci à l'avance. La manifestation de la sensibilité indique une augmentation ou une diminution de l'arrivée d'air, tandis que la valeur de la sensibilité indique une modification de la quantité d'air de la galerie latérale en question. La méthode développée sert à évaluer les modifications intentionnelles ou accidentelles de la répartition de l'air dans un système de ventilation minier.

DIE ANWENDUNG VON METHODEN ZUR BESTIMMUNG DER EMPFINDLICHKEIT DER LUFTZUFUHRMENGEN GEGEN VERÄNDERUNGEN IN DER FESTIGKEIT VON NEBENZWEIGEN ZUM ZWECK DER STEUERUNG DER BEWETTERUNG IN GRUBEN

A. Frycz, M. Kolarczyk

Die Wetterverteilung in Gruben wird durch Veränderung des Lüfterdruckes und die Errichtung von Wetterbauten reguliert. Bisher wurde der Einfluß dieser Veränderungen durch Berechnungen oder Messungen vor Ort bestimmt. Um die Vorausbestimmung der Veränderungen zu verbessern, wurde eine Methode zur Bestimmung der Empfindlichkeit von Wetterströmen gegen Veränderungen der Festigkeit von Nebenzweigen entwickelt. Die Empfindlichkeit wurde als die Grenze des Differentialquotienten vj/ Rj definiert, wobei vj die Änderung des Wetterstromes in dem j-sten Nebenzweig darstellt und eine Veränderung des Widerstandes Rj in dem j-sten Zweig hervorruft, definiert. Die Empfindlichkeiten werden durch Differenzierung eines Systems von Netzwerks-Gleichgewichtsgleichungen bestimmt, die ein System von linearen Gleichungen ergeben. Die Änderungen, die in den Parametern der Seitenzweige auftreten, fließen in die Berechnung ein als Störquelle für den Luftstrom in einem untergeordneten Netzwerk mit der gleichen Struktur wie das Ausgangssystem. Die Luftströme in den untergeordneten Netzwerken sind die gesuchten Empfindlichkeiten. In Netzwerken mit normalen Strukturen wie parallel/serien sind die Anzeichen der Empfindlichkeiten klar. In Netzwerken mit diagonalen Strukturen kann es zu zweideutigen Situationen kommen, z. B. einer Veränderung des Widerstandes in einem betrachteten Seitenzweig erhöht oder verringert die Luftzufuhr in einem anderen Seitenzweig. Ohne Netzwerkberechnung kann dieser Effekt nicht vorherbestimmt werden. Das Vorzeichen (+ / -) der Empfindlichkeit zeigt an, ob die Luftzufuhr erhöht oder verringert wird, während die Größe der Empfindlichkeit die Veränderung der Luftmenge in dem Seitenzweig anzeigt. Die entwickelte Methode ist zur Bestimmung der Auswirkungen von absichtlichen oder zufälligen Änderungen in der Luftzufuhr in Grubenwettersystemen anzuwenden.

ИСПОЛЬЗОВАНИЕ МЕТОДОВ ОПРЕДЕЛЕНИЯ ЗАВИСИМОСТИ КОЛИЧЕСТВА ПОСТУПАЮЩЕГО ВОЗДУХА ОТ ИЗМЕНЕНИЙ СОПРОТИВЛЕНИЯ В БОКОВЫХ ВОЗДУХОВОДАХ ДЛЯ РЕГУЛИРОВАНИЯ ВЕНТИЛЯЦИИ В ГЛУБОКИХ ШАХТАХ

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Распределение воздушных потоков в шахтной вентиляционной системе варьируется регулированием заслонок и давления от вентиляторов. До настоящего времени влияние этих переменных на количество воздуха определялось расчетами или измерениями на местах. Для улучшения оценки имеющих место изменений разработан метод определения чувствительности воздушного потока к изменениям сопротивления в боковых воздуховодах. Чувствительность определяется как предел дифференциального уравнения $\Delta Vj/\Delta Rj$; где ΔVj, изменение расхода воздуха в j-ом боковом воздуховоде, служит причиной изменения ΔRj в сопростивлении i-ого бокового воздуховода. Чувствительность определяется дифференцированием системы уравнений при условии равновесия в шахтной вентиляционной системе и решением системы результирующих линейных уравнений. Изменения параметров, происходящие в боковых воздуховодах, моделируются в расчетах как источник, нарушающий поток во вспомогательной сети такой же структуры, как и основная сеть. Определяется чувствительность воздушного потока во вспомогательной сети. В системе воздуховодов обычной структуры (параллельно-последовательной) признаки чувствительности очевидны. В диагональной системе воздуховодов могут встретиться двусмысленные ситуации, когда изменение сопротивления в рассматриваемом боковом воздуховоде может увеличить или уменьшить приток воздуха в другом боковом воздуховоде; такая ситуация не может быть заранее предсказана без расчетов вентиляционной сети. Знак чувствительности показывает увеличение или уменьшение притока воздуха, в то время, как величина чувствительности указывает на изменение количества воздуха в рассматриваемом боковом воздуховоде. Разработанный метод является полезным для оценки эффекта преднамеренных или случайных изменений в распределении воздуха в шахтных вентиляционных системах.

以供气量对支道阻力变化的灵敏度测定方法来调整深井煤矿的通风问题

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矿井里通风网路的空气流动分布情形是由调整风门和风扇排气压力来控制的。到目前为止，通风空气量变化的影响全凭从计算上来估计或靠就地测定。为了改进估计的准确性，我们发展了一种方法来测定供气量对支道阻力变化的灵敏度。灵敏度的定义是微商$\Delta\dot{V}j/\Delta Ri$的极限值。$\Delta\dot{V}j$是第 j 条支道空气量的变化，它便产生了第 i 条支道阻力的变化ΔRi。将网路平衡方程式微分后得到一组线形方程式，解出线形方程式便可测定所需要的灵敏性。支道参数的变化是用流源来模拟成为主网路的附属网路。附属网路里的空气流量便是所要求的灵敏度。在普通并联－串联网路这些灵敏度比较显而易见，但在斜交网路里情形并不明确。某一条支道的阻力对另一条支道供气量可能增加，也可能减少。未经计算，不能预知。灵敏度的符号为正为负代表供气量是增加，或是减少。灵敏度的数值则代表所考虑的支道空气量的变化。这种灵敏度的测定方法可以用来研究矿井里通风网路有意或无意的空气分布变化。

THE EFFECTS OF AIRFLOW ON METHANE AND AIRBORNE DUST IN ADVANCED BORD AND PILLAR HEADINGS

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ABSTRACT

High airborne dust counts in continuous miner operations resulted in the Coal Mining Laboratory of the Chamber of Mines Research Organization initiating a series of trials to study airflow patterns in advancing bord and pillar sections. As the information from these trials was studied it became clear that the airflow patterns created by conventional ventilation systems, greatly influenced the methane dissipation capabilities of such systems. This paper deals with some of the findings resulting from subsequent trials involving on-board dust suppression systems and the interaction of these with various auxiliary ventilation systems; the subsequent conclusions are outlined. Brief descriptions of the instrumentation used during the trials are included.

INTRODUCTION

High airborne dust counts are still a cause for concern, particularly with regard to continuous miners in bord and pillar workings. Airborne dust research work for South African coal mines has thus far concentrated, to a great extent, on improving the on-board dust suppression equipment on continuous miners. During one such investigation definite air recirculation zones were noticed within a coal heading. This phenomenon led the Coal Mining Laboratory of the Chamber of Mines of South Africa to launch a more detailed study into the airflow patterns that are created as a result of the interaction between the dust suppression systems on continuous miners and the auxiliary ventilation systems used in bord and pillar headings.

It was soon realized that these patterns would also influence the methane dispersion capabilities of the ventilation systems and that, in contrast to the requirement for dust control, methane dispersion depended largely on highly turbulent airstreams. Conversely, however, dust suppression would depend on airflow being as laminar as possible.

As the trials were conducted in producing sections it was impractical to conduct methane tests. The evaluation of the effects of particular ventilation systems on methane dispersion was therefore more subjective than measured.

Dust suppression systems such as the Peabody and water powered scrubbers, as well as conventional on-board water spray arrangements, were studied in headings with no auxiliary ventilation as well as headings ventilated by force, exhaust and force/exhaust systems. Readings were taken of section air volumes, crosscut air velocities, air velocities within the headings, airflow directions and dust conditions.

This paper highlights some of the more important findings of the studies.

INSTRUMENTATION

Airflow Measurement

Duct and roadway air velocities were measured with a digital, direct reading vane anemometer. The air velocities in the headings were measured with a J-Tec vortex meter which is capable of measuring air velocities as low as 0,1 m/s with acceptable accuracy. The instrument uses an ultrasonic technique to measure a form of turbulence within the instrument aperture. It is intrinsically safe and also features a direct digital velocity readout. Confirmation of airflow directions in the headings was obtained by means of chemical smoke generating tubes.

Dust Measurement

MRE Type 113 A gravimetric sampler

This instrument is used in most countries as the airborne dust reference instrument. The reason for using these instruments is to have recognized, known, standard readings to which the other instrument results could be related.

HSE personal sampler

As with most personal samplers this unit is basically a belt mounted, battery driven pump and control system. The pump is connected to a dust collection cyclone by means of a 6 mm diameter flexible hose. Dust is collected on a fibreglass filter disc of which the mass is measured before and after sampling. Three personal samplers were used during the trials.

Hund TM data sampler

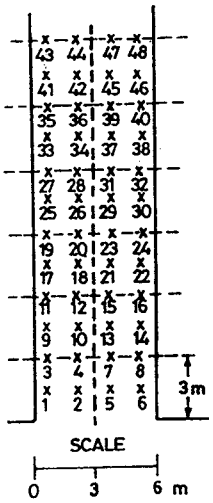
The fraction of fine dust in the total airborne dust quantity is indicated by measuring diffused light trapped after the passage of light beams through the dust laden air. This instantaneous reading instrument has been described by Oberholzer (1987) and was particularly suitable for the trials due to its ancillary computerized facilities. Two or more of these units were used to establish the dust release

patterns (peaks and valleys) during the trials.

FIELD TRIALS

Airflow and Dust Readings

In order to determine the resultant airflow patterns caused by the interaction between ventilation and dust suppression systems, the total mining sequence (from the start of mining at a solid coal face, up to a point where the next crosscut would intersect a particular heading) was monitored. The approach to airflow and dust measurement can be seen in Figure 1, the recording sheet used during the field trials. The sheet shows a diagram of an advancing heading with grid dimensions of 3,0 m by 3,0 m.



MINE:_____

SECTION:_____

DATE:_____

TIME:_____

VENTILATION:

DUCT VELOCITY._____m/s

DIAMETER._____mm

X-CUT VELOCITY._____m/s

DIMENSIONS._____mx_____m

DUST INSTRUMENTS PLACED AT:

1._____

2._____

3._____

4._____

5._____

6._____

GENERAL OBSERVATIONS AND REMARKS:

_____

_____

FIGURE 1. SHEET USED IN FIELD TRIALS FOR RECORDING RAW TRIAL DATA

The dust measuring instruments were placed as follows:

1. HSE 1 attached to the continuous miner at the operator position,
2. HSE 2 worn by a shuttle car driver,
3. HSE 3 worn by the feeder attendant,
4. Hund TM Data 1 in the intake air, and
5. Hund TM Data 2, 3 and 4 used for instantaneous readings at various positions.

During each cut sequence Hunds 2, 3 and 4 were used to take readings at the continuous miner, shuttle car and feeder operator positions.

Two MRE 113A instruments were used as reference samplers. One was positioned in front of the continuous miner operator and the other was used in conjunction with the Hund 1 sampler in the intake airstream. Mining was started and a 3,0 m deep cut was taken, giving a mined block of 3,0 m (depth of cut) by 3,4 m (the machine width). The machine was then stopped and air velocity and airflow direction readings were taken in the particular block as shown in the recording sheet. Mining then continued through the next block and all readings were repeated. Readings were then taken block by block through the mining sequence to the normal heading advance distance (between 12,0 m and 24,0 m).

Dust measurement was discontinued each time the continuous miner stopped cutting.

Dust Suppression Systems

The continuous miners used in the trials were equipped with three different types of dust suppression system. The trials were not conducted in the same section because it would have greatly impeded production. A total of 28 shifts were expended during the tests in four different mines.

Conventional spray system

This system consists of water sprays mounted singly or in banks above, below and to both sides of the cutting

drums and directed towards the gathering arms and the chain conveyor. The spray arrangement was such that a curtain of water enclosed the high dust producing areas around the continuous miner.

Water powered scrubber

This arrangement, developed by the Coal Mining Laboratory of the Chamber of Mines of South Africa Research Organization, incorporates three scrubber units placed above and to both sides of the machine to assist with dust suppression and to encourage airflow at the working face. Airflow is generated by means of water sprays discharging water within open-ended tubes of a larger diameter than the sprays. The discharged water induces airflow through the tubes by entrainment. Each scrubber unit consists of a number of spray/tube combinations. The main advantage of the water powered scrubber is that it has no moving parts and thus needs very little maintenance. The air moving capability of this scrubber is not as great as that of the Peabody scrubber, but, because it is water driven, the combined effect of face-directed air and the high water content of the discharged air, makes it extremely cost effective.

Peabody dust scrubber

This well-known system normally incorporates two hydraulic fans, drawing air through intake ports, via steel ducts through a flooded bed scrubber. The air then passes through an eliminator before the air is discharged.

Ventilation Systems

Three different types of ventilation system were examined. The resultant airflow patterns created by the combination of the dust suppression systems and force, exhaust and overlap systems were determined. In addition, the performance of the scrubber systems was assessed in headings that were otherwise totally unventilated. A variation of the overlap system, using an innovative vortex fan designed to do away with standard ducting, was incorporated in the overlap system tests. Vortex fan impellers were originally developed for use in vertical take-off aircraft and, after conversion for use in mines (Figure 2), were found to be remarkably efficient in converting input energy into air velocity. The fan's unique delivery vortex action may be likened to that of a tornado, but operating in a horizontal, rather than a vertical plane. The result is that the diffusion of the delivered air is sufficiently gradual so that the fan 'throws' the discharged airstream unusually far.

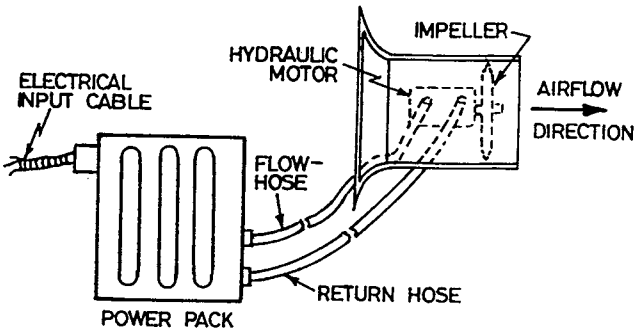


FIGURE 2. VORTEX FAN (SCHEMATIC)

DISCUSSION OF TRIAL RESULTS

At the end of the field trials the airflow patterns were plotted over sketches of the working headings (the sketches include schematic representations of the continuous miners and the shuttle cars). By subsequent examination of these diagrams, in conjunction with the measured airflow and dust results, conclusions as to the efficiency of each dust suppression/ventilation system permutation could be reached.

No Auxiliary Ventilation System

Peabody scrubber mounted on continuous miner

Although the powerful fan in the Peabody scrubber (supplying between 2,0 and 3,5 m^3/s) (Figure 3), draws a significant quantity of air into the heading and across the continuous miner, turbulent zones are, nevertheless, created and dust is not removed from within the confines of the heading. Air recirculation in the heading

is taking place and there is a danger of drawing methane laden air over the continuous miner. A mean equivalent dust count of 5,0 mg/m³ was recorded at the continuous miner operator.

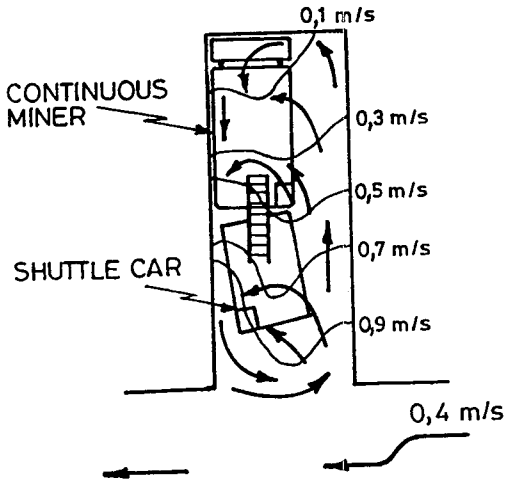


FIGURE 3. AIRFLOW PATTERNS - NO AUXILIARY VENTILATION AND A PEABODY SCRUBBER

Water powered scrubber mounted on continuous miner

The recorded airflow patterns are shown in Figure 4. The left-hand scrubber forces air along the side of the machine and creates turbulence in the mined-out void. The same methane conditions would prevail as for the Peabody scrubber. The mean dust count at the operator position was 6,3 mg/m³.

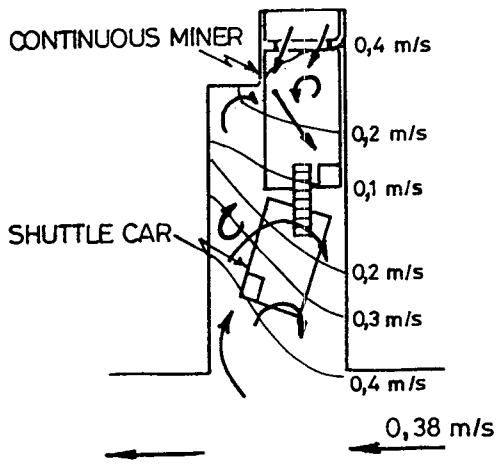


FIGURE 4. AIRFLOW PATTERNS - NO AUXILIARY VENTILATION AND A WATER POWERED SCRUBBER

Forcing Ventilation

Conventional spray system

With this system dust laden air is entrained at the discharge of the ventilation duct as shown in Figure 5. Whilst this turbulence, created by the air entrainment, is totally unsuitable for dust control, it is effective in preventing methane roof layering and could be applied in situations of high methane emission. A measured mean dust count of 7,7 mg/m³ was recorded.

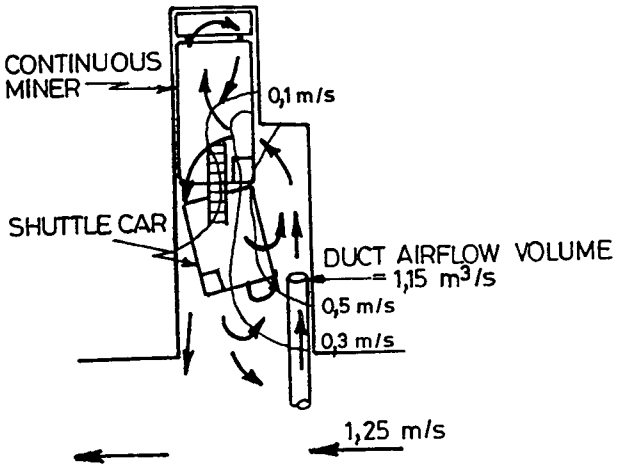


FIGURE 5. AIRFLOW PATTERNS - FORCING VENTILATION AND CONVENTIONAL SPRAYS

Water powered scrubber mounted on continuous miner

Clearly defined recirculation zones and air entrainment around the discharge of the force duct can be seen in Figure 6. The figure shows how dust laden air from the face mixes with the intake air and how this mixture then passes over the shuttle car and continuous miner operators. The mean dust count in this instance was 5,1 mg/m³ at the operator. Methane layering is unlikely to occur.

Peabody scrubber mounted on continuous miner

Very high turbulence was experienced with this arrangement (Figure 7). The forced air current reaches the working face, but some fresh air, together with contaminated air, is recirculated

above the continuous miner and the shuttle car. This system is thought to be unsuitable for dust control purposes but could be used in high methane risk areas. The mean dust result for this system was 6,3 mg/m³.

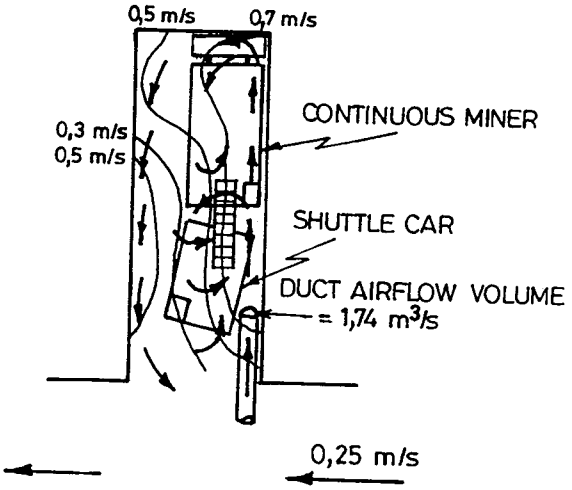


FIGURE 6. AIRFLOW PATTERNS - FORCING VENTILATION AND A WATER POWERED SCRUBBER

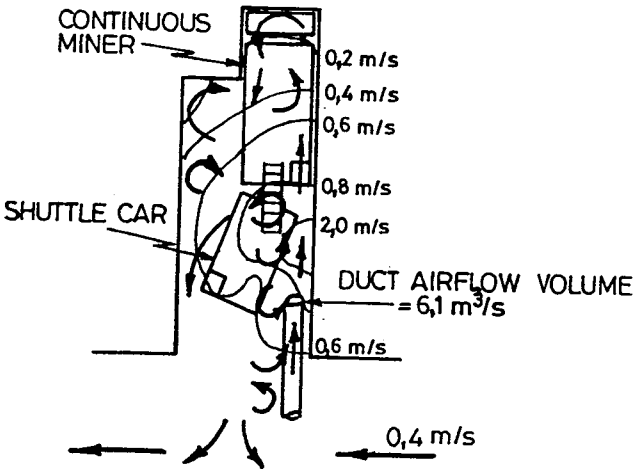


FIGURE 7. AIRFLOW PATTERNS - FORCING VENTILATION AND A PEABODY SCRUBBER

Exhaust Ventilation

Conventional spray system

The airflow in this heading was less turbulent than for previous tests but it was of unacceptably low velocity beyond a point approximately 5,0 m from through ventilation. The bulk of the air entering the heading flows directly towards the inlet of the exhaust column and thus by-passes the working face (Figure 8). The lack of airflow in the face area resulted in a mean dust count of 11,2 mg/m³ at the continuous miner operator. This system is also totally ineffective in the removal of methane gas or the prevention of methane roof layering.

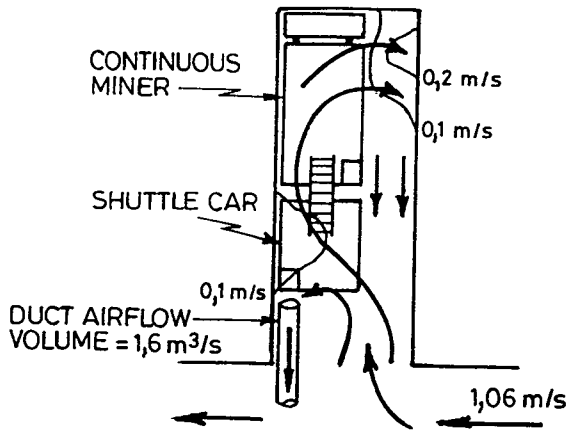


FIGURE 8. AIRFLOW PATTERNS - EXHAUST VENTILATION AND CONVENTIONAL SPRAYS

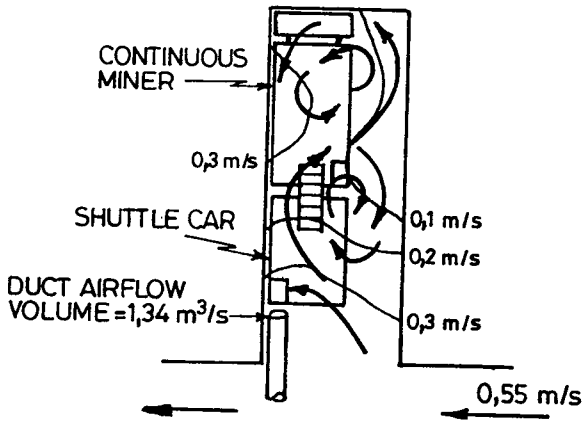


FIGURE 9. AIRFLOW PATTERNS - EXHAUST VENTILATION AND SCRUBBERS

Scrubbers mounted on continuous miner

The Peabody and water powered scrubbers produced almost identical results. A secondary recirculation pattern developed over the continuous miner resulting in little or no dilution of dust laden air by the fresh air entering the heading (Figure 9).

The resultant mean dust count was 8,1 mg/m³. The low air turbulence makes this system unsuitable for methane control.

Force/Exhaust (Overlap) System Using Standard Axial Flow Fans

Conventional sprays mounted on continuous miner

Recirculation occurs in two different spots within the heading (Figure 10). Almost laminar airstreams are shown all the way to the working face. Air velocities are adequate at the operator positions of both the continuous miner and the shuttle car, and the heading is generally well ventilated. The force fan provides sufficient turbulence to prevent methane roof layering and dust is reasonably well controlled. This is verified by a mean dust count of 3,7 mg/m³.

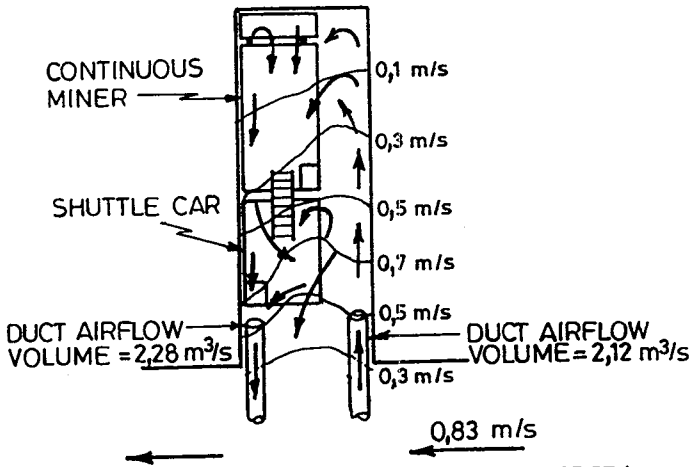


FIGURE 10. AIRFLOW PATTERNS - FORCE/ EXHAUST VENTILATION AND CONVENTIONAL SPRAYS

Scrubbers mounted on continuous miners

Airflow appears reasonably laminar. Figure 11 shows how the scrubbers assist in achieving air velocities in excess of 0,7 m/s close to the continuous miner operator. It also appears that very little recirculation is taking place. The measured mean dust level was 2,1 mg/m³.

VORTEX FANS

The vortex fans used in the tests were of 457 mm diameter and were mounted on either wheeled or skidded undercarriages. The fan motors are hydraulically powered which simplified the adjustment of the delivered air volumes by speeding up or retarding the motors. A typical section layout, using vortex fans, is shown in Figure 12.

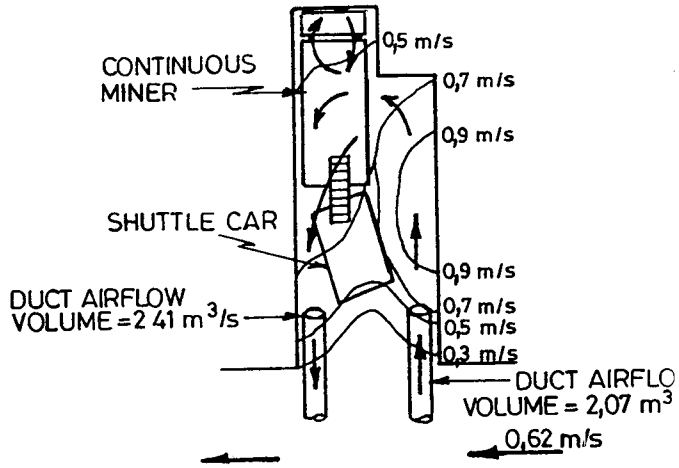


FIGURE 11. AIRFLOW PATTERNS - FORCE/ EXHAUST VENTILATION AND SCRUBBERS

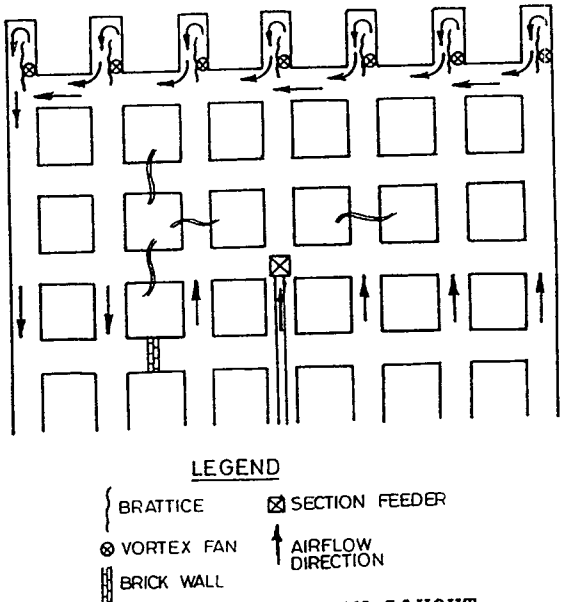


FIGURE 12. TYPICAL SECTION LAYOUT

Vortex Fan as Force Ventilation System

Free-standing vortex fan

It was found, during the vortex fan tests, that the airflow direction in the last crosscut had a definite influence on the performance of the fan. Figure 13 shows how the fan, when placed on the right-hand side of

the heading, causes the air to recirculate almost totally.

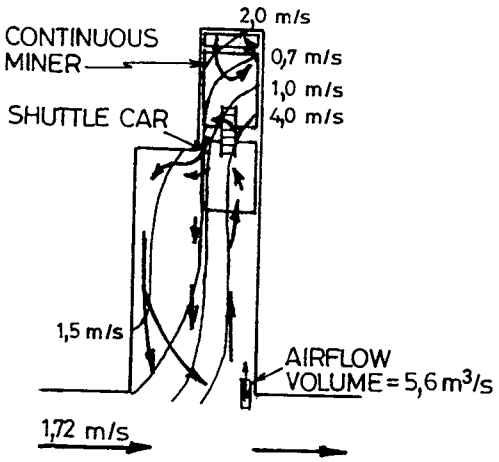


FIGURE 13. VORTEX FANS (FREE-STANDING) AIRFLOW FROM LEFT TO RIGHT

In Figure 14, however, where the airflow direction is from right to left across the heading entry, no air recirculation was observed at the fan. With reference to Figure 14, it was noticed that high air velocities were achieved throughout the heading, and that the turbulence created by the spiral airstream from the fan would make it particularly suitable for use in areas of high methane emission. Where dust control is concerned, the mentioned high degree of turbulence creates a problem in that dust laden air is blown over the shuttle car operator at a rate of 16,3 mg/m³ (mean). The conventional water spray system fitted to the continuous miner during this test could not control the airborne dust effectively.

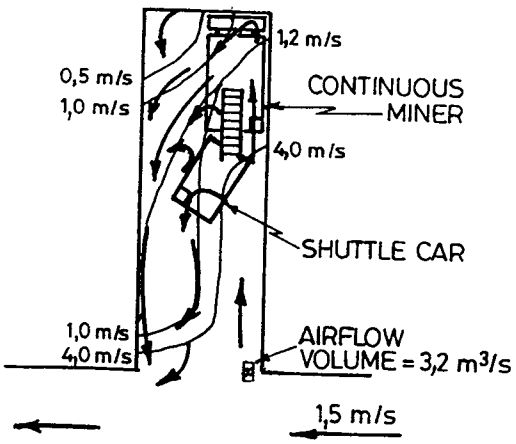


FIGURE 14. VORTEX FAN (FREE-STANDING) AIRFLOW FROM RIGHT TO LEFT

Vortex fan and ducting

For this test, the vortex fan was utilized in almost the same manner as conventional axial flow fans. A 400 mm diameter duct was suspended ahead of the fan outlet (Figure 15) leaving a gap of approximately 70 mm between the fan outlet and the intake end of the duct. This gap allowed for air to be entrained into the duct. The results were much the same as for a standard force fan (Figure 5). A mean dust count of 7,6 mg/m³ was measured.

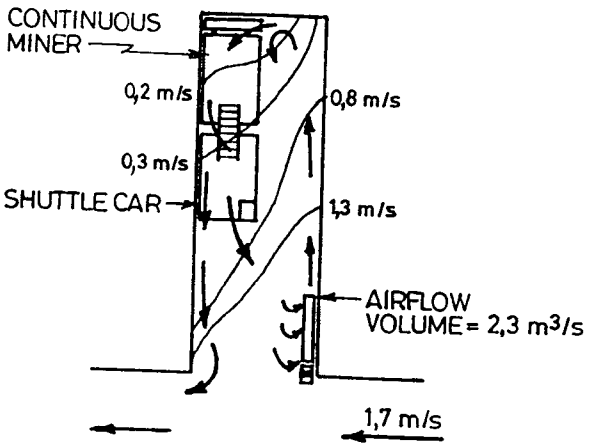


FIGURE 15. VORTEX FAN AND DUCT

Vortex fan and brattice

A ventilation curtain was suspended at about 1,0 m away from the right-hand sidewall for a distance of 6,0 m into the heading (Figure 16). This brattice acted as an air duct into the heading and it also allowed fairly accurate air quantity measurements to be taken. Again the results were much the same as for standard force fans (Figure 5). The mean dust count at the operator positions was 8,3 mg/m³.

Vortex Fan and Brattice in a Force/Exhaust (Overlap) System

Very high air velocities were achieved with only minimal air recirculation (Figure 17). The exhaust fan was handling 3,6 m³/s and the total airflow volume induced by the vortex fan was 3,1 m³/s. Airflow was fairly laminar from the working face to the inlet of the exhaust column, which is

conducive to good dust control. The presence of the vortex fan should ensure that methane can be controlled due to the high degree of turbulence created at the working face. This ventilation system was tested in conjunction with scrubbers as well as conventional water sprays and the mean dust counts at the operator positions were 2,5 and 2,9 mg/m³ respectively.

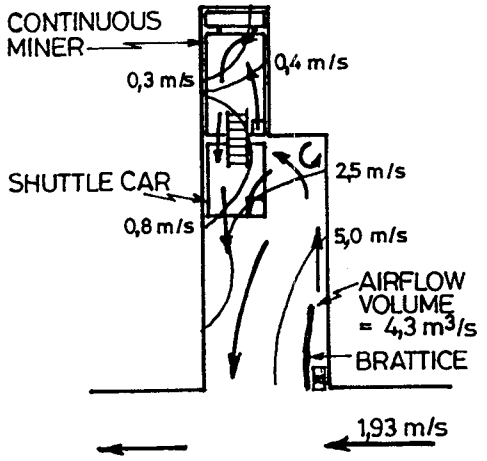


FIGURE 16. VORTEX FAN AND BRATTICE

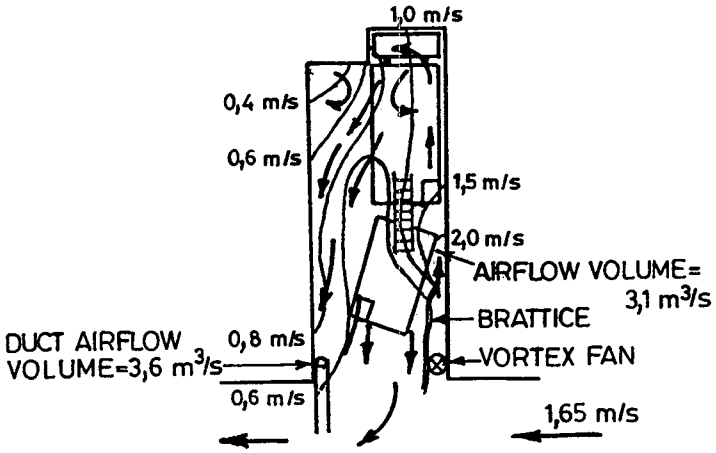


FIGURE 17. VORTEX FAN AND BRATTICE IN A FORCE/EXHAUST APPLICATION

CONCLUSIONS

The series of trials verified that, in order to reduce dust levels in a continuous miner application, an adequate volume of air, with as little turbulence as possible, should be supplied in advancing bord and pillar headings. The trials indicated that, for dust control purposes, the overlap system of ventilation was the most effective. As far as methane control is concerned it is clear that any successful ventilation system should incorporate a force fan where air can be delivered at the point of probable methane emission, namely, the working face.

No Auxiliary Ventilation Provided

While the scrubber systems induced some air into the headings, this induced air was insufficient to suppress dust or control methane.

Forcing Ventilation

This type of ventilation system is far from ideal for dust control because of the high degree of turbulence and recirculation found. However, where high methane gas emissions are prevalent, this method will be effective in preventing methane roof layering.

Exhaust Ventilation

The fact that this method causes most of the air to by-pass the working face makes it an unacceptable ventilation system for both methane and dust control. For this system to be more effective the inlet of the exhaust column should be kept ahead of the continuous miner operator at all times. High production demands, however, and the absence of means of suspension for the ducting, result in undue production losses where the machine has to be stopped to extend the column every 20 or 30 minutes.

Force/Exhaust (Overlap) System

This system incorporates all the attributes required to control both methane and airborne dust. There are, however, practical limitations because two sets of ducting are required to ventilate headings using this method. Not only does this arrangement congest the working space within the heading, it is also very expensive due to the high capital outlay for ducting. Mechanized mining results in rapid linear section advance which means that two sets of ducting per heading have to be advanced with every routine

equipment move. Additional labour will therefore have to be provided to keep production time losses at reasonable levels.

Vortex Fans

This system of ventilation has similar air provision capabilities to conventional axial flow fans, but has the added advantage of not relying on ducting to deliver air to the working face. In the free-standing mode, it performs as well as a normal axial flow fan and its ducting and, as such, is capable of controlling methane emissions adequately. Used as a force fan in a force/exhaust situation, it can be successfully applied to control airborne dust as well as methane emissions.

For methane control in bord and pillar workings a forcing system of ventilation is preferred due to the turbulent airflow caused by the force fan in the system. Paradoxically, in the case of dust control, the airflow needs to be as laminar as possible.

The best ventilation system tested for both methane and dust control, is the one in which a vortex fan is used in a force/exhaust situation.

ACKNOWLEDGEMENTS

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EFFETS DE L'ECOULEMENT D'AIR SUR LE METHANE ET LA POUSSIERE EN SUSPENSION DANS L'AIR DANS LES AVANCEES

E.J. Meets

L'ingénieur de l'environnement doit prévoir une alimentation d'air de qualité et en quantité suffisantes pour minimiser le taux de poussière en suspension dans l'air et maintenir de manière efficace le méthane à un niveau non dangereux. Ceci s'avère particulièrement nécessaire lorsqu'on doit prévoir les besoins en ventilation à l'intérieur des avancées.

Le contrôle de la poussière en suspension dans l'air est un facteur important de la prévention des explosions de poussières de charbon; en effet, le méthane est la cause la plus fréquente de ces explosions. Pour diminuer leur fréquence, de strictes mesures de contrôle doivent être prises.

Pendant un certain temps, l'on a cru qu'en fournissant d'importantes quantités d'air aux avancées de bord et de piliers, on pouvait contrôler de manière efficace les deux contaminants mentionnés ci-dessus.

Des travaux de recherche ont mis en évidence le paradoxe entre la ventilation qui minimise la poussière et celle permettant de contrôler la quantité de méthane. Dans le premier cas, l'écoulement d'air doit être aussi laminaire que possible; dans le deuxième, au contraire, il doit être turbulent.

Nous présentons des techniques d'études et des résultats de recherches appropriés pour le contrôle de la poussière et du méthane. Nous suggérons également certaines méthodes de ventilation alternatives pour contrôler à la fois la poussière et le méthane.

DIE AUSWIRKUNG DER BEWETTERUNG AUF METHANGAS UND FLUGFÄHIGEN STAUB IN VORGESETZTEN STRECKENVORTRIEBEN.

E. J. Meets

Der Umweltingenieur befaßt sich mit der Aufgabe, einen qualitativ und quantitativ ausreichenden Luftstrom zu schaffen, um den im Luftstrom getragenen Staub zu minimieren und die Methangaskonzentration innerhalb sicherer Grenzen zu halten. Das ist besonders wichtig bei der Planung der Bewetterung von weit vorgesetzten Streckenvortrieben.

Die Beherrschung von flugfähigem Staub ist ein wichtiger Faktor in der Verhütung von Kohlenstaubexplosionen, wobei Methangas der haüfigste Auslöser dieser Explosionen ist. Um diese Vorfälle zu vermindern, müssen strenge Kontrollmaßnahmen ergriffen werden.

Seit langem wird angenommen, daß die beiden oben genannten Probleme durch Zufuhr großer Luftmengen zum Strecken- und Pfeilerbau effektiv gelöst werden können.

Forschungsarbeiten haben das Paradox zwischen der Bewetterung zur Staub-Kontrolle und zur Methangasbeherrschung hervorgehoben. Im ersten Fall sollte die Wetterströmung so laminar wie möglich sein, während im zweiten Fall eine sehr turbulente Strömung erforderlich ist.

Der Vortrag beschreibt Untersuchungsmethoden und relevante Forschungs-Ergebnisse und schlägt bestimmte alternative Bewetterungsmethoden zur Staub - und Methangasbeherrschung vor.

ВОЗДЕЙСТВИЕ ВЕНТИЛЯЦИИ НА УРОВЕНЬ СКОПЛЕНИЯ МЕТАНА И ВЗВЕШЕННОЙ ПЫЛИ В НАПРАВЛЯЮЩИХ ШТОЛЬНЯХ

Е.Дж. Митс

Задача инженера по технике безопасности - свести к минимуму количество взвешенной пыли и концентрацию метана с тем, чтобы обеспечить наилучшее качество и количество воздуха в шахте. Эта задача особенно важна при планировании вентиляционных норм для направляющих штолен.

Система контроля над количеством пыли, содержашейся в воздухе, является существенным фактором в предупреждении взрывов угольной пыли. Метан, в свою очередь, является наиболее частой причиной этих взрывов. Вследствие этого, необходимо применение строгих контрольных мер.

В течение продолжительного времени существовало убеждение, что закачка больших количеств воздуха в горизонтальные и целиковые штольни является достаточной мерой борьбы с обоими вредными факторами. Исследования последних лет пролили свет на парадоксальные различия между вентиляционными мерами, принимаемыми для уменьшения количества взвешенной пыли и концентрации метана. Для уменьшения количества пыли необходим ламинарный поток воздуха, в то время, как для эффективного контроля над уровнем метана, необходим турбулентный поток.

Дается описание методов и основных результатов исследований. Предлагаются различные вентиляционные системы контроля над количеством взвешенной пыли и уровнем метана.

超前巷道中空气流对甲烷及煤尘的影响

E. J. 米兹

环境工程师有责任计划供给足够质量和数量的空气，以使空气中飞尘浓度保持在最低限度和有效地将甲烷含量保持在安全量之下．这个问题在超前巷道通风设备需求计划时尤为突出．

飞尘浓度的控制是预防煤尘粉末爆炸的重要因素部分，而甲烷则是最常见的煤尘引爆者．要杜绝此类爆炸事故，必须有严厉的控制措施．

过去有段时间里，人们曾认为向残柱式和柱式矿巷中罐入大量的空气就能有效地解决上述飞尘和甲烷的污染．

研究工作指出飞尘和甲烷的控制所要的通风情形是互相矛盾的，前者要求空气流尽可能在层流状态，而后者则需要紊流．

本文根据有关研究成果，说明了研究方法并提出一些不同的甲烷与飞尘控制的通风方式．

A STUDY INTO FACE GAS EMISSION DURING HEADING DEVELOPMENT

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ABSTRACT

The gas emission from the freshly mined face and from the cut coal during heading driveage is examined in terms of the coal-gas characteristics and mining layout. A gas emission model is developed. The model relates the gas quantity at the face at any one time to the seam gas content, coal diffusivity and the face advance rate. A "Face Cut-Out Cycle" (FCC) index, which is the gas volume emitted above the background gas level during the coal cutting phase is used to characterise the face emission and can be calculated from the above model.

Face emission data from a working face were measured using a continuous on-line gas monitoring system. Correlation with the above model shows that it can be used to accurately predict the peak emission during heading driveage.

The model can be used to predict the FCC for a given situation. Then if borehole drainage should be used the drainage time for FCC reduction to an admissible level can be computed. The above results have been used to assess the efficiency of drainage long-holes in reducing the face gassiness.

INTRODUCTION

During roadway driveage with continuous miners, seam gas is emitted into the working from three main sources:

- the freshly exposed face
- the ribside, and
- the cut coal while being loaded onto shuttle cars.

The gas quantity liberated at the face depends on the gas content of the coal, the advance rate and seam permeability. The actual emission fluctuates during mining depending on the mining method. For the continuous miner/shuttle car operation, a typical variation over a cutting/loading/bolting cycle is shown in Figure 1. The highest emission occurs at the end of the cutting operation. The level decays quickly to the background level when cutting ceases. Understanding and characterisation of the emission process is essential in designing outburst and gas control strategies.

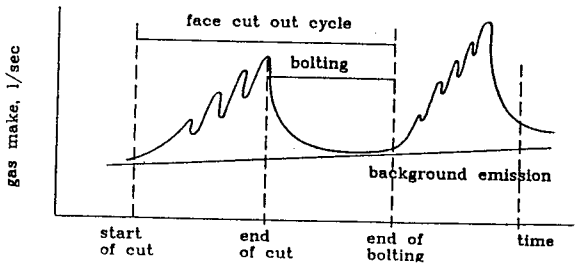


Figure 1. A Face Cut Out Cycle

Several methods for prediction of heading emission have been developed and used in various countries (C. Jeger, B. Bruyet , 1978, Vutukuri and Lama, 1986). These methods are for the calculation of the background emission under steady advance rates. There are cases where the peak emissions are significantly higher than the background emission or the production fluctuates so that steady state calculations become inadequate in predicting the emission.

In this paper, the dynamic process of the gas emission is examined and a mathematical model representing the transient emission is described. Validity of the model is verified by matching with the actual gas emission data.

DEFINITION

The analysis of emission is based on a quantity called Face Cut Out Cycle Emission. It is defined as the volume of gas per tonne of mined coal emitted above the background emission during one cycle of mining. The cycles of mining are limited by roof bolting before new advances. The end of a cut out cycle may be conventionally decided to be 30 minutes after advance stopping. This length of time corresponds to the average bolting duration. In current modelling the end of each cycle is assumed to be simply the starting of the next cycle.

Gas emission in a cut out cycle has two components: gas emission during advance of the face and gas emission after the face stopped.

GAS EMISSION DURING ADVANCE

During mining, the gas is emitted from,
- existing rib being exposed since commencement of mining,
- the newly exposed face and rib,
- the cut coal during mining and loading.

The background emission over the whole length of the heading is subtracted from the total gas volume.

Gas Emission Rate from the Ribsides

If the rib is partitioned in segments each corresponding to a single advance of the cutting machine during a cycle (Figure 2), then the gas emission rate **qi** from each single segment can be expressed by a decay exponential function:

$$qi = q0.D.e^{-D.ti} \qquad (1)$$

where, **D** is a diffusion constant, and **q0** is the total available gas volume contained in one rib segment.

The time **ti** represents the duration of exposure of the rib segment **i**. The total flow rate of gas into the heading is the sum of all **qi** emitted of the **n** segments of rib, $qr = \Sigma\, qi$.

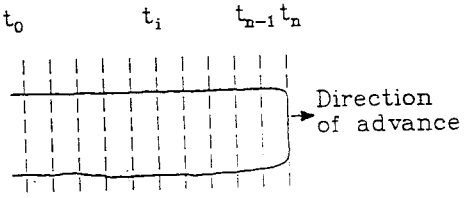


Figure 2. Idealised Mining Sequence in a FCC.

It can be shown that,

$$qr = q0\ (1- e^{-D.t}) \qquad (2)$$

where **t** is the age of the oldest segment of the cycle.

The value of **q0** can be approximated as follows:

the majority of gas emitted from the rib originates from the relaxed zone around the heading. In Figure 3 the assumed distribution of relaxed zones and gas content for an advance heading is given. If the initial gas content is **Ci**, then after passing the face the gas content in the relaxed zone will decrease to an average residual value **Cr**.

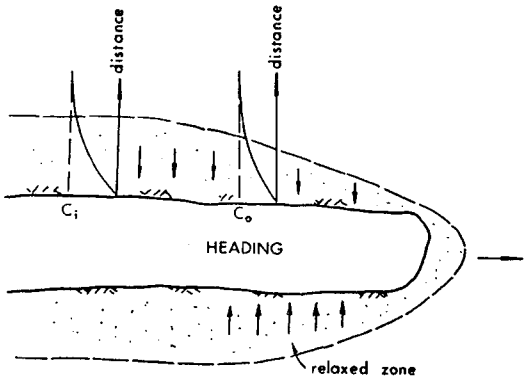


Figure 3. Relaxed zone and gas content in the vicinity of the advancing

The maximum evolvable volume of gas from rib and floor for each segment is:

$$q0 = \beta.r.(2h+w)\ .d.Ci.dx \qquad (3)$$

where

$$\beta = 1 - \frac{Cr}{Ci}$$

dx : length of each segment
d : coal density
h : height of the heading
r : width of relaxed zone

The term **w** expresses the eventual contribution from the floor or roof in total gas emission.

The average mining advance rate **v** for the cycle can be given in terms of coal production **P** during the cycle,

$$v = \frac{1}{h.w.d}.\frac{P}{T}$$

where **T** is duration of cutting period in the cycle. Therefore

$$q0 = \beta.r.\frac{2h + w}{h.w}.Ci.\frac{P}{T} \qquad (4)$$

Gas Emission from New Face

If the same law which was adopted for the rib emission rate is applied to the face emission, the face emission rate can be expressed by,

$$qf = k.q0.Df.e^{-Df.t} \qquad (5)$$

where **k.q0 is** the available gas volume in front of face (gas contained in relaxed zone):

$$K.q0 = \beta.r.h.w.d.Ci \qquad (6)$$

where **Df** is the diffusion constant in the direction of the heading.

As long as the face advances, it remains fresh and a constant gas flow rate is emitted from the face. This constant flow rate is (**t=0** in equation 5):

$$qf = k.q0.Df \qquad (7)$$

The coefficient **k** expressed in unit of time is the ratio between available gas volume in front of face to available gas volume in a segment of rib created in unit of time. It is given by,

$$k = \frac{h.w}{2h + w} \; h.w.d. \; \frac{T}{P} \qquad (8)$$

The influenced zone in front of the fresh face as well as residual gas content are assumed to be equal to that of the ribs. But there could be different residual gas content value **Cr** and relaxed zone width **r** for the rib and face.

Gas Emission from the Cut Coal

A part of the gas contained in the mined coal will be released during cutting and loading. The ratio **α**, i.e. the percentage of gas released during mining to the total desorbable gas contained in the extracted coal depends on coal properties, particle size and the time of exposure after cutting. The rate of released gas from the coal during cutting and loading is approximated by

$$qc = \alpha.Ci.\frac{P}{T} \qquad (9)$$

Total Gas Flow Rate into the Heading for Period of Mining

Total flow rate **Q** is the sum of the values of **qb, q0, qr, qf** and **qc** . The gas emission rate during mining is therfore:

$$Q = qb + A - q0.e^{-D.t} \qquad (10)$$

where **A** is a constant equal to

$$A = qc + qf + q0 =$$

$$qc+(1+kD).q0 \qquad (11)$$

If equation 10 is developed in its primary terms it can be seen that the gas flow rate above the background emission during face advance is proportional to the product of the initial gas content and the production rate.

GAS EMISSION DURING PERIODS OF NO MINING

Between two FCC's when mining has ceased for roof bolting or other activities, gas continues to flow out from the ribs and from the aging face. There is no new rib and face created. To calculate the gas emission from the rib, equation (2) still can be used provided that the portion of gas contributed from new ribs expressed in that equation is discarded.

If cutting were to continue beyond time **T**, then during the period **(t-T)** the new ribs would emit gas at a rate of:

$$q0 .[1- e^{- D.(t-T)}]$$

where **T** is the instant when mining ceases. It follows that for periods of no mining, the gas emission rate from the rib is:

$$\bar{qr} = q0 \; [e^{-D.(t-T)} - e^{-D.t}], \quad t>T \qquad (12)$$

Similarly, the gas emission rate from the face is:

$$\bar{qf} = k.q0.D.e^{-D.(t-T)} \quad t>T \qquad (13)$$

Total Gas Flow Rate into Heading for Period of No Mining

The gas emission rate during the period of no mining, **(t>T)**, until the starting of the next cycle is:

$$\bar{Q} = qb + \bar{qr} + \bar{qf}$$

Therefore the emission rate above background is,

$$\bar{Q} - qb = q0\,[(1+ kD).e^{-D.(t-T)} - e^{-D.t}] \quad (14)$$

where T is the moment mining stops and t is the elapsed time since the start of the cycle.

TOTAL GAS VOLUME LIBERATED DURING A FACE CUT OUT CYCLE

Between the instant t0 = 0, when a face cut out cycle starts and instant t, at the end of the face cut out cycle, Qt, the cumulated volume of released gas above the background emission is,

$$Qt=\int_0^T Q\,dt + \int_T^t \bar{Q}\,dt - \int_0^t qb\,dt \quad (15)$$

Coal cutting stops at instant T, and the next cycle begins at instant t (Figure 4).

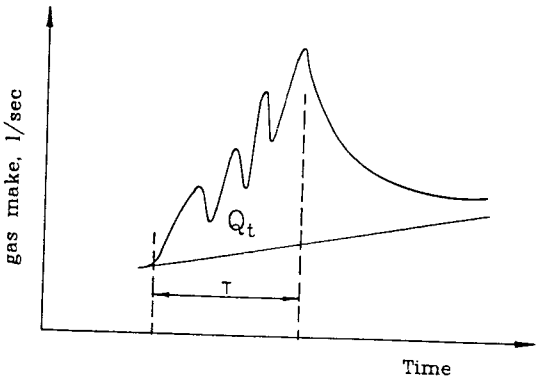


Figure 4. Gas volume Qt released above the background emission

It yields,

$$Qt = A.T - \frac{q0}{D}(1- e^{-D.t}) + (1+k.D).\frac{q0}{D}.[1- e^{-D.(t-T)}] \quad (16)$$

Therefore FCC, i.e. the gas volume emitted per tonne of extracted coal is:

$$FCC = Qt/P$$

or,

$$FCC = A.\frac{T}{P} - [1- e^{-D.t} - (1+k.D).(1- e^{-D.(t-T)})].\frac{q0}{D.P} \quad (17)$$

Using equation (17), the gas volume produced per tonne of coal can be calculated for a period of time commencing at the beginning of the cycle and finishing at any time t. For instance if it is assumed that the end of the cycle be 30 minutes after stop of cutting then in equation (17) t-T=30.

VERIFICATION OF THE MODEL

The above model was used to evaluate the effectiveness of long-hole gas drainage ahead of headings at No 2 Mine, Moura.

The gas monitoring was set up as part of a NERDDP (National Energy Research Development & Demonstration Program) research project to evaluate the effectiveness of long-hole drainage in reducing the gas emission. Both the gas concentration and air quantity were monitored with continuous read-out sensors located in the return roadway to give the gas emission quantity (Figure 5). The signals were transmitted to a surface datalogger and stored for analysis. Details of the production rates, mining sequences and face gas contents were also recorded.

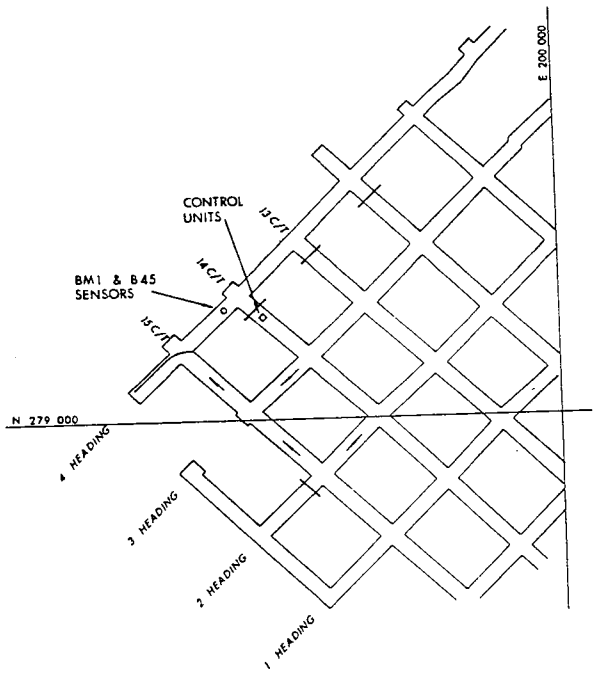


Figure 5. Location of monitoring station during face cutting in Heading

The recorded gas emission rate and FCCs before in-seam, long-hole drainage were matched against the simulated results.

Typical emission data for six consecutive cycles and simulated emission is shown in Figure 6 and Table 1. The in-situ coal properties used in the model were then evaluated. In the simulation it was assumed that the base emission rate increases linearly with face distance.

Drainage was undertaken by drilling parallel in-seam long-holes ahead of future heading (Figure 7).

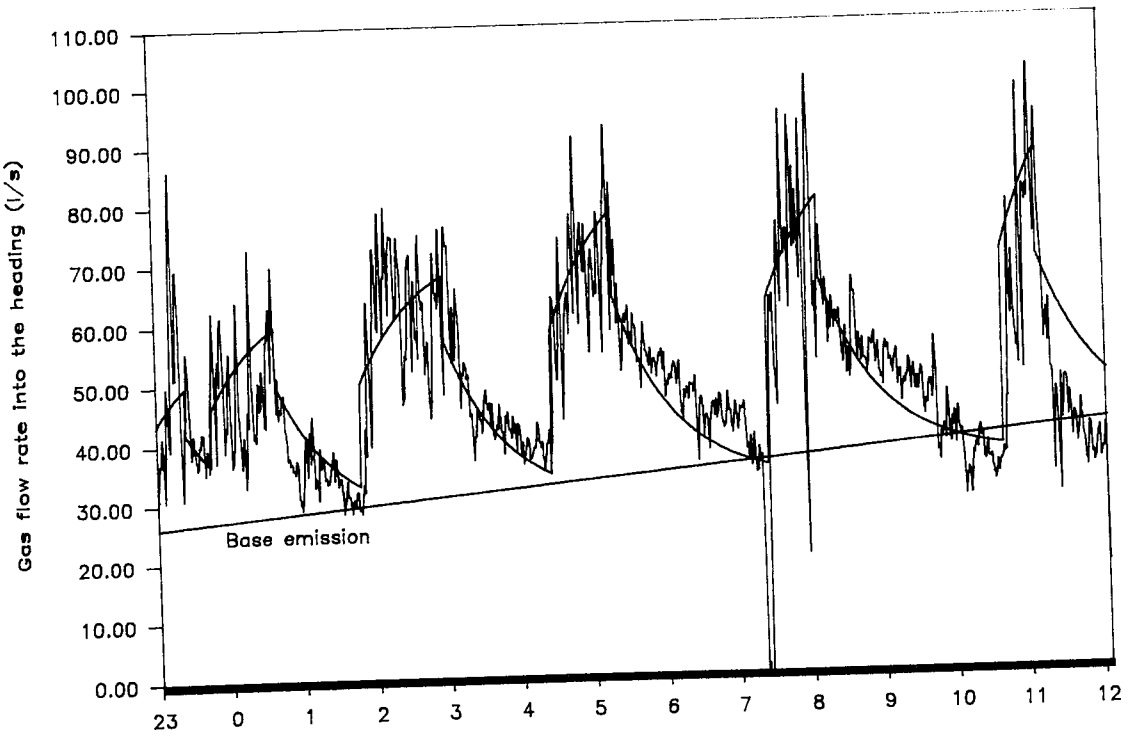


Figure 6. Simulated and actual emission rate for cutting cycles prior to borehole drainage

Table 1: Face Cut out Cycle Emission (4 Heading)

Cycles	Prod. tonne	Distance (m)	Measured FCC (m3/t)	Predicted FCC (m3/t)
1	30	1.4	1.66	1.55
2	80	5.1	1.25	1.64
3	120	10.6	1.66	1.56
4	100	15.2	1.93	1.86
5	70	18.5	2.39	2.28
6	60	21.2	1.04	2.10

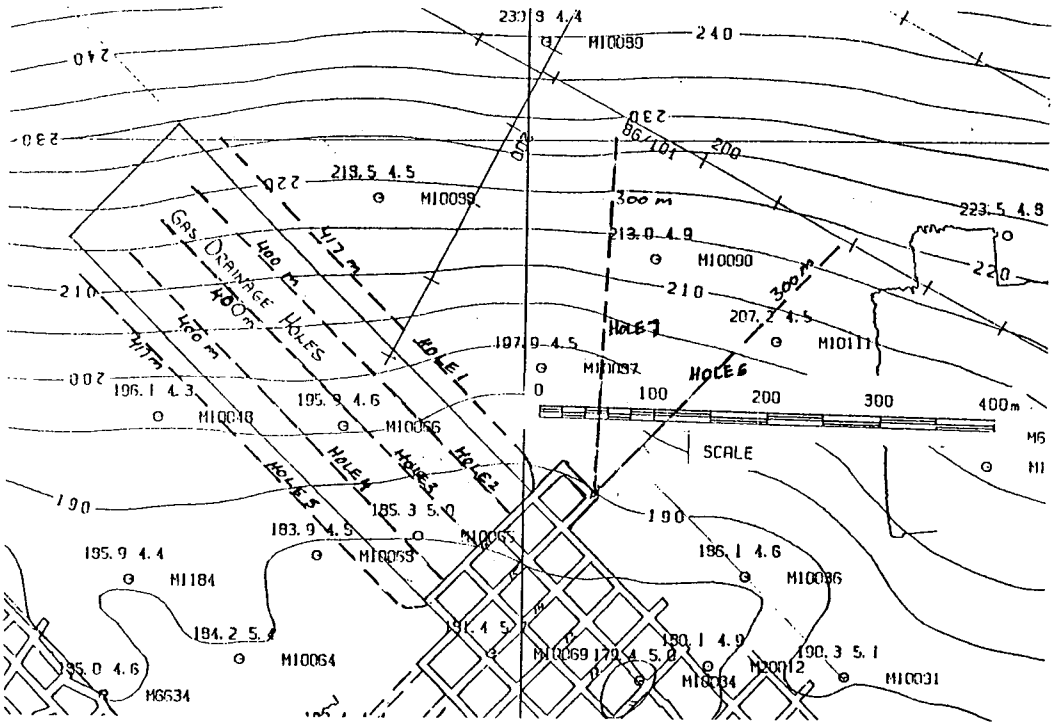


Figure 7. Long-hole drilling ahead of future development

After five months drainage, the average residual gas content in the drainage zone was estimated using the gas flow rate history from the drainage boreholes.

The likely emission rate was then simulated by the model for the residual gas content (Figure 8). It was used to assess the maximum gas concentration in return and as a guide for decision making on restart of mining.

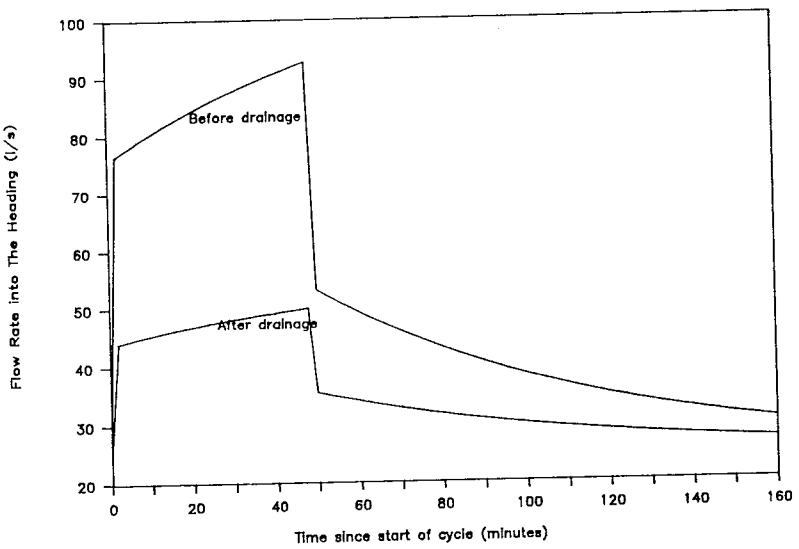


Figure 8. Predicted gas emission rate before and after drainage

CONCLUSIONS

Face Cut Out Cycle emission was identified as a representative index of gassiness of a given seam. This concept was found to be accurate and an attempt was made to develop a method of calculation of the gas emission pattern during a face cutting cycle. A model of gas emission during mining cycle using physico-emperical knowledge of gas movement was established.
The model gives in time flow rate and FCC during each cycle.

A number of in-situ properties of the coal seam (β, α and **D**) and mining layout parameters are used in the model. For a given coal and mining layout the gas emission during a cycle depends on gas content (**Ci**) and advance rate (**P/T**). By back analysis of an actual gas emission pattern, the in-situ coal properties can be estimated. Then with these properties different possible mining scenario could be studied. The gas content can be changed as well as production rate or stopping time during a cycle. The model then calculates the instantaneous gas flow into the heading as well as the overall emission called FCC emission (volume of emitted gas per tonne of coal production).

The model was applied to a number of consecutive cycles during face cutting in a heading. Gas diffusion properties of **D** seam were estimated. When five months predrainage of the seam ahead of the headings was completed, the seam gas content was estimated by material balance method and was used in the model to predict the possible gas emission rate during the future mining operations.

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ETUDE DE L'EMISSION GAZEUSE D'UN FRONT DE TAILLE AU COURS DU DEVELOPPEMENT D'UNE AVANCEE.

A. Saghafi, D. Truong, R. Williams

L'émission gazeuse d'un front de taille récemment exploité et du charbon taillé au cours du creusement de l'avancée est examinée en termes de caractéristiques charbon-gaz et de disposition de l'exploitation minière. Puis, on développe un modèle d'émission de gaz, qui établit un rapport entre la quantité de gaz existant dans le front de taille à n'importe quel moment et la teneur en gaz du filon, la capacité de diffusion du charbon et la vitesse d'avancement du front de taille. Un indice du "cycle de coupure du front de taille" (FCC), représentant le volume d'émission de gaz au-delà de son niveau normal pendant la phase de taille du charbon, sert à caractériser l'émission du front de taille; il se calcule sur la base du modèle décrit ci-dessus.

Les données d'émission d'un front de taille ont été mesurées au moyen d'un système de surveillance de gaz continu et en direct. Leur corrélation avec les modèles mentionnés ci-dessus indique qu'on peut utiliser ce système pour calculer correctement l'émission de pointe lors du creusement de l'avancée.

Ce modèle peut calculer le FCC pour une situation donnée. En outre, si on utilise un drainage par trou de mine, on peut calculer le temps de drainage nécessaire pour diminuer le FCC jusqu'à un niveau admissible. Les résultats ci-dessus ont été utilisés afin d'évaluer l'efficacité des trous de mine de drainage pour diminuer la teneur en gaz du front de taille.

UNTERSUCHUNG DER GASAUSSTRÖMUNG IN STREBEN WÄHREND DES STRECKENVORTRIEBES.

A. Saghafi, D. Truong, R. Williams

Die Ausgasung aus dem frisch verritzten Kohleflöz und der gewonnenen Kohle während der Streckenauffahrung wurde im Hinblick auf die Gasbeschaffenheit der Kohle und der Feldauslegung untersucht. Ein Modell für die Gasausströmung wurde entwickelt. Das Modell zeigt die Gasmenge im Streben im Verhältnis zu der Gasmenge im Kohleflöz, der Kohleausbreitung und zu dem Abbaufortschritt zu einem gegebenen Zeitpunkt. Ein 'face cut-out cycle" (FCC) Index, der das Volumen des ausströmenden Gases über den normalen Ausgang des Flözes während der Gewinnphase darstellt, wird benutzt, um die Ausgasung aus Streben zu kennzeichnen und kann aus dem oben genannten Modell errechnet werden.

Ausgasungsdaten aus laufenden Streben wurden mit einem kontinuierlichen 'on-line' Gasmonitor System gemessen. Korrelationen mit dem oben genannten Modell beweisen, daß das System zur genauen Voraussage der Spitzenausgasung während des Vortriebes angewandt werden kann.

Das Modell kann verwendet werden, um den FCC für eine gegebene Situation vorauszusagen. Die Absaugzeit kann bis zum Erreichen einer zulässigen Gas-Konzentration errechnet werden, falls eine Gasvorabsaugung eingesetzt wird. Die Ergebnisse wurden bereits zur Bewertung des Leistungsgrades der Drainage-Löcher bei der Gasabsaugung benutzt.

ИЗУЧЕНИЕ ВЫДЕЛЕНИЯ ГАЗА В ЗАБОЕ ВО ВРЕМЯ РАЗРАБОТКИ ПЕРЕДОВОЙ ШТОЛЬНИ

А. Сагафи, Д. Труонг и Р. Вильямс

Рассматривается выделение газа из свежевыработанного забоя и из нарезанного угля во время проходки штольни с точки зрения характеристик угля и газа и расположения горных выработок. Разработана модель газовой эмиссии. Модель дает соотношение количества газа в забое к содержанию газа в пласте в отдельно взятый промежуток времени, диффузивность угля и скорость подвигания забоя. Индекс цикла вскрытия забоя, который понимается как разница между первоначальным уровнем газа и количеством газа, выделенным в процессе проходки, используется при описании эмиссии в забое и может быть вычислен из вышеупомянутой модели.

Данные об эмиссии газа в действующем забое были получены с помощью системы непрерывного слежения. Корреляция с вышеописанной моделью показывает, что эта модель может быть использована для точного предсказания максимальной эмиссии газа во время проходки передовой штольни.

Эта модель может быть использована для предсказания индекса эмиссии газа в любой отдельно взятой ситуации. В случаях, когда необходима откачка газа посредством бурения, индекс помогает вычислить количество времени откачки, нужное для достижения допустимого уровня газа. Вышеописанные результаты использовались для оценки эффективности откачки через скважины для уменьшения содержания газа в забое.

巷道掘进时工作面煤气溢出的研究

A. 沙加飞， D. 特孺翁，R. 威廉士

这里用煤与气体的关系特性和矿井布局的观点来分析掘进时新开发的工作面和新采出煤的煤气散发。本文提供一个气体散发模型。这模型建立了工作面任意时间的气体排量与煤层气体含量，煤层扩散率和工作面掘进速度之间的关系。我们用"工作面挖掘周期" 指数（FCC），即煤炭挖掘阶段中在背景气体量以上的散发气体体积，来描述工作面气体的散发，此指数可由上述模型计算而得。

工作面的气体散发量数据是用一连续联机监测系统测得。与上述模型一并使用，可准确地预测巷道挖掘中溢气高峰期。

此模型可用于确定一给定挖掘环境的 FCC 指数。如需用钻孔排泄时，可计算出将 FCC 降至允许量所需要的排泄时间。以上结果，已应用于估计钻孔排泄降低工作面含气量的效率。

НАДЕЖНОСТЬ ШАХТНЫХ ВЕНТИЛЯЦИОННЫХ СИСТЕМ

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РЕФЕРАТ

Шахтные вентиляционные системы (ШВС) относятся к классу систем жизнеобеспечения, надежность функционирования которых имеет особое значение. В СССР ведутся системные исследования надежности ШВС. Надежность ШВС определяется достоверностью исходных данных, методов расчета, аэродинамическим сопротивлением элементов вентиляционной сети и ее топологией, надежностью работы вентиляторов. Оценка надежности ШВС может производиться структурным методом (по относительным коэффициентам готовности элементов) или имитационным моделированием. Эффективность работы ШВС оценивается через среднюю глубину отказа системы и ее коэффициент готовности. Существенно влияет на надежность аэродинамическое старение выработок. Скорость старения больше у выработок участковых, чем у капитальных, у пластовых, чем у полевых, с конвейерным транспортом, чем с рельсовым. Вероятность безотказной работы за 720 ч. для выемочных участков угольных шахт 0,4-0,9, рудных 0,85-0,95, для главных вентиляторных установок 0,75-0,96. Надежность ШВС - категория экономическая и социальная. Повысить ее можно увеличением разветвленности вентиляционной сети, резервированием вентиляторных установок и линий энергоснабжения, ремонтом выработок по специально разработанным графикам, улучшением их охраны, учетом процессов старения выработок при проектировании вентиляции.

ВВЕДЕНИЕ

Шахтная вентиляционная система - основная система жизнеобеспечения работающих в шахте людей. Надежность ее функционирования имеет решающее значение в деле обеспечения безопасности труда в шахте. Системные исследования надежности ШВС в СССР ведутся с конца 60-х гг. В настоящее время разработаны методы оценки надежности ШВС, получены ее статистические характеристики и ведется разработка раздела "Надежность" в системе автоматизированного проектирования вентиляции шахт.

Исходя из общего определения надежности системы, можно сказать, что надежность ШВС - это ее свойство сохранять в процессе эксплуатации те значения параметров, которые обеспечивают поддержание в шахте требуемой чистоты воздуха. Из этого следует, что надежность ШВС зависит от двух групп факторов: факторов, определяющих воздухораспределение в вентиляционной сети, и факторов, определяющих выделение в сеть загрязняющих воздух субстанций. В настоящем докладе исследуется надежность ШВС только по фактору воздухораспределения, т.е. надежность собственно вентиляционной системы как конструктивного элемента.

В соответствии со сказанным под надежностью ШВС понимается ее свойство сохранять в процессе эксплуатации шахты значения

параметров, обеспечивающих требуемое воздухораспределение. Отказом ШВС будем называть событие, состоящее в потере работоспособности системы. Под работоспособностью ШВС понимается такое ее состояние, когда воздухораспределение в сети соответствует установленным требованиям (содержание вредностей в атмосфере выработок, а также скорости движения воздуха в выработках находятся в допустимых пределах).

Шахта как вентиляционная система имеет глубокое резервирование. Отказ (один или даже несколько одновременных отказов) ее элемента как правило не приводит к отказу вентиляции шахты в целом: в большинстве случаев происходит лишь частичная потеря работоспособности системы. Поэтому характеристика отказа ШВС должна содержать также оценку его масштаба, или значимости. По значимости отказы ШВС делятся на три ранга: отказы I ранга охватывают шахту в целом; отказы II ранга охватывают значительную часть шахты (пласт, крыло, залежь); отказы III ранга охватывают отдельные участки шахты (Ушаков К.З. 1976; Рудничная вентиляция: Справочник 1988).

КРИТЕРИИ НАДЕЖНОСТИ

Вероятность безотказной работы – вероятность того, что в течение заданного времени отказ объекта не возникает:

$$\tau(t) = \exp\left(-\int_0^t \lambda(\tau)\,d\tau\right), \qquad (1)$$

где t – время; $\lambda(\tau)$ – интенсивность отказов (условная плотность вероятности отказов в момент времени τ, определяемая при условии, что до момента τ отказа не было).

Среднее время безотказной работы

$$T_{rb} = \int_0^\infty \tau(t)\,dt. \qquad (2)$$

Для экспоненциального распределения времени безотказной работы (1) при $\lambda = const$, $T_{rb} = 1/\lambda$.

Среднее время восстановления работоспособного состояния:

$$T_{rs} = \int_0^\infty t_{rs} f_{rs}(t)\,dt, \qquad (3)$$

где t_{rs} – время восстановления; $f_{rs}(t)$ – плотность распределения случайной величины t_{rs}.

Коэффициент готовности восстанавливаемых объектов, к которым относятся и ШВС:

$$K = \frac{T_{rb}}{T_{rb} + T_{rs}} \qquad (4)$$

ОПРЕДЕЛЯЮЩИЕ ФАКТОРЫ

В соответствии с принятым выше определением отказа в ШВС к факторам надежности следует отнести те, которые влияют на воздухораспределение в системе. В наиболее общей постановке это: исходные данные; методы расчета; аэродинамическое сопротивление элементов вентиляционной сети и ее топология; надежность работы главных вентиляторных установок (ГВУ). Ошибка в исходных данных и недостоверные методы расчета могут привести к запланированным отказам высокого ранга. Известны случаи, когда по этим причинам ШВС подвергались реконструкции уже в первые годы эксплуатации шахты.

С позиций надежности важна достоверность исходных данных, определяющих аэродинамические сопротивления элементов ШВС, т.е. данных, характеризующих аэродинамические свойства крепей, поперечные сечения выработок, утечки воздуха в шахте. Из методов расчета важна достоверность методов расчета депрессии выработок и воздухораспределения в сети. Аэродинамическое сопротивление элементов ШВС (выработки, выработанные пространства, вентиляционные сооружения)

является одним из главных факторов, определяющих общее количество воздуха, поступающее в шахту. Оно также определяет относительное воздухораспределение в сети. Изменение аэродинамического сопротивления элементов сети во времени должно учитываться уже на стадии проектирования вентиляции шахты и быть объектом постоянного внимания в процессе эксплуатации.

Режим работы ГВУ определяет поступление воздуха в шахту. Из этого следует важность стабилизации режима ГВУ и адекватного его изменения в процессе жизни шахты. Отказы в работе ГВУ связаны с нарушениями в их механических системах (элементы вентилятора, средства управления и резервирования) и системах энергоснабжения.

Рассмотренные четыре группы определяющих факторов составляют подсистемы общей системы надежности ШВС. В смысле надежности они соединены между собой последовательно: отказ в любой из подсистем приводит к отказу системы в целом. Схема надежности ШВС представлена на рис. I.

со сказанным можно записать, что общая надежность ШВС

$$\gamma_S = \gamma_a \gamma_m \gamma_n \gamma_f. \qquad (5)$$

МЕТОДЫ ОЦЕНКИ

Структурный метод основан на определении относительных коэффициентов готовности K выработок единичной длины: $K = K_i / K_0$, где K_i – абсолютный коэффициент готовности участка единичной длины i-ой выработки, – то же некоторой базовой выработки.

В качестве показателя надежности принимается величина K, пропорциональная коэффициенту готовности ШВС:

$$K = (W - n + 1)/M, \qquad (6)$$

где W – количество выработок в сети; n – количество узлов в сети;

$$M = \sum_{i=1}^{W} M_i; \qquad (7)$$

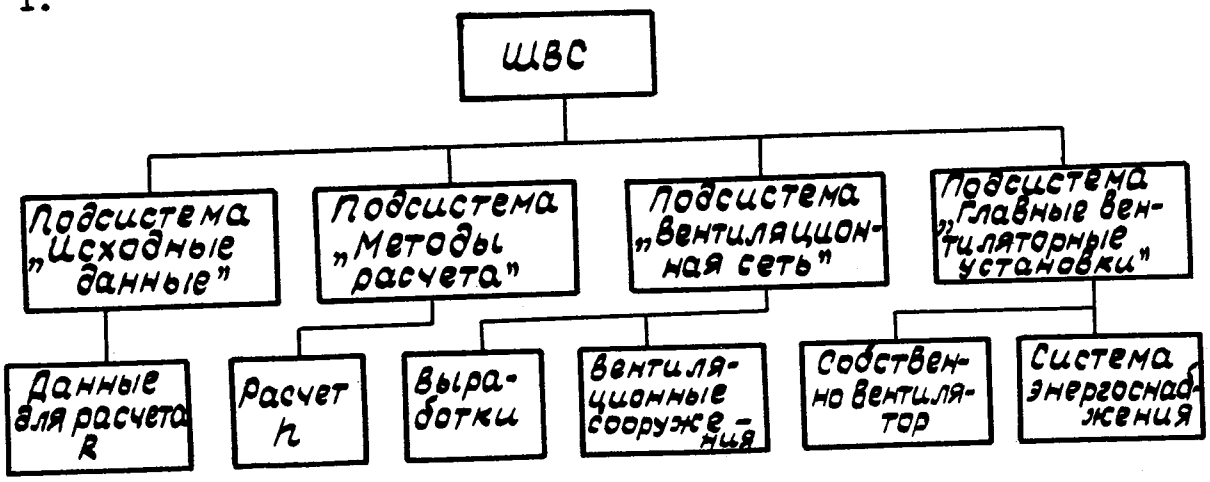


Рис. 1

Если надежность подсистемы "Исходные данные" обозначить через γ_a, подсистемы "Методы расчета" через γ_m, подсистемы "Вентиляционная сеть" через γ_n, подсистемы "Главные вентиляторные установки" через γ_f, то в соответствии

$$M_i = L_i P_{wi} + \sum_{s=1}^{a} P_{vci}^{(s)}; \qquad (8)$$

$$L_i = L_i / \ell; \qquad (9)$$

L_i – длина i-ой выработки;
ℓ – длина участка выработки, принятая за единицу (например, ℓ = 100 м);

$$P_{wi} = \ell n K_{wi} / \ell n K_{wo} = \quad (10)$$
$$= 1 + \ell n \bar{K}_{wi} / \ell n K_{wo};$$

K_{wi} – абсолютный коэффициент готовности участка единичной длины i-ой выработки; K_{wo} – то же базовой выработки;

$$\bar{K}_{wi} = K_{wi} / K_{wo} \quad (11)$$

– относительный коэффициент готовности участка единичной длины i-ой выработки;

$$P^{(s)}_{vci} = \ell n K^{(s)}_{vci} / \ell n K_{wo} = \quad (12)$$
$$= 1 - \ell n \bar{K}^{(s)}_{vci} / \ell n K_{wo};$$

$K^{(s)}_{vci}$ – абсолютный коэффициент готовности s-го вентиляционного сооружения в i-ой выработке; $\bar{K}^{(s)}_{vci}$ – его относительное значение; a – число вентиляционных сооружений в i-ой выработке.

Следует заметить, что главные вентиляторные установки в этом методе рассматриваются как выработки единичной длины (т.е. для них L_i = 1) со своими коэффициентами готовности.

Сравнение получаемых по формуле (6) показателей надежности разных вариантов ШВС позволяет выбрать наиболее надежный. Аналогично оцениваются и мероприятия по повышению надежности ШВС.

Метод достаточно прост. Для его использования необходимо знание топологии ШВС, длин выработок, числа и видов вентиляционных сооружений, относительных коэффициентов готовности основных групп выработок и K_{wi} и основных видов вентиляционных сооружений K_{vci} и всего лишь одного абсолютного значения коэффициента готовности базовой выработки K_{wo}. Данные о топологии, длинах выработок, числе и видах вентиляционных сооружений легко доступны. Относительные коэффициенты готовности могут быть получены сравнительными наблюдениями на шахтах или путем анализа статистических данных о состоянии выработок или, наконец, методом экспертных оценок. Метод удобен для оперативных оценок.

В таблице приведены примерные значения относительных коэффициентов готовности некоторых типов выработок и вентиляционных сооружений в угольных шахтах.

При расчете значений относительных коэффициентов готовности, представленных в табл. , в качестве базовой выработки принят шахтный ствол с абсолютным коэффициентом готовности K_{wo} = 0,999.

Метод имитационного моделирования (Ушаков В.К. 1989) состоит в имитации возможных состояний ШВС (значений сопротивлений ее элементов и депрессий ГВУ) в соответствии с плотностями их распределений. Временная дискретизация процесса функционирования системы устанавливается с учетом плотностей распределения интервалов времени между профилактиками и внезапными разрушениями элементов. Число реализаций состояний ШВС определяется точностью расчета характеристик надежности системы. Для каждой реализации рассчитывается воздухораспределение в системе и фиксируются отказы. По результатам моделирования всех реализаций рассчитываются показатели надежности ШВС в целом. Блок-схема алгоритма моделирования представлена на рис. 2.

Метод базируется на решении сетевой задачи, поэтому корректен по своей постановке. Он обладает большими возможностями, поэтому является перспективным.

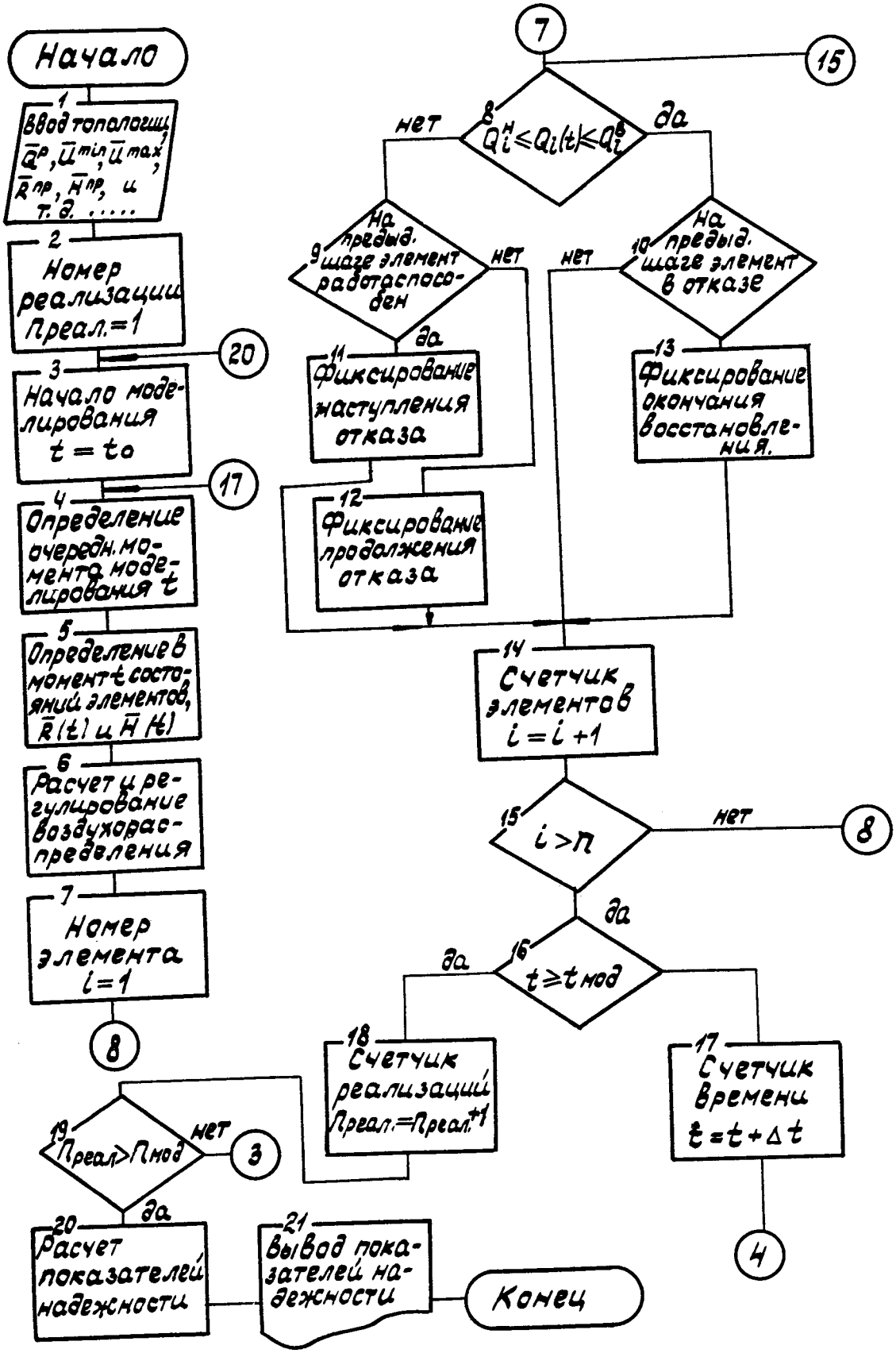

Начало
1 Ввод топологии, $\bar{Q}^{пр}$, $\bar{U}^{min}$, $\bar{U}^{max}$, $\bar{R}^{пр}$, $\bar{H}^{пр}$, и т.д. .....
2 Номер реализации Преал.=1
3 Начало моделирования t = to
4 Определение очеред. момента моделирования t
5 Определение в момент t состояний элементов, $\bar{R}(t)$ и $\bar{H}(t)$
6 Расчет и регулирование воздухораспределения
7 Номер элемента i=1
8 $Q_i^H \le Q_i(t) \le Q_i^B$
нет
да
9 На предыд. шаге элемент работоспособен
нет
да
10 На предыд. шаге элемент в отказе
нет
да
11 Фиксирование наступления отказа
12 Фиксирование продолжения отказа
13 Фиксирование окончания восстановления.
14 Счетчик элементов i = i + 1
15 i > n
нет
да
16 t ≥ tмод
да
нет
17 Счетчик времени t = t + Δt
18 Счетчик реализаций Преал.=Преал.+1
19 Преал > Пмод
нет
да
20 Расчет показателей надежности
21 Вывод показателей надежности
Конец
7
15
20
17
8
3
4

Рис. 2

Таблица: Относительные коэффициенты готовности элементов вентиляционных систем угольных шахт

Элементы	Относительные коэффициенты готовности
Стволы, капитальные квершлаги	I,0
Капитальные штреки, бремсберги, уклоны полевые	0,9
То же, пройденные по пласту	0,8
Ходки полевые	0,54
Ходки, пройденные по пласту	0,48
Участковые штреки в целике	0,I5
Участковые откаточные штреки в выработанном пространстве	0,0I
То же, вентиляционные штреки	0,07
Лавы	0,I
Вентиляционные двери одинарные	0,9
Вентиляционные шлюзы с двумя дверями	0,99
Главные вентиляторы	I,0

ЭФФЕКТИВНОСТЬ РАБОТЫ ШВС В АСПЕКТЕ НАДЕЖНОСТИ

Выше отмечалось, что отказы в ШВС обычно не вызывают полного прекращения функционирования системы. При этом масштабность отказов характеризуется их рангом. В то же время отказ любого ранга может иметь различную глубину σ:

$$\sigma = \frac{|\Delta Q|}{Q_R}, \qquad (I3)$$

где $|\Delta Q|$ - абсолютное отклонение фактического расхода воздуха в выработке от его нормативного значения Q_R в сторону недопустимых значений.
Для отдельной выработки среднее значение глубины отказа (его математическое ожидание) равно

$$\bar{\sigma}_i = \int_0^\infty \sigma_i f_i(\sigma) d\sigma, \qquad (I4)$$

где $f_i(\sigma)$ - плотность распределения случайной величины σ в i-ой выработке. Для всей ШВС имеем

$$\bar{\sigma}_S = \frac{1}{N} \sum_{i=1}^{N} \bar{\sigma}_i, \qquad (I5)$$

где N - количество элементов в ШВС.
При дискретном изменении эффективности функционирования системы ее суммарный выходной эффект E можно оценить через частные выходные эффекты e, свойственные ее отдельным состояниям (Надежность технических систем: Справочник I985):

$$E = \sum_{j=1}^{h} e_j p_j, \qquad (I6)$$

где p_j - вероятность нахождения объекта в j-ом состоянии; h - число состояний системы.
Если под эффективностью работы ШВС понимать степень обеспеченности системы воздухом, т.е.

оценивать ее через среднюю глубину отказов в системе δ_S, то для случая нахождения системы в работоспособном состоянии (отказов нет) $e_1 = 1$; для случая нахождения системы в состоянии отказа $e_2 = 1 - \delta_S$. Других состояний ШВС не имеет. Если, далее, вероятность нахождения системы в одном из возможных состояний оценивать через ее коэффициент готовности K_S, то $p_1 = K_S$, $p_2 = 1 - K_S$ и согласно (16)

$$E = 1 \cdot K_S + (1 - \delta_S)(1 - K_S) = \quad (17)$$
$$= 1 - \delta_S (1 - K_S).$$

При сравнении различных вариантов ШВС следует сопоставлять показатели E для отказов одинаковых рангов.

СТАТИСТИЧЕСКИЕ ДАННЫЕ

Оценка достоверности исходных данных и применяемых методов расчета может рассматриваться как величина, пропорциональная интенсивности отказов или коэффициенту готовности системы. По имеющимся данным наблюдений и экспертным оценкам, различие между проектными и фактическими значениями коэффициентов аэродинамического сопротивления выработок составляет в угольных шахтах 25–40%. Многочисленные коэффициенты, используемые при расчетах количества воздуха, являются результатом обобщения большого статистического материала, имеющего невысокую корреляцию. Они могут отличаться от фактических в 1,5–2 раза.

Метод расчета депрессии в целом имеет высокую достоверность, хотя использование квадратичной формулы $h = RQ^2$, где h – депрессия, R – сопротивление, Q – количество воздуха, в общем требует уточнения области ее применения, особенно когда дело касается расчета депрессий шахт со значительным удельным весом фильтрационных течений. В целом можно считать, что изменение показателя степени при Q на 0,01 вызывает погрешность в расчете h на 4–7% при Q = = 100 ÷ 1000 м^3/с.

В качестве первого приближения можно принять, что интегральная ошибка Δ (в долях единицы) подсистемы "Исходные данные" ("Методы расчета") связана с коэффициентом готовности подсистемы соотношением

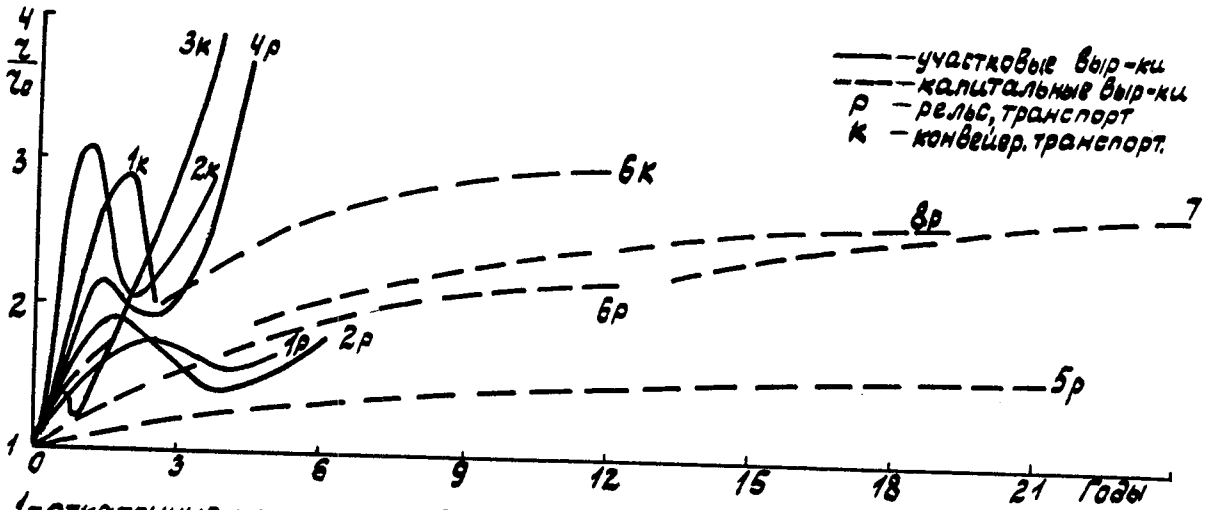
$$K_{d(m)} = e^{-\beta \Delta}, \quad (18)$$

где β – некоторый коэффициент. Надежность функционирования вентиляционных сетей определяется устойчивостью сопротивлений их элементов. Изменчивость во времени, или аэродинамическое старение, – свойство сопротивлений элементов. Характеристики процесса старения зависят от вида элемента и условий его работы. На рис. 3 приведены графики для расчета сопротивлений горных выработок разного типа, полученные на основании наблюдений на шахтах пологого падения Донбасса. Сопротивления выработок всех типов во времени увеличивается. Наиболее сильно изменяются сопротивления участковых выработок, наиболее устойчивы сопротивления капитальных полевых выработок. Ориентировочную оценку влияния старения выработок на надежность функционирования вентиляционной сети можно произвести с помощью критерия

$$K_a = \frac{\sum_{i=1}^{N} R_i}{\sum_{i=1}^{N} R_{oi}} \cdot \quad (19)$$

$$\cdot \left(1 + \sqrt{\frac{\sum_{i=1}^{N} \left(\frac{1}{N} \sum_{i=1}^{N} \frac{|R_i - R_{oi}|}{R_{oi}} - \frac{|R_i - R_{oi}|}{R_{oi}} \right)^2}{N}} \right)$$

где R_i, R_{oi} – соответственно фактическое и проектное сопротивление i-ой выработки; N – число выработок в сети. Критерий (19) получен из предположения о пропорциональности обще-



1-откаточные, целик-целик; 2-откаточные, целик-выр. пр-во; 3-Вентиляционные, целик-целик; 4-вентиляционные, целик-выр. пр-во; 5-откаточные, целик-целик; 6-наклонные, целик-целик; 7-откаточные полевые; 8-наклонные полевые.

Рис. 3

го сопротивления сети сумме сопротивлений составляющих сеть выработок, что можно допустить при значительном числе выработок в сети. Предполагается, что различие между проектным и фактическим воздухораспределением в сети пропорционально относительному изменению сопротивлений выработок и различию этих изменений между собой.

Коэффициент готовности горных выработок по внезапным разрушениям находится в пределах 0,995 ÷ 0,999. Эти отказы имеют экспоненциальный закон распределения.

Для шлюзов коэффициенты готовности имеют порядок 0,988, закон распределения времени отказов также экспоненциальный.

Показатели надежности подсистемы "Главные вентиляторные установки" зависят от способа резервирования энергоснабжения вентилятора и в меньшей степени от его типа. Для вентиляторов типа ВОКР, ВУПР, ОДВ и ВШЦ τ_4 = 0,75 ÷ 0,96 за 720 ч. работы. Для систем местной вентиляции этот показатель изменяется в пределах 0,5 ÷ 0,9 за 720 ч. работы в зависимости от схемы резервирования их элементов.

Вероятность безотказной работы выемочных участков угольных шахт за 720 ч. имеет порядок 0,4-0,9, рудных - 0,85-0,95. Аналогичные показатели для шахт в целом могут быть значительно ниже.

НАДЕЖНОСТЬ ШВС - КАТЕГОРИЯ ЭКОНОМИЧЕСКАЯ И СОЦИАЛЬНАЯ

Надежность ШВС может оказывать существенное влияние на экономику шахты. Отказы в ШВС вызывают остановки работ и связанные с этим потери добычи. Отказы могут приводить к авариям, что связано с потерей добычи, с затратами на ликвидацию последствий аварий, с несчастными случаями и связанными с этим расходами на лечение, содержание инвалидов труда и т.п.

Как система жизнеобеспечения шахты, ШВС призвана обеспечить здоровые и безопасные условия труда во всех горных выработках. Отказы ШВС влияют на жизнь и здоровье большого количества людей. Надежность ШВС не просто техническая характеристика. Она является одновременно и важной характеристикой безопасности шахт. Поэтому как таковая надежность ШВС имеет и социальное значение. Несомненна связь между уровнем надежности ШВС и уровнем безопасности труда в шахте. Повышение надежности ШВС будет способст-

вовать снижению травматизма, а также улучшению здоровья занятых на подземных работах людей.

ПУТИ ПОВЫШЕНИЯ НАДЕЖНОСТИ ШВС

Одним из важных и эффективных способов повышения надежности является резервирование элементов системы. Степень резервирования ШВС характеризуется прежде всего разветвленностью вентиляционной сети. Распределение добычи на большее число участков делает шахту менее зависимой от отдельных отказов и должно способствовать снижению их ранга. Резервирование путем проведения параллельных выработок способствует повышению характеристик надёжности ШВС. Так,расчет,выполненный для одной из шахт СССР,показал, что проведение лишь трех дополнительных параллельных выработок при начальном их числе в шахте 62 позволит повысить надёжность ШВС в I,3 раза. С помощью различных способов резервирования линии энергоснабжения можно повысить надёжность ГВУ в I,25 раза, а различные схемы резервирования элементов вентиляторных установок местного проветривания повышают их надёжность в I,25-I,9 раза.

Большое значение имеет повышение надёжности отдельных элементов ШВС, прежде всего выработок и вентиляционных сооружений. Важно уменьшить влияние процессов старения на надёжности этих элементов. Последнее можно достичь разработкой графиков профилактических ремонтов на основе закономерностей старения элементов. Характеристики старения должны учитывать уже на этапе проектирования ШВС.

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RELIABILITY OF MINE VENTILATION SYSTEMS

K.Z. Ushakov, V.K. Ushakov and O.V. Skopintseva

A mine ventilation system (MVS) is considered inefficient if it fails to provide sufficient quantities of air for ventilating all areas of the mine. In practice, the probability of MVS failures varies, depending on local conditions. MVS reliability is determined by four interrelated elements, i.e., input data, design methodology, ventilation networks and main fans. A quick estimate of the reliability of a mine ventilation network, which is the main MVS component, can be obtained using a relative reliability method based on relative reliability of separate workings from the point of view of the mine ventilation network topology.

Sudden failures of a MVS, caused by rock falls, are rare. Failures develop gradually as a result of worsening of aerodynamic properties of workings and structures due to aging and changes in airflow resistance of MVS components depend on deposit geology, roadway drivage and maintenance methods, and rock pressure. Generally, such changes can be adequately expressed using logarithmic-normal law. Decreases in MVS reliability adversely affect mine cost productivity and safety. Available methods contribute to ensuring MVS reliability. Steps involve an increased number of airways in parallel, a larger share of high stability workings (driven in stone, with monolithic lining, etc.) in the mine ventilation network, systematic maintenance of roadways and ventilation network, adequate stand-by MVS facilities, as well as increased reliability of input data and improved designs.

FIABILITE DES SYSTEMES DE VENTILATION MINIERS

K.Z. Ushakov, V.K. Ushakov et O.V. Skopintseva

On considère qu'un système de ventilation minier (MVS) est inefficace lorsqu'il est incapable de fournir des quantités suffisantes d'air pour ventiler toutes les zones d'une mine. En pratique, la probabilité d'échec d'un système de ventilation varie en fonction des conditions locales. On détermine la fiabilité d'un système de ventilation au moyen des quatre éléments corrélatifs suivants : données d'entrée, méthodologie de l'aménagement, réseaux de ventilation et ventilateurs principaux. On peut évaluer rapidement la fiabilité d'un réseau de ventilation minier, qui est le composant principal du système de ventilation minier, en utilisant une méthode de fiabilité relative basée sur la fiabilité relative de différentes exploitations, du point de vue de la topologie du réseau de ventilation minier.

Il est rare que des pannes soudaines de systèmes de ventilation soient provoquées par des chutes de roches. Les pannes se préparent peu à peu; elles sont dûes à l'aggravation des propriétés aérodynamiques des excavations et des structures; cette aggravation est elle-même provoquée par le vieillissement; les modifications de la résistance des composants des systèmes de ventilation miniers à la circulation d'air sont fonction des tassements géologiques, des méthodes de creusage et de maintenance des voies, ainsi que de la pression rocheuse. On exprime en général ce type de changements par la loi des logarithmes ordinaires. Lorsque la fiabilité des systèmes de ventilation miniers diminue, cela provoque un impact négatif sur la productivité et la sécurité minières. Les méthodes existantes contribuent à assurer la fiabilité de ces systèmes de ventilation. Elles comprennent l'augmentation du nombre de couloirs d'aération en parallèle, un pourcentage plus élevé d'exploitations à haute stabilité (creusées dans la pierre, à parois monolithiques, etc...) dans le réseau de ventilation minière, la maintenance systématique des voies et du réseau de ventilation minière, des systèmes de ventilation minière auxiliaires adéquats, ainsi que l'amélioration de la fiabilité des données d'entrée et des conceptions.

DIE ZUVERLÄSSIGKEIT VON GRUBEN-BEWETTERUNGSSYSTEMEN

K. Z. Ushakov, V. K. Ushakov, O. V.Skopintseva

Ein Grubenbewetterungssystem (GBS) wird als ineffektiv angesehen, wenn es nicht in der Lage ist, ausreichende Mengen von Luft zu Ventilationszwecken in alle Bereiche der Grube zu schaffen. Die Wahrscheinlichkeit, daß ein GBS in der Praxis versagt, variiert in Abhängigkeit von den jeweiligen lokalen Bedingungen. Die Zuverlässigkeit eines GBS wird von vier von einander abhängigen Elementen bestimmt: von den Eingangsdaten, der Vorgehensweise bei der Auslegung, von den Bewetterungsnetzen und den Hauptlüftern. Eine schnelle Abschätzung der Zuverlässigkeit eines Bewetterungsnetzes, welches die Hauptkomponente in einem GBS darstellt, kann durch Anwendung der Methode der relativen Zuverlässigkeit, welche auf der relativen Zuverlässigkeit separater Teile vom Gesichtspunkt der Topologie des Bewetterungsnetzes beruht, gewonnen werden.

Ein plötzliches Versagen des GBS, bedingt durch hereinbrechendes Gebirge, ist sehr selten. Ein Versagen entwickelt sich langsam als Auswirkung von sich verschlechternden aerodynamischen Eigenschaften der Grubenbaue, verursacht durch Alterung und Änderungen im Widerstand der Bewetterungsnetz-Komponenten, was von der Geologie der Lagerstätte, der Auffahrungs- und Instandhaltungsmethode der Strecken und dem Gebirgsdruck abhängig ist. Im allgemeinen können solche Änderungen hinreichend durch die Anwendung der Normallogarithmen-Gesetze ausgedrückt werden. Ein Rückgang in der Zuverlässigkeit eines GBS wirkt sich negativ auf Wirtschaftlichkeit und Sicherheit aus. Die verfügbaren Methoden tragen dazu bei, die Zuverlässigkeit der GBS zu gewährleisten. Die Schritte hierzu umfassen folgende Punkte: vermehrte Anzahl von parallelen Wetterwegen, eine höhere Anzahl von stabileren, in festem Stein aufgefahrenen Strecken mit gemauerter Auskleidung in einem Bewetterungsnetz, adäquate Reserveeinrichtungen und eine gesteigerte Zuverlässigkeit der Eingangsdaten sowie verbesserte Auslegungen.

矿井通风系统的可靠性

K。Z。乌霞可夫， V。K。乌霞可夫，
及 O。V。司可品济伐

矿井通风系统（MVS）如果不能供给所有矿井区域通风所需的空气量便被认为失效。实际上MVS失效的可能性随当地情形而变化。MVS可靠性由四个相关的因素所决定：输入数据，设计方法，通风网路及主风扇。矿井通风网路是MVS的主要元件，它的可靠性可用其中分件在主件中所占位置情形而定的分件可靠性来求出。MVS由于岩石塌落而失效的情形极少。MVS失效常：由于通道和结构因年深日久而使空气动力性能减弱。MVS元件的空气流动阻力的变化则视地质条件，道路开筑和保养，及岩石压力而定。通常这种变化可用对数法线律表示。MVS可靠性的减少对矿井成本出产及安全均有不良影响。要增加MVS的可靠性可用下列各种办法：增加并联通风道，在矿井通风网路中增加高稳定的巷道（打入楔石，用整体的镶衬等），系统性的保养巷道和通风网路，足够的MVS备用设备，增加输入数据的可靠性及改进设计。

БОРЬБА С ПЫЛЬЮ В ШАХТАХ

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РЕФЕРАТ

В Советском союзе разработан широкий комплекс способов и средств борьбы с пылью в шахтах, позволяющий эффективно снижать интенсивность пылеобразования и концентрацию взвешенной пыли в воздухе.

Значительные исследования проведены по предотвращению пылеобразования путем гидромеханического и импульсного воздействия на угольный массив высоконапорными струями жидкости под давлением 30 МПа, что позволяет снизить выход пыли и обеспечить концентрацию ее на рабочем месте в пределах 7-15 мг/м^3. Установлены эффективные параметры предварительного увлажнения массива растворами различных поверхностно-активных веществ (ПАВ), обеспечивающего эффективность пылеподавления до 90%. Расход ПАВ составляет 0,1-0,15 % по объему.

Теоретически и экспериментально установлены параметры орошения и оросительноэжекционных систем, работающих под давлением 1,0-1,5 МПа. Доказана эффективность электроподзарядки жидкости в пределах 10^3 нКл/л, что позволяет достичь эффективности пылеподавления до 95%. При этом эффективность осаждения тонких фракций пыли (менее 10 мкм) увеличивается в среднем на 15-20%. Теоретически обоснованы, разработаны и созданы конструкции оросительно-эжекционных систем для проходческих комбайнов, работающие на гидравлической энергии оросителей, позволившие снизить запыленность воздуха на рабочем месте до 10-20 мг/м^3 по общей массе.

Обоснованы параметры пылеотсоса для различных технологических процессов, в увязке с пылединамикой в горных выработках, координатами всаса и кратностью отсоса. Показано, что эффективная кратность отсоса находится в пределах 1-1,4 в зависимости от условий применения пылеотсасывающих установок. Для достижения ПДК следует применять комплекс противопылевых мероприятий. Показано, что при

известных ПДК, начальной концентрации пыли и возможной эффективности пылеподавления различных способов и средств можно выбрать необходимый комплекс противопылевых мероприятий для достижения ПДК при различных технологических процессах.

ВВЕДЕНИЕ

Запыленность воздуха в шахтах значительно превосходит предельно-допустимые концентрации (ПДК) пыли, что приводит к различным профессиональным заболеваниям дыхательной системы, глаз, кожи.

Снижение заболеваемости обеспечивается предупреждением пылеобразования и осаждения пыли путем гидромеханического и импульсного разрушения горного массива воздействием высоконапорными струями жидкости под давлением 30 МПа и выше. Проведенные исследования показали, что при этом достигается снижение концентрации пыли до 7-15 мг/м^3. Показано, что предварительное увлажнение угольного массива жидкостью с различными поверхностно-активными веществами (ПАВ) эффективно в режиме фильтрации и капиллярного насыщения. Продолжительность увлажнения составляет несколько часов. Очень важно делать перерывы между увлажнениями в пределах 70 часов. Высоконапорное увлажнение позволяет расширить область воздействия жидкости путем раскрытия трещин, что позволяет увлажнить больший объем. При этом давление должно быть в пределах $P_н$=0,025 $H_з$.

Для осаждения взвешенной пыли применяется орошение полидисперсным водяным аэрозолем с диаметром капель 10-100 мкм и при давлении 1,5-2,5 МПа. Большое влияние оказывает подзарядка водного аэрозоля, позволяющая повысить эффективность осаждения тонкодисперсной пыли. Величина подзарядки составляет 10^3 нКл/л.

Оросительные эжекционные системы, использующие эффект захвата и коагуляции, позволяют снизить запыленность воздуха на 90% и более. Эффективный режим работы оросителей при давлении 1,0-1,5 МПа. Эти системы могут быть использованы и как пылеотсасывающие. Очень важно увязать параметры пылединамики с геометрическими параметрами пылеотсасывающей установки и кратностью отсоса, которая должна быть в пределах 1-1,4.

С учетом известных коэффициентов эффективности пылеподавления различных способов и

средств и ПДК определяется комплекс мероприятий, обеспечивающий достижение ПДК.

ПРЕДУПРЕЖДЕНИЕ ПЫЛЕОБРАЗОВАНИЯ

Современное угольное предприятие характеризуется высокой насыщенностью высокопроизводительным, мощным оборудованием, машинами, механизмами, работа которых приводит к образованию различных вредных факторов, создающих опасность травмирования горнорабочих, ухудшение санитарно-гигиенических условий труда, возможность возникновения взрыва. Одним из опасных факторов является пыль, вызывающая профессиональные заболевания, снижение видимости на рабочем месте и создающая опасность взрыва.

Рудничная пыль является причиной различных профессиональных заболеваний дыхательной системы трудящихся (пневмокониоз и пылевой бронхит), глазных заболеваний (коньюктивит), кожных заболеваний (папулезная сыпь, пиодермия, дерматит) и причиной загрязнения окружающей среды. Борьба с пылью в шахтах является важной общегосударственной задачей, имеющей социально-экономическое значение. Вот почему успешное ее решение настоятельно диктуется самой жизнью.

Предупреждение пылеобразования и эффективное осаждение образовавшейся пыли возможно только на основе внедрения принципиально новых способов разрушения массива, исключающих его измельчение, и высокоэффективных способов и средств пылеподавления.

Эффективное снижение удельного пылеобразования достигается применением гидромеханического и импульсного способов разрушения горного массива, разработанных в СССР.

Гидравлическое воздействие на массив осуществляется высоконапорными струями под давлением 30 МПа и выше. Выход фракций угля 0-6 мм при этом не превышает 20%, а фракций 50 мм - от 35 до 40%. При этих режимах разрушения концентрация пыли в направлении вентиляционной струи не превышает 5 мг/м^3.

Гидромеханическое разрушение массива осуществляется одновременно высоконапорной струей воды под давлением до 30 МПа и резцом. Запыленность воздуха уменьшается в 200-300 раз по сравнению с запыленностью при механическом разрушении массива угля. При этом концентрация пыли в зоне нахождения горнорабочего составляет 7-15 мг/м^3 (рис.1)

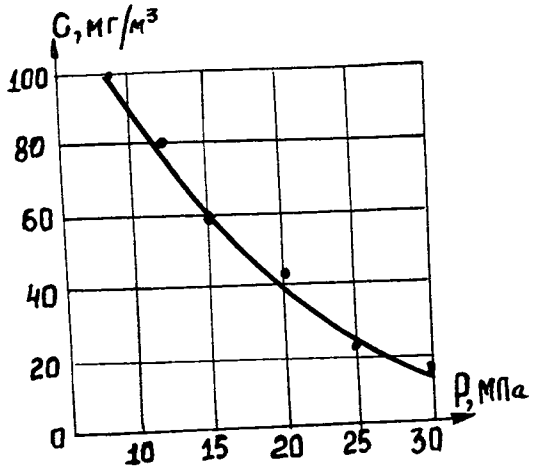


Рис. I

Изменение запыленности воздуха С от давления Р при гидромеханическом разрушении массива.

Однако такие способы разрушения массива возможны при незначительной крепости угля. При крепких углях используются традиционные механические способы их разрушения, что приводит к высокому удельному пылевыделению, достигающему I5кг/т отбиваемого угля. А это в свою очередь способствует образованию концентрации пыли 5000–I5000 мг/м^3.

Разработка высокоэффективных способов и средств борьбы с пылью основывается на знании физико-химических свойств ее, параметров пылединамики, физико-химической сущности взаимодействия пыли с воздействующим фактором.

Эффективное пылеподавление осуществляется жидкостями путем придания им свойств, улучшающих смачиваемость пыли, и задания оптимальных режимов работы способов и средств осаждения пыли.

Известно, что угольная пыль в большинстве своем гидрофобна. Поэтому для повышения смачиваемости ее применяются ПАВ различного типа (ПО–I2, Синтанол и другие).

Многочисленные исследования показали, что ПАВ эффективны при предварительном увлажнении массива в режиме фильтрации и капиллярного насыщения. При этих условиях происходит хорошее насыщение угольного массива жидкостью и повышается его влажность на 3% и более. Фильтрация может осуществляться при незначительных давлениях (низконапорное увлажнение при давлении до I МПа). Продолжительность нагнетания составляет несколько часов. Целесообразно делать циклы нагнетания с перерывом до 70 часов, что позволяет равномерно насыщать массив.

Высоконапорное увлажнение производится при давлениях $P_н \leq 0{,}025\, H_з$, где $H_з$ – глубина залегания пласта, м. Эффективность снижения пылеобразования достигает 70–80%.

Применение ПАВ позволяет увеличить эффективность пылепо-

давление на 10-20% по сравнению с чистой водой. Расход ПАВ составляет 0,1-0,15% по объему. Схемы увлажнения могут быть различные.

Широко распространенное орошение с точки зрения физико-динамического и химического взаимодействия трехфазной среды изучено слабо. По нашему мнению, основным в процессе взаимодействия капель жидкости и частиц пыли имеют место три явления - инерционное осаждение частиц пыли на капельках жидкости, зависящее от относительной скорости движения их; диффузионно-конвективное перемешивание внутри факела орошения частиц пыли и капель жидкости; конденсация тончайших капель жидкости на частицах пыли, являющихся ядрами конденсации. Как показали исследования, основное количество пыли смачивается и коагулирует внутри факела орошения и выпадает в виде шлама. Для усиления всех эффектов следует создавать полидисперсный водяной аэрозоль с диаметром капель 10-100 мкм (рис.2) и давлением жидкости 1,5-2,5 МПа. Добавление ПАВ позволяет увеличить степень диспергирования жидкости и создавать подзарядку водного аэрозоля. С точки зрения смачиваемости пыли при орошении растворами ПАВ, последние не оказывают существенного

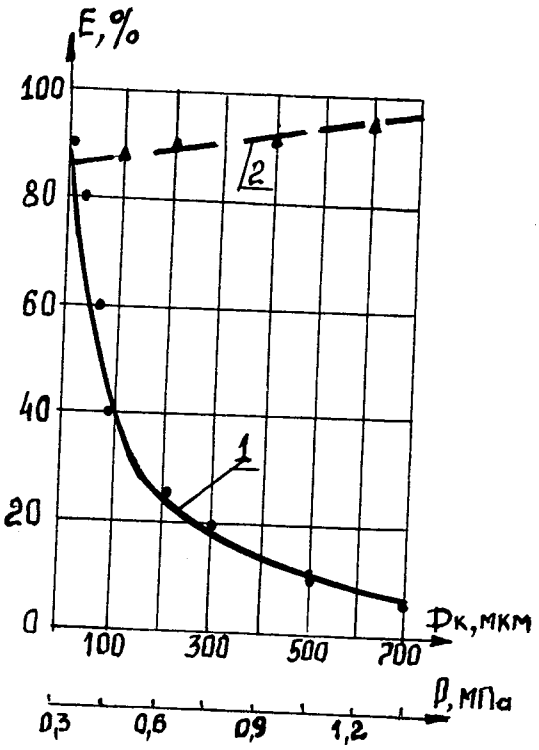


Рис.2

Эффективность пылеподавоения
1- при изменении диаметра капель;
2- при изменении давления.

влияния, так как время контакта капли и пылинки значительно меньше времени адсорбции ПАВ на поверхности пылинки.

Существенное влияние оказывает искусственная подзарядка водного аэрозоля. Она может осуществляться как контактным способом, так и индукционным. При этом следует учитывать естественную зарядку частиц пыли угля и знак заряда. Нами установлено, что пыль угля заряжается как положительно так и отрицательно. Величина удельной напряженности пылево-

го аэрозоля достигает

$$0{,}15\text{–}3{,}0 \ \frac{\text{в/м}}{\text{мг/м}^3}.$$

Величина заряда водного аэрозоля, заряжаемого контактным или индукционным способами при давлениях воды 1,5–2,0 МПа, может быть доведена до 10^3 нКл/л. Применение оросителей с подзарядкой воды позволяет повысить эффективность пылеподавления до 95%. Очень важно то, что значительно повышается эффективность пылеподавления фракций пыли менее 10 мкм. Применение подзарядки водного аэрозоля и обеспечение давлений воды в пределах 1,5–2,5 МПа позволяет создать высокотурбулизированное движение запыленного воздуха внутри факела с электрокоагуляцией пылевых частиц. Учитывая эффект эжекции, турбулизации и подзарядки капель, следует идти по пути создания на основе этих эффектов оросительных систем для пылеподавления в подготовительных и очистных забоях. Они позволяют эффектно осаждать пыль в местах ее образования, например, при работе проходческих комбайнов с рабочим органом избирательного действия и добычных комбайнов. Нами разработаны теоретические основы и созданы конструкции оросительно-эжекционных систем для проходческих комбайнов, позволяющие обеспечить снижение запыленности на рабочем месте до 10–20 мг/м^3. Сущность их заключается в том, что для отсоса запыленного воздуха, эжектирования его внутрь факела орошения и дальнейшего осаждения пыли используется гидравлическая энергия орошающей жидкости. Разработана конструкция системы орошения с таким расчетом, чтобы запыленный воздух не смог распространяться от места образования пыли по длине выработки. Схема устройства представлена на рис.3.

Эффективный режим работы оросителей находится в пределах 1,0–1,5 МПа. Все устройство конструктивно располагается на рабочем органе и работает от обычного водопровода.

Как было отмечено ранее, ПАВ не оказывают существенного влияния на смачивание пыли, но они оказывают влияние на коэффициент захвата частиц пыли каплями жидкости, что способствует повышению эффективности пылеподавления. Для гидрофобной пыли захват происходит только при условии, если частицы обладают достаточной кинетической энергией, отнесенной к единице поверхности частицы. Она может быть определена по формуле

$$\varepsilon_к = \frac{m_п v_п^2}{2 S_п} = \frac{1}{6} \gamma_п \rho_п v_п^2, \ \text{Дж/м}^2 \quad (1)$$

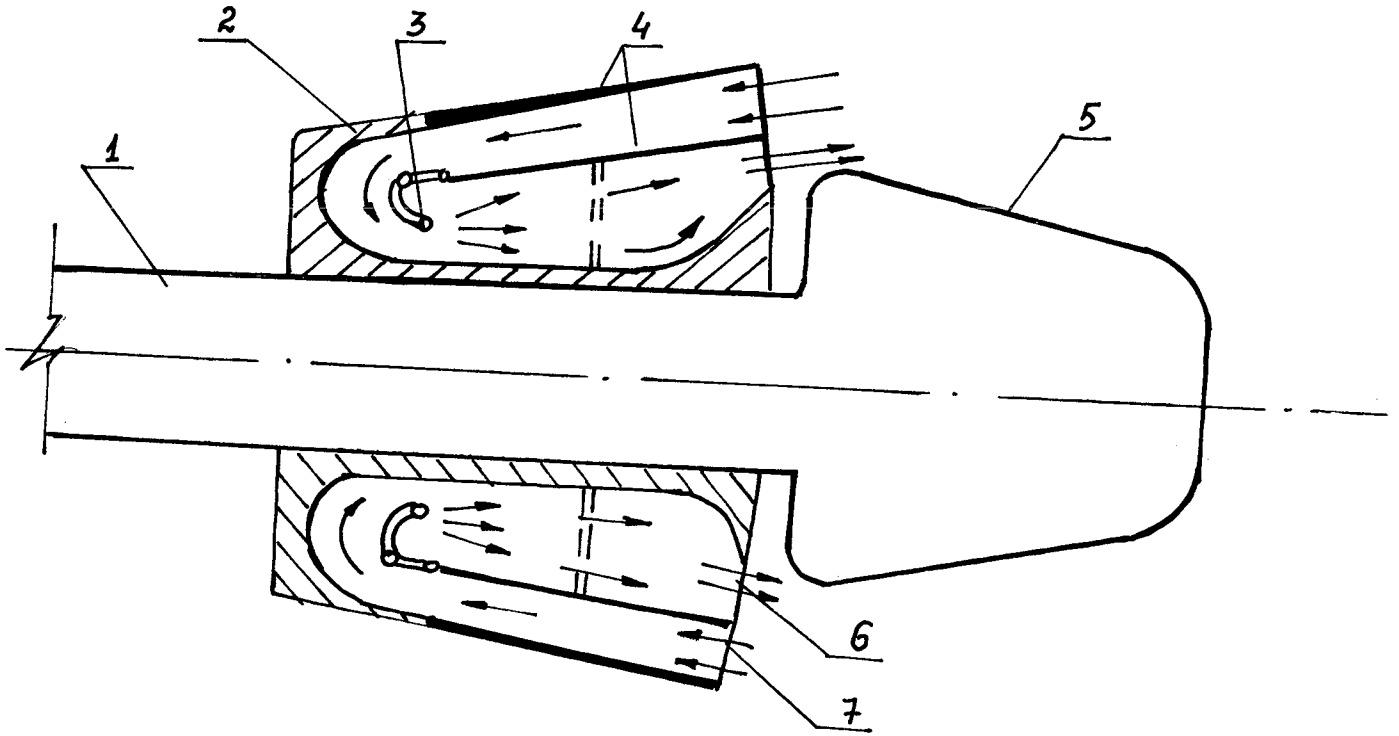


Рис.3. Схема эжекционно-оросительного устройства для проходческого комбайна с избирательным рабочим органом.

I- стрелы комбайна; 2 - корпус устройства; 3-оросители;
4-резиновые щитки; 5- рабочий орган комбайна; 6-водовоздушная смесь;
7-эжектируемый запылённый воздух.

где $m_п$ – масса пылинки, кг;

$V_п$ – скорость движения пылинки, м/с; $S_п$ – поверхность пылинки, $м^2$; $\rho_п$ – плотность пылинки, кг/$м^3$; $r_п$ – радиус пылинки, м.

Захват пылинки каплей произойдет тогда, когда величина кинетической энергии окажется большей или равной энергии захвата $W_з$; или большей или меньшей энергии налипания $W_н$.

$$\varepsilon_к \geq W_з$$

$$\varepsilon_к \geq W_н$$

Энергия захвата может быть определена по формуле

$$W_з = -2\sigma_{т.г}\cos\theta, \quad (2)$$

где $\sigma_{ж.г}$ – удельная поверхностная энергия на границе жидкое-газ;

θ – краевой угол смачивания.

Энергия налипания определяется по формуле

$$W_н = -\sigma_{ж.г}\cos\theta \quad (3)$$

Минимальные значения скоростей встречи и налипания частиц пыли за счет инерции и адгезии определяются по формулам

$$V_з = \sqrt{\frac{6W_з}{\rho_п r_п}}, \ м/с \quad (4)$$

$$V_н = \sqrt{\frac{6W_н}{\rho_п r_п}}, \ м/с \quad (5)$$

При условии, когда пыль полностью не смачивается $\cos\theta = -1$. Тогда получим, что $V_з \sim \sigma_{ж.г}^{1/2}$. Отсюда видно, что для повышения эффективности захвата необходимо либо понизить поверхностное натяжение жидкости, либо повысить скорость встречи.

Для разработанной эжекционно-оросительной системы целесообразно применение ПАВ, так как в этом случае имеют место оба явления. Применение ПАВ эффективно еще и потому, что они повышают электризацию водного аэрозоля. Наиболее эффективными с этой точки зрения являются ПАВ неионогенного типа, которые повышают напряженность электростатического поля в 2–4 раза по сравнению с чистой водой.

Широкие исследования проведены по изучению процесса пылеотсоса и пылеудавливания. Пылесос является тогда эффективным, когда параметры его работы увязаны с пылединамикой в горных выработках. Эффективность пылеотсоса является функцией координат всаса и кратности отсоса, т.е.

$$\eta = f(L_x, L_y, \varphi), \quad (6)$$

где L_x и L_y – координаты всаса; φ – кратность отсоса.

Теоретическая эффективность пылеотсоса определяется из выражения

$$\eta = \int_0^{\frac{L_y}{B}} u c \, dS, \quad (7)$$

где u – скорость движения воздуха, м/с;

L_y – безразмерное расстояние

до всаса, м;

B – ширина выработки, м;

C – безразмерная концентрация пыли; S – площадь всаса, $м^2$.

Эффективная кратность отсоса определяется из выражения

$$\varphi = \int_0^{\frac{L_y}{B}} u\,dS \qquad (8)$$

Научно и практически доказано, что кратность отсоса $\varphi \geqslant 1$. При $\varphi = 1$ $\eta = 90\%$ и более, при $\varphi = 1{,}4$ $\eta = 98{,}5\%$, т.е. практически весь запыленный воздух проходит через отсасывающее устройство.

Для осаждения пыли из отсасываемого воздуха целесообразно иметь пылеулавливающие установки с фильтрующими элементами из синтетических материалов, обеспечивающих эффективность пылеосаждения 90% и более.

Рассмотренный нами комплекс способов осаждения пыли позволяет снизить концентрацию ее в диапазоне фракций менее 10 мкм до ПДК, а по общей массе до 10–20 $мг/м^3$.

Выбор способов производится исходя из условий интенсивности образования пыли и горнотехнических условий работы технологического оборудования.

Суммарная эффективность пылеподавления при применении комплекса противопылевых мероприятий определяется по формуле

$$Э_к = 1-(1-Э_1)(1-Э_2)(1-Э_3)\cdots$$

$$\cdots(1-Э_n)\,100 \qquad (9)$$

где $Э_1$, $Э_2$, $Э_3$ ... $Э_n$ – эффективность отдельных противопылевых мероприятий.

Зная первоначальную концентрацию пыли C_0 и значение ПДК n определяется требуемая эффективность пылеподавления по формуле

$$Э_{тр} = 1 - \frac{n}{C_0}\,100 \qquad (10)$$

Таким образом, комплекс противопылевых мероприятий должен обеспечить требуемую эффективность пылеподавления, т.е.

$$Э_к = Э_{тр}$$

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DUST SUPPRESSION IN MINES

B.F. Kirin

Coal mine dust is responsible for respiratory diseases, explosions, environmental pollution, and has negative economic and social consequences. Dust control involves highly efficient methods that use chemically active substances (CAS) and surface active substances (SAS), which can reduce dust concentrations by 98%. CAS are used for preliminary saturation of the rock and coal mass and lower dust formation and emission by 50-60%; SAS enhance the wettability of dust in the air, and on mine surfaces and the fissures; SAS also decrease dust formation by 60-70%; they also increase settling of fine airborne dust by 20-40%. The efficiency of CAS and SAS for dust suppression has been confirmed experimentally and in practice. A new dust-suppression system for heading machines, using standard spray functioning at pressures of 1-2 5 MP, has been developed. It is based on a triple concept: dust extraction, wetting and settling by the sprays, partial recirculation and additional preventive zones where the mass is destroyed by the functioning organ, which also prevents the formation of methane concentration zones.

SUPPRESSION DES POUSSIERES MINIERES

B.F. Kirin

Les poussières de charbon sont la cause de maladies respiratoires, d'explosions, de la pollution de l'environnement; elles ont en outre des conséquences économiques et sociales négatives. Le contrôle des poussières minières requiert l'emploi de méthodes très efficaces utilisant des substances chimiques actives (CAS) et des substances tensio-actives (SAS) pouvant réduire de 98% les concentrations de poussières. On utilise les CAS pour la saturation préliminaire des masses rocheuses et charbonnières et pour diminuer de 50 à 60% la formation et l'émission des poussières; les SAS améliorent la mouillabilité des poussières en suspension dans l'air ou à la surface des mines et dans les fissures; elles diminuent de 60 à 70% la formation de poussières et augmentent de 20 à 40% le tassement des poussières fines en suspension dans l'air. Des essais des méthodes CAS et SAS et l'expérience pratique ont permis de confirmer leur efficacité pour la suppression des poussières. On a développé un nouveau dispositif de suppression des poussières pour les machines d'avancement, qui utilise un vaporisateur standard fonctionnant à des pressions de 1-2 5 MP. Ce dispositif est basé sur un triple concept: l'extraction des poussières, l'arrosage et le tassement par les vaporisateurs, la recirculation partielle et la création de zones préventives supplémentaires dans lesquelles la masse est détruite par l'organe de fonctionnement, ce qui empêche également la formation de zones de concentration de méthane.

STAUBBEKÄMPFUNG IN GRUBEN

B. F. Kirin

Staub in Kohlegruben ist für Erkrankungen der Atemorgane, Explosionen und Umweltverschmutzung verantwortlich und hat negative ökonomische und soziale Konsequenzen. Staubbekämpfung umfaßt höchst effektive Methoden, welche chemisch aktive Substanzen (chemically active substances - CAS) und oberflächenaktive Substanzen (surface active substances - SAS) einschließen. Diese Methoden können die Staubkonzentrationen um 98% reduzieren. CAS-Methoden werden für die vorläufige Sättigung der Stein- und Kohlemasse angewendet und verringern die Staubbildung und -aufwirbelung um 60 - 70%. Die SAS-Methoden verbessern die Benetzbarkeit von Staub in der Luft und von Grubenoberflächen und -spalten; SAS-Methoden verringern ebenso die Staubbildung um 60 - 70%, sie beschleunigen aber auch das Absetzen von feinem, flugfähigem Staub um 20 - 40%. Die Effizienz von CAS- und SAS-Methoden zur Staubbekämpfung wurde im Versuch und in der Praxis bestätigt. Ein neues Staubbekämpfungs-System für Streckenvortriebsmaschinen, welches standard Sprühdüsen mit einem Druck von 1 - 2.5 MPa verwendet, wurde entwickelt. Es gründet sich auf ein dreifaches Konzept: Staubabsaugung, Befeuchtung und Niederschlagung durch die Düsen, teilweise Wiederumwälzung und zusätzliche Schutzzonen, in denen der Staub unschädlich gemacht wird, was auch die Bildung von Methankonzentrations-Gebieten verhütet.

矿中尘埃抑制

B。F。克林

煤灰尘埃会引起呼吸器官疾病，爆炸和环境的污染，而且有不良的经济和社会方面的影响。尘埃控制使用化学作用物质（CAS）和表面作用物质（SAS）作有效的处理，可减少灰尘浓度98%之多。CAS用在岩石和煤碳初步饱和的情形下，可以减低尘埃生成和放射50-60%；SAS可增加尘埃在空气中的浸湿性，能减低尘埃生成60-70%；它们并能增加空中尘埃下沉20-40%。CAS及SAS对尘埃抑制的效率曾由实验和经验证明。开采机上曾装上尘埃抑制设备，用1-2。5兆帕斯卡压力的标准喷雾器施行。尘埃抑制是根据下列三个作用即尘埃抽除，浸湿和沉淀。沉淀是由喷雾，部分回流及产生防尘范围所造成。这范围里尘粒被作用物质所消除，同时也可以防止甲烷浓度集中区域的产生。

WAERMEBILANZ DER GRUBENWETTER IM BRAUNKOHLENBERGWERK KREKA

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ZUSAMMENFASSUNG

Die Temperatur der Grubenwetter ist einer der wichtigsten Faktoren, der auf das Grubenklima einwirkt. Auf die Temperatur der Grubenwetter wirken viele Faktoren ein, von denen die bedeutendsten sind: die Temperatur der atmosphär. Luftfeuchtigkeit, Luftverdichtung und Luftverdünnung, Elektrogeräte, Sprengen, Gestein, chemische Prozesse, Luftreibung, Grubenbeleuchtung sowie die Grubenarbeiter.

Beim Untersuchen der einzelnen Faktoren, die auf die Temperatur der Grubenwetter einwirken, wurde festgestellt, dass einige Faktoren die Temperatur der Grubenwetter herabsetzen, während sie die anderen erhöhen, und zwar verschiedentlich in einzelnen Grubenteilen und Jahreszeiten.

Chemische Prozesse, Luftfeuchtigkeit, Elektrogeräte sowie Gestein sind die Faktoren, die auf die Grubenwettertemperatur im Braunkohlenbergwerken am höchsten einwirken.

VORWORT

Je tiefer die Gruben werden und je mehr die Mechanisierung eingefürt wird, desto mehr wird der Arbeitskomfort bedroht und demgemäss werden die Arbeitsbedingungen immer schwerer und gesundheitsgefährlicher.

Durch Verschlechterung der klimatischen Arbeitsbedingungen in den Kohlenbergwerken war es notwendig, ein neues wissenschaftliches Fach zu entwickeln, das sich mit der Konditionierung der Grubenweter befasst.

Unter dem Begriff "ungünstige klimatische Arbeitsbedingungen in den Gruben"versteht man gewönnlich gemeinsame Einwirkung der Temperatur - und Feuchtigkeiterhöhung der Grubenwetter und kleine Bewegungsgeschwindigkeit der Wetter, wodurch Wärmeaustausch zwischen dem menschlichen Organismus undder Arbeitsumgebung erschwert wird. Auf diese Probleme stossen vorwiegend die Kohlenbergwerke mit grossen Abbautiefen, in denen das Wärmeproblem die sonstigen Probleme verbunden mit anderen Spezifitäten der Naturbedingunen der Kolengewinnung wie z.B.Gase, Staub übersteigt.

Die Grubenwettertemperatur ist sicher einer der wichtigsten Faktoren, der auf das Grubenklima einwirkt. In den Kohlenbergwerken mit grösseren Tiefen und grösseren Wärmequellen erschwert die Temperatur der Grubenwetter nicht nur die Grubenarbeits bedingungen, sondern stellt in meisten Fällen der begrenzende Faktor für die Produktion und Entwicklung des Kohlenbergwerks dar. Ausser in tiefen Gruben kommt es zur bedeutenden Erhöhung der Grubenwettertemperatur auch in Gruben, wo Erzoxidation erscheint.

In jugoslawischen Kohlenbergwerken ist die Wärme der Grubenwetter, als Faktor der Grubenklima, ungenügend untersucht. Die Entwicklung des Bergwesens verlangt, dass man dieses

Problem tiefer erforscht, damit sich die Arbeitsbedingungen der Grubenarbeiter winigstens den Arbeitsbedingungen der anderen Tätigkeiten annähern, in denen der Varbesser-

ung der Arbeitsbedingungen immer grossere Aufmerksamkeit gaschenkt wird.

Im Zusammenhang mit dem oben Erwähnten hat diese Arbeit die Aufgabe, auf dem Beispiel der Grube"II Krovni sloj" des Braunkohlen bergwerks Lipnica die Hauptcharakteristiken des Wärmeregimes und ihre einzelne Einwirkung auf Wärmebilanz der Grubenwetter zu bestimmen.

2. MESSERGEBNISSE UND UNTERSUCHUNG DER FAKTOREN, DIE AUF DIE GRUBENWETTERTEMPERATUR EINWIRKEN

Zwecks Untersuchung der Temperaturänderung der Grubenwetter und der Einwirkung der einzelnen Faktoren wurden folgende Messungen vorgenommen:

Auf der Fläche:

- Lufttemperatur
- barometrischer Druck

In der Grube:

- Grubenwettertemperatur
- relative Luftfeuchtigkeit
- barometrischer Druck
- guantitative Luftbilanz
- Gesteinetemperatur
- Wetterdepression
- Leistung und Anordnung der Elektrogeräte
- Menge und Anordnung der Grubenkohle
- Explosivverbrauch und
- Zahl der Arbeiter

Um die Einwirkung der Paramerer der Eingangsluft auf das Wärmeregime in der Grube untersuchen zu koennen, wurden Lufttemperatur und berometrischer Druck auf der Fläche jeden Tag im Laufe eines Kalenderjahres gemessen.

Um die Einwirkung der einzelnen Faktoren auf die Grubenwettertemperatur so real wie möglichfeststellen zu können, wurden fast alle parameter während normales technologisches Prozesses der Kohlengewinnung gemessen.

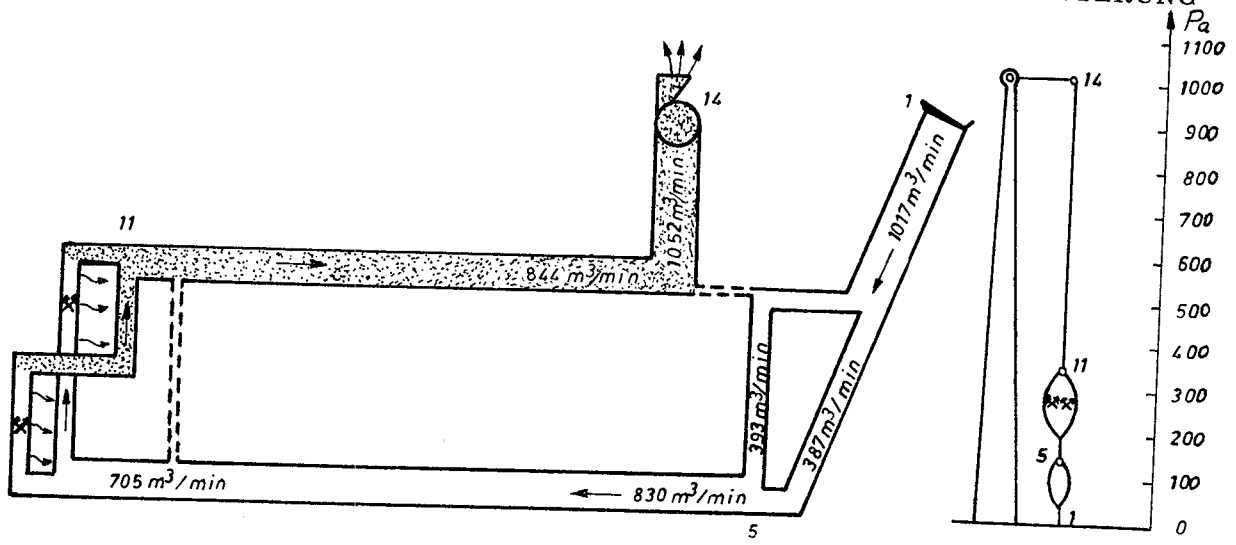
Alle variablen Parameter wurden alle 15 Tage im Laufe eines Kalendarjahres gemessen.

2.1 Temperaturänderungen der Grubenwetter durch atmosphärischen Druck

Durch Aenderung der Höhenlage der Luft beim Durchgehen durch die Grube wird atmosphar. Druck geändert, der die Dichte und somit die Temperatur der Grubenwetter ändert.

Unter der Voraussetzung, dass es keine Wärmeaustausch zwischen der Luft und den Wänden der Grubenbaue gibt, kann man diesses Prozess als adiabatische Zustandsänderung betrachten.

BILD 1.- LINEARER MENGENSTAMMBAUM DER GRUBENBEWETTERUNG



Temperaturänderung der Grubenwetter durch Luftverdichtung (Temperaturgradient) wurde nach folgender Formel berechnet:

$$\frac{dt}{dH} = \frac{\frac{c_p}{c_v} - 1}{\frac{c_p}{c_v} \times R}, \ ^{o}C/m$$

c_p - spezif.Wärme der Masse bei konstantem Druck
c_v - spezif.Wärme der Masse bei konstantem Volumen
R - Gaskonstante der Luft

Tabelle 1 - Temperaturänderung der Grubenwetter durch Luftverdichtung und -verdünnung

	T °C		
Monat	1-5	5-11	11-14
Jan.	2,103	- 1,051	- 0,958
Feb.	2,104	- 1,052	- 0,958
März	2,103	- 1,052	- 0,957
Apr.	2,102	- 1,051	- 0,958
Mai	2,101	- 1,050	- 0,958
Juni	2,097	- 1,048	- 0,956
Juli	2,097	- 1,048	- 0,956
Aug.	2,097	- 1,048	- 0,956
Sept.	2,097	- 1,048	- 0,957
Okt.	2,100	- 1,050	- 0,956
Nov.	2,104	- 1,051	- 0,954
Dez.	2,104	- 1,052	- 0,957
Durchschnnitt	2,101	- 1,050	- 0,957

Beim Durchgehen durch die Grube wurde die Luft von Punkt 1-5 um 2,101 °C durch Luftdichteerhöhung erwärmt. Gehend zu Punkt 14 wurde die Lufttemperatur durch Luftverdünnung erniedrigt. Aufgrund der Messergebnisse ist Wärmegradient in diesem Kohlenbergwerk

$$\frac{dh}{dt} = 102 \ m/C^{o}$$

2.2 Einwirkung des Gesteins auf die Temperatur der Grubenwetter

Wärmeaustausch zwischen dem Gestein und der Grubenwetter wurde nach folgender Formel gerechnet:

$$\Delta t = t_{st} - \frac{t_{st} - t_v}{e^{cx}}, \ ^{o}C$$

t_{st} - Gesteinetemperatur
t_{sv} - Lufttemperatur am Anfang des Luftweges
e - Basis natürliches Logarithmus
x - Länge der Grubenbau

$$c = \frac{k \times U}{c_p \times \gamma_v \times V}$$

k - Wärmedurchlässigkeit zwischen dem Gestein und der Luft
U - Rauminhalt
c_p - Spezif.Wärme der Masse bei konstantem Druck
γ_v - spezif.Luftdichte
V - Luftmenge

Tabelle 2 - Temperaturänderung der Grubenwetter durch Einwirkung der Grubenbaugestein

	T°C		
Monat	1-5	5-11	11-14
Jan.	5,286	4,479	- 7,177
Feb.	3,755	5,507	- 6,886
März	2,477	5,515	- 9,153
April	1,542	1,761	- 8,073
Mai	1,542	1,761	- 8,073
Juni	0,054	0,064	-11,256
Juli	- 1,762	- 0,145	-12,095
Aug.	- 3,192	0	-11,139
Sept.	0,132	0,536	- 8,867
Okt.	1,761	2,244	-11,321
Nov.	4,630	6,256	-13,223
Dez.	4,051	5,685	- 8,385
Durchscjmott:	1,683	2,882	- 9,644

Vom Eingang in die Grube (punkt 1) bis Punkt 11 geht die Wärme vom Gestein auf Luft über. Vom Punkt 11 bis zum Ausgang aus der Grube(Punkt 14) ist der Austausch umgekehrt.

Der Wärmeübergang vom Gestein auf die Luft ist grösser in kälteren Jahreszeiten, während er inwärmeren Jahreszeiten kleiner ist. Im Juli und August geht die Wärme von der Luft auf dem Gestein durch die ganze Grube über.

2.3 Einwirkung der chemischen Prozesse auf Temperatur der Grubenwetter

Die Wärme wird durch Kohlenoxidation, Holzoxidation usw.nach dem exothermen Entwicklungsprozess freigeworden.

Die Temperaturerhöhung der Grubenwetter durch Oxidationsprozess wird nach folgender Formel berechnet:

$$\Delta t = \frac{q}{c_p \times \gamma_v}, \ ^oC$$

q - Wärmemenge

Tabelle 3 - Temperaturänderung der Grubenwetter durch Einwirkung chemischer Prozesse

	T°C		
Monat	1-5	5-11	11-14
Jan.	1,466	15,491	1,542
Feb.	0	14,234	0
März	0	18,589	0
April	2,507	11,027	0
Mai	3,089	7,969	
Juni	0	10,727	
Juli	0	7,663	0
Aug.	0	8,086	0
Sept.	0,761	8,322	2,706
Okt.	2,708	18,771	4,387
Nov.	0	29,538	0
Dez.	0	18,713	0
Durchschnitt	0,878	14,094	0,720

Die höchste Temperaturerhöhung der Grubenwetter (14,094°C)erfolgte auf dem Luftweg zwischen Punkt 5 und 11.Auf diesem Luftwegteil befindet sich das Abbaurevier sowie das Fördersystem in der Länge vom 2000 m. Die Temperaturerhöhung in anderen Grubenteilen war nur um 1,598°C. Die Temperaturerhöhung durch Oxidation der Kohle ist höher in Winter - und Uebergangsperioden als in Sommermonaten.

2.4 Einwirkung der Feuchtigkeit auf die Grubenwetter

Durch Erhöhung des Wasserdampfgehalts wird die Temperatur der Grubenwetter herabgesetzt. Dieses prozess ist adiabatisch. Falls Wasserdampf unter der Einwirkungs der Gestein-oder Wasserwärme ausgeschieden wird, ist dann dieses Prozess unadiabatisch.

Aufgrund der Angaben über die Wärme der Verdunstung ist es möglich, festzustellen, in welchem Masse die Lufttemperatur geändert wird:

$$\Delta t = \frac{C}{\gamma_v \times c_p} (X_2 - X_1), \ ^oC$$

C - spezifische Wärme des Wasserdampfs

X_1 - apsolute Feuchtigkeit im Anfang des Luftweges

X_2 - apsolute Feuchtigkeit am Ende des Luftweges

Tabelle 4.-Temperaturänderung der Grubenwetter durch Feuchtigkeit

	T °C		
Monat	1-5	5-11	11-14
Jan.	-5,089	-16,047	4,874
Feb.	-3,415	-14,714	5,036
März	-3,560	-17,492	5,043
April	-4,849	-11,683	4,566
Mai	-4,105	- 9,105	4,644
Juni	-1,365	-11,079	7,536
Juli	-0,526	- 9,048	7,845

Aug.	0,372	- 9,511	6,592
Sept.	- 2,876	- 9,495	3,051
Okt.	- 4,233	-17,028	3,691
Nov.	- 3,862	-26,131	8,460
Dez.	- 3,217	-18,018	4,799
Durchschnitt:	-3,206	-14,161	5,481

Der Einfluss der Feuchtigkeit auf die Temperatur der Grubenwetter auf dem Luftweg von Punkt 1 bis 11 ist durch die Temperaturerniedrigung, Erhöhung relativer Feuchtigkeit, Erniedrigung apsoluter Feuchtigkeit und Erniedrigung des Wärmeinhalts gekennzeichnet.

Von Punkt 11 bis 14 ist die Luft gesättigt, ihre relative Feuchtigkeit ist gegen 100%, apsolute Feuchtigkeit wird erhöht und die Temperaturer und Wärmeinhalt steigen an.

2.5 Einfluss der Luftreibung gegen die Grubenbauwänden auf die Grubenwettertemperatur

Die Temperaturerhöhung der Grubenwetter durch Reibung wird nach folgender Formel berechnet:

$$\Delta t = \frac{q}{c_p \times \gamma_v \times V}, \ ^oC$$

Tabelle 5-Temperaturänderung der Grubenwetter durch Reibung gegen Grubenbauwänden

	t°C		
Monat	1-5	5-11	11-14
Jan.	0,1103	0,1634	0,5620
Feb.	0,1092	0,1515	0,5552
März	0,1096	0,1614	0,5570
April	0,1110	0,1630	0,5615
Mai	0,1107	0,1618	0,5536
Juni	0,1131	0,1642	0,5617
Juli	0,1140	0,1648	0,5619
Aug.	0,1136	0,1642	0,5580
Sept.	0,1133	0,1544	0,5624
Okt.	0,1209	0,1532	0,5631
Nov.	0,1089	0,1624	0,5645
Dez.	0,1094	0,1632	0,5569
Durchschnitt	0,1120	0,1630	0,5598

Die Reibung, als Einflussfaktor, ist nicht von besonderer Bedeutung für die gesamte Wärmebilanz der Grubenwetter.

Es ist charakteristich, dass die Temperatur durch Erhöhung der Grubenluftdepression erhöht wird, und durch Erniedrigung der Grubenluftdepression erniedrigt wird.

2.6 Einfluss der Elektrogeräte auf die Grubenwettertemperatur

$$\Delta t = \frac{q_e + q_{tr} + q_{ke} + k_{ktr}}{c_p \times \gamma_v \times V}, \ ^oC$$

q_e - Wärmemenge von Elektromotoren
q_{tr} - Warmemenge von Transformatoren
q_{ke} - Wärmemenge von Kabeln für Elektromotoren
q_{ktr} - Wärmemenge von Kabeln für Transformatoren

Tabelle 6-Temperaturänderung der Grubenwetter durch Einfluss von Elektrogeräten

	t°C		
Monat	1-5	5-11	11-14
Jan.	1,735	4,675	0,805
Feb.	1,705	4,570	0,785
März	1,705	4,656	0,795
April	1,705	4,590	0,795
Maj	1,730	4,505	0,795
Juni	1,685	4,555	0,815
Juli	1,765	4,655	0,830
Aug.	1,695	4,580	0,810
Sept.	1,660	4,680	0,845
Okt.	1,645	4,760	0,840
Nov.	1,715	4,970	0,850
Dez.	1,690	4,515	0,795
Durchschnitt:	1,703	4,635	0,813

Lufterhitzung durch den Einfluss der Elektromotoren ist in der Funktion der installierten Leistung der Elektrogeräte.

2.7 Einfluss von Beleuchtung auf die Grubenwettertemperatur

$$\Delta t = \frac{q_s \times n}{c_p \times \gamma_v \times V}, \ ^oC$$

Wärmemenge, die von einer Glühlampe freigeworden wird:

$q_s = 0,14 \times N_s \times \eta$

N_s - Glühlampestärke

n - Glühlampezahl

η - Grad der Glühlampeausnützung

Die Beleuchtung, als Wärmequelle in der Grube, wirkt nicht bedeutend auf die Temperaturänderung der Grubenwetter ein.

Tabelle 7 - Temperaturänderung der Grubenwetter durch Einfluss von Beleuchtung

	t °C		
Monat	1-5	5-11	11-14
Jan	0,000247	0,001048	0,000177
Feb.	0,000244	0,001025	0,000172
März.	0,000243	0,001024	0,000174
April	0,000242	0,010290	0,000176
Mai	0,000247	0,001021	0,000174
Juni	0,000240	0,001022	0,000179
Juli	0,000251	0,001044	0,000182
Aug.	0,000241	0,001027	0,000178
Sept.	0,000237	0,001051	0,000186
Okt.	0,000235	0,001068	0,000185
Nov.	0,000245	0,001115	0,000186
Dez.	0,000241	0,001013	0,000176
Durchschnitt:	0,000243	0,001040	0,000178

2.8 Einwirkung des Sprengens auf die Grubenwettertemperatur

Beim Gebrauch des Sprengstoffs wird die Wärme in der Grube frei. Die Wärmezunahme in den Grubenwttern wird nach folgender Formel berechnet:

$$\Delta t = \frac{(1-\varphi)\ q}{c_p \times \gamma_v \times V}, \ ^oC$$

$q = \delta \times i_e \times A$

δ = Koeffizient der Wärmeverluste beim Sprengen (0,6-0,7)

i_e - die von 1 kg Sprengstoff freigewordene Wärmemenge

A - Sprengstoffverbrauch

φ - Koeffizient(0,3-0,4) fuer Vorrichtungsort, (0,6-0,8) fur den Abbaustoss

Tabelle 8- Temperaturänderung der Grubenwetter durch Sprengen

	t°C		
Monat	1-5	5-11	11-14
Jan	-	0,2754	-
Feb	-	0,2473	-
Marz.	-	0,2563	-
April	-	0,2286	-
Mai	-	0,1649	-
Juni	-	0,1836	-
Juli	-	0,1534	-
Aug.	-	0,2336	-
Sept.	-	0,2319	-
Okt.	-	0,2126	-
Nov.	-	0,1927	-
Dez.	-	0,1927	-
Durchschnitt:	-	0,2152	-

2.9 Einfluss von dem Arbeitspersonal auf die Grubenwettertemperatur

Die von dem menschlichen Leib freugewordene Wärmemenge ist von der Schwere der Arbeit abhängig. Ais Durchschnittswärmemenge bei den Bergarbeiten nimmt man 1 050 kJ/ Stunde:

$$\Delta t = \frac{q \times n}{c_p \times \gamma_v \times V}, \ ^oC$$

q - die von dem menschlichen Leib bei den Bergarbeiten freigewordene Wärmemenge

n - Zahl der Arbeiter

Tabelle 9 - Temperaturänderung der Grubenwetter durch den Einfluss von dem Arbeitspersonal

	t °C		
Monat	1-5	5-11	11-14
Jan.	0,0964	1,5980	0,4610
Feb.	0,0951	1,5644	0,4482
Marz.	0,0947	1,5624	0,4545
Apr.	0,0944	1,5707	0,4579
Mai	0,9965	1,5428	0,4548
Juni	0,0938	1,5599	0,4682
Juli	0,0981	1,5931	0,4747
Aug.	0,0943	1,5681	0,4642
Sept.	0,0925	1,6019	0,4846
Okt.	0,0917	1,6292	0,4831
Nov.	0,0941	1,7014	0,4864
Dez.	0,0942	1,5457	0,4575
Durch-schnitt:	0,0946	1,5864	0,4663

3. DAS ERGEBNISS DER UNTERSUCHUNGEN DER EINFLUSSFAKTOREN AUF DIE TEMPERATUR DER GRUBENWETTER

Nach Untersuchung der Wärmebilanz der Grubenwetter kann man folgendes konstatieren:

- Die Temperatur der Grubenwetter erhöht sich von dem Eingang in die Grube bis zum Ausgang aus der Grube. Die Temperaturerhöhung ist in Wintermonaten höher als in Sommermonaten.

- Die Temperatur der Eingangswetter hängt von der atmosphärischen Temperatur ab und bewegte zwischen 1 und 17,8°C.

- Die Temperatur des Ausgangswetter zeigt viel kleinere Unterschiede und bewegt sich zwischen 16,6 und 21°C.

- Relative Luftfeuchtigkeit ist ziemlich hoch und bewegt sichzwischen 90 und 100 %.

Die Grube "II Dachschichten" des Braunkohlenbergwerks "Lipnica" wird mit 1052 m^3/Minute Luft mehanisch, depressiv mit ziemlich einfacher Wetterverteilung bewettert. Die Länge des Windwegs beträgt 3800 m.

Die Untersuchungen, d.h. das Messen aller Parameter, Bestimmen der Einflussfaktoren auf die Grubenwettertemperatur und Errechnung der Ergebnisse, wurden auf vier charakteristiche Punkte vorgenommen, u.z. 1 und 14, die die ganze Grube umfasst und 5 und 11, die die Förderräume und Abbauorte umfassen.

Neun faktoren, die in der Grube "II Dachschichten" Einfluss auf die Temperaturänderung der Grubenwetter ausüben, wurden antersucht.

Der Einfluss von der Grubenwetterreibung gegen Räumewänder, dann Einfluss von Luftverdichtung und verdünnung, sowie Wärmeerhöhung durch Sprengen und das Arbeitspersonal sind ziemlich klein, während Temperaturerhöhung durch Grubenbeleuchtung ist sehr klein und unbedeutend.

Das Gestein der Grubenbaue, chemische Prozesse, Luftfeuchtigkeit und Elektrogeräte tauschen am hohsten die Wärme mit der Grubenwetter aus.

Einige Faktoren erniedrigen und die anderen erhöhen die Grubenwettertemperatur, u.z. verschiedentlich in eizelnen Grubenteilen und Jahreszeiten. Die Gesamtbilanz zeigt, dass die Temperatur von 9,4°C am Eingang in die Grube bis Punkt 5 um 3,366°C, dann bis Punkt 11 um weitere 8,374°C erhöht wird damit sie bis Punkt 14 um 2,552 °C erniedrigt wird.

Aus den Untersuchungen der Wärmebilanz der Grubenwetter in der in dieser Arbeit untersuchten Grube "II Dachschichten" des Kohlenbergswerks "Lipnica" kann man folgendes schliessen:

- Die Tiefen der Braunkohlengruben in dem Kohlenrevier "Kreka"stellen, was Erhöhung bzw. Erniedrigung der Grubenwetterdichte betrifft, keinen bedeutenden Faktor dar, der auf die Temperaturänderung der Grubenwetter einfliesst. Es wurde berechnet, dass der Thermalgradient (m/°C) 102 m beträgt. Die Gruben sind bis 400 m tief.

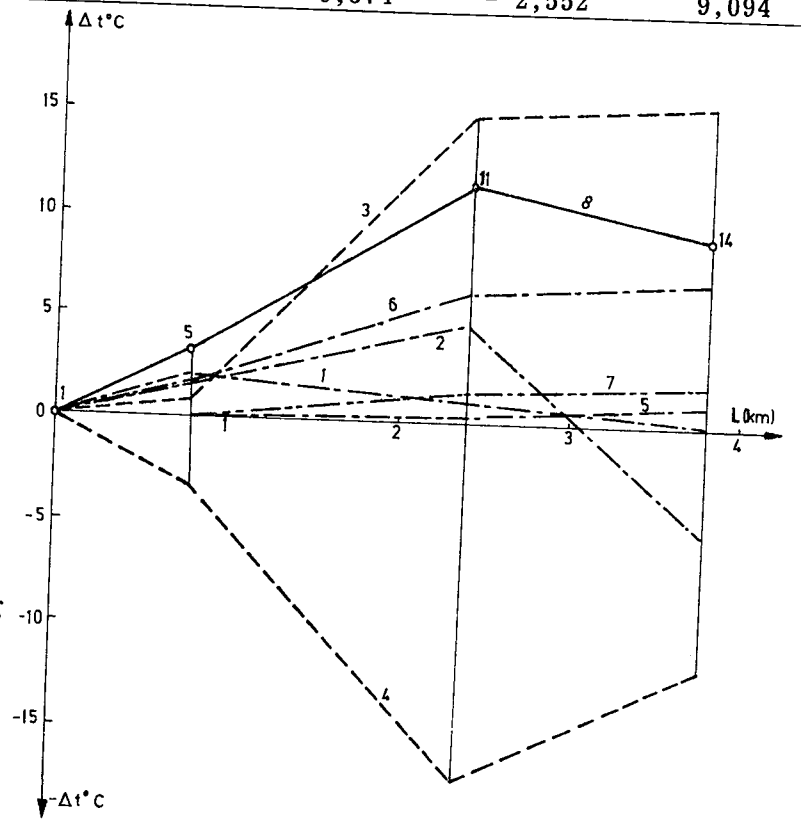
- Die Gesteinetemperatur wirkt im grösseren Masse auf die Abkuhlung der Grubenwetter ein, besonders auf dem Luftwegteil, wo die Einwirkung der Atmosphäre reduziert ist.
- Kohlenoxidation erhöht die Grubenwettertemperatur, besonders im Grubenteil, wo Abbauorte und Fordersystem sind.

Die durch Oxidation verursachte Temperaturerhöhung ist von der Berührung der Luftfläche mit der Kohle abhängig.

- Die Feuchtigkeitserhohung erniedrigt die Grubenwettertemperatur, bis die Grubenwetter gesattigt werden, dannach erhoht sich die Lufttemperatur.

Tabelle 10 - Uebersicht der Durchschnittswerte der im Laufe eines Jahres gewonnenen Ergebnisse

Luftwegteil	1 - 5	5 - 11	11 - 14	1 - 14
Gemessene Lufttemperaturen	9,4 - 13,0	13,0-21,4	21,4 - 18,4	9,4 - 18,4
Verdichtung-Verdünnung	2,101	- 1,050	- 0,957	0
Gestein	1,683	2,882	- 9,644	-5,069
Chemische Prozesse	0,878	14,094	0,720	15,692
Feuchtigkeit	- 3,206	- 14,161	5,481	-11,886
Luftreibung	0,112	0,163	0,560	0,835
Elektrogerate	1,703	4,635	0,813	7,151
Beleuchtung	0,000243	0,001040	0,000278	0,001461
Sprengen	-	-	-	0,2152
Arbeitspersonal	0,0946	1,5864	0,4663	2,1473
I n s g e s a m t:	3,366	8,374	- 2,552	9,094



Legende:
1. Verdichtung. Verdünnung
2. Gestein
3. Chemische Prozesse
4. Feuchtigkeit
5. Reibung
6. Elektrogeräte
7. Arbeitspersonal
8. Resultante

Beld 2 - Graphische Darstellung der Einflussfaktoren auf Grubenwettertemperatur

- Durch Modernisierung der Technologije der Kohlengewinnung wird Elektroausrüstung in den Gruben immer mehr eingeführt, so dass der Einfluss dieses Faktors auf die Temperaturerhöhung immer grösser wird.

Die Grubenwettertemperatur ist einer der wichtigsten Faktoren des Grubenklimas. Ihr Einfluss ist von vielen Elementen abhängig, wie z.B.

- Art und Menge der Erzgewinnung und Arbeitsbedigungen,
- Grösse und Tiefe der Grube
- Bewetterungsart (Länge des Luftweges, Luftmenge, Bewetterungsnetz, usw.)
- Mechanisierung der Grube

In vielen grösseren Kohlengruben wird grosse Aufmerksamkeit darauf gerichtet, dass die Temperatur am Arbeitsplatz in der Grube auf das normale Mass zurückgeführt wird, so dass z.b.verschiedene Lösungen fur Luftabkühlung in tiefen Gruben Anwendung finden.

Obwohl die Temperaturen der Grubenwetter an den Abbauorten der Braunkohlengruben gegen 21°C sind, kann man durch Bewetterungsystem noch günstigere Grubentemperaturen erzielen.

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HEAT BALANCE OF MINE ATMOSPHERE IN LIGNITE MINE KREKA

H. Tanovic

The temperature of the mine atmosphere is one of the most important factors in mine air conditioning. A number of factors affect the temperature of the mine atmosphere, the most important being: humidity, density and dilution of the air; electrical equipment; blasting; rock; chemical processes; friction of air on the mine surfaces; mine illumination; and mine personnel. The investigation of the individual factors acting on the temperatures of the mine atmosphere showed that some lower the temperature, whereas others raise it, in different ways depending on the area and time of year. Chemical processes, humidity of the air, electrical equipment and rock are the factors that have the greatest effect on the temperature of the atmosphere in the lignite mine.

STABILISATION DE LA CHALEUR DE L'ATMOSPHERE MINIERE DE LA MINE DE LIGNITE DE KREKA

H. Tanovic

La température de l'atmosphère minière est l'un des principaux éléments de la climatisation de l'air minier. En effet, un certain nombre de facteurs modifient la température de l'atmosphère minière; les plus importants sont les suivants : humidité, densité et dilution de l'air, équipement électrique, tirages, roches, processus chimiques, friction de l'air à la surface des mines, éclairage des mines et personnel minier. L'examen des facteurs individuels qui ont une incidence sur la température de l'atmosphère minière a permis de démontrer que certains d'entre eux abaissent la température tandis que d'autres l'élèvent, ceci de différentes manières, selon la région et l'époque de l'année. Les facteurs des processus chimiques, de l'humidité de l'air, de l'équipement électrique et des roches sont ceux dont l'impact est le plus important sur la température de l'atmosphère des mines de lignite.

ТЕПЛОВОЕ РАВНОВЕСИЕ В АТМОСФЕРЕ ШАХТЫ НА ПРИМЕРЕ ЛИГНИТОВОЙ ШАХТЫ КРЕКА

Х. Танович

Температура воздуха в шахте - один из важнейших факторов, влияющих на кондиционирование воздуха подземных горных выработок. На температуру воздуха в шахте влияют многие факторы. Важнейшие из них - это влажность, плотность и разреженность воздуха, электрическое оборудование, взрывные работы, состав пород, химические процессы, трение воздуха о шахтные поверхности, освещение шахты и состав сотрудников шахты. Исследование индивидуальных факторов, влияющих на температуру воздуха в шахте, показывает, что некоторые из них понижают температуру, в то время, как другие повышают ее в различной степени в зависимости от района и времени года. Химические процессы, влажность воздуха, электрооборудование и состав коренных пород являются факторами, оказывающими наибольшее влияние на температуру в лигнитовой шахте.

克瑞卡褐色煤矿内大气热力平衡

H。塔诺维奇

矿内大气温度为矿井空调中最重要的因素．影响矿内大气温度的因素很多，最具重要性的如下：空气湿度，密度及稀释度；电力设备；爆破；岩石；化学过程；空气在矿表面的磨擦；矿井的照明；及矿井人员．这些因素对矿内大气温度有的能使它增高，有的则使它减低．作用情形因地点和时间而异．其中化学过程，空气湿度，电力设备及岩石情形对褐色煤矿内大气的温度最有影响．

Session VI

RESCUE AND RECOVERY

RESQPACS: THE ROLE OF THE CHAMBER OF MINES RESEARCH ORGANIZATION IN THE IMPLEMENTATION OF SELF RESCUERS IN SOUTH AFRICAN COLLIERIES

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ABSTRACT

The Research Organization of the Chamber of Mines of South Africa played a central role both in the search for a suitable self rescuer for use in South African collieries and in assisting the industry with their implementation.

Feasibility studies established belt-worn self contained self rescuers (ResQpacs) as the most suitable for use in local collieries. Extensive field and laboratory tests were then conducted to ascertain the practical viability of the ResQpac concept, identify infrastructural requirements and evaluate the products developed for South African requirements.

The Research Organization maintained a supportive role throughout the implementation process, assisting mines to resolve the problems that arose in areas such as training, surface infrastructure, and the establishment of integrated escape systems.

This paper presents a review of the various developments associated with the introduction of ResQpacs into South African collieries with particular emphasis on the role of the Chamber of Mines.

INTRODUCTION

Continuing efforts to minimize the loss of life due to fires and explosions in South African collieries led the coal mining industry to reassess the measures needed to prevent such occurrences and to protect workers.

Although the best form of protection for the underground workforce would be prevention of all fires and explosions this philosophy can never be totally effective and wherever possible additional measures should be taken to ensure the minimum loss of life in the event of an underground fire or explosion.

The Coal Mining Laboratory of the Chamber of Mines Research Organization commenced studies into the use of self rescuers in 1982. These studies involved assessing both the requirements of the local industry and the effectiveness of the self rescuer systems in use elsewhere in the world.

By 1984 it had become apparent that the requirements of the local coal mining industry would best be served by the development of a totally new product embodying the best features of the self contained and filter units in use overseas.This led to the development of a short duration, personally worn, self contained self rescuer, now referred to by the generic name ResQpac.

These devices would provide the primary protection in an overall escape strategy designed to move personnel safely from their workplaces to the surface. In most instances this strategy would involve secondary and tertiary elements, incorporating features such as specially designed escape routes, refuge bays and the mine's rescue teams.

It should be emphasized that the ResQpac is not, and never was, intended to be a stand alone device. An escape strategy based upon the ResQpac will, by definition, involve several stages of which only the first will be the protection of escapees by the ResQpac.

The industry moved quickly towards acceptance of this concept and the Chamber of Mines remained involved through to the final stages of implementation. It was necessary to firstly set a specification for the type of device that was required, then monitor development to ensure compliance with the specification and where necessary modify the specification as new requirements became apparent. It was also necessary to conduct extensive studies on the mines to determine all infrastructural requirements, particularly with regard to the secondary and tertiary elements of escape strategies, training and surface storage and distribution facilities.

More recently, the Research Organization has been monitoring and assisting the industry to overcome the problems following implementation in areas such as the support offered by suppliers, post implementation training and the withdrawal of samples for performance checks.

The objective of this paper is to review the contribution of the Research Organization of the Chamber of Mines to the implementation of ResQpacs on South African collieries and through this, to illustrate how many of the major obstacles to implementation were overcome.

FEASIBILITY STUDIES

Studies into the feasibility of introducing self rescuers into South African collieries began in 1982 with a review of the systems in use overseas.

Two systems were identified:

i) filter self rescuers (FSR's) - used widely throughout the world and the only device considered viable until the USA began introducing self contained self rescuers in the early 1980's

ii) self contained self rescuers (SCSR's) - introduced into USA collieries from 1980 onwards.

Despite the general satisfaction with FSR's, it was felt that most South African collieries, which are predominantly bord and pillar workings, were faced with problems that favoured SCSR's. For instance , in bord and pillar workings many workers are sheltered by the pillars from the direct effects of the explosion and there is also a tendency for most workers to be in blind headings. A common feature of past methane ignitions was therefore a high fatality rate amongst workers who survived methane explosions but were then overcome as a result of being cut off from safety by accumulations of afterdamp with oxygen levels below those needed to sustain life using FSR's

The self contained self rescuers in use in the USA would have overcome this type of problem but were considered unsatisfactory because of their size and weight. The units were all too large to be kept on the person at all times. Consequently, most of these devices would be deposited at some convenient location upon reaching the work-place.

It was felt that the combined effects of lack of visibility, disorientation and the tendency to panic, in the event of a methane explosion, would result in the SCSRs not being found quickly enough to save a survivors life or even being ignored in favour of making all possible haste towards the shaft. A device carried continually on the person was essential.

Consideration was therefore given to the ResQpac concept: a SCSR small enough to be carried on the person at all times but of sufficient duration to enable a person to reach safety.

Consultation with manufacturers led to the belief that the longest life-support duration possible for a device small enough to be carried on the person would be approximately 30 minutes (for a breathing rate of about 30 l/min). Although this was clearly not sufficient to enable people to escape to surface in most cases, it could provide sufficient protection to enable survivors to proceed to some point at which a long duration air supply would be provided. This supply could be a longer duration SCSR or refuge bay.

CASE HISTORY STUDIES

The next phase of the research involved assessing the life saving potential of the different types of self rescuer by studying past fires and explosions in South African collieries.

Six cases involving 113 fatalities were reviewed, as being typical examples of different types of incident. The analyses were performed in terms of two different scenarios. Firstly, maximum life saving potential - which assumed all survivors, excluding the severely injured but including those with minor or moderate injuries, were able to don self rescuers carried on the person. Secondly, the least likely survival rate which assumed various percentage survival chances for different degrees of injury.

The findings were as follows:

Self Rescuer Type	Maximum Life Saving Potential	Least Likely Survival Rate
FSR	21	1
SCSR	21	21*
ResQpac	51	34

* It was evident in the analysis involving SCSR's that, if training had been carried out effectively, some workers who survived the real event by good fortune, quick thinking or by whatever means, would have been killed as a result of returning to, or remaining in, critically contaminated areas in order to locate their SCSR's. The number of fatalities that would have arisen in this way could not be quantified and so the figure shown here ignores this factor. The relevance of this factor has however been confirmed by an incident that took place following this study.

Given that 68 of the 113 fatalities occurred in one incident where self rescuers of any type would have been of little use, these findings represented a strong argument in favour of the ResQpac concept.

PRE-IMPLEMENTATION PROCEDURES

A working party comprising personnel from the various mining groups as well as several branches and laboratories of the Chamber of Mines and its Research Organization, was formed to examine all aspects of the protection of underground workers in collieries. This working party was asked inter alia, to formulate a specification covering the basic characteristics for self rescue devices to be used in South African mines. The purpose of this specification was to try to ensure

that the best use was made of existing technology and knowhow in developing products for local use.

In addition to the requirements indicated by proposed legislation, namely that the device had to have a life support duration of 30 min. at a breathing rate defined as 30 l/min, the industry specification dealt with durations at other breathing rates, carbon dioxide build up, breathing resistance, ergonomics, the donning time, maintenance, and shelf life.

As manufacturers began developing prototypes in accordance with the specification, the Research Organization acquired a new role. Although the ResQpacs used proven operating principles, no information existed regarding their durability in the belt-worn configuration or the degree to which the devices might interfere with the worker's ability to carry out his tasks or cause discomfort. There was also an urgent need to fully assess the infrastructural requirements associated with the implementation of ResQpacs.

The Research Organization proposed a series of field trials that would enable the new products to be evaluated, the viability of the concept to be established, and information on infrastructural requirements to be obtained. Later, the field trial procedure became the basis for approving devices for underground use.

It was over two years before sufficient information was obtained to enable the introduction of ResQpacs onto South African collieries to proceed. However, the significant benefits of the field trials were ample justification for the delays.

Many products were found to be unacceptable and in some cases dangerous. Important factors not included in the specification were identified such as the potential for high temperature external components to burn users and the need for extremely simple donning procedures.

Training of employees was identified as a major problem (Hardman 1987) and the Chamber of Mines prepared guidelines for both suppliers and the mines, as well as becoming involved in the production of preliminary training material and assessing the material from suppliers for effectiveness and consistency. It was at this point that the ResQpac generic and associated logo were defined. This was considered crucial in an industry where a migratory workforce would otherwise be confronted by numerous company logos and product names for different devices provided for the same function.

Regarding the underground infrastructure, it was not possible to define single solutions due to the diverse conditions in different mines. A working party was however set up and a methodology devised for the establishment of a suitable and comprehensive infrastructure. The areas covered included:-

i) Hazard detection: for the full life saving potential of self rescuers to be realized it is necessary to identify any hazard timeously. Both central monitoring systems and personal toxic gas monitors were considered. The interpretation of readings, the alarm levels for the different types of device and the location of sensors for central monitoring systems, are all crucial to both survival and the prevention of false alarms.

ii) Communications: effective alarms to notify workers of the need to don their ResQpacs are also vital. These must be unequivocal cues to cease all other activity, don a ResQpac and escape.

iii) Escape routes: means must be provided for identifying escape routes and procedures defined for

using them to reach the designated safe places.

iv) Back up facilities: refuge bays or some alternative means of providing a guaranteed supply of breathable air must be sited at points that can be reached by survivors using ResQpacs. The design of these facilities and their integration into comprehensive rescue strategies were studied. A test protocol was devised (Van Rensburg 1987) (Kielblock 1987) for the assessment of escape routes in terms of breathing rates so that safe locations for back-up facilities could be determined. Similarly, means of evaluating the life-support capabilities of refuge bays were established and tests conducted on several designs.

Surface requirements were also assessed. Means of storing, distributing, collecting and maintaining ResQpacs were reviewed.

IMPLEMENTATION ON COLLIERIES

Legislation was gazetted on 31 October 1986 and the implementation process commenced on collieries early in 1987.

Three ResQpacs were approved for use in underground collieries. These were :-

(a) The MSA/Auer SSR 30/100 - oxygen generated chemically using potassium peroxide - stainless steel outer casing - donned unit worn on the chest.

(b) The Draeger Oxybooks K - oxygen generated chemically using potassium peroxide - plastic casing with stainless steel lid and reinforcement - chlorate candle starter - donned unit worn on head.

(c) The Fenzy Spiral 1 - oxygen generated chemically using potassium peroxide - plastic casing - compressed oxygen starter - unit (excluding lid) remains on belt when in use.

In general three types of "back-up" facility were favoured for use in local collieries:-

i) Refuge bays were generally accepted to be the simplest, most effective and most economical form of back-up. These would be created either by walling up an area between coal pillars or by cutting a room in a rib pillar. The relatively shallow depths of many workings and ease of access to the surface above, meant that these refuge bays could be supplied with fresh air via boreholes from the surface. These boreholes could also be used to supply fresh water and food, and for communication. The normal pressure differential between surface and the workings would ensure a continual flow of air through the chamber when not in use, and fans could be used to provide the necessary airflow in the event of the disruption of ventilation that often accompanies methane explosions. These fans could either be fixed within the refuge bay, fixed at the surface end of the borehole or portable units to be moved to the surface end of the borehole as required.

ii) An alternative to refuge bays supplied with fresh air via boreholes is to ensure a sufficient volume of air within the refuge bay to support life until all occupants can be moved to surface. Tests were conducted on a refuge bay of this type in order to determine likely oxygen depletion rates and carbon dioxide accumulation rates, and it was found that the concept is viable (Kielblock 1988). However, it clearly depends upon the competence of the rest of the escape strategy in that there must be absolute certainty that the final rescue phase can be effected within the

life support duration of the refuge bay. The chamber must also be protected against the ingress of contamination during entry by survivors and by natural leakage. This type of facility may however be the best option where boreholes from the surface are not possible or practicable.

iii) The third alternative is to use long duration SCSR's to supplement the protection afforded by the ResQpac. The ResQpac would be used to protect the survivor until he is able to reach a cache of long duration SCSR's. He would then use the SCSR as protection in order to make his way to the surface. This arrangement will, however, require the provision of some means of protecting escapees from toxic fumes during the change over from the primary device to the secondary unit. This concept does have the major advantage of ensuring that all survivors make their way to the surface rather than barricading themselves underground.
A modification of this arrangement is the so called "piggy-back" system which would involve plugging a supplementary chemical cannister onto the ResQpac, thereby overcoming the need to protect escapees during the switch over from the ResQpac to the secondary device. However no designs currently exist for such an arrangement for use in conjunction with any of the currently approved ResQpacs.

The design of escape routes varies considerably from mine to mine. Currently no single arrangement can be specified as being universally ideal. However, some interesting features have been conceived such as:-

(a) steel ropes attached to walls to enable escapees to identify and follow escape routes in limited visibility.

(b) stroboscopic lights and sirens, activated by the pressure waves resulting from explosions, to help escapees to find escape routes and refuge bays or SCSR caches.

(c) reorientation procedures to enable survivors to locate an escape route in bord and pillar workings during limited visibility, by following pillars in a specific set direction until some prescribed "landmark" is found from which an escape route can be defined.

POST-IMPLEMENTATION PROCEDURES

Having developed research techniques for the evaluation and study of the various aspects associated with the introduction of ResQpacs, the Research Organization continues to serve the industry by utilizing these techniques for the resolution of unique problems on individual mines.

A ResQpac User's Committee was established to provide a forum for discussing problems and exchanging ideas. Where necessary the Research Organization would assist mines with problems pertaining to any of the products or suppliers.

Several problems were addressed through this committee, which although apparently minor, could have had serious repercussions on the productivity of the mines or the successful implementation of ResQpacs.

* Inappropriate arrangements for the distribution and collection of ResQpacs could lead to the disruption of the flow of traffic to and from the workplace. Visits to different mines to view lamprooms and manpower deployment areas gave an insight into various alternatives that could be utilized to eliminate bottlenecks and other problems.

* Belt designs were found to be

consistently inadequate, and the choices of materials unsuitable. This led to discomfort and in some cases minor injury, which in turn brought about a negative attitude in wearers. Suppliers were advised of these problems and alternative designs reviewed.

* Suppliers were initially poorly equipped to deal with the problems associated with the implementation of ResQpacs and many complaints were voiced about the lack of after sales service. Follow up meetings were held with those concerned and better service procured.

* One supplier consistently failed to produce acceptable training material and training programmes were assembled using components from other sources under guidance from the Research Organization.

* The training sets to be used for hands-on practice of donning were all adaptations of the original designs and as such were often not capable of withstanding the repeated opening and repacking for training. Suppliers were asked to modify designs and ensure an adequate supply of replacements for those components prone to failure.

* Great concern was expressed about ensuring adequate sterilization of training sets to prevent the possible transmission of contagious diseases, in particular AIDS. The Research Organization, in consultation with self-rescuer and chemical manufacturers, drew up guidelines in this regard and where possible training programmes were modified to facilitate the use of disposable components or components that could more readily be thoroughly sterilized.

CONCLUSIONS

The processes of introducing self rescuers into South African collieries demonstrate clearly the importance of the various roles of a research organization. It is not sufficient merely to identify a theoretical solution to a problem through research. The organization concerned must continue to be involved at every stage of implementation to ensure that its findings are viable in practice and to be on hand to resolve problems as they are identified during the implementation process.

The Research Organization of the Chamber of Mines identified ResQpacs as the best means of protecting underground workers against the aftermaths of fires and explosions in coal mines. It then went on to identify all of the requirements for the associated infrastructure, where necessary assisting the industry to overcome problems with the planning and design of various elements. Aspects covered included equipment performance, escape route designs, the design and evaluation of back up facilities, underground hazard detection and communication, surface storage and distribution facilities, training and dealing with suppliers with regard to common problems with after sales service.

Without the involvement of the Chamber of Mines Research Organization it is likely that the introduction of self rescuers into South African collieries as part of integrated escape strategies, would have been a very protracted and more difficult process.

ACKNOWLEDGEMENT

This paper is based on work undertaken within the research programme of the Chamber of Mines Research Organization on behalf of the coal mining members.

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RESQPACS: ROLE DE L'ORGANISATION DE RECHERCHES DE LA CHAMBRE DES MINES EN CE QUI CONCERNE LA MISE EN PLACE D'APPAREILS DE SAUVETAGE AUTONOMES DANS LES HOUILLERES SUD-AFRICAINES

D.R. Pardoe

L'Organisation de Recherches de la Chambre des Mines d'Afrique du Sud a joué un rôle central, à la fois dans la recherche d'un appareil d'auto-sauvetage adéquat destiné aux houillères d'Afrique du Sud et par son appui à l'industrie en ce qui concerne la mise en place de ces appareils.

Sur la base des études de faisabilité effectuées, on a établi que les appareils autonomes d'auto-sauvetage portés à la ceinture (ResQpacs) sont les mieux adaptés à l'utilisation dans les houillères locales. Des essais approfondis ont ensuite été effectués sur le terrain et en laboratoire afin de s'assurer par la pratique de la fiabilité du concept des ResQpacs, afin de déterminer les exigences d'infrastructures et d'évaluer les produits développés pour les besoins de l'Afrique du Sud.

L'Organisation de Recherches a joué un rôle de support pendant tout le processus de mise en place en aidant à résoudre les problèmes qui ont surgi dans les domaines de la formation, de l'infrastructure de surface et dans l'établissement de systèmes de sauvetage intégrés.

Notre communication passe en revue les divers développements relatifs à l'introduction des ResQpacs dans les houillères d'Afrique du Sud, en insistant particulièrement sur le rôle de la Chambre des Mines.

RESQPACS: DIE ROLLE DER 'CHAMBER OF MINES RESEARCH' ORGANISATION BEI DER EINFÜHRUNG VON SELBSTRETTERN IN SÜDAFRIKANISCHEN GRUBEN.

D. R. Pardoe

Die Forschungsorganisation der 'Chamber of Mines of South Africa' spielte eine zentrale Rolle in der Suche nach einem brauchbaren Selbstretter für den Gebrauch in südafrikanischen Zechen, sowie in der Unterstützung der Industrie bei der Indienststellung dieser Geräte.

Durchführbarkeitsstudien zeigten, daß am Lampenriemen getragene Selbstretter (ResQpacs) sich am besten für den Gebrauch in den lokalen Gruben eignen. Danach dienten ausführliche Feld- und Labortests dazu, die praktische Anwendbarkeit des ResQpac-Konzeptes zu untersuchen, Anforderungen an ihre Infrastruktur zu bestimmen und die für Südafrika bereits entwickelten Erzeugnisse zu beurteilen.

Die Forschungsorganisation übte während des gesamten Einführungsprozesses eine unterstützende Rolle aus, indem sie den Minen bei der Lösung von Problemen wie Training, Tageseinrichtungen und Erstellung von umfassenden Fluchtsystemen zur Seite stand.

Dieser Vortrag stellt eine Zusammenfassung der verschiedenen Entwicklungen im Zusammenhang mit der Einführung der ResQpacs in südafrikanischen Gruben dar, und zwar mit besonderem Schwerpunkt auf dem Einfluß der 'Chamber of Mines'.

РОЛЬ ИССЛЕДОВАТЕЛЬСКОГО ОТДЕЛА ГОРНОЙ ПАЛАТЫ ВО ВНЕДРЕНИИ "САМОСПАСАТЕЛЕЙ" НА ЮЖНО-АФРИКАНСКИХ УГОЛЬНЫХ ШАХТАХ

Д.Р. Пардо

Исследовательский отдел Горной Палаты Южной Африки сыграл центральную роль в нахождении подходящего самоспасателя для применения в Южно-Африканских шахтах, а также оказал большую помощь угольной промышленности в его внедрении. Предварительные исследования выявили, что прикрепленный к поясу автономный самоспасатель является наиболее подходящим для местных условий. Затем были проведены интенсивные полевые и лабораторные испытания с целью выяснения практической жизнеспособности самой идеи самоспасателей, определения структурных требований и оценки самоспасателей, созданных специально для местных условий. Исследовательский отдел оказывал постоянную поддержку промышленности в ходе внедрения самоспасателей, помогал разрешать проблемы, возникшие при обучении и установке вспомогательных сооружений на поверхности и сложных спасательных путей. Доклад представляет собой обзор различных путей внедрения самоспасателей в Южно-Африканских шахтах; подчеркивается особая роль Исследовательского Отдела Горной Палаты в этом процессе.

ResQpacs：南非煤矿中煤矿部研究所供给自救器

D。R。帕尔多

南非煤矿部研究所着力于选出一种适当的自救器，并且帮助煤矿工业配备应用这种自救器。对于这种束带式设备齐全的自救器（ResQpacs）我们作了试用实验，认为是最适当的。然后经过非常广范的实验室和实地的试验，以检定这种自救器的实际可行性，认定它基本的要求，和判断它对南非煤矿情形是否适合。这研究所对配备这种自救器和培训，地面连系及建立一套整个逃脱系统给与很多支援。本文将述将ResQpacs应用于南非煤矿的详细过程，特别强调煤矿部的协助立场。

SELF-CONTAINED SELF-RESCUERS: NEW DEVELOPMENTS IN TECHNOLOGY AND TRAINING

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ABSTRACT

One purpose of this paper is to discuss new developments in Self-Contained Self-Rescuer (SCSR) technology. It appears to be technically possible to produce an SCSR that is approximately twice the size and weight of a Filter Self-Rescuer (FSR) and that has a rated duration of 60-minutes. Such an apparatus can be worn on a miner's body, making it immediately available in the event of a mine emergency. The other purpose of this paper is to discuss the development of an innovative SCSR donning method that has been shown to cut SCSR donning time in half, and to significantly reduce errors made by trainees. Because the new method consolidates the task of donning an SCSR into 3 critical steps and 3 secondary steps, it is called the "3+3" method. The 3+3 method is expected to play a significant role in training miners to use the new apparatus.

INTRODUCTION

When a mine disaster occurs, the basic survival technique for a miner is to escape from the mine. After a mine fire or explosion, the atmosphere inside the mine may become oxygen deficient or filled with smoke and toxic gasses. Under these circumstances, escape is nearly impossible unless a miner is equipped with a self-rescue device that supplies oxygen without the need for breathing mine air.

Federal regulations (30 CFR 75.1714) require that every person who goes into an underground coal mine in the United States must be supplied with a Self-Contained Self-Rescuer (SCSR). An SCSR is an emergency breathing apparatus designed for the purpose of mine escape. It must be capable of providing at least a 60-minute supply of oxygen, regardless of the condition of the mine atmosphere. Only SCSR's approved by the National Institute for Occupational Safety and Health (NIOSH) and the Mine Safety and Health Administration (MSHA) can meet the provisions of the regulations.

SCSR's have been deployed operationally in U. S. coal mines since 1981. During that time, two practical problems have emerged. First, because of the large size and weight of the current SCSR's, in most cases miners and mine operators have elected either to store or carry and store SCSR's in daily operational use, rather than wear them as personal protective equipment. In the event of a mine emergency, SCSR's deployed in either of these two ways would not be immediately available to an escaping miner. The other problem deals with training in the use of an SCSR. The ability to put on an SCSR in the event of a mine emergency is one of the most critical and non-routine of all mine health and safety skills. However, donning an SCSR is not an easy task. The purpose of this paper is to discuss new developments in SCSR technology and training that addresses these problems.

PERSON-WEARABLE, SELF-CONTAINED SELF-RESCUER TECHNOLOGY

The Bureau, through contract and in-house research, has refined SCSR technology to the point where it is now possible to provide miners with apparatus small enough to be worn on the body with sufficient oxygen for escaping miners to reach fresh air. These "second generation" SCSR's are called Person-Wearable, Self-

Contained Self-Rescuers (PWSCSR's). (Sometimes these devices are referred to as "belt-wearable.")

The Bureau, in 1986, awarded a research contract to CSE Corporation (U. S. Bureau of Mines Contract H0368013) to develop a prototype unit called the SRX-80 (Self-Rescuer Experimental - 80 Liters of Usable Oxygen). This contract was based on technical recommendations of the joint Bureau of Mines/Mine Safety and Health Administration/National Institute for Occupational Safety and Health PWSCSR task force (Kravitz and Kovac 1985) and had the approval of the mining industry.

The SRX-80 uses a solid chemical bed (a mixture of potassium superoxide and lithium hydroxide, to generate oxygen and to absorb carbon dioxide) and is equipped with a small oxygen bottle that serves as a quick-start mechanism. The SRX-80 is approximately 70 percent smaller by volume and 40 percent lighter than the average SCSR currently in use. In practical terms, the SRX-80 is about twice the size and weight of a filter self-rescuer (FSR). The size and weight limits were determined during discussions with the Bituminous Coal Operators Association and the United Mine Workers of America. Table 1 compares the size and weight of an FSR, PWSCSR, and average SCSR.

Table 1.

	FSR	SCSR	PWSCSR
Wt	2.5 lbs	8.97 lbs	4.86 lbs
Vol	75.4 in^3	469.25 in^3	130.00 in^3

The research is being conducted in two phases. Phase 1 has been successfully completed. The contractor fabricated ten prototype SRX-80's. The Bureau evaluated the life support performance of the prototypes. Test results indicated that the prototypes met or exceeded all life support performance requirements of the contract and marginally passed the performance requirements of 30 CFR 11 regulations as a 60-minute duration SCSR.

Because of the technical promise of the SRX-80, as well as the interest shown by MSHA and the coal mining industry in deploying practical PWSCSR's, the Bureau has initiated Phase 2. The objectives of Phase 2 are: to obtain MSHA/NIOSH certification of the PWSCSR prototype as a 60-minute duration SCSR, and to fabricate 200 production-quality, approved units to be subjected to an in-mine safety hazard evaluation by the Bureau. Research has been conducted successfully to increase the usable oxygen generated by the SRX-80 by 25 percent, from 80 to 100 liters of oxygen. The new unit is designated the SR-100. Figure 1 contains photographs of the SR-100 comparing it to an FSR, and showing it deployed and ready for donning.

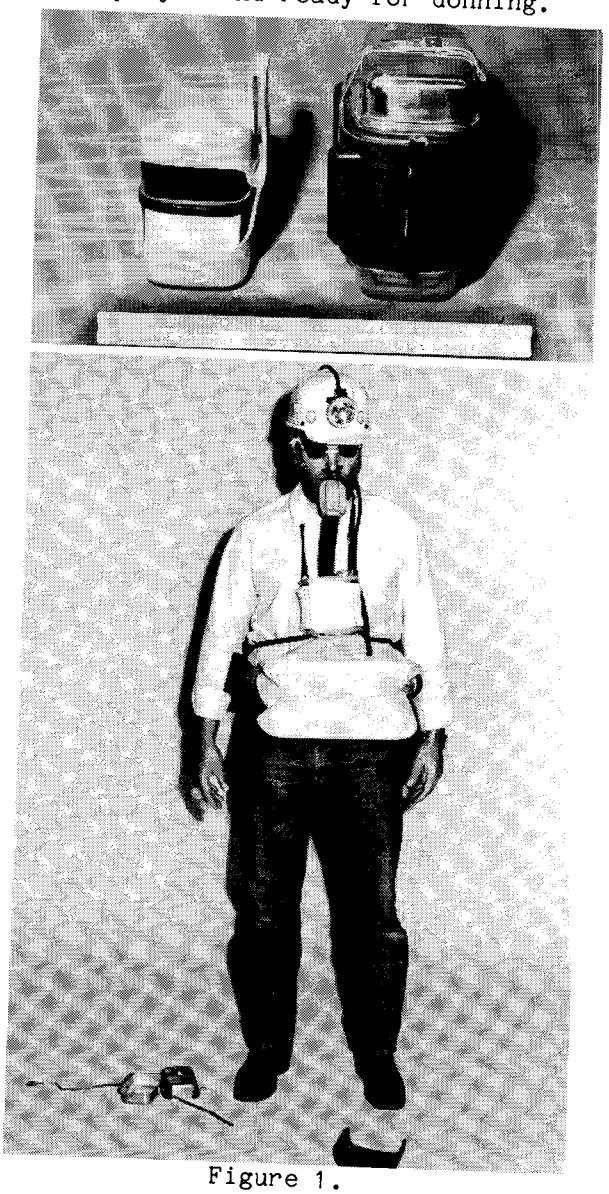


Figure 1.

Figure 2 is an engineering drawing of the SR-100. The SR-100 is being certified under current 30 CFR 11 regulations as a 60-minute duration SCSR. The modifications do not affect the size or weight of the PWSCSR.

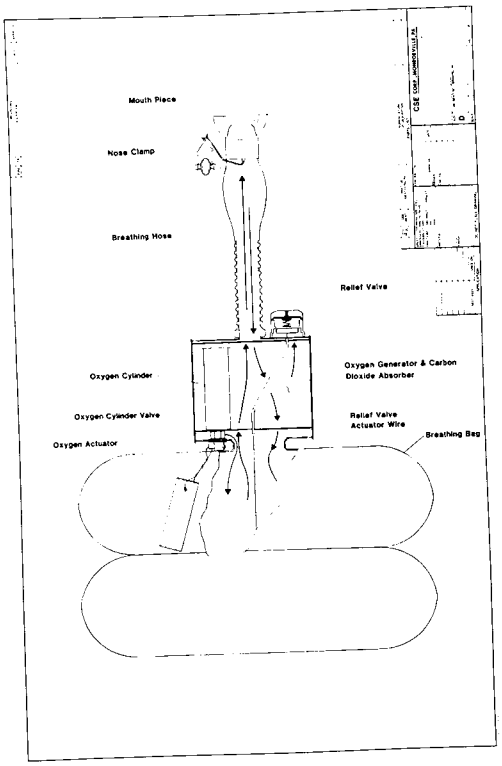


Figure 2.

A joint Bureau/MSHA planning effort is underway to decide how to evaluate the production-quality apparatus. These plans will be presented to the mining industry for discussion and comment. Our current thinking is to subject the PWSCSR's to a series of laboratory environmental treatments designed to simulate various conditions in underground coal mines (Stengel 1982; Kyriazi 1983). The environmental treatments consist of extremes of temperature, shock and vibration. These tests will be used as predictors of the ability of the prototype to survive the in-mine environment with no degradation in protection to the wearer. The PWSCSR's will also be examined and destructively tested to evaluate potential fire and explosive hazards.

Successful mine ruggedness evaluation will set the stage for field trials. There will be limited field deployment of the SR-100 for in-mine testing. This will involve cooperative ventures with the mining industry. The purpose of this work is: (1) to further establish the credibility of the new technology, (2) to explore practical deployment options, (3) to demonstrate performance under simulated mine escape, and (4) to assess the transferability of recently developed SCSR training methods to the new apparatus. As commercial versions of SR-100 - type apparatus are introduced into the industry, the Bureau's role will change to long-term monitoring of user proficiency and of how well these devices survive in-mine deployment.

THE ISSUE OF SCSR DONNING PROFICIENCY

The introduction of SCSR's into underground coal mines in 1981 made possible a new survival skill: the ability of miners to don the apparatus in order to escape an unbreathable atmosphere. Under MSHA's existing training regulations (in 30 CFR Part 48) was the requirement to provide "...instruction in the use, care, and maintenance of self-rescue and respiratory devices ..." (Code of Federal Regulations 1979). To comply with this regulation as it applied to the newly deployed SCSR's, mine trainers relied primarily upon the use of audiovisual programs or demonstrations by an instructor instead of upon performance trials by the trainees.

The prevailing assumption among most instructors was that workers were generally proficient in donning SCSR's. In the absence of empirical evidence to the contrary, conventional instruction was accepted as being sufficient. Reputed failures of workers to don the devices in situations calling for their use were most often attributed to poor judgment, panic, or both. Evidence mounted, however, that conventional SCSR task training, lacking a hands-on component, was inadequate and that lack of procedural skills might be an important concern. For instance, a

report on the Greenwich Collieries explosion (Fesak and Cavanaugh 1984) alerted Bureau of Mines researchers to the fact that workers might not know how to don SCSR's in spite of the demonstrations they had seen in annual refresher training.

A 1985 experiment conducted by the University of Kentucky and the Bureau of Mines with a group of underground miners who had received hands-on training in 1981, and a demonstration in annual refresher class each year since, resulted in a 39% failure rate. In November, 1986, MSHA completed a nationwide evaluation of the effectiveness of SCSR training. They found that 1,780 of the 8,904 miners tested, or nearly 20%, failed (Federal Register 1987). These findings are hardly surprising when one considers what is known empirically about infrequently used procedural skills in general and procedural skills having a motor component in particular.

First, it has been demonstrated that procedural knowledge is different from declarative knowledge, or knowledge about facts and things (Anderson 1985). Procedural knowledge deals with the process of performing subtasks that will lead to the accomplishment of a desired goal. Further, it is known that increasing the availability of relevant knowledge will help an individual carry out a procedure, while the presence of irrelevant knowledge can inhibit the process and hence impede fast, accurate, and flawless performance. The best way to teach a procedure, then, is to specify clearly the actions that are necessary to accomplish a particular goal (such as how to don an SCSR) and avoid superfluous information, even when that information has value in a declarative context (Mayer 1983).

Mayer's assertion is illustrated in the research reported by Cole, Vaught, Kellner, and Chafin (1987). After reviewing existing SCSR training materials, the authors noted that in addition to mixing declarative (factual) information with procedural information "...the training materials present no simplified, easy to remember overall procedural rules that can cue miners to recall and properly order the complex array of multiple tasks needed to don an SCSR." In other words, the donning procedure was not presented in such a way that the needed actions were easily understood (Anderson 1985).

A second fact that has been established empirically concerns procedural tasks that have a motor component. Motor learning has been defined as "...a set of processes associated with practice or experience leading to relatively permanent changes in the capability for responding." (Schmidt 1988). The key to learning a motor task such as SCSR donning, therefore, is practice combined with knowledge of results, as the definition suggests. The goal of practice is to reach some criterion of proficiency in which correct actions become more or less automatic. Schmidt has hypothesized the existence of schemas, or general rules governing the performance of a task, which the individual builds up in the course of practice. According to Schmidt, practice movements benefit the learner, because they contribute to building up the schema regarding a particular procedure.

Schemas also guide and direct practice. Without some overall directing framework within which people can assimilate their knowledge and experience, learning of complex procedural skills (motor or otherwise) may focus on small units, and not the whole performance. Learning to be proficient on such tasks usually takes longer without such an initial organizing schema.

There is little doubt that a person learning an infrequently used motor skill should have the opportunity to repeat the procedure and receive feedback on the adequacy of his or her performance once he or she reaches an established criterion. The reason for this "overlearning," especially in the case of seldom-used procedures, is that the level of performance achieved during training

is the best predictor of skill retention. While Johnson (1981) summed up what is known empirically about retention by stating that "...people can't remember what they didn't learn, and they forget over time...," the fact is that they retain more if they have learned more. Repetition combined with knowledge of results does enhance the level of task learning, and that in turn helps sustain an individual's procedural knowledge over periods when there is no practice (Hagman & Rose 1983).

One way to determine the value of training is through systematic assessment. Evaluation of performance itself is a major element in making training effective. This is true in the case of SCSR training for two reasons: First, in the process of developing training techniques, it is necessary to evaluate the performance of trainees in order to determine where in the donning procedure errors and wasted time are likely to occur. Once this has been done, it is fairly easy to devise the most efficient strategy for performing the task. It is then possible to determine the most effective way to teach the procedure. Second, since SCSR donning is largely a motor task and one of the most crucial elements of motor learning is knowledge of results, assessment and feedback to the learner are key to training effectiveness (Schmidt 1988).

INNOVATIVE SCSR DONNING METHOD

The preceding paragraphs suggest clearly that there is an extensive body of literature dealing with issues that are in every way comparable to those that have arisen regarding SCSR training. This research base was used directly in development of the Bureau's new training method.

Cole, Vaught, Kellner, and Chafin (1987) developed evaluation techniques as part of their effort to show what an optimized SCSR training program would include. The researchers first conducted a detailed task analysis using a controlled experiment in which 36 working miners who had recently gotten refresher training were videotaped in performance trials with the SCSR model in use at their mine. Assessment of the tapes allowed the investigators to target those steps in the procedure where most errors occurred and where most time was lost. It was found that individuals spent a majority of their time adjusting straps and locating goggles that had been dropped on the floor. In addition, many of the subjects became confused and omitted tasks such as putting on the nose clips. Only 22 individuals (61 pct) were able to complete the minimum of steps necessary to isolate their lungs, and approximately half of these required over a minute to do so.

Based on the experimental findings, an instructor's manual and short videotape demonstration were prepared for field testing. This package presents a generic procedure for the four SCSR's in common use (CSE, Draeger, MSA, and Ocenco). It offers the following: (1) a donning position (kneeling) that is easy and efficient; (2) a donning sequence that moves critical steps (those tasks necessary to isolate one's lungs) ahead of the others; and (3) a set of "chunked" procedural rules that facilitate easy retention of the many detailed subtasks in a single overall schema. This "3+3" donning method (see figure 3) has been promoted as an efficient means for providing thorough hands-on SCSR training, and has become the norm for the industry because it is a highly generalizable procedure. It works equally well with each SCSR currently being marketed, and the areas in which miners are prone to have errors are well-known. Individuals who will be expected to don the PWSCSR will have had training in performing the 3+3 for several years.

TASK TRANSFER

Although SCSR's are rugged and well-built, they do not have unlimited service lifetimes. Some units are



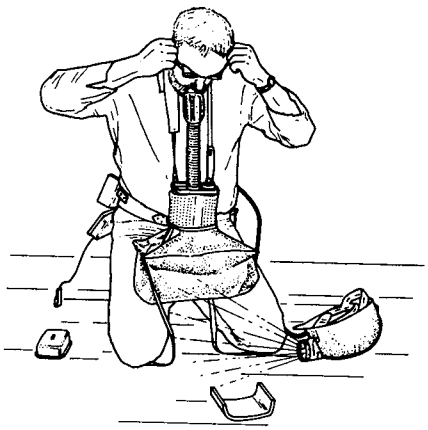
1. Activate Oxygen



2. Insert Mouthpiece



3. Put on Noseclips



+ 1. Put on Goggles



2. Adjust Straps



3. Replace Cap

Figure 3. The "3+3" sequence

lost or expended, others are damaged through misuse, while some simply wear out. To complicate matters, certain SCSR manufacturers have gone out of business or no longer manufacture apparatus for the mining industry. In other words, mine operators face the real problem of replenishing their SCSR's in order to remain in compliance with Federal regulations. Given advantages in both comparative cost and technology, we anticipate that the mining industry over time will purchase and gradually deploy PWSCSR's as replacements for SCSR's. Thus, there may be a need to train miners in the use of two apparatus. To illustrate the potential motor learning problems engendered by this situation, consider the following:

The first one hour person-wearable SCSR to be developed is CSE's SR-100. A worst-case scenario was projected in which a miner would activate the small oxygen bottle that initially fills the bag on the SR-100, and would then inadvertently dump all or part of the bag before getting the mouthpiece in. Dumping the entire bag would result in the loss of eight percent of the total oxygen supply. For this reason, it was suggested that it might be advisable to change the 3+3 sequence to teach users of the SR-100 to insert the mouthpiece before activating the oxygen. This seems a minor modification, and not anything to be concerned about.

Yet, it is known empirically that people are easily confused when they omit steps or get them out of order. For example, out of 624 SCSR donning trials conducted by the Bureau, there were 275 omitted steps. Only 90 of these omissions were subsequently corrected by the subject. The conclusion is that when individuals skip a step, they usually continue with the donning procedure and do not go back and correct themselves. Changing the donning sequence for the SR-100 would, in effect, tell people trained previously in the 3+3 method to skip activating the oxygen until the mouthpiece is in and the noseclips are on, and then to go back and activate the oxygen. This would actually build in error and would result in many initial failures to activate the oxygen.

The ability to transfer task learning is crucial, especially if PWSCSR's are phased in gradually. While it is easy, given the above illustration, to predict the amount of confusion that would result from mixing the PWSCSR with the SCSR if each had a different recommended donning procedure, it is not known how well the 3+3 extends to the PWSCSR. Schmidt (1988) has hypothesized that there might be low transfer, even among very similar devices. The only data collected by the Bureau concerning this issue deals with 25 subjects' ability to don the Draeger after having been trained and evaluated on the Ocenco. For this group, at least, performance with the Ocenco seems to be a fairly good predictor of performance with the Draeger (see figure 4). It is hoped that future research with the PWSCSR will yield comparable results.

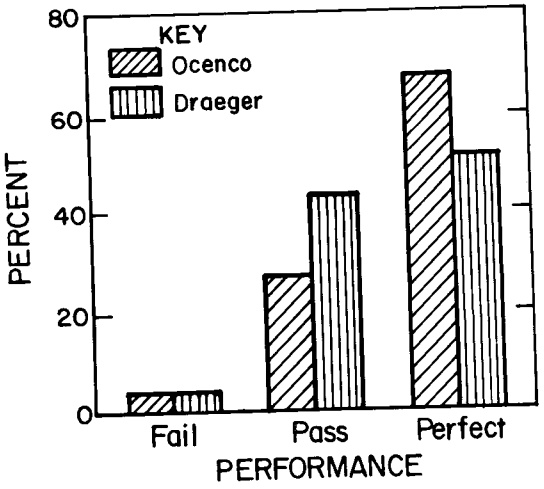


Figure 4.

DISCUSSION

PWSCSR technology based on the SR-100 design is intended as a strict substitute for current SCSR's, without any changes in Federal regulations. Because the SR-100 is belt-wearable, it is also a replacement for the filter self-rescuer. The cost of SCSR's meeting the Federal regulations for mine escape is approximately $500. A

commercial version of the SR-100 is estimated to cost less than $350. It is quite likely, therefore, that as existing apparatus are retired they will be replaced with PWSCSR's.

Since 1985 the Bureau of Mines has been engaged in research aimed at evaluating the adequacy of SCSR training, the development of an improved donning strategy, and the assessment of effects of training strategy on SCSR donning performance. With the development of PWSCSR's, attention must be devoted to questions of task transfer. It is becoming increasingly apparent that trainers themselves must be prepared to use appropriate teaching and evaluation techniques in order to insure that miners develop proficiency in donning whatever apparatus they will be expected to use in the future.

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APPAREILS AUTONOMES D'AUTO-SAUVETAGE: NOUVEAUX DEVELOPPEMENTS TECHNOLOGIQUES ET FORMATION

John G. Kovac, Charles Vaught

L'objectif du présent rapport est de résumer deux importantes voies de recherches sur le développement et l'utilisation d'appareils autonomes d'auto-sauvetage (SCSRs). Le Bureau des Mines effectue des recherches pour développer un appareil autonome d'auto-sauvetage à porter sur soi, conçu pour les sauvetages miniers. L'appareil, dont la taille est le double environ de celle de l'appareil d'auto-sauvetage à filtre, fournit une heure d'oxygène. Des recherches supplémentaires ont examiné les résultats de techniques de formation sur l'aptitude des mineurs à mettre ces appareils. Des études empiriques portant sur cette aptitude ont démontré que les méthodes d'entraînement influencent de manière significative et durable la façon dont les personnes accomplissent les gestes nécessaires, alors qu'en l'absence de pratique périodique ces techniques ont tendance à se détériorer assez rapidement.

GRUBEN-SELBSTRETTER: NEUE ENTWICKLUNGEN IN TECHNOLOGIE UND SCHULUNG.

Hohn G. Kovac, Charles Vaught

Der Vortrag ist eine Zusammenstellung von zwei Richtungen in der Forschung, Entwicklung und des Gebrauchs von unabhängigen Selbstrettern (SCSRs). Das Bureau of Mines führt zur Zeit eine Studie zur Entwicklung eines persönlichen unabhängigen Selbstretters durch, der zur Flucht aus Gruben geeignet ist. Der Selbstretter ist fast doppelt so groß wie der Filterselbstretter und hat einen Sauerstoffvorrat von einer Stunde. Zusätzliche Forschungsarbeiten befassen sich mit der Bestimmung der Wirkung von Schulungsprogrammen auf die Fähigkeit der Grubenarbeiter, die Handhabung der SCSR nicht zu vergessen. Empirische Studien über Anwendungsvermögen ergaben, daß Schulungsmethoden einen bedeutenden Einfluß auf die Art haben, mit welcher der Grubenarbeiter die Filter anlegt, und daß dieser Effekt lange anhält, während er gewöhnlich sehr rasch nachläßt, wenn Nachschulungen unterbleiben.

АВТОНОМНЫЕ САМОСПАСАТЕЛИ: ПОСЛЕДНИЕ УСПЕХИ В ТЕХНОЛОГИИ И ОБУЧЕНИИ

Джон Г. Ковач и Чарльз Вот

Задача настоящего доклада - обобщение двух главных направлений в исследованиях, относящихся к разработке и использованию автономных самоспасателей (SCSR). Горное Бюро США проводит исследования по разработке переносного шахтного автономного самоспасателя. Этот аппарат по размеру в два раза превосходит фильтровый самоспасатель. Он вмещает часовой запас кислорода. Дополнительные исследования были направлены на определение эффективности обучения шахтеров быстро и правильно приводить самоспасатели в готовность. На основании этого исследования было определено, что большое значение имел конкретный метод обучения людей, и что люди сохраняли способность правильно обращаться с оборудованием в течение длительного времени, хотя эти знания в значительной степени терялись без периодических тренировок.

设备齐全的自救器：技术和培训的新发展

约翰 G．柯凡克　及　查理 福特

本文扼要地叙述有关设备齐全的自救器（SCSRS）两项研究项目．矿业局正在研究发展一种单人穿的设备齐全的自救器用来逃出煤矿．这种自救器约为过滤式的自救器一倍大小，它可以供给一小时的氧气．另外一项研究项目为针对培训矿工保持穿戴自救器的熟练性．根据穿戴熟练的经验，培训方法对矿工熟练的程度和持久性有很大的关系．但是如果不经常练习则也很容易生疏．

MENSCHLICHE, TECHNISCHE UND ORGANISATORISCH-TAKTISCHE VORAUSSETZUNGEN FÜR GRUBENWHREINSÄTZE UNTER ERSCHWERTEN BEDINGUNGEN

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ZUSAMMENFASSUNG

Die Entwicklung des Bergbaues in der Deutschen Demokratischen Republik erfordert, die Grubenwehren auch für Einsätze in trocken-heißen Klimaten vorzubereiten. Durch systematische Forschungsarbeiten wird die Weiterentwicklung der Ausbildung, der Ausrüstung und der Einsatzfähigkeit der Wehren für diese Bedingungen betrieben.
In Klimakammerversuchen wurde der Einfluß der körperlichen Leistungsfähigkeit auf die Einsatzzeit bestimmt und mit Hilfe des PWC 170 eine Homogenisierung der Gruppen erreicht. Für die Trockentemperaturbereiche 40 - 55 Grad Celsius, $\varphi < 15$ % wird eine Einsatzzeittabelle vorgestellt. Mit dem getesteten Einsatz einer Wassereisweste können im Bereich $\tau_{tr} > 45^{0}C$ die Zeiten um 65 - 70 % gesteigert werden. In diesen Klimaten ist eine Konditionierung der Einatemluft von Regenerationsgeräten mit Drucksauerstoff erforderlich. Ergebnisse verschiedener Varianten werden im Beitrag angegeben.

AUSGANGSSITUATION

Der moderne untertägige Bergbau, durch Massenproduktion, hohen Mechanisierungs- und zunehmenden Automatisierungsgrad, spezialisierte Arbeitskräfte und eine wachsende Computeranwendung in allen Teilprozessen gekennzeichnet, weist in der Regel einen hohen Sicherheitsstandard auf. Das belegen für den Bergbau der DDR die Unfallstatistiken im Vergleich zur übrigen Industrie (Bild 1). Trotzdem bleiben die bergbauspezifischen Gefährdungen bestehen, die in den einzelnen Bergbauzweigen und -revieren nach Art und Grad sehr verschieden sein können. Die Anstrengungen zur Prophylaxe, Beherrschung und Beseitigung von Havarien sind daher auch weiterhin zentrale Aufgaben der Bergbausicherheit.

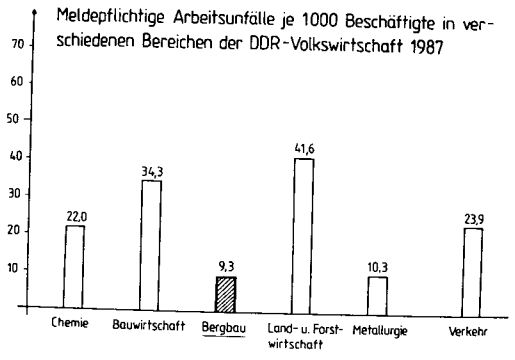


Bild 1

Ein wichtiger Garant ihrer Lösung ist das Grubenrettungswesen. Bei zahlreichen schweren Unglücken im internationalen Bergbau haben Grubenwehren, oft unter Einsatz des eigenen

Lebens, Bergleute gerettet und wertvolle Produktionsausrüstungen erhalten. Diese Einsätze waren stets auch wichtige Quellen für technische und taktische Weiterentwicklungen im Grubenrettungswesen. Eine Reihe von Entwicklungstendenzen weist darauf hin, daß diese spontanen Anstöße für die Entwicklung allein nicht mehr genügen:

- Die positive, weil planmäßig tendenziell rückläufige Unfall- und Vorkommnisentwicklung im Bergbau führt zwangsläufig zu sinkenden Einsatzfällen der Wehren. Damit gehen die aus ihnen gewinnbaren Erkenntnisse, Erfahrungen und Fertigkeiten zurück (Bild 2).

Entwicklung des Einsatzgeschehens der Grubenwehren der DDR von 1951 - 1987 (nach Riedel)

Zeitraum	Brände / Explosionen	Gasausbrüche	Sonstige	Gesamt
1951 - 1960	336	172	33	541
1961 - 1970	346	135	72	553
1971 - 1980	141	156	25	322
1981 - 1987	36	149	9	194

Bild 2

- Die natürlichen und technisch-technologischen Einflußfaktoren im untertägigen Bergbau (Geologie, Teufe, Grubenzuschnitt, Abbauverfahren u.a.) bedingen neue, zum Teil wesentlich kompliziertere Einsatzfälle für die Grubenwehren.
- Die bei Ernstfalleinsätzen zu bewältigenden Aufgaben, insbesondere die mit ihnen verbundenen Arbeitsenergieumsätze, sind für die überwiegend nebenberuflichen Wehrmitglieder zunehmend schwerer als ihre normalen Arbeitstätigkeiten.
- Die Ausrüstungen der Wehren unterliegen in ihrer Gesamtheit dem Einfluß der wissenschaftlich-technischen Entwicklung sowie ständig wachsenden Anforderungen hinsichtlich ihrer Anpassung an die veränderten Einsatzbedingungen.
- Die im Ergebnis der Mechanisierung und Automatisierung an untertägigen Arbeitsplätzen planmäßig eintretende Reduzierung der Arbeitsschwere führt gemeinsam mit allgemeinen Zivilisationserscheinungen (Bewegungsarmut, Übergewicht) zu einer Abnahme der körperlichen Leistungsfähigkeit der Bergleute, die sich auch bei nebenberuflichen Wehren auswirkt.

Aus diesen Entwicklungstendenzen ergeben sich zwingend neue Forderungen an die Qualität der Ausbildung sowie an Umfang und Inhalt grubenwehrspezifischer Forschung und Entwicklung. Grundsätzlich ist davon auszugehen, daß Grubenwehreinsätze stets unter erschwerten Bedingungen stattfinden, da folgende Erschwernisse im Einsatz immer vorhanden sind:

- hohe Arbeitsenergieumsätze,
- Tragen und Beatmen eines Atemschutzgerätes,
- Mitführen von Werkzeugen und Zusatzausrüstungen,
- starke psychische Beanspruchungen.

Weitere erschwerende Einflüsse ergeben sich bei spezifischen Einsätzen, deren Anteil an potientell möglichen, steigende Tendenz aufweist:

- Vorgehen und Handeln unter starker Sichtbehinderung,
- Einsatz in ungünstigen klimatischen Bedingungen,
- Erfordernis zum Tragen spezieller Schutzbekleidung,
- Anforderungen bei Spezialeinsätzen (Taucher, Abseilgruppen).

Die zur Gewährleistung wirksamer Wehreinsätze notwendige Beherrschung dieser Bedingungen ist prinzipiell über die drei Hauptfaktoren eines Einsatzes, den eingesetzten Wehrmann, die diesem zur Verfügung stehende Ausrüstung und die angewendete Taktik, zu erreichen. Das dabei erforderliche komplexe Vorgehen soll methodisch am Beispiel von

Forschungsergebnisse der DDR zum Grubenwehreinsatz in trokken-heißen Klimaten gezeigt werden. Die Schwerpunkte sind:
- die geistig und körperliche Vorbereitung der Wehrmänner auf Hitzeeinsätze (menschliche Komponente),
- die Ausrüstung und zusätzliche Hilfsmittel in ungünstigen Klimaten (technische Komponente),
- die Grundsätze und Verhaltensregeln für Wehreinsätze unter Klimabelastung (taktische Komponente).

ZUR TAUGLICHKEIT UND VORBEREITUNG VON GRUBENWEHRMÄNNERN FÜR HITZEEINSÄTZE

Die starke Inanspruchnahme der Thermoregulation führt bei Hitzearbeit zu zusätzlichen Herz-Kreislaufbeanspruchungen. Ihre Höhe und ihre Auswirkungen werden maßgeblich von der körperlichen Leistungsfähigkeit des Hitzearbeiters bestimmt. In umfangreichen Klimakammerversuchen mit schwerarbeitenden Grubenwehrmännern (BEU 500 W) bei trocken-heißen Klimabelastungen (40 - 60°C 15 %, v = 0,5 m s^{-1}) konnte der dominierende Einfluß der körperlichen Leistungsfähigkeit auf die bei Hitzeeinsätzen auftretenden Beanspruchungen sicher nachgewiesen werden. Für die Versuche wurden die Probanden in zwei Gruppen unterschiedlicher Leistungsfähigkeit eingestellt. Als Einteilungskriterium diente eine Physical Work Capacity (PWC 170) von 3,0 W kg^{-1}. Diese Größe wurde durch einen submaximalen ergometrischen Belastungstest ermittelt. Wehrmänner mit Werten 3,0 W kg^{-1} bildeten die leistungsstärkere Gruppe 1, Wehrmänner mit PWC 170 3,0 kg^{-1} die Leistungsgruppe 2. Dabei entspricht das gewählte Grenzkriterium der Gruppeneinteilung keineswegs Maximalanforderungen. Der Wert liegt im oberen Drittel der für die entsprechende Altersstruktur zutreffenden Referenzbereiche der körperlichen Leistungsfähigkeit für untrainierte Männer (Bringmann 1980). Der Einfluß der Klimabelastung ist bei den leistungsstärkeren Wehrmännern nur halb so groß, wie bei denen der Gruppe 2 (Bild 3), wobei der nachweisbare Klimaeinfluß auf die gemessenen Beanspruchungskriterien bei der Gruppe 1 erst bei höheren Belastungen beginnt, als das bei der anderen Gruppe feststellbar ist.

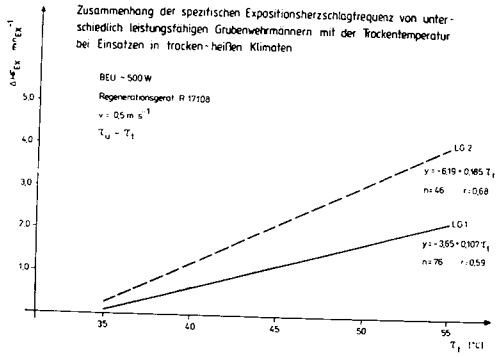


Bild 3

Wie in Bild 4 ersichtlich, treten bei der leistungsschwächeren Gruppe 2 signifikant höhere Beanspruchungen während der Hitzeeinsätze auf, als bei der Leistungsgruppe 1.
Da die o.g. Entwicklungstendenzen negativ auf die Leistungsfähigkeit nebenberuflicher Wehren einwirken (Spioch 1987), Faster 1987), (Thien 1981), (Goedecke u.a. 1989), ihr andererseits eine solche Bedeutung zukommt, müssen planmäßige, kontinuierliche Maßnahmen des körperlichen Trainings zu notwendigen Ausbildungsinhalten werden. Mit der neuen Zentralen Ausbildungsanweisung für Grubenwehren der DDR (--- 1989) wird dieser Forderung entsprochen. Durch die gleichzeitige Modifizierung und Neufassung der seit Jahren bewährten speziellen Tauglichkeits- und Überwachungsrichtlinie des Ministeriums für Gesundheitswesen (---1973) steht für die-

τ_t [°C] / Phys. Größen / Leistungsgruppen	40 °C		45 °C		50 °C		55 °C	
	$\Delta HF/min_{EX}$	$\Delta\tau_{REC}/h_{EX}$	$\Delta HF/min_{EX}$	$\Delta\tau_{REC}/h_{EX}$	$\Delta HF/min_{EX}$	$\Delta\tau_{REC}/h_{EX}$	$\Delta HF/min_{EX}$	$\Delta\tau_{REC}/h_{EX}$
Gruppe 1	0,752 ± 0,32	0,93 ± 0,49	1,00 ± 0,53	1,00 ± 0,46	1,762 ± 0,62	1,36 ± 0,26	2,278 ± 0,91	1,71 ± 0,32
Gruppe 2	1,444 ± 0,72	1,08 ± 0,43	2,127 ± 1,30	1,48 ± 0,21	2,661 ± 0,75	1,46 ± 0,41 *	4,443 ± 2,29	1,61 ± 0,44 *
Signifikanz ΔG2-G1	P = 5 %	-	P = 1 %	P = 0,1 %	P = 0,5 %	-	P = 0,5 %	-

* Wegen der z. T. sehr kurzen Expositionen der LG 2 in diesen Temperaturen sind die Mittelwerte der τ_{REC} nur bedingt interpretierbar.

Bild 4

sen Ausbildungsteil ein hochwirksames Kontroll- und Differenzierungsinstrumentarium zur Verfügung.
Die internationalen Erfahrungen aus Grubenwehreinsätzen in ungünstigen Klimaten und aus Klimakammeruntersuchungen (Frycz 1983), (Funkemeyer und Stocklassa 1989), (Groß 1973) unterstreichen die Wichtigkeit von ernstfallnahen Hitzegewöhnungsübungen in allen potentiell vorkommenden Bereichen der Klimabelastung. Diese Forderung gilt insbesondere angesichts der bei größeren Havarien traditionell selbstverständlichen Hilfeleistung durch Wehren nicht betroffener Gruben. Damit können z.B. Wehren aus dem Salzbergbau, die primär mit Hitzeeinsätzen in trocken-heißen Klimaten konfrontiert werden, in feuchtwarmen Klimaten zum Einsatz kommen. Die Hitzegewöhnungsübungen müssen daher differenziert nach Haufigkeit und Inhalt, aber nie auf ein Klimabelastungsintervall isoliert durchgeführt werden. Der Vermittlung individueller Beanspruchungsgrenzen und ihre Abhängigkeit von den Einsatzbedingungen kommt daher ein Hauptaugenmerk zu.

SPEZIELLE ERKENNTNISSE ZUR AUSRÜSTUNG BEI HITZEEINSÄTZEN

Durch die Differenzierung der Testgruppen hinsichtlich ihrer körperlichen Leistungsfähigkeit war es möglich, auch mit normal ausgerüsteten Wehrmännern der Gruppe 1 längere Einsätze in hohen Klimabelastungen zu realisieren bzw. in höhere Klimabelastungen vorzudringen. Dadurch ergaben sich wichtige Hinweise auf notwendige Veränderungen und Ergänzungen für Hitzeeinsätze und Erkenntnisse zu den Obergrenzen wirksamer Einsätze ungeschützter Wehrleute in hohen Klimabelastungen.
Da in potentiellen Ernstfällen (z.B. bei Rettungswerken)

auch Einsätze oberhalb dieser "Grenzen" erforderlich sind, müssen entsprechende Ausrüstungen ein gefahrloses Vorgehen ermöglichen. Eine derartige Schutzvariante, für Einsätze in feucht-warmen Klimaten inzwischen weltweit erprobt und zunehmend angewendet, ist Kühlkleidung, vor allem Teilkörperkühlkleidung.
(Semljanki u.a. 1986), (Funkemeyer 1987), (Pasternack 1979), (Webbon et al. 1977), (Schutte et al. 1987), (Krüger und Scholz 1985).
Da ähnliche Aussagen für trokken-heiße Klimate kaum vorliegen (des Rosa und Stein 1976), der Salzbergbau größerer Teufen aber derartige Erkenntnisse benötigt, wurden in der DDR mit den gleichen Versuchsinhalten und -methoden, wie im vorangehenden Abschnitt vorgestellt, umfangreiche Tests zur Wirksamkeit von Wassereiskühlwesten bei Grubenwehreinsätzen in diesen Klimaten durchgeführt. Die Ergebnisse bestätigen die positiven Aussagen aus dem Steinkohlen- und Erzbergbau und rechtfertigen folgende Feststellungen:

- Durch das Tragen von Wassereiswesten werden signifikant niedrigere Beanspruchungen und damit längere Einsatzzeiten erreicht, als bei ungeschützten Wehrleuten (Bild 5), (Bild 6).

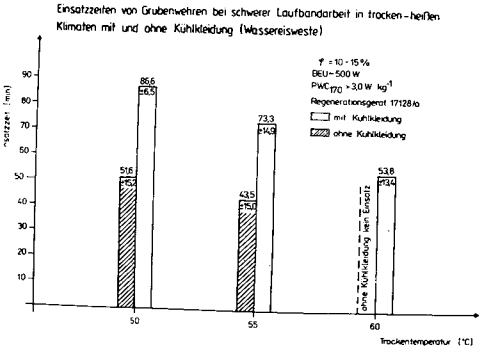


Bild 5

- Es gibt eine Untergrenze des wirksamen Einsatzes von Teilkörperkühlkleidung bei Grubenwehreinsätzen.
Sie ergibt sich aus technologischen (Westenwechsel - Einsatzzeit) und physiologischen Gründen. Unter den getesteten trocken-heißen Bedingungen sollte eine Wassereiesweste nicht unterhalb einer Trockentemperatur von 45oC eingesetzt werden.
- Ebenso exestiert eine Obergrenze für den wirksamen Einsatz von Wassereiswesten für Grubenwehren. Nach den derzeitigen Erkenntnissen, die Versuche sind noch nicht abgeschlossen, liegt sie bei Trockentemperaturen von 65 - 70°C.
- Hitzeeinsätze unter Verwendung von Wassereiswesten sind nur für Wehrleute der Leistungsklasse 1 (PWC 170 3,0 W kg -1) sinnvoll.
- Das Tragen einer Wassereisweste führt bei allen denkbaren Tätigkeiten während eines Grubenwehreinsatzes zu keinen nennenswerten Behinderungen und ist auch unter Einsatzbedingungen sicher sowie technologisch beherrschbar.

Die durch Verwendung der Wassereisweste möglichen Einsätze in sehr hohen Klimabelastungen erbrachten erstmals konkrete Ansatzpunkte für notwendige Modifizierungen der Standard-Regenerationsgeräte R 17128/a bei Hitzeeinsätzen in hohen Trockentemperaturen (45°C). Nach Einsatzzeiten von 45 - 60 Minuten (60°C, 150 %, BEU 500 W) traten verstärkt Versuchsabbrüche wegen unerträglich empfundener Einatembedingungen auf. Ergänzende Prüfstanduntersuchungen bei vergleichbaren Bedingungen belegten, daß zum Zeitpunkt des Abbruchs sehr ungünstige Einatemverhältnisse vorlagen (= 59°C, 50 %) und sich bei den simulierten Belastungsbedingungen nach ca. 20 Minuten eine kontinuierliche wachsende Wärmeaufnahme über

Reduzierung der Beanspruchung von Grubenwehrmitgliedern durch das Tragen einer Wassereisweste bei schwerer Hitzearbeit in trocken-heißen Klimaten

Beanspruchungsgröße u. Einsatzart [°C]	ohne Wassereisweste			mit Wassereisweste		
	$\Delta\tau_R$/h Exp. [°C]	ΔHF_{EX}/min Exp. [Schläge · min^{-1}]	Schweißrate [g · h^{-1}]	$\Delta\tau_R$/h Exp. [°C]	ΔHF_{EX}/min Exp. [Schläge · min^{-1}]	Schweißrate [g h^{-1}]
50 (n = 10)	1,36 ± 0,26	1,76 ± 0,62	1300 ± 300	0,71 ± 0,22	0,95 ± 0,27	1166 ± 312
55 (n = 9)	1,71 ± 0,32	2,28 ± 0,91	1440 ± 410	0,78 ± 0,25	1,21 ± 0,50	1302 ± 286
60 (n = 11)	- keine Versuche -			1,12 ± 0,24	1,71 ± 0,40	1562 ± 401

$PWC_{170} \geq 3{,}0\ W \cdot kg^{-1}$
$\varphi < 15\,\%$
$v = 0{,}5\ m \cdot s^{-1}$
BEU ~ 500 W

Die Differenzen für Korntemperatur und Expositionsherztrequenz sind hochsignifikant (P = 0,1 %).

Bild 6

die Atmung vollzieht, die wesentlich durch eine Veränderung der Wasserbilanz der Regenerationspatrone mit fortgeschrittener Gebrauchsdauer verursacht wird (Bild 7).

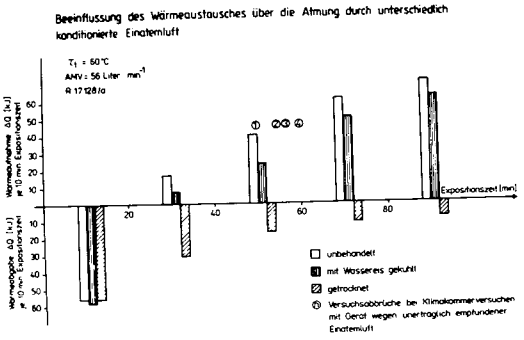


Bild 7

Die Veränderung dieses Tatbestandes ist für wirksame Hitzeeinsätze unerläßlich. Dazu sind aus der internationalen Praxis sowohl die Kühlung der Einatemluft bei klassischen Regenerationsgeräten (Shvartz und Strydom 1975), (Brandis und Suchowoj 1961), (Pasternack 1987), als auch der Einsatz alternativer Geräte (Flüssigsauerstoff- und Behältergeräte) bekannt geworden. (Ley u.a. 1964), (Didenko 1971).

Eigene Untersuchungen zur Verbesserung der Einatembedingungen beim Regenerationsgerät 17128/a in trocken-heißen Klimaten zeigen, daß einer Trocknung der Einatemluft größerer Bedeutung zukommt, als einer Kühlung (vergl. Bild 7). Werden beide Varianten gemeinsam angewendet, sind die mit Wassereisweste erreichbaren Einsatzzeiten in trocken-heißen Klimaten (vergl. Bild 5) auch bei Verwendung der Standard-Regenerationsgeräte realisierbar. Gerätetechnisch sind Trocknungs- und Kühlvarianten sowohl extern als auch in das Gerät integriert ausführbar.

Die gegenwärtig noch mit organisatorischen und technischen Nachteilen und zum Teil mit Vorbehalten behaftete Verwendung von alternativen Geräten muß für Hitzeeinsätze systematischer untersucht werden, dürfte aber auf wenige Einsatzfälle beschränkt bleiben.

TAKTISCHE BESONDERHEITEN BEI HITZEEINSÄTZEN

Wegen des großen Einflusses der körperlichen Leistungsfähigkeit auf die bei Hitzeeinsätzen auftretenden Beanspruchungen der Grubenwehrmänner ist ihre jährlich aktualisierte Dokumentation für jeden Wehrmann und deren praktische Beachtung bei der Einteilung von Einsatzgruppen eine Grundforderung der Taktik für Hitzeeinsätze. Mit Hilfe einer zentralisierten Wehrstammdatei, die diese Daten enthält, ist auch bei großen Grubenunglücken, zu deren Beherrschung Wehrmänner vieler Gruben und Bergbaubereiche eingesetzt sind, eine Homogenisierung der Einsatzgruppen nach den Kriterien der körperlichen Leistungsfähigkeit möglich. Dabei werden speziell bei Hitzeeinsätzen die vorgegebenen Richtzeiten für den Oberführer sicherer handhabbar, vorzeitige Abbrüche der Einsätze oder eventuellen Gefährdungen der Wehrmänner geringer und damit der operative Spielraum größer.
Diese Richtzeiten sind ein unverzichtbares taktisches Hilfsmittel bei Hitzeeinsätzen. Die für den feucht-warmen Klimabereich gültigen, international gut übereinstimmenden Zeiten sind in trockenheißen Klimaten immer weniger zutreffend (Bild 8). Aus den o.g. Klimakammerversuchen wurden daher in der DDR eine spezielle Zeittabelle für diese Klimate ermittelt und in das Regelwerk der Grubenwehr aufgenommen (Bild 9).

Richtzeiten für Grubenwehreinsätze in trocken-heißen Klimaten

φ < 20 %
BEU ~ 500 W

Trockentemperatur [°C]	Richtzeiten [min]	
	LG 1 ($PWC_{170} \geq 3{,}0\ W \cdot kg^{-1}$)	LG 2 ($PWC_{170} < 3{,}0\ W \cdot kg^{-1}$)
40	70	50
45	60	30
50	40	-
55	30	-

Bild 9

Sie ist für Hitzegewöhnungsprüfungen verbindlich und für Ernstfalleinsätze (z.B. Rettungswerke) als Richtschnur zu betrachten. Die in den Zeiten enthaltenen Sicherheitsreserven gestatten unter konkreten Einsatzbedingungen deren operative Modifizierung durch den Einsatzleiter. Angesichts der beispielsweise mit der Teufenentwicklung der Gruben zunehmenden und differenzierten potentiellen Hitzeeinsätze der Grubenwehren sind die Richtzeiten für derartige Einsätze zu erweitern und zu vervollkommnen. Dabei stehen zur Zeit folgende Schwerpunkte:
- Zeiten für Mehrfacheinsätze während einer Schicht unter Beachtung einsatztypischer Erholungsbedingungen,
- Zeitnormative für Einsätze mit Kühlkleidung,
- Dauer von Einsätzen unter Verwendung isolierender Schutzkleidung (Gasschutzanzüge, Wärmestrahlungsschutzkleidung),
- Einsatzdauer in hohen CO_2 Konzentrationen.

Untersuchungen zur Ermittlung dieser Zeitvorgaben werden gegenwärtig in der DDR durchgeführt. Durch eine begründete Homogenisierung der Einsatzgruppen hinsichtlich der körperlichen Leistungsfähigkeit,

Ermittlung von Einsatzzeiten für trocken-heiße Klimate aus Vorgaben für feucht-warme Klimate

Beispiele: τ_t: 45 °C ; τ_f: 22 °C
τ_t: 50 °C ; τ_f: 24 °C
τ_t: 55 °C ; τ_f: 25 °C
} $v \leq 0{,}5$ m
schwere Arbeit (z.B. Schleifkorbeinsatz)

Tabelle feucht-warm

τ_f [°C]	Zeit [min]
31	70
33	50
35	40
37	30
39	25
40	25

- Wenn $\tau_t - \tau_f > 10$ °C
  → Zeit lt. Tabelle - 10 min
- Bei besonders schwerer Arbeit
  → Zeit lt. Tabelle - 10 min

- Da τ_f = 22°, 24° und 25 °C nicht in der Tabelle enthalten, kommt τ_f = 31 °C zum Ansatz
  = 70 Minuten
- Da $\tau_t - \tau_f$ in jedem Fall > 10 °C → - 10 Minuten
  70 - 10 = 60 Minuten
- Da schwere Arbeit → - 10 Minuten
  60 - 10 = 50 Minuten

Es ergibt sich also für deutlich unterschiedliche thermische Belastungen (siehe Beispiele) stets die gleiche Einsatzzeit und damit ein zu nehmender Widerspruch zwischen Belastung und Beanspruchung

Bild 8

eine theoretisch und praktisch auf hohem Niveau stehende Ausbildung der Wehren, spezielle Ausrüstungen sowie ein geschlossenes System von Richtzeiten für Hitzeeinsätze, sind keine individuellen Beanspruchungskontrollen der Wehrmänner im Einsatz erforderlich. Die meßtechnische, inhaltliche und organisatorische Realisierbarkeit einer solchen, im Grubenrettungswesen immer wieder auftauchenden Forderung, ist ohnehin sehr kritisch zu beurteilen. Nicht der bestüberwachte, sondern der optimal vorbereitete und ausgerüstete Grubenwehrmann ist bei Hitzeeinsätzen am sichersten vor gesundheitlichen Gefährdungen geschützt.

SCHLUSSFOLGERUNGEN

Die Ergebnisse unserer Forschungsarbeiten zur Gewährleistung sicherer und wirksamer Hitzeeinsätze belegen überzeugend die methodische Ausgangsthese, daß die isolierte Beeinflussung und Veränderung von Teilkomponenten eines Grubenwehreinsatzes keinen dauerhaften Erfolg garantiert. Die technische Ausrüstung allein ist kein Garant für wirksame Einsätze in ungünstigen Klimaten, wenn nicht speziell vorbereitete Wehrmänner zur Verfügung stehen. Das Gleiche gilt umgekehrt. Anwendbare Richtzeiten sind ohne Bezug zur Ausrüstung und Einsatztechnologie eher ein Unsicherheitsfaktor als ein taktisches Hilfsmittel. Diese Zusammenhänge vorausschauend immer detaillierter zu erforschen und umgehend in der Wehrpraxis zu beachten, ist die beste Garantie für eine wachsende Schlagkraft der Wehren, trotz abnehmender Häufigkeit der Ernstfalleinsätze. Daß eine leistungsfähige Grubenwehr auch und gerade im "Bergwerk der Zukunft" zu den unverzichtba-

baren Elementen der Bergbausicherheit zählt und keine sich überlebende historische Einrichtung ist, müssen nicht erst immer tragische Grubenunglücke schmerzhaft deutlich machen.

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HUMAN, TECHNICAL AND ORGANIZATIONAL-TACTICAL CONDITIONS OF RESCUE CREW ENGAGEMENTS UNDER COMPLICATED CIRCUMSTANCES

M. Goedecke, D. Hesse, J. Schmidt

Mine rescue systems are an indispensable part of mine safety. Due to new mining techniques and technological developments, as well as to the continuous worsening of mining conditions, rescue operations are becoming more complex.

Reasons for this trend are analyzed and its effect on the rescue team's operations are discussed. Comprehensive experimental investigations have been carried out for the mine rescue system of the GDR in the mines and in special simulation chambers during the last years. Their emphasis was on:

- Definition of the minimum requirements imposed upon the physical strength of the rescue team members;
- Establishment of a time schedule for rescue team operations in arid and torrid climates (temp: 35 to 55° C);
- Efficiency and limits of a cooling dress for rescue teams at temperatures of 40 to 60° C and low atmospheric humidity;
- Modification of rescue respirators for hot operations.

Methods for a quick practical application of this knowledge are presented and explained.

CONDITIONS HUMAINES ET TECHNIQUES, DE L'ORGANISATION ET DES TACTIQUES DE SAUVETAGES. INTERVENTION DES EQUIPES DE SAUVETAGE DANS DES CIRCONSTANCES COMPLEXES

M. Goedecke, D. Hesse et J. Schmidt

Les systèmes de sauvetage miniers sont un aspect indispensable de la sécurité minière. Les nouvelles techniques d'exploitation utilisées, les développements technologiques et l'aggravation continuelle des conditions minières ont rendu les opérations de sauvetage plus complexes. Nous en analysons les raisons et en examinons les effets sur les opérations effectuées par les équipes de sauvetage. Ces dernières années, on a effectué des études expérimentales sur le système de sauvetage minier en RDA, à la fois dans les mines et dans des chambres spéciales de simulation. Les objectifs de ces essais étaient les suivants :

- Définitions des exigences minimum imposées à la force physique des membres des équipes de sauvetage;
- Etablissement du temps nécessaire pour les opérations des équipes de sauvetage dans des climats arides et torrides (temp. : 35 à 55 degrés C);
- Efficacité et limites des tenues refroidissantes des membres des équipes de sauvetage par des températures de 40 à 60 degrés C avec faible humidité atmosphérique;
- Modification des appareils respiratoires de sauvetage pour les opérations effectuées par des températures élevées.

Nous présentons et analysons les méthodes permettant l'application pratique et rapide de ces connaissances.

ПСИХОЛОГИЧЕСКИЕ, ТЕХНИЧЕСКИЕ И ОРГАНИЗАЦИОННО-ТАКТИЧЕСКИЕ КОМПОНЕНТЫ РАБОТЫ СПАСАТЕЛЬНОЙ КОМАНДЫ В СЛОЖНЫХ УСЛОВИЯХ

М. Гедеке, Д. Хессе и Х. Шмидт

Система горноспасательных работ является неотъемлемой частью техники безопасности в шахтах. Спасательные операции в шахтах все более усложняются из-за усовершенствований горной техники и технологии, а также в связи с ухудшением горных условий. Обсуждаются причины существования этого тренда и вытекающие из него последствия для работы спасательной команды. В последние годы в ГДР, в действующих шахтах и в специально оборудованных камерах, моделирующих шахтные условия, были проведены всесторонние исследования систем горноспасательных работ. Особое значение придавалось следующим обстоятельствам: определение минимальных требований по отношению к физической выносливости членов спасательных команд; расписание операций спасательной команды в сухом и жарком климате (от 35°C до 55°C); эффективность и ограничения охлаждающей спецодежды при температуре от 40°C до 60°C и низкой влажности воздуха; модификация спасательных дыхательных аппаратов для использования при высоких температурах. Представляются и объясняются методы быстрого практического применения полученных данных.

在复杂情况下援救人员的人为，技术及组织策略问题

M。哥达克， D。赫塞， 及 斯密特

矿工援救系统是矿业安全必不可少的一部分。由于新的开采技术，技术上的进步，和煤矿情形渐渐恶化，援救系统变得非常复杂。我们先分析这种趋势然后再讨论它对援救工作的影响。过去几年中，民主德国对煤矿援救系统在煤矿及特别模拟室里做了各种实验。它们着重在下列诸点：

- 援救队人员所规定的体力最小限度的定义；
- 建立援救工作者在干燥和酷热气候（温度摄氏３５至５５度）中的工作时间表；
- 援救工作者在摄氏４０至６０度低湿度中冷爽衣着的效率及限度；
- 援救呼吸器在酷热操作中的变更。

上述事项的实际施行办法在本文中亦有所讲述。

LONG-TERM ANALYSIS OF A SEALED MINE POST-FIRE ATMOSPHERE

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ABSTRACT

In early June, 1986, an underground fire of unknown origin erupted in one of several entries of a coal mine located in southwestern Colorado. For a time mine personnel aggressively fought the fire, but because of a lack of extinguishing foam, control of the fire was lost. To prevent further damage and to reduce the possibility of an explosion, all portals and shafts to the mine were sealed.

The mine operator requested the Bureau of Mines' assistance in obtaining and analyzing mine fire gases taken from sampling ports placed through the mine seals, as well as down boreholes drilled after the fire. This report recounts activities within the first two years that the mine was sealed. It includes a description of gases analyzed as well as mine atmosphere status equations generated from the analyzed gases. The gases were analyzed by chromatography and included standard atmospheric gases as well as higher hydrocarbons that are found in post-fire atmospheres. Nine equations were used to evaluate what was occurring within the sealed mine.

INTRODUCTION

The Orchard Valley Mine, an underground coal mine, is located 8 km north of Paonia, Colorado. It is operated by Colorado Westmoreland, Inc., a wholly owned subsidiary of the Westmoreland Coal Company. The mine was opened in 1976 and produced steam coal which was primarily used for electric power generation. Approximately 2200 tons of coal were produced daily from two unit shifts.

Portal elevation for this drift mine was just under 2,150 m. The coal seam mined is known as the D seam. It is a highly friable, vertically cleated bituminous coal and varies in thickness from 2.5 to 8 m. The seam pitches downward toward the north and is interlaced with numerous vertical fault zones that have limited the practicality of development mining in several sections. The seam is prone to spontaneous heating.

All mining was by the room-and-pillar method, using electrically-powered continuous miners. Primary coal haulage and ancillary vehicular services were provided by diesel-powered equipment. Because of the diesel equipment and the propensity of the seam to heat spontaneously, a mine-wide carbon monoxide (CO) monitoring system continuously measured CO concentrations. This system included monitors and data storage centers located in buildings at the entrance to the mine property. Security personnel were responsible

for monitoring CO during periods when the mine was idle.

An intake shaft and a fan shaft were sunk on the right side of the main entry, approximately 150 m northeast of the belt and intake portals. Two parallel axial-vane fans were the ventilation sources for the mine. Because of Federal coal leasing arrangements, as mining advanced, individual sections were developed on either side of the main entries, creating mined-out abandoned areas that required bulkhead structures to isolate them from the active mine. Four sealed gobs abutted the left side; eight sealed gobs abutted the right. In addition, about 2 km inby the portal an active submain extended southeast from the main entry. This submain led to the only working section, located approximately 5.3 km from the intake portal.

THE FIRE

At 4:40 am, Sunday, June 1, 1986, an underground fire was reported by the on-duty security guard. The portion of the mine involved in the fire is shown in Figure 1. Mine exploration to locate the fire and fire fighting techniques used to combat the fire have been documented (Timko, Derick, and Thimons 1987). Because of insufficient quantities of high expansion fire fighting foams, the fire raged out of control. The inability to directly fight the fire ultimately forced evacuation of the mine, which was completely sealed at 4:30 am, Tuesday June 3, 1986.

Determining the Quantity of Coal Undergoing Combustion

Once it was determined that a fire existed, it became important to determine how much coal was burning. The following equation (Mitchell and Burns, 1979) enabled personnel to make this estimate:

$$F = (\ O_2)(k)$$

where:
F = quantity burning, kg/sec
k = 0.03 (coal), 0.08 (wood)
$O_2 = Q(O_2' - O_2)$

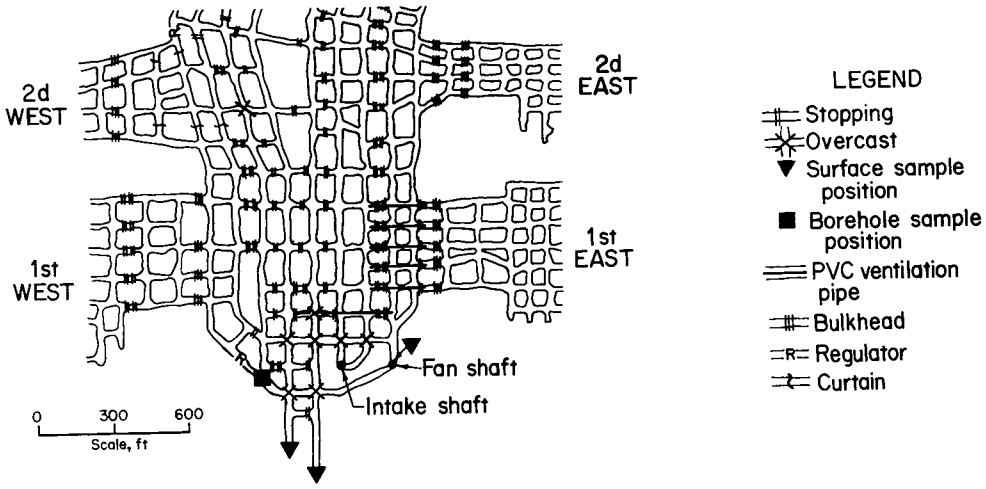


Figure 1.

Q = airflow across fire, m^3/sec
O_2' = intake oxygen concentration, pct
O_2 = return oxygen concentration, pct

This equation required a knowledge of air quantities and oxygen concentrations that were not available once the mine was sealed. It was important that adequate samples were obtained while fighting the fire, so that information concerning coal combustion quantities could be generated.

Analysis Using the Combustion Equation

In Figure 2 coal combustion rates remained slightly above 0.8 kg/sec as long as the high expansion fire-fighting foam was being applied.

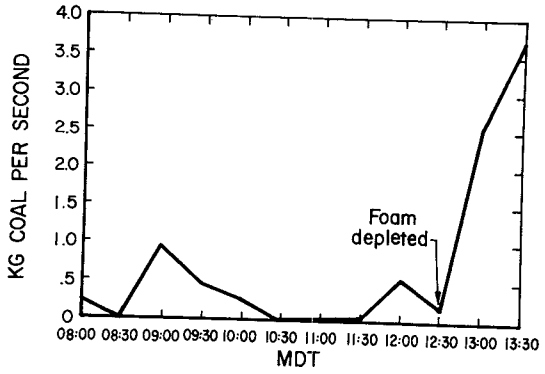


Figure 2.

Within 30 min after the first depletion of foam, the quantity of coal undergoing combustion increased to approximately 1.8 kg/sec. Following the initial foam supply exhaustion, the quantity of burning coal was never reduced to the combustion rates found while the foam was being applied.

POST-FIRE SAMPLE LOCATIONS

Prior to sealing both portals, a 0.64 cm ID copper gas sample line was placed inby the seals. An empty water pipe in the fan shaft ran from the fan shaft collar to the mine level. This pipe was used to lower a 0.64 cm ID semi-rigid sample line to the mine level. The top of the pipe was then sealed to prevent ingassing from the surface. Initially, samples were taken twice daily from each of these three positions.

Colorado Westmoreland, Inc. personnel and their mine fire consultant determined that at least two boreholes were needed. A 76 m deep borehole was drilled in the 2 left return on June 13, 1986. On June 18 a second borehole, this one 335 m deep, was drilled into the mine near an old borehole in the left return at crosscut 33. Sampling in this borehole was at two levels, the mine level and approximately 1.8 m below the borehole collar. Semi-rigid sample tubing was used to transport the borehole samples to the surface.

DEVELOPING A POST-SEALING GAS ANALYSIS

Gas Sampling Strategy

A sampling strategy was developed to analyze mine fire gases remotely from the surface. Gas samples were only taken when the barometer was falling and the mine was outgassing, which ensured a positive pressure differential across the seal. The differential was measured by connecting a Magnehelic pressure measuring instrument to the end of the sampling line and recording the value. Waiting for an outgassing period eliminated the possibility of erroneous samples due to atmospheric air mixing with mine gases (Fauconnier, 1980).

Temperature also plays an important role in determining when to obtain mine gas samples. If outside temperatures are lower than internal sealed atmosphere temperatures, gas flow into the mine is more likely to occur. Conversely, if outside temperatures are higher than internal temperatures, the potential for airflow from the mine to atmosphere is greater (Forbes and Grove, 1948).

All gas sampling was done with 20 ml evacuated glass sample tubes. A 2 L/min personal sampling pump was used to draw the mine gas to the

sampling tube. Two types of gas chromatographs were used: flame ionization detectors for low gas concentrations (ppm range), and thermal conductivity detectors for higher gas concentrations (pct range).

MINE FIRE GAS ANALYSIS

The gases examined were divided into two general categories: atmospheric gases, and oxidation or combustion byproducts. The atmospheric gases were those typically found in an atmospheric gas sample; they included nitrogen (N_2), oxygen (O_2), carbon dioxide (CO_2), and argon (Ar). Oxidation or combustion byproducts included carbon monoxide (CO), hydrogen (H_2), and the alkane (methane [CH_4], ethane [C_2H_6], propane [CC_3H_8], butane [C_4H_{10}], and pentane [C_5H_{12}]), alkene (ethylene or ethene [C_2H_4], and propylene or propene [C_3H_6]), and alkyne (acetylene [C_2H_2]) series of hydrocarbon gases.

MINE FIRE EQUATIONS

Several equations have been developed to aid in establishing the status of a mine atmosphere. These predictive models are used to determine:

1) if a fire has occurred
2) if the atmosphere surrounding a fire is explosive
3) if the fire continues to burn after sealing the affected area
4) if a potential exists for reignition once the sealed area is reopened

All of the models described in this section were used to evaluate the sealed atmosphere in the Orchard Valley Mine. A brief description of each model is given, along with an analysis of values obtained for the first two years of sampling. Since most sampling locations had similar gas concentrations throughout the evaluation period, only the intake sample position is discussed. The intake sample position was located approximately 80 m inby the seal. It should be noted that these models are used to develop trends, not measure individual values. Only through long term analysis can a realistic picture of the mine atmosphere be created.

Is There a Fire?

Jones-Trickett Ratio

The Jones-Trickett Ratio (JTR) (Jones and Trickett, 1954) is used primarily to determine sample reliability. The equation:

$$JTR = (CO_2 + (0.75 \times CO) - (0.25 \times H_2))/((0.265 \times (N_2 + Ar)) - O_2)$$

is based on the principle that the molecules of matter consumed in a fire are proportional to the volumes of gas produced. It was originally developed to help researchers differentiate between the causes of explosions, whether gas or dust. It has since been adapted, with mixed success, to mine fire situations.

The JTR varies with the type of fire, depending on the fuel. If the JTR is less than 0.4, there is no combustion. When the JTR is between 0.4 and 0.5, methane is the fuel. JTR values between 0.5 and 1.0 indicate that the fuel is coal, oil, conveyor belting, insulation, or urethane foam. If the JTR is between 0.9 and 1.6, wood is burning. JTR values above 1.6 are impossible, and the gas chromatograph must be recalibrated.

The problem with the JTR is that air dilution affects the results. This model relies on a quantity of air moving down an entry, over a fire, and directly to a sampling point. If air that has not passed over the fire zone mixes with oxygen-depleted air from the fire, the sample is diluted and incorrect values are obtained.

Analysis Using Jones-Trickett's Ratio

Figure 3 initially showed a JTR value decreasing to less than 0.4, which normally indicates that combustion has ceased. However, because of air dilution, the JTR originally

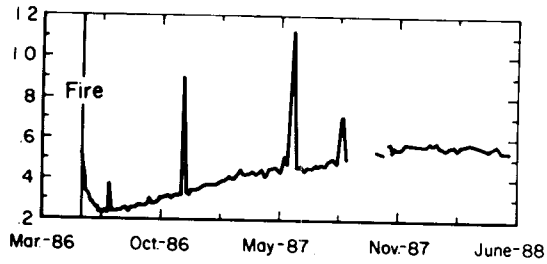


Figure 3.

indicated a methane fire, when it was evident that coal was the primary fuel. Following the initial decline in the JTR, values have trended upward throughout the two-year period. Because of apparent dilution, the data from this equation alone cannot reliably indicate the status of the mine atmosphere.

Indices of Carbon Monoxide and Carbon Dioxide

Carbon Monoxide

The Index of Carbon Monoxide (ICO), or Graham's index (Graham, 1920), is a dependable indicator of conditions underground, even if there is dilution of the sample. ICO is temperature dependent, i.e. its concentration rises with increasing temperature. The index must have a fresh-air O_2 to inert gas ratio of 0.265. For all equations in this paper, inert is a combination of N_2 and Ar:

$$ICO = (CO \times 100)/[(0.265 \times (N_2 + Ar)) - O_2]$$

where:
CO = carbon monoxide, in pct(0.265 x (N_2 + Ar)) - O_2 = oxygen deficiency, in pct

The ICO can signify the onset of a fire or at least increasing temperatures if, after stabilizing for a time, the value suddenly begins to rise. CO concentrations will rise with increasing temperatures until actual ignition takes place. With combustion, the rate of increase then begins to slow and, in many cases will actually stabilize. Once a mine is sealed and the oxygen available for combustion is reduced, CO levels again begin to climb until temperatures fall to near ambient levels.

Inherent problems exist for the ICO as the CO levels decrease and O_2 concentrations increase. Values in the denominator are forced toward zero when O_2 levels increase, because of the natural O_2-to-inert gas proportion. If CO levels are decreasing at a slower rate, the ICO may actually increase. Fortunately, this error will always be on the conservative side, suggesting that conditions underground are worse than they actually are.

When the mine atmosphere approaches ambient gas levels, other indicators should be monitored in parallel, since very small quantities of CO can be produced by normal oxidation and mineral desorption. This occurs as the sealed area goes from smoldering combustion at slightly elevated temperature to ambient temperature oxidation. During this time, CO values change very little so that, when ambient oxidation is reached, this usually goes unnoticed.

Other difficulties exist with the ICO. There is no single CO value, ratio, or index that can be uniformly applied to all mines. At times, small concentrations of CO continue for extended periods. On the other hand, no CO, a lack of CO does not guarantee that the fire will not reignite once air flows past it (Dougherty, 1969). As in most equations relating to the status of a mine atmosphere, it is important to develop a trend by obtaining a reliable data base of ICO values and noting if successive values follow or deviate from the trend.

Carbon Dioxide

Carbon dioxide (CO_2) gas is a major product of combustion formed underground in a variety of ways. It can be liberated by microorganisms oxidizing the coal. It can also be formed by mixing acidic mine water with calcium carbonate, either in the surrounding strata or in rock dust. Samples may therefore contain more

CO_2 than is actually being generated by combustion. On the other hand, CO_2 is soluble in water, so samples could also contain less CO_2 than is actually being evolved.

A Carbon Dioxide Index (ICO_2) (Mitchell and Burns, 1979) has been developed to monitor elevated CO_2 levels:

$$ICO_2 = [(CO_2 - 0.03) \times 100]/(100 - O_2)$$

This equation indicates elevated CO_2 values in an oxygen-free atmosphere. In an ambient atmosphere it would give the proportion of CO_2 to total inerts. In a mine fire atmosphere, elevated CO_2 would be compared to total inerts and gaseous byproducts of combustion.

If CO_2 is a product of combustion, an ICO_2 versus temperature plot will closely parallel an ICO versus temperature plot. Where no temperature information exists, an ICO/ICO_2 ratio is developed. ICO_2 values increase once ignition occurs and continue to increase as long as the temperature or fuel quantity increases. In sealed areas having no combustion, the ICO_2 value rises, and then stabilizes with time.

Analysis Using the ICO and ICO_2 Index

If combustion is continuing or increasing in intensity, the ICO and ICO_2 indices should increase. Figure 4 showed both the ICO and ICO_2 values to be initially decreasing. The ICO_2 rapidly stabilized while the ICO continued to decrease, then widely fluctuate over the entire sample period. The intake results suggested that combustion was slowing, however it was impossible to determine from these equations alone whether combustion had actually slowed. All values trended toward zero, which can only occur with reduction in the CO concentration and a cessation of elevated temperature oxidation. CO concentrations fell to zero wherever the data appears to be missing on the ICO plot.

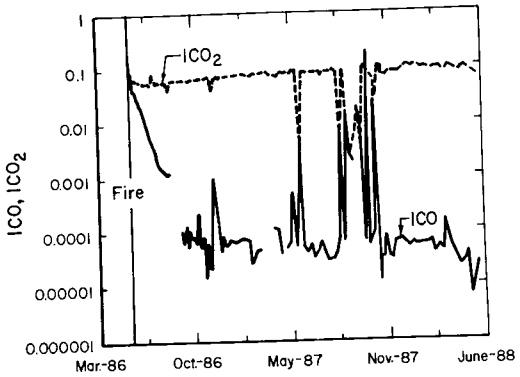


Figure 4.

Is the Atmosphere Explosive?

Explosibility

Once an area is sealed and combustion begins to slow, the gas components of the atmosphere change. It is important to determine whether the atmosphere is explosive, or if it has the potential to become explosive. If combustion restarts and the atmosphere is explosive, an explosion could result. This problem is frequently encountered during mine reopening when oxygen is reintroduced into a sealed mine.

A diagram has been developed (Zabetakis, 1959) to monitor the explosibility of mine atmospheres having a maximum hydrogen (H_2) concentration of 5 pct (50,000 ppm) and a maximum CO concentration of 3 pct (30,000 ppm). The diagram has three major parts: Effective Inerts (EI), Effective Combustibles (EC), and a ratio of methane (CH_4) to total combustibles (R):

$$EI = \text{excess } (N_2 + Ar) + (1.5 \times CO_2)$$

where:

$$\text{excess } N_2 = \text{sample } (N_2 + Ar) - (3.8 \times \text{sample } O_2)$$

$$EC = CH_4 + (1.25 \times H_2) + (0.4 \times CO)$$

(The values 1.25 and 0.4 are the lower explosive limit ratios of CH_4 to H_2 and CO, respectively.)

$$R = CH_4/(CO + CH_4 + H_2 + \text{alkanes} + \text{alkenes} + \text{alkynes})$$

These values can then be diagrammed to determine the potential for explosibilty as shown in Figure 5.

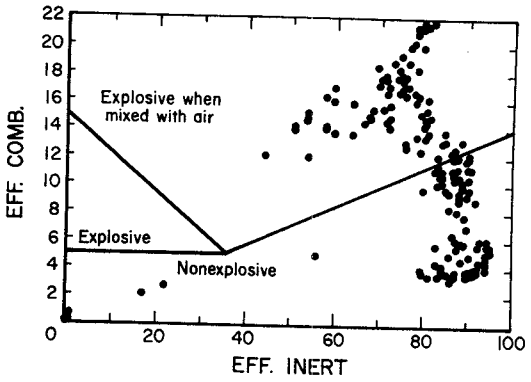


Figure 5.

EI is plotted along the horizontal axis, EC is then found on the vertical axis. The R value varies between 0.0 and 1.0. Data are fitted into one of three areas: nonexplosive, explosive when mixed with air, and explosive. The area into which the sample point fits depends directly on R.

To determine when the atmosphere could become explosive, a graph plotting explosibility (E) versus time is generated:

$$E = EC(O_2/MAO)$$

where:

MAO = Maximum allowable oxygen = $5 + 7R$
$= 5 + 7[CH_4/(CH_4 + H_2 + CO)]$

MAO is the maximum quantity of oxygen required for flame propagation; an O_2 sample below the calculated MAO means that insufficient oxygen is present for ignition and the mixture is either nonexplosive or explosive when mixed with air. If the oxygen concentration exceeds the MAO, the explosibility diagram must be used to determine if the mixture is explosive.

Analysis Using Explosibility Equations

In Figure 6 the intake sampling location appeared to be moving toward a period of increased explosibility. Methane and nitrogen

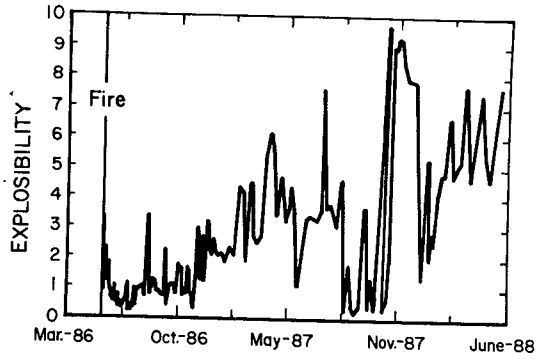


Figure 6.

concentrations steadily increased, while oxygen, carbon monoxide, carbon dioxide, hydrogen, and higher hydrocarbons decreased. The higher methane concentration was the reason for the apparent increase in explosibility.

Is the Fire Still Burning?

Relative Intensity and Relative Efficiency

The Relative Intensity (RI) (Mitchell, 1984) equation is used by power industry engineers to determine boiler and fuel efficiency, oxygen excess or deficiency, water vapor production, and heat losses. In fire analysis it is used as a tool to determine the proximity of the sampling location to the fire; it performs best when many sampling locations, spread throughout the mine, are available. RI measures the quantity of air available to burn a unit mass of fuel, the percentage of oxygen consumed, and the effect of combustion on temperature. The stoichiometric relationship requires 11.6 kg of air to completely burn one kg of coal. Mathematically:

$$RI = [(1 - 3.83\ O_2)/(N_2 + Ar)]ICO$$

where:

O_2 = oxygen, in pct,
$N_2 + Ar$ = nitrogen plus argon, in pct,
ICO = Index of Carbon Monoxide

The Relative Efficiency (RE) (Mitchell, 1986) equation was also originally developed by the power industry. It was derived from basic fuel combustion equations. It is an effective indicator of thermal reactions in the earliest and latest stages, which are of greatest interest immediately after a mine is sealed.

$$RE = [0.262(N_2 + Ar - 3.83\ O_2)]/(CO_2 + CO)$$

Both the RI and RE equations are designed to show the location of the sampling point relative to the fire. If RI or RE increases, the fire is either growing or moving closer to the sampling location. Conversely, if the RI or RE decreases, the fire is either being throttled, or is moving away from the sampling location.

Analysis Using RI and RE Equations

The intake sampling location in Figure 7 exhibited a visible decrease in RI since the beginning of gas analysis, whereas the relative efficiency values showed an increase, followed by a general flattening of the curve.

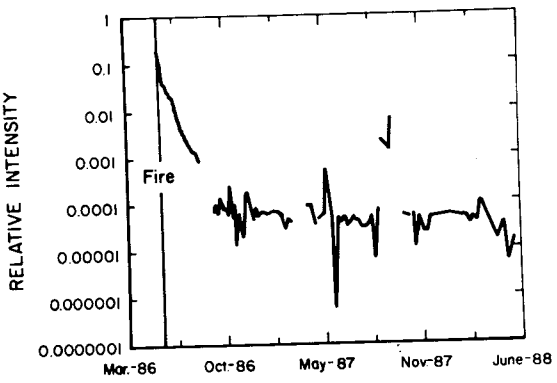


Figure 7.

The RE curve in Figure 8 showed that the oxygen available for combustion was being reduced over time. This lag with respect to the RI curve could show that, because of the pitch of the mine, the samples continued to have gaseous fire byproducts. In other words, there may have been a higher oxygen concentration that was continuously being consumed by combustion somewhere deep within the

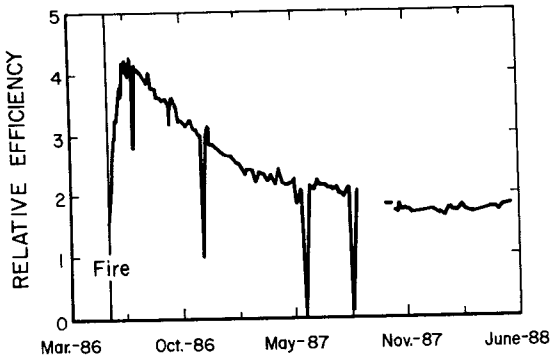


Figure 8.

mine. The intake sample location was at one of the highest elevations, simulating a partially capped exhaust stack that was slowly emitting fire byproducts. The only way to prove this theory would be to 1) sink a borehole well inby any potential fire areas, 2) begin obtaining samples, and 3) determine if combustion was still occurring.

Hydrocarbon Ratio

The Hydrocarbon Ratio (R1) (Justin and Kim, 1988) was developed after researchers (Kim, 1974, 1978, 1986) showed that the desorption of low molecular weight hydrocarbon gases from coal was directly related to temperature. Primarily methane is liberated at ambient temperatures. If coal is heated, other hydrocarbons begin to be desorbed. In effect, all hydrocarbons increase with rising temperatures.

Knowing this direct relationship between hydrocarbon concentrations and temperature, the Hydrocarbon Ratio equation was developed. It is defined as:

$$R1 = [(1.01[THC] - CH_4)/([THC] + c)](1000)$$

where:
THC = total hydrocarbon concentration, in ppm
CH_4 = methane concentration, in ppm
c = constant, 0.01 ppm

This equation was designed to equal zero when no hydrocarbons are detected, 10 when methane is the only measured hydrocarbon and, 100 at the upper limit. The only limiting factor is that the methane concentration must be greater than 20 ppm.

For bituminous coal, the R1 values obtained are closely dependent on temperature. Because of this, limits have been derived to aid in determining the temperature of coal being analyzed. R1 values from zero to 50 indicate that only normal temperature oxidation is occurring. Values from 50 to 100 indicate possible elevated temperatures. Above 100, the coal is undergoing elevated temperature oxidation. This equation, like all others, is designed to be examined over time, so that trends can be established.

Analysis Using the Hydrocarbon Ratio

R1 values at the intake sampling position shown in Figure 9 rapidly fall from the elevated oxidation range to normal temperature oxidation values. A large deviation that

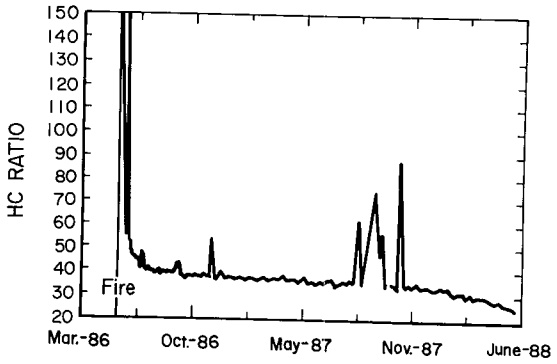


Figure 9.

occurred not long after the fire was caused by a large increase in measured total hydrocarbons. Seasonal changes in the sealed mine atmosphere appear significant enough to drive the R1 values back into the elevated temperature oxidation range.

Is There Potential for Reignition?

Litton RATIO

The Litton RATIO (Litton, 1986) is used to monitor a sealed mine atmosphere. It was designed to eliminate the potential for reignition of a mine fire when the mine is reopened and oxygen is reintroduced. Only four gases are measured to successfully obtain results: oxygen, carbon monoxide, methane, and ethane. The atmosphere is divided into three parts: air, methane and ethane, and residual gas.

$$Air = a(O_2)$$

where:

$$a = 100.0/20.946 = 4.774$$

Residual gas (Rg) is the volume percent of gas within a sealed coal mine after air, methane, and ethane have been subtracted from the sample. Rg contains excess nitrogen (not part of air), products of combustion (CO, CO_2, H_2, and hydrocarbons), along with products formed by chemical processes not related to combustion.

$$Rg = 100.0 - Air - CH_4 - C_2H_6$$

The concentration of CO in ppm of air within the residual gas is:

$$(CO)_{Rg} = [(CO)_S/Rg] \times 100$$

where:

$(CO)_S$ = sampled CO, in ppm

From these values the R Index (R_I) is developed:

$$R_I = (CO)_{Rg}/Rg = [(CO)_S/Rg^2] \times 100.0$$

If R_I remains constant, one of two things is occurring: sustained low-temperature combustion or a sustained low rate of ambient CO production. If combustion is taking place, the average R_I value will be higher than if ambient CO production is occurring, because CO production increases as the temperature increases.

In deriving a ratio, the actual measured R_I is referred to as $(R_I)_{act}$. A second R_I, known as the maximum equilibrium value $(R_I)_{eq}$, is also obtained:

$$(R_I)_{eq} = 300(Rg/O_2)^{-1/2}$$

$(R_I)_{eq}$ represents the maximum equilibrium value of the R Index for a gas sample if only ambient temperature processes are responsible for CO production.

An equation, referred to as RATIO has been created to determine the state of a sealed area:

$$RATIO = (R_I)_{act}/(R_I)_{eq} = 1/3(CO)_S(Rg)^{-3/2}(O_2)^{-1/2}$$

If RATIO is greater than one, ambient temperatures have not been reached. There is either smoldering combustion or above ambient temperature oxidation. This condition is considered unsafe for reentry.

If RATIO is less than or equal to one, it is possible that ambient temperatures have been reached. However, equilibrium exists only if RATIO stabilizes at a value less than one. The minimum time for the value to remain stabilized at less than one has been established at 30 days. If the value remains below one for a time and then rises to a value greater than one, the 30 day time period must begin anew.

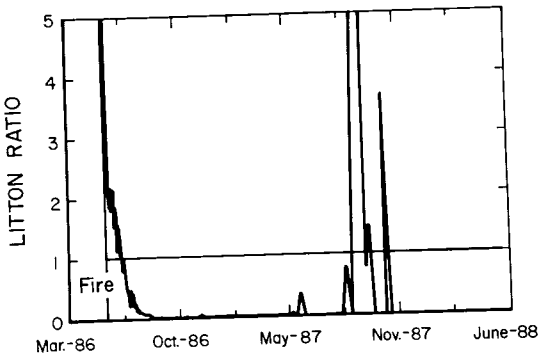


Figure 10.

Analysis Using Litton RATIO

In Figure 10 intake combustion appeared to be slowing. Values fell to less than one and stabilized. Variations in the long term stability were caused by the effects of seasonal changes (barometric pressure fluctuation, temperature variation, etc.) on the mine atmosphere.

DISCUSSION

A fire of unknown origin occurred in a coal mine in southwestern Colorado. After successfully combating the fire for a time, control was lost and the mine had to be sealed. The Bureau was requested to monitor and analyze the atmosphere in the mine. Several different atmospheric and combustion gases were monitored. In addition, the status of the mine has been continuously followed by applying equations developed to analyze various post sealing mine problems. This report details information from the first two years of gas sampling following the sealing of the mine. There were six sampling locations but, since all locations generally presented the same information, only the intake is discussed. The results over a two-year period indicated that seasonal changes may influence the status of the atmosphere. Overall, combustion has abated and the atmosphere appears stable. The results of two years of data show that individual values of any equation are worthless, trends must be developed to obtain a realistic picture of the sealed atmosphere, and no one equation can provide all the answers regarding the sealed mine atmosphere.

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ANALYSE A LONG TERME DE L'ATMOSPHERE D'UNE MINE OBTUREE APRES UN INCENDIE

Robert L. Derick, Robert J. Timko, Edwàrd D. Thimons

Au début de juin 1986, un incendie souterrain d'origine inconnue s'esr produit dans l'une des entrées d'une mine de charbon située dans le sud-ouest du Colorado. Pendant un certain temps, le personnel de la mine combattit activement l'incendie mais, en raison du manque de mousse d'extinction, il en perdit le contrôle. Toutes les entrées et tous les puits de la mine furent obturés pour empêcher d'autres dégâts et diminuer les risques d'explosion.

L'exploitant de la mine demanda l'assistance du Bureau des Mines pour obtenir et analyser les gaz d'incendie de mine récupérés aux endroits de prélèvements qui se situaient dans les obturations et dans les trous de forage percés après l'incendie. Ce rapport donne le détail des activités qui eurent lieu les deux premières années après l'obturation de la mine: description des gaz analysés, équations de l'état de l'atmosphère de la mine obtenues par l'analyse de ces gaz, première repénétration dans la mine obturée et évaluation continue pendant de nombreuses heures. L'analyse des gaz, effectuée par chromatographie, comprenait celle des gaz atmosphériques standard ainsi que celle des hydrates de carbone plus élevés présents dans les atmosphères post-incendies. Neuf équations ont permis d'établir ce qui se produisait à l'intérieur de la mine obturée. Une description de ce qui eut lieu lors de la repénétration initiale relate ce que les premières équipes de sauvetage autorisées à descendre dans la mine ont trouvé. Enfin, une évaluation continue sur 60 heures a permis de déterminer les effets à court terme des fluctuations de la pression barométrique sur les gaz obturés de la mine.

LANGZEITANALYSE VON GRUBENWETTERN IN EINER VERSIEGELTEN GRUBE NACH EINEM FEUER.

Robert L. Derick, Robert J. Timko

Anfang Juni 1986 brach in einem der zahlreichen Zugänge einer Grube im südwestlichen Kolorado ein Feuer unbekannter Ursache aus. Versuche des Grubenpersonals, das Feuer zu bekämpfen, scheiterten wegen Mangels an Löschschaum, und das Feuer geriet außer Kontrolle. Um weiteren Schaden zu vermeiden und die Möglichkeit einer Explosion zu verringern, wurden alle Öffnungen und Schächte versiegelt.

Der Betreiber bat das Bureau of Mines um Unterstützung bei der Gewinnung und Analyse von Verbrennungsgasen, die den durch Grubenverschlüsse und durch nach dem Feuer gebohrte Bohrlöcher eingesetzten Probesystemen entnommen wurden. Der Vortrag gibt eine Beschreibung der Aktivitäten in den ersten zwei Jahren nach der Verschließung der Mine. Der Bericht enthält eine Beschreibung der analysierten Gase, sowie Errechnungen der Atmosphäre in der Grube, das erstmalige Wiederbetreten der geschlossenen Grube und mehrstündige Dauer-Bewertungen. Die Gase wurden mit Gaschromatographie analysiert, dabei zeigten sich die üblichen Gase der Atmosphäre, sowie höhere Kohlenwasserstoffe, die in Nachbrandatmosphären gewöhnlich auftreten. Neun Gleichungen wurden zur Beurteilung der Vorgänge in der verschlossenen Grube angewandt. Der Bericht gibt eine Beschreibung dessen, was die ersten Grubenwehrmannschaften bei dem erstmaligen Wiederbetreten der Grube vorfanden. Eine 60-stündige Dauer-Bewertung ermittelte die Kurzzeiteffekte von barometrischen Druckänderungen auf eine versiegelte Grube.

ДОЛГОСРОЧНЫЙ АНАЛИЗ АТМОСФЕРЫ В ЗАКУПОРЕННОЙ ШАХТЕ ПОСЛЕ ПОЖАРА

Роберт Л. Дерик, Роберт Дж. Тимко и Эдвард Д. Тимонс

В начале июня 1986 года подземный пожар неизвестного происхождения возник в одном из нарезных штреков угольной шахты на юго-западе штата Колорадо. Работники шахты в течение некоторого времени энергично боролись с огнем, но из-за того, что у них не было достаточного количества противопожарной пены, борьба была проиграна. Все входы в штольни и стволы шахты были герметически закупорены, чтобы предотвратить дальнейшее разрушение и свести к минимуму возможность взрыва.

Компания, производившая добычу угля из этой шахты, запросила у Горного Бюро США помощи в опробовании и анализе газов. Пробы были взяты через отверстия, проделанные в герметических щитах, а также из скважин, пробуренных после пожара. Настоящий доклад описывает события, происшедшие в течение первых двух лет после герметизации шахты. Доклад включает описание проанализированных газов, уравнения состояния атмосферы в шахте, выведенные из анализированных газов, первый вход в закупоренную шахту и многочасовую непрерывную оценку. Газы послепожарной атмосферы были проанализированы хроматографическим методом, который показал наличие стандартных атмосферных газов и углеводородов более высокого порядка. Было использовано девять уравнений для оценки ситуации, имевшей место внутри закупоренной шахты. В деталях описывается все, что произошло в процессе первого захода и что было найдено первой спасательной командой. И, наконец, 60-часовой непрерывной оценкой было определено немедленное влияние изменений барометрического давления на газы в закупоренной шахте.

封闭煤矿火灾后空气的长期分析

罗伯特 L．德立克

罗伯特 J．丁姆柯 及 庆德华 D．铁门士

１９８６年６月初，科罗拉多西南部煤矿里其中有一个巷道发生了不知原因的地下火灾．煤矿人员积极地救火，但因缺少灭火泡末，未能熄灭．为了防止更多的损害和可能的爆炸，他们已把所有洞口及井口全部封闭．

矿上操作人员请求矿业局的协助，从取样孔及火灾后往下钻的孔中取得并分析了矿内火灾气体．本文将叙述封矿后两年内的工作．它包括气体分析报告，从分析出气体推导的矿内气态方程，封矿初次复进的情况，和连续很多小时的测定．气体用色层谱法分析，包括标准大气和火灾后大气里发现的较高炭氢化合物．本文并叙述首批救护队初次复进矿里所见到的详情．最后叙述６０小时连续的测定决定了气压起伏对封闭矿内气体短时间的影响．

LOCATION OF BURIED MINERS--NEW SYSTEMS

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In the seventies already electromagnetic systems were developed to find buried miners within short time.

In the Tremonia Experimental Mine systems were tested that had been developed in USA, Poland, Austria and the FRG. The test results showed that only systems using extremely low frequencies for the miners transmitters were successful, so that the location measurement being conducted in the near field: Only there inhomogenities of the environment are negligible.

The functional principle is described by means of the Austrian system SERLES: Every miner has a low-frequency transmitter powered pulsatively (to save energy) by his cap lamp battery. A highly sensitive narrow band receiver is used to measure field direction and field strength in three or more points of reception. The built in computer therefrom evaluates the miner's location. The location accuracy for the SERLES system is 5% for distances up to 30 m.

INTRODUCTION

Efforts have been made for several decades to find fast, efficient methods of locating buried miners. The reasons are first and foremost the hope of rescuing them, and also to show the rescue workers - who often have to work under extremely hazardous conditions - the shortest and safest way to the buried miners and, last but by no means least, to give relatives and the general public prompt, accurate information on the extent of any accident.

One such system concept is to provide each miner on his descent to the unterground mine with an electromagnetic transmitter or transponder of his own that automatically transmits signals that can be picked up by an appropriate locating receiver. Prototypes were developed in Czechoslovakia and in Poland in the 1960s. These were small transmitters powered by the cap lamp battery. They used a long wave frequency already in use in the mining industry in the range of approx. 400 kHz. The transmitter developed by the Barbara Experimental Mine in Poland was put into application at the time throughout the Polish mining industry and was responsible in many cases, according to the rescue headquarters in Bytom, for buried miners being located and in some instances rescued.

In the USA, initial efforts have been concentrated on the task of locating trapped miners who are still able to activate their transmitters. Reports on experiments with shortwave transmitters and transponders operating on approx. 25 MHz to help locate buried miners too were published in the early 1980s (Webb 1984).

The Polish transmitter underwent thorough testing at the Tremonia Experimental Mine in Dortmund in the mid-1970s. Its field characteristic was found to be

severely assymetric, which led to considerable deviations of up to about 80 degrees, so that it was considered unsuitable for general application in the West German mining industry.

RECENT DEVELOPMENTS

This problem was naturally recognized in Poland too, so that new theory-based fundamental investigations were initiated; these then resulted in a new system (Lozak 1980, 1986). The vital realization was that it is the wavelength of electromagnetic radiation that determines whether the direction to a transmitter can be specified precisely enough and the intervening material such as debris, solid rock, timbering and walling, moisture or cavities does not cause any distortion of the direction or distance.

It was found that only extremely long waves with frequencies of less than approx. 10 kHz may be selected if good results are to be achieved. The locating receiver is then in the utmost near field of the transmitter-generated electromagnetic field whose magnetic components permit precise location. This was one of the factors that resulted in development of the new GLON-LOP system in Poland.

A number of mining accidents in the Federal Republic of Germany had meanwhile led to a research programme being established with the financial backing of the German mining authority, and involving the Tremonia Experimental Mine company in Dortmund too, to find systems of locating buried miners. The objective was to test all systems currently available for their suitability for application in the German underground mining industry. Both well-established systems, including avalanche rescue devices, and the latest developments - some of which were publicized or initiated only during the investigations - were covered by the tests. Besides the USA projects, the following new systems were available:

1. the GLON-LOP system already undergoing underground testing in Poland,
2. the SERLES system developed in Austria at the University of Innsbruck and
3. the POS 102 system that had been embarked upon only recently by the Ruhrkohle AG mining company in Germany.

Each of these three systems had its own locating procedure, detailed below.

In 1986 staff at the Tremonia Experimental Mine were able to satisfy themselves of the efficiency of the new Polish system by performing tests of their own below ground in Poland. This testing was then continued at Tremonia.

TEST AREA

An area on the third level was selected as one of the test locations and in part specially set up. A plan of part of the level (Fig. 1) gives some idea of the situation there: all roadways are reinforced with steel arches and steel plate lagging. Roadway I was filled in for a distance of 20 m with debris into which 5 plastic pipes were let in order to place the test piece; these were distributed across the roadway section, running parallel along their entire length. The existing track was not removed, so that a long, possibly interfering waveguide was reproduced. The steel lagging plates of the roadway reinforcement were removed in every fifth field to provide sections that were electrically independent of each other.

In roadways II, III and IV there are also tracks and in addition supply lines for electric power, for water and for compressed air. In roadway IV there is moreover a large-format power distributor. Roadway V is partly filled in with debris. In this way the influence of the surroundings such as rock,

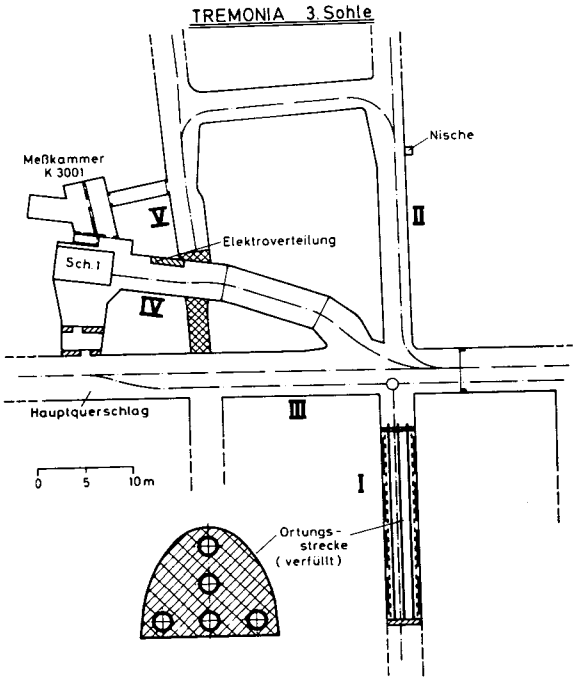


Fig. 1: Test area on 3rd level

cavities, debris, cables and supply lines, reinforcements, steel plate lagging, moisture and power distributor on locating precision among other things, i.e. on distance and direction, can be determined.

PRELIMINARY INVESTIGATIONS

Attenuation or penetration measurements through debris were first performed, using all frequencies of the equipment included in the preliminary considerations, from 9 kHz to 25 MHz and 425 MHz. The working range appeared adequate with all frequencies up to 25 MHz; only the REDAR protective transponder system used in the German mining industry has a range of less than 1 m in debris with a frequency of 425 MHz (Richtscheid 1982).

When determining direction, the previously recorded site error of 50 degrees or more with frequencies of, for example, 384 kHz was verified both above ground and in the test sections underground.

THE POLISH GLON-LOP SYSTEM

This system consists of the GLON transmitter that is installed in the battery housing of the cap lamp (Fig. 2) and the LOP locating device with two different aerials (Fig. 3). Its PAM magnetic measuring antenna is used for direction finding and the relatively insensitive "locating probe" antenna is suitable in particular for close-range locating in a circumference of 1 to 2 m.

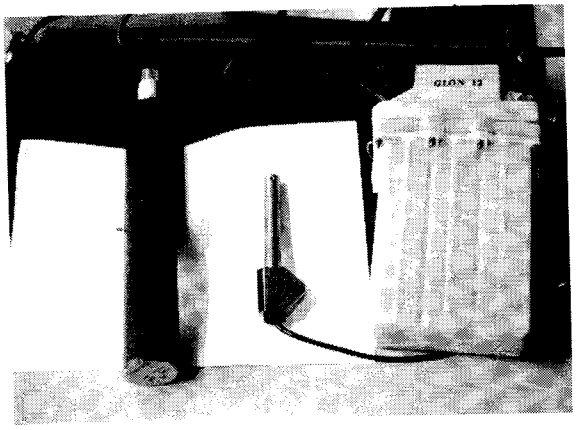


Fig. 2: GLON transmitter and locating antenna

The LOP locating receiver is a small, intrinsically safe, battery-powered unit (Fig. 3). The transmission frequencies used are distributed on 42 bands between 3925 and 5975 Hz.

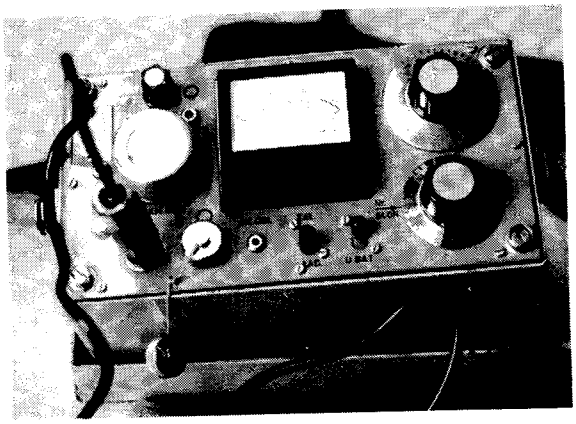


Fig. 3: LOP locating unit

The mode of operation is yielded by the Maxwell equations, that are reduced for amplitude H of the magnetic field component in the near field when aerials are aligned parallel to each other to the simple formula:

$$H = \frac{1}{4\pi} \times \frac{m}{r^3} \qquad (1)$$

with m being the magnetic moment of the transmitter and r its distance from the receiver. With a frequency of less than 10 kHz, the near field in underground hard coal mining has a range of up to approx. 20 m.

Under these circumstances a receiver can be scaled direct in distance meters when the transmitter strength is known. This results in "locating method I", in that the respective distances to a transmitter are determined from various locations in front of a caved-in area and the transmitter location can be specified from this data by triangulation.

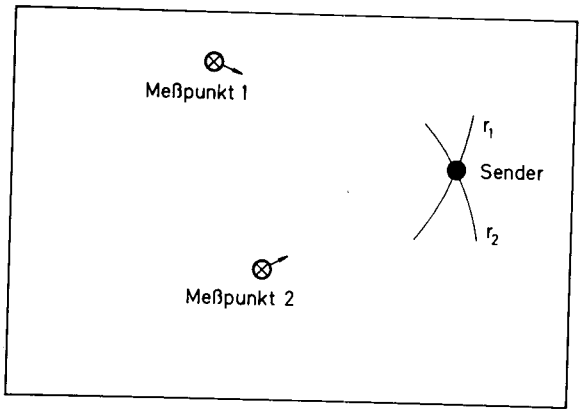


Fig. 4: GLON-LOP locating method I

If the transmission field strength has weakened, e.g. due to damage or excessive use, locating method II is of further assistance. This is a gradient method that exploits the typical drop in field strength depending on distance, with the direction to the transmitter being determined from a selected fixed point and the field strength in relation to the 1st point being read off on a scale at a 2nd point a fixed distance away on this direction line. The distance required is then taken direct from a further scale.

Initial experiments were performed during the visit to Poland, followed by more in the test area at Tremonia (Fig. 1). In each instance, both locating methods were used and compared with each other. The experiments were first performed in locating roadway I, where conditions were very easy to survey and the specified locations could be easily checked and remeasured. The position of the transmitter within the roadway section or its distance from the support material was found to be insignificant. At a distance of 15 m, the maximum deviation was 0.5 m.

To test the efficiency of the system under other, preferably unfavourable conditions too, roadways II to V were also used. Fig. 5 shows some selected tests. The starting point of the locating efforts in these cases was always roadway III. The location of the transmitter was not known in advance to those searching.

In case A, the transmitter was behind a steel lagging plate. As shown in Fig. 5, the measurements at the different points in roadway II indicated the location of the transmitter with marked precision. The locating probe then led the search team in roadway IV direct to the transmitter that was concealed behind the steel lagging plate A.

In case B, the transmitter was in roadway V on the floor beside a support arch B. Between the starting point of the search and the transmitter there were rock, cavities, debris, several courses of lagging plates and the power distributor. From point B_S, the rough direction was (correctly) determined, and furthermore a distance of somewhat more than 20 m. The search team then moved into

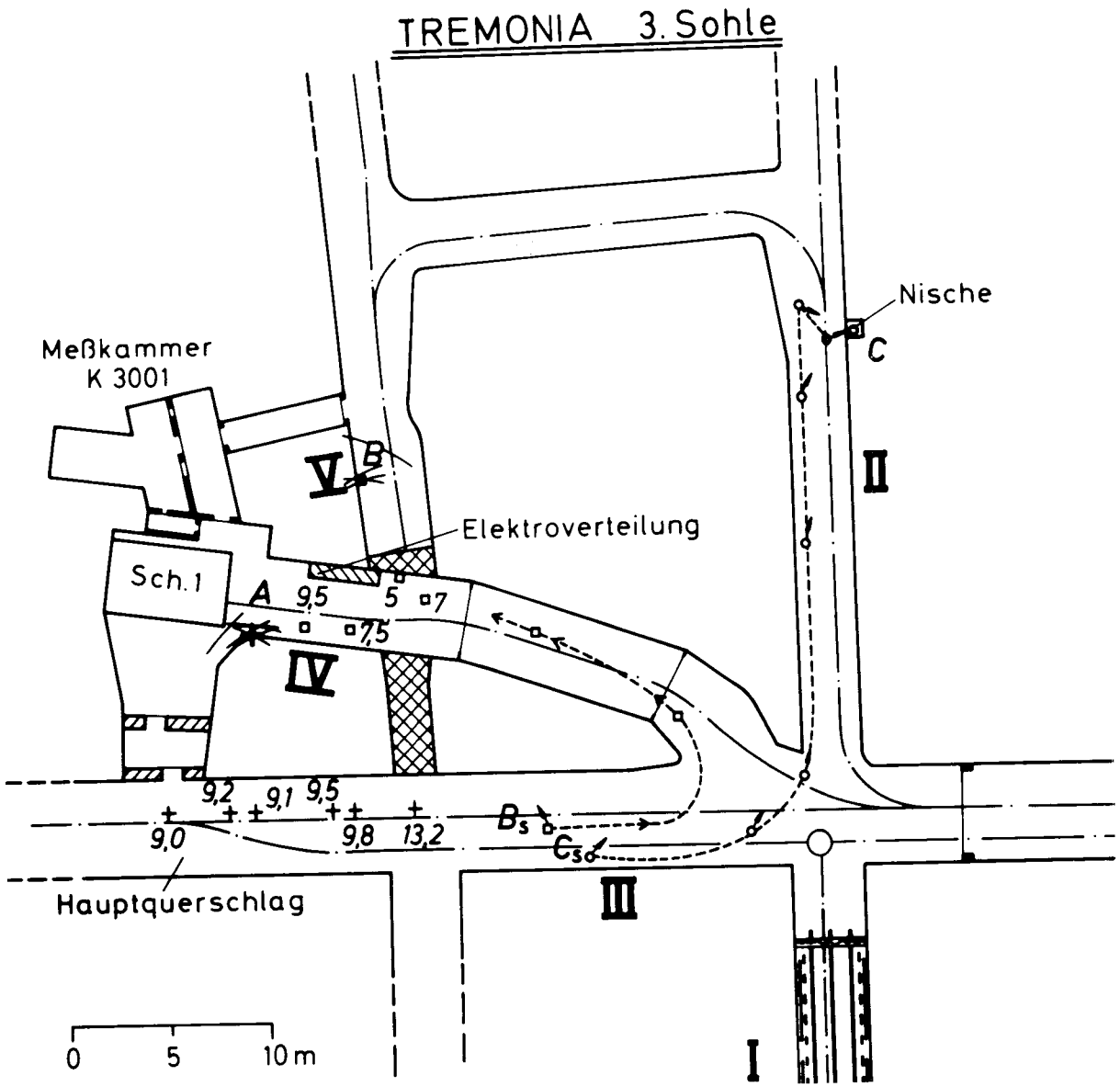


Fig. 5: GLON-LOP locating test (Numerical data in meters)

roadway IV as close as possible to the transmitter, constantly observing the indicator on the locating device. From there they were able to determine the exact location even through the power distributor.

In case C, the transmitter was in roadway II, in a concreted recess behind a steel door C at a distance of approx. 30 m from the starting point C_s. Once again, the search team determined the direction correctly and ascertained a distance of more than 20 m. They therefore suspected the transmitter to be in roadway II. They moved purposefully along the roadway, once again observing their locating device constantly, until they were close to the recess. Their locating probe antenna then led them direct to the steel door positioned at head height, behind which they found the transmitter.

THE POS 102 GERMAN SYSTEM

The POS 102 system currently being developed by the Ruhrkohle AG (Ruhrkohle 1986) works as follows:

The locating unit calls with a transmission frequency of 9 kHz a PKB 1 transmitter on the missing miner, so that this responds with a frequency of 444 kHz. This transmitter on the miner's body has its own small lithium battery that is used only when transmission is activated. The entire PKB 1 transmitter module is so small that it can be



Fig. 6: POS RAG locating system 102/PKB 1

accommodated in the heel of a miner's boot (Fig. 6).

It can furthermore emit a personal identification, using a coded signal, and on the other hand be used as a protective transmitter to shut off hazardous equipment when a person is in close proximity. Precise details of its operating method have yet to be released. In initial tests at the Tremonia Experimental Mine in locating roadway I (Fig. 1), a distance of only just under 5 m was attained. Subsequent to improvements, 7 to 8 m are now claimed to be possible. Fitting one transmitter in each of the miner's boots is favoured because boots are assumed to be less susceptible to destruction or loss in an underground accident than lamp batteries. Boots with PKB 1 modules vulcanized into the heels are currently being worn for test purposes at one pit.

THE AUSTRIAN SYSTEM SERLES - THEORETICAL BASIS

As stated above through the earth propagation and communication by means of electromagnetic signals is only possible in the VLF and ELF range. Since the wavelength in this frequency range is several km only magnetic antennas are appliable for transmitters and receivers. Unfortunately the radiation pattern of a magnetic dipole antenna (ferrite rod or loop antenna) in the near field range is not a sphere but has a more complex shape. Furthermore the conductivity of rock affects the field distribution the more, the higher the conductivity, the signal frequency and the distance from the antenna dipole. The field components of a magnetic dipole in a spherical coordinate system can be calculated from Maxwell's equations as follows.

$$Hr=m/2\pi\ \cos\theta/r^3(1+ikr)e^{-ikr} \quad (1)$$

$$H\theta=m/4\pi\sin\theta/r^3(1+ikr+k^2r^2)e^{-ikr} \quad (2)$$

$$H\phi = 0 \quad (3)$$

with $m = n.I.A$ .. magnetic moment of source dipole and

$$k^2 = -i\ \omega\ \mu\ \sigma + \omega^2\ \epsilon\ \mu \quad (4)$$

Since k contains all the medium parameters it is possible to chose the signal frequency low enough to fulfill the condition $kr \ll 1$. In this case the amount of field strength reads

$$H = m/4\pi\ 1/r^3\sqrt{1 + 3\ \cos^2\theta} \quad (5)$$

The field strength in the axis of the dipole $\theta = 0^\circ$ is twice as high as the field at $\theta = 90^\circ$, resulting in a source antenna pattern as shown in Fig. 7. The field direction is described by

$$\tan\psi = Ho/Hr = 1/2\ \tan\theta \quad (6)$$

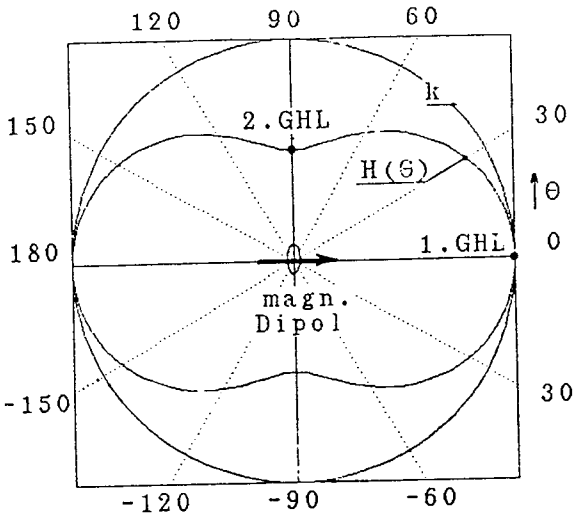


Fig. 7: Source antenna pattern

The dependence of fieldstrength versus distance or the spatial distribution of field lines or both features can form the basis of a location method. The first feature is used with the Polish system GLON-LOP which implies a worst case error in distance of 26% for undisturbed conditions because of neglecting the angular term in equ. 5. The second feature was used for a location method which is independent of the source strength (Nessler 1984 and 1988) but for larger distances is rather sensitive to field distribution by the inhomogeneous enviroment. A combination of both features, field strength and field direction, forms the basis of the SERLES method avoiding the above mentioned drawbacks (Nessler 1989).

LOCATION METHOD

For the application of this basic features as location method it is necessary to recalculate the antenna pattern for a fixed receiving antenna while the source dipole is moved around in such a way, that field strength and field direction at the receiving location remains constant. The resulting locus for the source position is described by

$$r = \text{const.} \sqrt[3]{2/(H \sqrt{1 + 3.\sin^2\phi})} \quad (7)$$

and is shown in Fig. 8 with H indicating the direction of the receiving antenna. SA represents one position and the corresponding direction of the source antenna dipole.

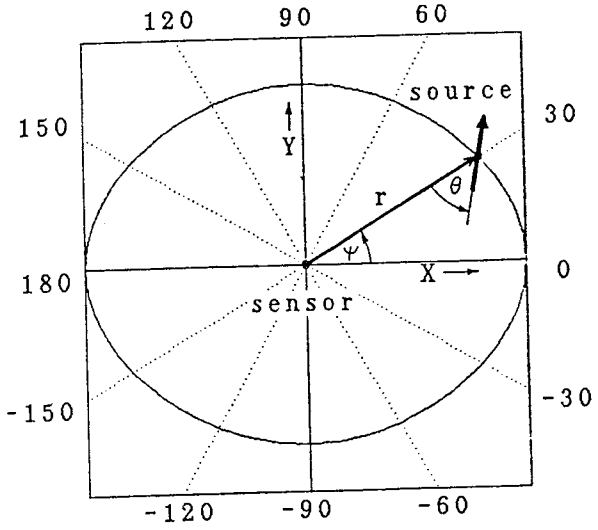


Fig. 8 : Locus of posible source locations for "fixed receiver"

This locus can be used in two different ways to solve location problems: 1) determination of unknown source antenna position and 2) determination of unknown receiver location.

UNKNOWN SOURCE POSITION

The source antenna consists of a single magnetic dipole antenna and is assumed to be located at an unknown position with unknown orientation. To find the source position it is necessary to measure the field amplitude and field direction emitted by the source in at least two properly spaced points of reception. For any point of reception therefore a locus according to equ. 7 can be calculated assuming that the dipole strength of the source is known. Two such locus curves intersect in two points which indicate two possible source locations (Fig. 9). A third locus curve, drawn from the

measured values in a third point of reception, eliminates the ambiguity.

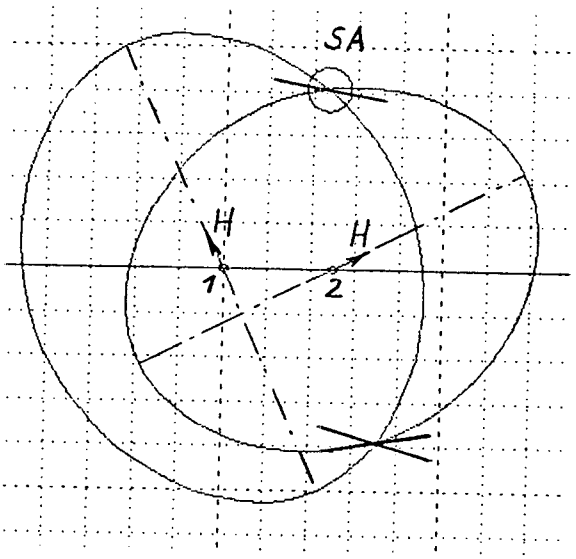


Fig. 9 : Two locus curves plus source direction determine the unknown source location.

In case the source dipole strength is not known sufficiently accurate a common zoom factor for the locus curves remains unknown. This unkown factor can be determined either by an increased number of measurements and variation of the zoom factor for best results or using the direction information according to equ. 6 as a second parameter. Since the source antenna is assumed to stay in its position during measurement, the source direction calculated for the "estimated" location must be the same for any locus.

The position coordinates of any receiver location are essential inputs for the calculation. For a quick measurement a straight line with equidistant points is marked forming the basic coordinate system. This method was used for the test measurements in the Experimental Mine Tremonia. Since this is not applicable everywhere a similar location technique was developed for automatic location of the receiving position using an auxiliary source antenna system.

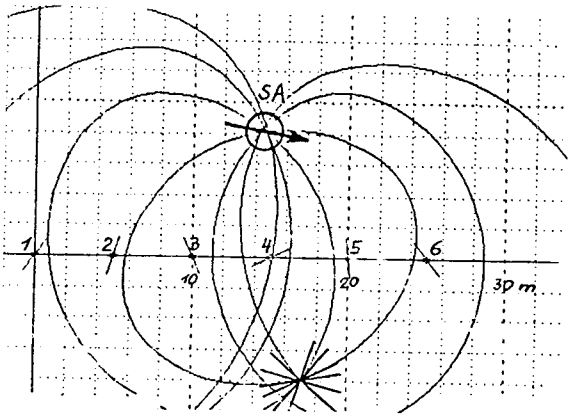


Fig. 10 : Location of a transmitter with unknown dipole strength. Result obtained by variation of zoom factor.

UNKNOWN RECEIVER POSITION

The inverse problem of that one described above is the determination of the receiver position relative to the known (auxiliary) source location and source orientation. In this case both, the source and the receiver remain at their respective positions. The movement of the receiver in this case is replaced by three source antenna orientations (Nessler 1987). Field strength and field direction at the receiver location for any of the source orientation determine one locus each. The locus curves are concentrically, but differ in orientation and size as shown in Fig. 11.

The source dipole strength must be known in this case. The common intersection of three such locus curves indicate two possible source locations. Because of the symmetry of the problem the ambiguity cannot be solved by increasing the number of source orientations.

In most applications physical constraints unable do distinguish, which position is the correct one. If this is not applicable, the

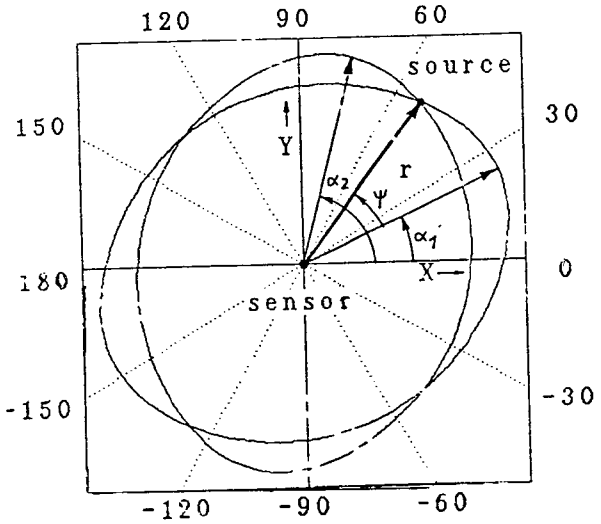


Fig. 11 : Two concentrical locus curves determine four possible SA locations.

strong distance dependence of field strength ($1/r^3$) is used as an indicator: moving the receiver towards the estimated source position must be accomplished with an increase of field strength amount.

TEST MEASUREMENTS WITH THE AUSTRIAN SERLES SYSTEM

Test measurements with the SERLES system were performed at the same site as with the Polish GLON-LOP system. For test 1 the transmitter was behind a steel lagging plate. For the search team only roadway III was assumed to be accessible thus a straight baseline with points every fife meters were marked (see Fig. 12, MP1..10). Measurements of field strength and fields direction in each of these points were taken semiautomatically measuring three orthogonal field components and a phase information. Already from these measurements it was clear that the transmitter was next to MP6 and MP7. The graphic evaluation on a portable computer proved this first result. Intersections of all the loci was within 0.8m of the true position (cf. Fig. 12). Using only measurements with highest field strengths (smallest loci, MP5,6,and 7) the accuracy was a few cm(!).

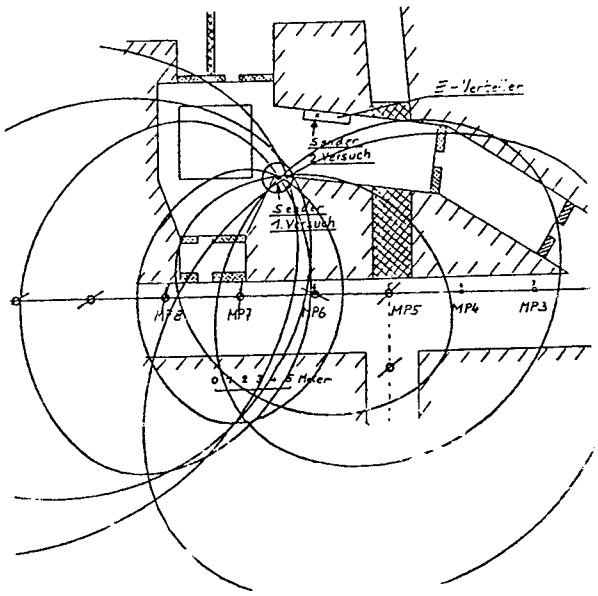


Fig. 12 : Result of test 1

For test 2 the transmitter was hidden between heavily armed electric power switches, additionally the transmitter power was reduced unknown to the search team. The measurement was performed in the same way as described in case 1, the graphic evaluation showed a rather distributed field if intersections, the center of gravity, however, coincided with the true source position (Fig. 13).

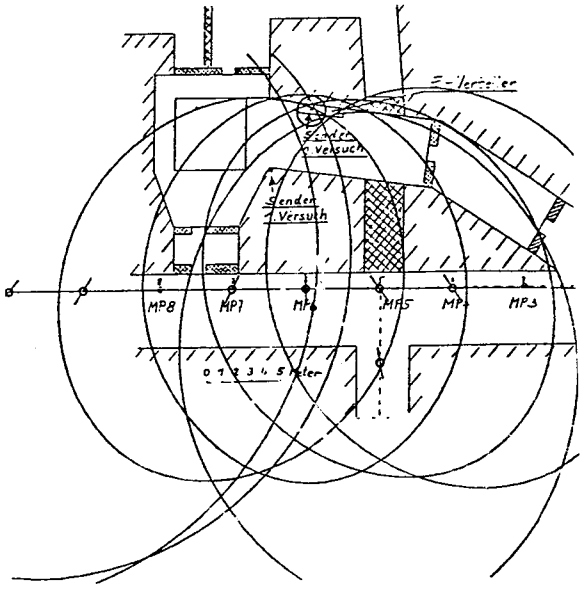


Fig. 13 : Result of test 2

Using the coinciding source antenna direction additionally, measurements in MP5, 6 and 7 gave best results. Calculated position was less than 50 cm to the true position.

CONCLUDING REMARKS

All three systems described - GLON-LOP, RAG-POS and SERLES - are capable of locating with marked precision persons wearing a corresponding transmitter. They differ, in part severely, in their range. The fact that the Polish system is able to discern concealed transmitters according to direction from as far away as approx. 50 m and to specify the location with great precision from distances below 20 m, and that the SERLES system can even surpass that range, enables mine rescue brigades to clear the way fast with high-power clearing equipment until they are close to trapped or buried miners and then to clear the last few meters carefully by hand, with constant reference to the locating unit. The German systems offers considerably lower detection distances of less than 10 m. Fairly precise advance information on the possible location of a trapped or buried miner is therefore essential before the range in which his transmitter can be detected can be reached. There are, however, advantages to be gained from further possible applications, e.g. as a protective transmitter. In particular, a transmitter accommodated in a miner's boot is assumed to be less susceptible to destruction or loss and, being independent of any power supply, should be more reliable.

The locating unit of the Polish and of the German system is worn by a rescue worker on his chest. He can then move quickly through the mine with the locating antenna in his hand and watch for reactions by the unit. The locating unit of the Austrian system consists of several small devices that can be set up quickly at preset distances; they are connected with cabels and then detect and specify the position of a locating transmitter by themselves under computer control. A change of position presents no problem either in this case.

No exact details of prices are available as yet, but they should not be much above DM 10.- for the transmitter with which each miner ought to be equipped. On the other hand, relatively few examples of the considerably more expensive locating units are needed.

A Polish locating unit with ten cap-lamp-mounted transmitters is currently at the disposal of the Tremonia Experimental Mine, so that further investigations including testing at active pits in the German underground mining industry can be undertaken.

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ORTUNG VERSCHÜTTETER BERGLEUTE - NEUE SYSTEME

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ZUSAMMENFASSUNG

Schon in den 70er Jahren wurden elektromagnetische Systeme entwickelt, um verschüttete Bergleute schnell aufzufinden.

Auf der Versuchsgrube Tremonia in Dortmund, Bundesrepublik Deutschland, wurden jetzt Systeme untersucht, die in den letzten Jahren in den USA, Polen, Österreich und Deutschland entwickelt worden sind. Die Testergebnisse zeigten, daß nur solche Systeme erfolgreich waren, die extrem niedrige Frequenzen für den am Mann getragenen Sender benutzten, so daß die Suchmessungen im Bereich des Nahfeldes stattfinden. Nur unter diesen Bedingungen sind die Inhomogenitäten des Umfeldes vernachlässigbar.

Das Funktionsprinzip solcher Systeme wird am Beispiel des österreichischen Systems SERLES beschrieben: Gekoppelt mit seiner Lampenbatterie führt jeder Bergmann einen Niederfrequenzsender bei sich, der zur Energieersparnis gepulst betrieben wird. Geortet wird mit Hilfe eines hochempfindlichen Schmalband-Empfängers, der Feldrichtung und -stärke an wenigstens vier Positionen mißt und registriert. Daraus berechnet der eingebaute Rechner automatisch den Aufenthaltsort des gesuchten Bergmannes. Die Genauigkeit beträgt ca. 5 % bei einer Distanz von ca. 30 m.

EINFÜHRUNG

Schon seit Jahrzehnten wird nach Methoden gesucht, verschüttete (buried) Bergleute zuverlässig und schnell zu lokalisieren. Gründe dafür sind in erster Linie die Hoffnung, Verunglückte vielleicht noch lebend bergen zu können, darüber hinaus der Rettungsmannschaft, die unter oft lebensbedrohenden Umständen vorgehen muß, den kürzesten und sichersten Weg zu den Verschütteten aufzuzeigen, und nicht zuletzt Angehörigen und der Öffentlichkeit schnell richtige Informationen über das Ausmaß eines Unglücks geben zu können.

Ein Gedanke des Systems liegt darin, jedem Bergmann einen eigenen elektromagnetischen Sender oder Transponder bei der Einfahrt ins Bergwerk mitzugeben, der ohne dessen Zutun Signale senden und durch einen geeigneten Suchempfänger aufgespürt werden kann. Erste derartige Geräte sind schon in den 60er Jahren in der CSSR und in Polen entwickelt worden. Es handelte sich um kleine Sender, die von der Kopflampenbatterie gespeist wurden. Ihre Frequenz lag im Bereich der im Bergbau bereits eingeführten Langen Wellen bei ca. 400 kHz. Der von der polnischen Versuchsgrube Barbara entwickelte Sender wurde seinerzeit im gesamten polnischen Bergbau eingeführt und hat nach Angaben der Hauptrettungsstelle in Bytom vielfach zur schnellen Bergung auch von noch lebenden verschütteten Bergleuten geführt. In den USA konzentrierte man sich zunächst auf die Aufgabe, eingeschlossene Bergleute zu lokalisieren, die die Sendeeinrichtungen noch selbst aktivieren konnten. Anfang der 80er Jahre wurden Berichte veröffentlicht über Untersuchungen mit Sendern und Transpondern auch für verschütte-

te Bergleute, die im Kurzwellenbereich bei ca. 25 MHz arbeiteten (Webb 1984).

Der polnische Sender wurde Mitte der 70er Jahre bei der Versuchsgrube Tremonia in Dortmund näher untersucht. Es erwies sich, daß seine Feldcharakteristik sehr stark unsymmetrisch war, was zu erheblichen Mißweisungen um bis zu ca. 80 grd führte, so daß er für eine allgemeine Übernahme auch im deutschen Bergbau nicht geeignet erschien.

NEUERE ENTWICKLUNGEN

In Polen selbst war dieses Problem selbstverständlich auch bekannt, so daß dort neue Grundsatzuntersuchungen von der Theorie her aufgenommen wurden, die dann zu einem neuen System führten (Lozak 1980, 1986). Die ausschlaggebende Erkenntnis war, daß die Wellenlänge einer elektromagnetischen Strahlung entscheidend bestimmt, ob die Richtung zu einem Sender genau genug angegeben werden kann und das dazwischen liegende Material wie Haufwerk, festes Gebirge, Ausbau, Feuchtigkeit oder Hohlräume keine Verfälschung von Richtung oder Entfernung verursacht.

Es zeigte sich, daß nur extrem lange Wellen mit Frequenzen unter ca. 10 kHz gewählt werden dürfen, um gute Ergebnisse zu erzielen. Der Suchempfänger befindet sich dann im extremen Nahfeld des vom Sender ausgehenden elektromgnetischen Feldes, dessen magnetische Komponente die genaue Ortung ermöglicht. Das führte u.a. in Polen zu dem neuen System GLON-LOP.

In der Zwischenzeit begann, veranlaßt durch einige Unglücksfälle in der Bundesrepublik Deutschland, auch bei der Versuchsgrubengesellschaft mbH auf ihrer Versuchsgrube Tremonia in Dortmund ein von der deutschen Bergbehörde finanziell unterstütztes Forschungsprogramm über Systeme zur Ortung verschütteter Bergleute. Dabei sollten alle seinerzeit verfügbaren Systeme auf ihre Einsatzfähigkeit im deutschen Steinkohlenbergbau geprüft werden. Es wurden sowohl die alten Systeme einschließlich Lawinenschutzfunk als auch die neuesten Entwicklungen, die zum Teil erst während der Untersuchungen bekannt oder begonnen wurden, einbezogen. An neuen Systemen standen außer den Arbeiten in den USA zur Verfügung:

1. das in Polen bereits in der untertägigen Erprobung befindliche System GLON-LOP,

2. das in Österreich an der Universität Innsbruck entwickelte System SERLES und

3. das erst kürzlich in Deutschland bei der Bergwerksgesellschaft Ruhrkohle AG in Angriff genommene System POS 102.

Jedes dieser drei Systeme realisiert die Ortung auf seine eigene Weise, wie noch dargelgt werden wird.

1986 konnten sich Mitarbeiter der Versuchsgrube Tremonia in Polen untertage durch eigene Experimente von der Leistungsfähigkeit des neuen polnischen Systems eingehend überzeugen. Diese Experimente wurden dann auf Tremonia fortgeführt.

VERSUCHSBEREICH

Als Versuchsort war u.a. ein Bereich auf der 3. Sohle ausgewählt und zum Teil besonders hergerichtet worden. Ein Ausschnitt aus dem Sohlen-Riß (Bild 1) gibt eine Vorstellung von der dortigen Situation: Alle Strecken haben stählernen Bogenausbau und Verzug aus Stahlblechen. Die Strecke I wurde auf einer Länge von 20 m mit Haufwerk verfüllt, in das zur Einbringung der Prüflinge fünf Kunststoffrohre über den Streckenquerschnitt verteilt in der gesamten Länge parallellaufend eingelassen waren. Das vorhandene Gleis wurde nicht ausgebaut, um dadurch einen langen, möglicherweise störenden Wellenleiter nachzubilden. Die stählernen Verzugbleche des Streckenausbaus wurden in jedem 5. Feld entfernt, um elektrisch voneinander unabhängige Sektionen zu erhalten.

In den Strecken II, III und IV befinden sich ebenfalls Gleise und außerdem Versorgungsleitungen für elektrische Energie, für Wasser und

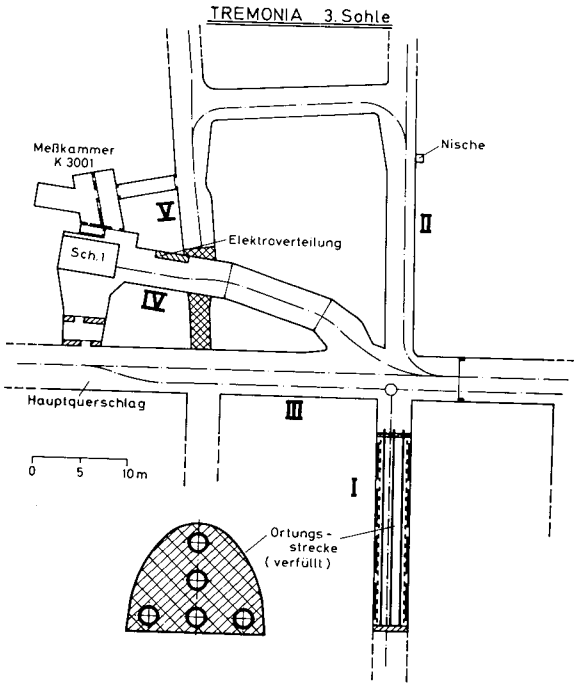


Bild 1. Untersuchungsgebiet 3. Sohle

für Druckluft. In der Strecke IV befindet sich zusätzlich noch eine großflächige Energieverteilungsanlage. Die Strecke V ist teilweise mit Haufwerk verfüllt. Auf diese Weise kann der Einfluß des Umfeldes wie Gebirge, Hohlraum, Haufwerk, Kabel und Leitungen, Ausbau, Stahlblechverzug, Feuchtigkeit, Elektroverteilung u.a. auf die Genauigkeit der Ortsbestimmung, d.h. auf Entfernung und Richtung untersucht werden.

VORUNTERSUCHUNGEN

Zunächst wurden Dämpfungs- bzw. Durchdringungsmessungen durch Haufwerk durchgeführt, und zwar mit allen Frequenzen der in die Vorbetrachtung einbezogenen Geräte: von 9 kHz bis 25 MHz und 425 MHz. Die Reichweite erschien bei allen Frequenzen bis 25 MHz ausreichend; nur das im deutschen Bergbau eingesetzte Arbeitsschutz-Transpondersystem REDAR hat bei einer Frequenz von 425 MHz im Haufwerk eine Reichweite von weniger als 1 m (Richtscheid 1982).

Bei der Richtungsbestimmung konnten die früheren Peilfehler von 50 Grad und mehr bei Frequenzen von z.B. 384 kHz sowohl über Tage als auch in den Teststrecken unter Tage verifiziert werden.

DAS POLNISCHE SYSTEM GLON-LOP

Das System besteht aus dem Sender GLON, der im Batteriegehäuse der Kopflampe eingebaut ist (Bild 2), und

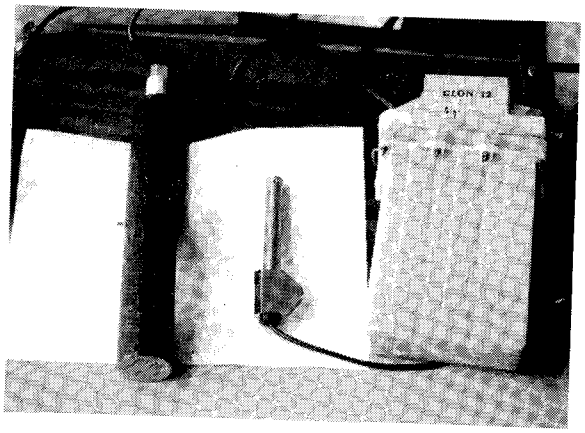


Bild 2. Sender GLON und Suchantennen

dem Suchgerät LOP mit zwei verschiedenen Antennen. Seine magnetische Meßantenne PAM wird für die Ortsbestimmung verwendet, und die relativ unempfindliche Antenne "Suchersonde" ist für eine Nahortung im Umkreis von 1 bis 2 m besonders geeignet. Der Ortungsempfänger LOP ist ein kleines, eigensicheres, batteriebetriebenes Gerät (Bild 3). Die verwendeten Sendefrequenzen sind auf 42 Kanäle zwischen 3925 und 5975 Hz verteilt.

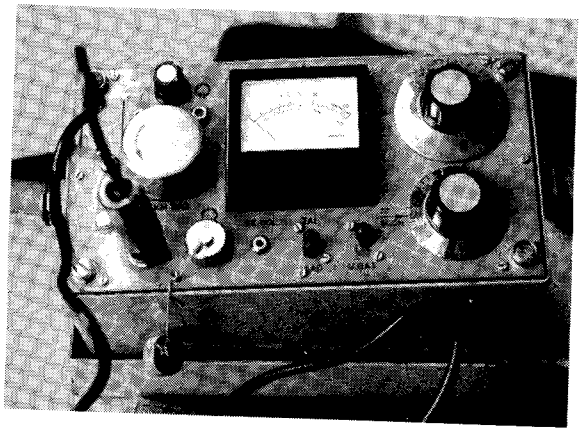


Bild 3. Suchgerät LOP

Die Funktionsweise ergibt sich aus den Maxwellschen Gleichungen, die sich für die Amplitude H der magnetischen Feldkomponente im Nahfeld bei parallel ausgerichteten Antennen auf die einfache Formel reduzieren lassen:

$$H = \frac{1}{4\pi} \cdot \frac{m}{r^3} \qquad (1)$$

wobei m das Magnetmoment des Senders und r seine Entfernung vom Empfänger ist. Bei einer Frequenz unterhalb von 10 kHz reicht im Steinkohlenbergbau das Nahfeld bis etwa 20 m.

Unter diesen Umständen läßt sich bei bekannter Senderstärke ein Empfangsgerät direkt in Entfernungsmetern skalieren. Dadurch ergibt sich die "Suchmethode I", indem von verschiedenen Standorten vor einem Bruchbereich die jeweiligen Entfernungen zu einem Sender ermittelt werden und daraus durch Triangulation der Senderort angegeben werden kann (Bild 4).

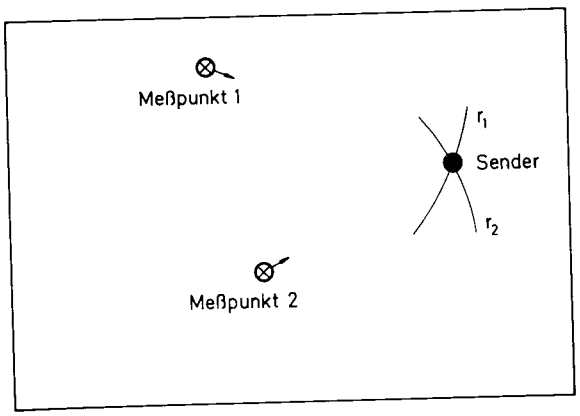


Bild 4. GLON-LOP Suchmethode I

Hat die Sendefeldstärke nachgelassen, z.B. durch Beschädigung oder zu hohen Verbrauch, hilft die Suchmethode II weiter: Sie ist eine Gradientenmethode, die den typischen Abfall der Feldstärke in Abhängigkeit von der Entfernung ausnutzt. Dabei wird von einem gewählten Fixpunkt aus die Richtung zum Sender bestimmt und an einem 2. Punkt in festem Abstand auf dieser Richtingslinie die Feldstärke im Verhältnis zum 1. Punkt auf einer Skala abgelesen. Die gesuchte Entfernung ist danach direkt von einem weiteren Maßstab abzunehmen.

Erste Experimente wurden noch während des Besuches in Polen gemacht, weitere dann in dem Versuchsgebiet auf Tremonia (Bild 1). Es wurden jeweils beide Suchmethoden angewendet und miteinander verglichen. Zunächst fanden die Versuche in der Ortungsstrecke I statt. Dort waren die Verhältnisse sehr übersichtlich und die angegebenen Orte konnten leicht überprüft und nachgemessen werden. Es zeigte sich, daß die Lage des Senders innerhalb des Streckenquerschnitts oder sein Abstand zum Ausbau keine besondere Rolle spielte. Bei einer Distanz von 15 m lag die maximale Abweichung bei 0,5 m.

Um die Leistungsfähigkeit des Systems auch unter anderen, möglichst ungünstigeren Bedingungen zu testen, wurden auch die Strecken II bis V mitbenutzt. Bild 5 zeigt einige ausgewählte Versuche. Ausgangspunkt der Suchaktionen war in diesen Fällen immer die Strecke III. Der Ort der Sender war den Suchenden vorher nicht bekannt.

Im Fall A befand sich der Sender hinter einem stählernen Verzugblech. Wie Bild 5 zeigt, ergaben die Messungen an den verschiedenen Punkten in der Strecke III sehr genau den Ort des Senders. Die Suchersonde führte dann in der Strecke IV die Suchmannschaft direkt zu dem nicht sichtbaren Sender hinter dem stählernen Verzugblech A.

Im Fall B befand sich der Sender in der Strecke V auf der Sohle neben einem Ausbaubogen B. Zwischen dem Ausgangspunkt der Suche und dem Sender lagen Gebirge, Hohlräume, Haufwerk, mehrere Lagen von Verzugblechen und die Elektroverteilung. Vom Punkt B_S aus wurde grob die Richtung (richtig) festgestellt und darüber hinaus eine Entfernung von etwas mehr als 20 m. Die Suchenden bewegten sich daraufhin unter ständiger Beobachtung der Anzeige auf dem Suchgerät bis in die Strecke IV so nah wie möglich auf den Sender zu. Von dort konnten sie - auch durch die Elektroverteilung hin-

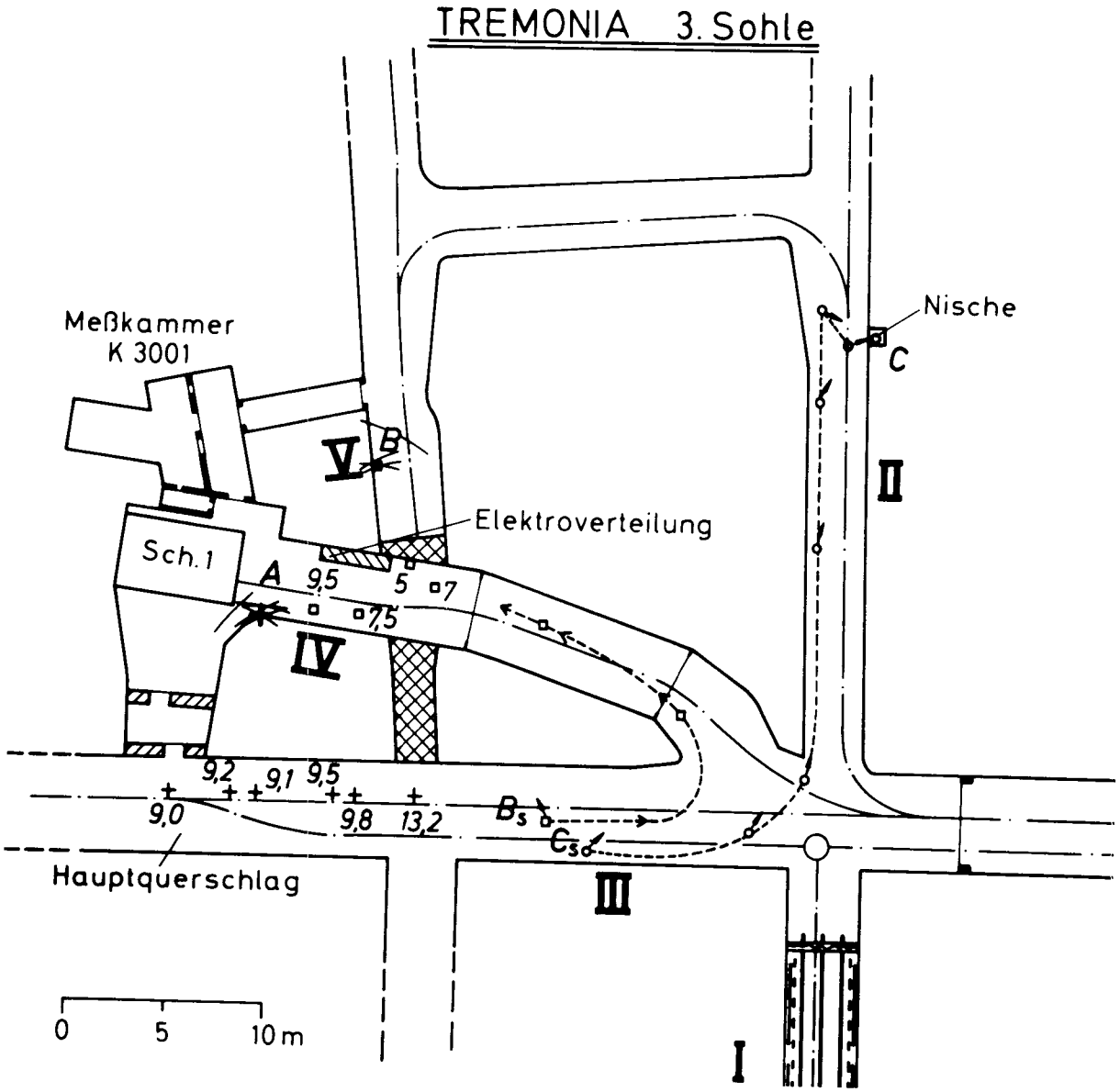


Bild 5. GLON-LOP, Ortungsversuche (Zahlenangaben in Metern)

durch - eine exakte Ortsbestimmung vornehmen.

Im Fall C befand sich der Sender in der Strecke II, und zwar in einer betonierten Wandnische hinter einer Stahltür C in etwa 30 m Entfernung vom Startpunkt MC_s. Die Suchenden stellten wieder die Richtung richtig fest und ermittelten eine Entfernung von mehr als 20 m. Sie vermuteten deshalb den Sender in der Strecke II. Sie bewegten sich zielsicher durch die Strecke ebenfalls unter ständiger Beobachtung ihres Suchgerätes bis in die Nähe der Nische. Ihre Spezialantenne führte sie dann direkt zu der in Kopfhöhe sich befindenden stählernen Tür, hinter der sie den Sender fanden.

DAS DEUTSCHE SYSTEM POS 102

Das zur Zeit bei der Ruhrkohle AG (Ruhrkohle 1986) in Entwicklung befindliche System POS 102 arbeitet folgendermaßen: Das Suchgerät ruft mit einer Sendefrequenz von 9 kHz einen Sender PKB 1 am zu suchenden Bergmann auf, damit dieser seinerseits mit einer Frequenz von 444 kHz sendet. Dieser Sender am Mann besitzt eine eigene, kleine Lithiumbatterie, die nur bei aktivierter Sendung benutzt wird. Der gesamte Senderbaustein PKB 1 ist so klein, daß er im

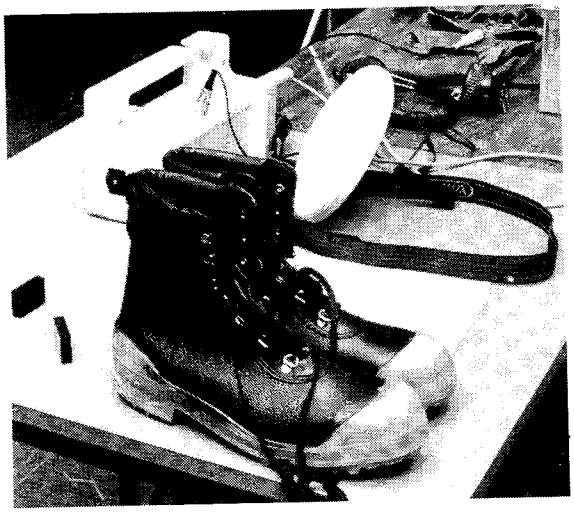


Bild 6. RAG-Ortungssystem POG 102/PKB 1

Absatz eines Arbeitsschuhs Platz findet (Bild 6). Außerdem kann er eine Personenkenn-Nummer über ein kodiertes Signal abstrahlen und zum anderen als Arbeitsschutzsender Verwendung finden, um bei Annäherung von Personen an gefährliche Geräte diese stillzusetzen. Über seine Funktionsmethode ist noch nichts genaues bekannt. Bei ersten Versuchen auf der Versuchsgrube Tremonia in der Ortungsstrecke I (Bild 1) wurde nur knapp 5 m Distanzweite erreicht. Nach Verbesserungen sollen 7 bis 8 m möglich werden. Die Unterbringung je eines Senders in jedem Schuh eines Bergmanns wird deshalb favorisiert, weil angenommen wird, daß bei einem Unglück unter Tage Schuhe eher verloren gehen oder zerstört werden als Lampenbatterien. Zur Zeit finden Trageversuche der Schuhe mit einvulkanisierten PKB 1-Bausteinen auf einer Schachtanlage statt.

DAS ÖSTERREICHISCHE SYSTEM SERLES - THEORETISCHE GRUNDLAGEN -

Wie schon oben erwähnt, ist die Ausbreitung und Kommunikation mit Hilfe elektromagnetischer Signale nur im VLF- und ELF-Bereich möglich. Da die Wellenlänge in diesem Frequenzbereich mehrere Kilometer beträgt, kommen für Sender und Empfänger nur magnetische Antennen in Frage. Ungünstigerweise ist die Strahlungscharakteristik einer magnetischen Dipolantenne (Ferritantenne oder Schleifenantenne) im Nahfeldbereich nicht kugelförmig, sondern hat eine kompliziertere Form. Weiters beeinflußt die Gesteinsleitfähigkeit die Feldausbreitung um so mehr, je höher die Leitfähigkeit, die Signalfrequenz und die Distanz vom Senderdipol ist. Die Feldkomponenten eines magnetischen Dipols in sphärischen Koordinaten können aus den Maxwell-Gleichungen wie folgt hergeleitet werden:

$$Hr = m/2\pi . \cos\theta / r^3 . (1+ikr) . e^{-ikr} \quad (1)$$

$$H\theta = m/4\pi \sin\theta / r^3 . (1+ikr+k^2 . r^2) . e^{-ikr} \quad (2)$$

$$H\phi = 0 \quad (3)$$

Mit m = n.I.A magnetisches Moment des Senderdipols.

$$k^2 = i . \omega . \mu . \sigma + \omega^2 . \varepsilon . \mu \quad (4)$$

Da k alle Mediumparameter enthält, ist es möglich, die Signalfrequenz tief genug zu wählen, damit die Bedingung k r 1 erfüllt werden kann. In diesem Fall lautet der Betrag der Feldstärke wie folgt:

$$H = m/4\pi . 1/r^3 . \sqrt{1 + 3.\cos^2} \quad (5)$$

Die Feldstärke in der Achse des Dipols = 0° ist doppelt so hoch wie das Feld bei = 90°, woraus sich ein Sendeantennendiagramm wie in Bild 7 gezeigt, ergibt. Die Feldrichtung wird beschrieben durch

$$\tan\psi = Ho/Hr = 1/2 . \tan\theta \quad (6)$$

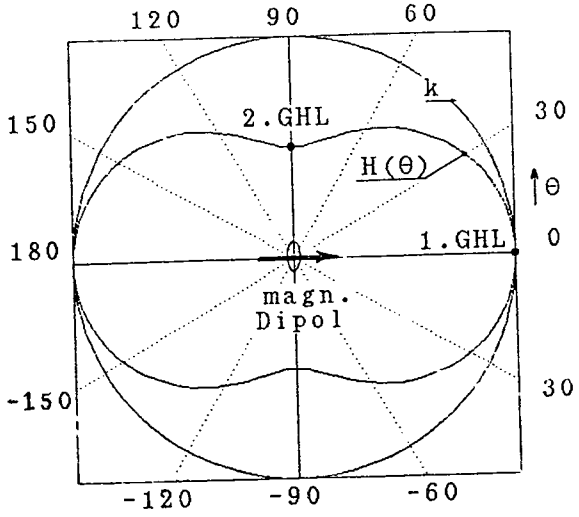


Bild 7. Sendeantennendiagramm

Die Abhängigkeit der Feldstärke von der Entfernung oder die räumliche Verteilung der Feldlinien oder beide Eigenschaften können die Basis für eine Ortungsmethode bilden. Die erste Eigenschaft nützt das polnische System GLON-LOP, womit wegen der Vernachlässigung des Winkelterms in Gleichung 5 ein worst-case-Fehler in der Entfernungsbestimmung von 26 % auftritt. Die zweite Eigenschaft wurde für eine Ortungsmethode benützt, die unabhängig von der Senderstärke arbeitet (Nessler 1984 und 1988), die aber für größere Entfernungen etwas empfindlich auf Feldstörungen durch die inhomogene Umgebung reagiert. Eine Kombination beider Eigenschaften, Feldstärke und Feldrichtung, bildet die Basis für die Serles-Methode und vermeidet die oben angeführten Nachteile (Nessler 1989).

ORTUNGSMETHODE

Zur Anwendung dieser Basiseigenschaften als Ortungsmethode ist es notwendig, das Antennendiagramm für eine feststehende Empfangsantenne neu zu berechnen, wobei der Senderdipol derartig um den Empfänger bewegt wird, daß die Feldstärke und die Feldrichtung am Empfangsort konstant bleibt. Die sich damit ergebende Ortskurve für die Senderposition wird durch

$$r = \text{const.} \sqrt[3]{2/(H.\sqrt{1+3.\sin^2\phi})} \qquad (7)$$

beschrieben und ist in Bild 8 dargestellt, wobei H die Richtung der Empfangsantenne angibt. In Sa ist eine mögliche Position und die zugehörige Richtung der Sendeantenne gezeigt.

Diese Ortskurve kann auf zwei verschiedene Arten zur Lösung von Ortungsproblemen verwendet werden.

1. Bestimmung einer unbekannten Sendeantennenposition

2. Bestimmung einer unbekannten Empfängerposition

UNBEKANNTE SENDERPOSITION

Die Sendeantenne besteht aus einem einzelnen magnetischen Dipol, von dem weder die Position noch die Orientierung bekannt sei. Um die Senderposition zu finden, ist es notwendig, die Feldamplitude und die Feldrichtung, welche vom Sender erzeugt wird, in mindestens zwei geeignet gewählten Empfangspunkten zu messen. Für jeden Empfangspunkt kann daraus eine entsprechende Ortskurve gemäß Gleichung 7 berechnet werden, wobei die Senderdipolstärke als bekannt vorausgesetzt wird. Zwei solche Ortskurven schneiden sich in zwei Punkten, welche mögliche Senderpositionen anzeigen (Bild 9). Eine dritte Ortskurve, die gemäß

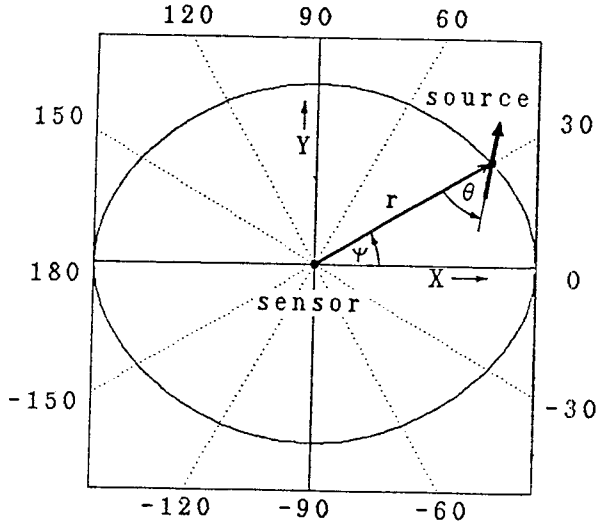


Bild 8. Ortskurve für mögliche Senderpositionen bei fixiertem Empfänger

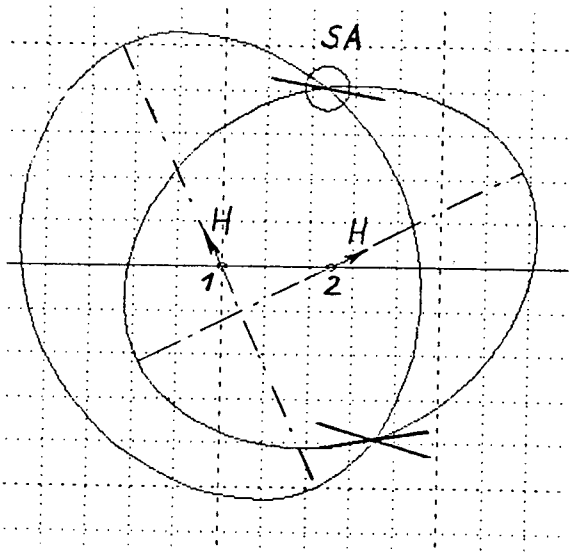


Bild 9. Zwei Ortskurven und die Senderorientierung

den Meßwerten in einem dritten Punkt gezeichnet wird, eliminiert die Zweideutigkeit.

Falls die Senderdipolstärke nicht ausreichend genau bekannt ist, bleibt ein gemeinsamer Vergrößerungsfaktor für die Ortskurven unbekannt. Dieser unbekannte Faktor kann entweder durch eine größere Anzahl von Meßwerten und Variation des Faktors bis zum genauen Ergebnis (ein Schnittpunkt) bestimmt werden oder es werden die Richtungsinformationen entsprechend Gleichung 6 als zweiter Parameter verwendet. Da angenommen werden kann, daß die Sendeantenne während einer Messung ihre Position beibehält, muß die berechnete Richtung der Sendeantenne für eine geschätzte Position auf allen Ortskurven die gleiche sein.

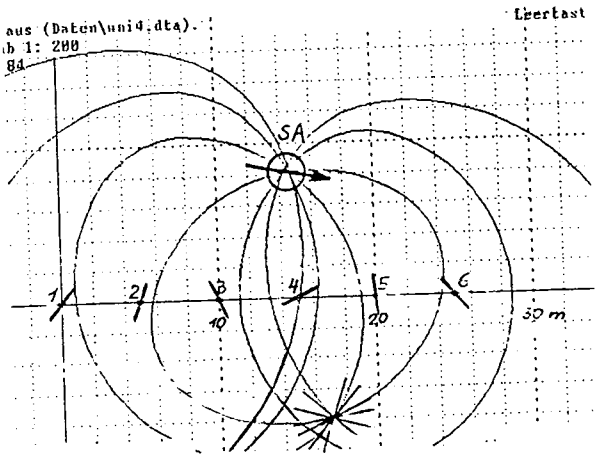


Bild 10. Ortsbestimmung für einen Sender mit unbekannter Dipolstärke. Das Ergebnis wurde durch Variation des Vergrößerungsfaktors erzielt

Die Ortskoordinaten jedes Empfängerstandortes sind für die Berechnung wesentliche Eingaben. Für eine schnelle Messung wird eine gerade Linie mit äquidistanten Punkten markiert, welche die Basis eines Koordinatensystems bildet. Diese Methode ist bei den Testmessungen in der Versuchsgrube Tremonia benützt worden. Da dieses Verfahren nicht überall anwendbar ist, wurde eine ähnliche Ortungsmethode entwickelt, mit der mittels eines Hilfsantennensystems die Empfängerposition automatisch bestimmt werden kann.

UNBEKANNTE EMPFÄNGERPOSITION

Das inverse Problem zu dem oben beschriebenen ist die Bestimmung einer Empfängerposition relativ zur bekannten Position einer (Hilfs-)quelle (Nessler 1987). In diesem Fall bleiben sowohl Sender als auch Empfänger an ihren Positionen stehen. Die Bewegung des Empfängers wird in diesem Fall durch drei verschiedene Sendeantennen-Orientierungen ersetzt. Für jede Senderorientierung bestimmt die Feldstärke und Feldorientierung an der Empfängerposition je eine Ortskurve. Die Ortskurven sind jetzt konzentrisch, unterscheiden sich aber in ihrer Orientierung und Größe wie in Bild 11 gezeigt.

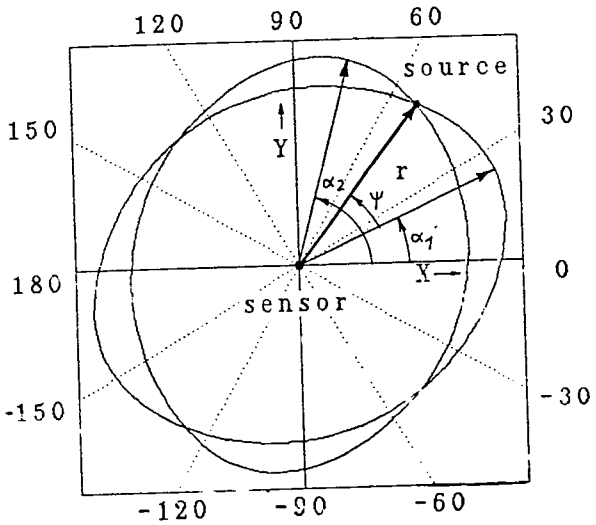


Bild 11. Zwei konzentrische Ortskurven bestimmen 4 mögliche Senderpositionen

In diesem Fall muß die Senderdipolstärke bekannt sein. Der gemeinsame Schnittpunkt dreier solcher Ortskurven bestimmt zwei mögliche Senderpositionen. Wegen der Symmetrie des Problems kann die Zweideutigkeit nicht durch eine erhöhte Zahl der Senderorientierungen beseitigt werden. In den meisten Fällen ist es möglich, durch physikalische Einschränkungen die richtige Position zu erkennen. Ist das nicht möglich, so muß die starke Entfernungsabhängigkeit der Feldstärke ($1/R^3$) als Merkmal herangezogen werden: wird der

Empfänger in Richtung der vermuteten Senderposition bewegt, so ergibt sich eine Feldstärkezunahme.

TESTMESSUNGEN MIT DEM ÖSTERREICHISCHEN SYSTEM SERLES

Mit dem Serles-System wurden Testmessungen an den gleichen Stellen wie schon mit dem polnischen System GLON-LOP durchgeführt. Für Test 1 war der Sender hinter einem stählernen Verzugblech befestigt. Es war angenommen, daß für das Suchteam nur Strecke III zugänglich sei. Daher wurde eine Basislinie mit Meßpunkten je 5 m markiert (s. Bild 12 MP 1 = 10). Die Messung der Feldstärke und Feldrichtung in jedem dieser Punkte wurde halbautomatisch durch Messung von drei orthogonalen Feldkomponenten und einer Phaseninformation durchgeführt. Bereits aus diesen Messungen war klar erkennbar, daß sich der Sender in der Nähe der Meßpunkte MP 6 und MP 7 befinden mußte. Die graphische Auswertung mit einem tragbaren Computer bestätigte dieses erste Ergebnis. Die Schnittpunkte aller Ortskurven lagen innerhalb von 0,8 m von der wahren Position (vergleiche Bild 12). Benützt man nur Meßwerte mit den höchsten Feldstärken (kleinste Ortskurven, MP 5,6 und 7), so betrug die Genauigkeit wenige cm.

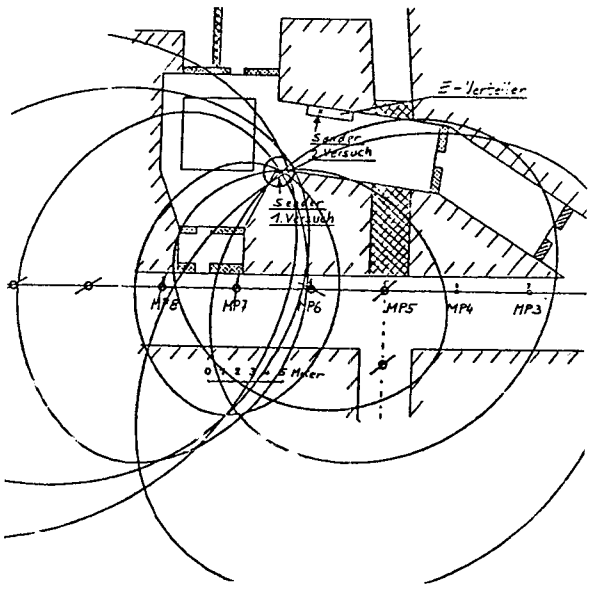


Bild 12. Graphische Auswertung von Test 1

Für den Test 2 war der Sender zwischen massiv gepanzerten Elektroverteilern versteckt, und zusätzlich war die Senderleistung für das Suchteam unbekannt reduziert. Die Messungen wurden in der gleichen Weise, wie für Test 1 beschrieben, durchgeführt; die graphische Auswertung zeigte ein eher verteiltes Feld von Schnittpunkten, deren Schwerpunkt jedoch mit der wahren Senderosition übereinstimmte (Bild 13).

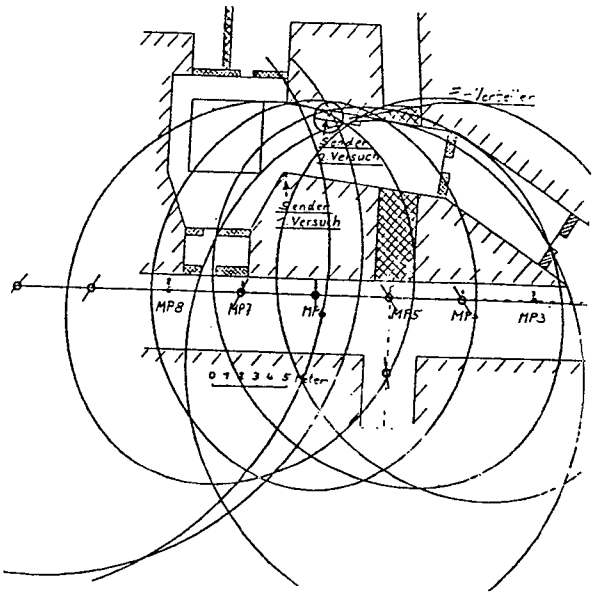


Bild 13. Graphische Auswertung von Test 2

Wurde zur Auswertung zusätzlich die Übereinstimmung der Sendeantennenrichtung verwendet, so ergaben wieder die Messungen in MP 5, 6 und 7 das beste Ergebnis. Die berechnete Position war nur 50 cm von der wahren Position entfernt.

ABSCHLIESSENDE BEMERKUNGEN

Alle drei beschriebenen Systene: GLON-LOP, RAG-PKB und SERLES sind in der Lage, Personen recht genau zu orten, die einen entsprechenden Sender tragen. Sie unterscheiden sich zum Teil gravierend in ihrer Reichweite. Dadurch, daß das polnische System versteckte Sender schon aus ca. 50 m der Richtung nach wahrnehmen und unterhalb von 20 m sehr genaue

Ortsangaben machen und daß das System SERLES diese Reichweite noch übertreffen kann, ist bei einem Unglück die Möglichkeit gegeben, sich zunächst mit starkem Räumgerät rasch bis in die Nähe von Verschütteten vorzuarbeiten, um dann die letzten Meter unter ständiger Kontrolle durch das Suchgerät vorsichtig per Hand freizuräumen. Beim deutschen System sind mit weniger als 10 m die Erkennungsdistanzen erheblich geringer. Das erfordert daher genauere Vorabinformationen über die mögliche Lage eines Verschütteten, um so in den Erkennungsbereich seines Senders zu kommen. Dagegen stünden Vorteile durch weitere Einsatzmöglichkeiten z.B. als Arbeitsschutzsender. Insbesondere wird angenommen, daß ein im Arbeitsschuh mitgebrachter Sender weniger anfällig für Zerstörung oder Verlust ist und die Stromversorgung zuverlässiger sein könnte.

Das Suchgerät des polnischen und des deutschen Systems trägt ein Mann vor der Brust. Dieser kann mit der Suchantenne in der Hand zügig durch die Grube gehen und auf Reaktionen des Geräts achten. Das Suchgerät des östereichischen Systems besteht aus mehreren Kleingeräten, die schnell in bekannten Abständen aufgestellt werden können, mit Leitungen verbunden sind und dann rechnerkontrolliert selbständig die Lage eines Ortungssenders herausfinden und angeben. Ein Positionswechsel ist dabei ebenfalls kein Problem.

Über die Preise liegen noch keine genauen Angaben vor, doch dürften sie für die Sender, mit dem jeder Bergmann ausgerüstet sein sollte, nicht viel mehr als DM 10,-- betragen. Von den erheblich teureren Suchgeräten werden dagegen nur relativ wenige Exemplare benötigt.

Auf der Versuchsgrube Tremonia ist zur Zeit ein polnisches Suchgerät mit zehn Lampensendern vorhanden, so daß weitere Untersuchungen, einschließlich Erprobungen auf aktiven Schachtanlagen im deutschen Steinkohlenbergbau durchgeführt werden können.

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"NOUVEAUX SYSTEMES DE REPERAGE DES MINEURS ENTERRES".

Dr. N. Nessler, Dr. D. Fischer

Des systèmes électromagnétiques de repérage des mineurs enterrés ont été développés dans les années 1970. Toutefois, des essais effectués dans la mine de Tremonia à Dortmund, n'avaient pas donné de résultats satisfaisants.

L'année dernière, des essais de systèmes développés aux Etats-Unis, en Pologne, en Autriche et en République Fédérale d'Allemagne ont été effectués à Tremonia. Leurs résultats ont démontré que seuls réussissaient les systèmes utilisant des fréquences extrêmement faibles pour les émetteurs des mineurs et exigeant des mesures de champs rapprochés. Dans ces cas, seule l'influence de l'environnement hétérogène était négligeable.

Ces systèmes fonctionnent sur le même principe que le système autrichien SERLES: combiné à son accumulateur de lampe frontale, le mineur porte un émetteur à faible fréquence et à impulsions pour économiser les piles. Le repérage de sa position s'effectue au moyen d'un récepteur à bande étroite très sensible qui mesure la direction et la force du champ dans quatre emplacements au moins. Un ordinateur intégré calcule la position du mineur. La portée de l'opération va jusqu'à 30 m, l'exactitude est de 5% de la distance environ.

НОВЫЕ СИСТЕМЫ ДЛЯ ОБНАРУЖЕНИЯ ЗАСЫПАННЫХ ШАХТЕРОВ

Н. Несслер и Д. Фишер

Электромагнитная система обнаружения засыпанных шахтеров была разработана уже в 70-е годы. Однако испытания на экспериментальной шахте Тремония в Дортмунде, ФРГ, показали неудовлетворительные результаты.

В прошлом году недавно разработанные системы из США, Польши, Австрии и ФРГ. были испытаны в Тремонии. Результаты испытаний подтвердили, что успешными были только те системы, в которых использовались чрезвычайно низкие частоты для шахтерских передатчиков, т.к. они оказались способными принимать сигнал в пределах близлежащей зоны. Только в этом случае можно было пренебречь воздействием неоднородной окружающей среды. Принцип работы таких систем продемонстрирован в описании австрийской системы "СЕРЛЕС". Каждый шахтер должен носить низкочастотный передатчик, получающий питание от батареи шахтерской лампы и работающий, для сохранения батареи, на импульсном режиме. Поиски местоположения шахтера производятся высокочувствительным приемником, измеряющим направление и силу поля по меньшей мере в четырех точках. Встроенный в приемник компьютер вычисляет местоположение шахтера. Зона действия - до 30м, точность - приблизительно 5% расстояния.

第二十三届矿业安全研究所国际会议

第八节 救护和复原

"困在矿里矿工的位置，新探测系统"

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在七十年代时，用电磁学探测被困矿工的系统已经发明。但在联邦德国多特蒙得屈裘尼亚矿业研究所做的试验结果并不良好。

去年由美国，波兰，奥地利和联邦德国发展出来的系统曾在屈裘尼亚试验，试验结果指出，只有用极低的矿工发送机频率和远距离就地测量的系统成绩圆满，只有在这种情形下，不均匀的环境影响可以忽略。

这种系统的操作原理用描述奥地利 "塞尔斯" （"SERLES"）系统来示范。矿工可以将一个低频率的发送机与他的帽灯蓄电池连接在一起，用脉冲操作以节省电池。方位探测用一个非常灵敏的窄频带接收机至少在四个位置测量方向和强度。仪器内附有的计算机可以计算出矿工的位置。操作距离可达 30 米，精度约为距离的百分之五。

MONITORING SYSTEM OF PERSONNEL WORKING UNDERGROUND

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ABSTRACT

The objective of this research is to identify the location of underground workers continuously from a monitoring station and to obtain useful information in an emergency for controlling disasters and rescuing sufferers. This system is composed of personal transmitters, antennae, frequency converters, fixed stations underground and monitoring stations on surface. The personal transmitter is in the safety cap lamp and emits a ID code by 400MHz. The ID code is received by the antennae, converted to 70MHz by the frequency converters, arranged by the fixed station and transmitted to the monitoring station at a speed of 1,200bps. Whole data are processed synthetically at the monitoring station and displayed on C.R.T. so that informations of workers location underground can be obtained. Now the system to detect 50 persons has developed and has been demonstrated at Ikeshima Colliery without serious problem.

INTRODUCTION

At present there are important and high skilled techniques to identify the location of workers underground in any mines in Japan, and the location of them are only done by the allotment of jobs to workers when they were entering a pit other than direct confirmation by induced wireless communication. Especially when accident occurs, it is very important to identify the exact location of workers considering the panic underground. In order to solve those problems, The Coal Mining Research Centre, Japan are developing underground personal monitoring system as one of the projects by entrusted fee of Development of Coal Mine Safety Technology since 1985 fiscal year. The purpose of this work is to research and develop the advanced underground personal monitoring system to identify the location and trace of workers continuously and exactly in real time at centralized monitoring station and also by utilizing those informations to established the system to coping with the suppression of disasters and the rescue of sufferers. So the position of this system is a important part of comprehensive monitoring system underground shown in Fig. 1. The reliability and outline of this system obtained from field test are introduced in this report.

BASIC DESIGN

Considering a size of colliery, maximum registration of workers, maximum workers underground per shift and workers movement, basic design of this system was made as follows;

1. Emitting distance
The personal transmitter emits the ID signal at minimum distance of 12.5m. (Workers underground location must be identify with carrying the personal transmitter when they pass the check point.)

2. Workers
Maximum registration of workers

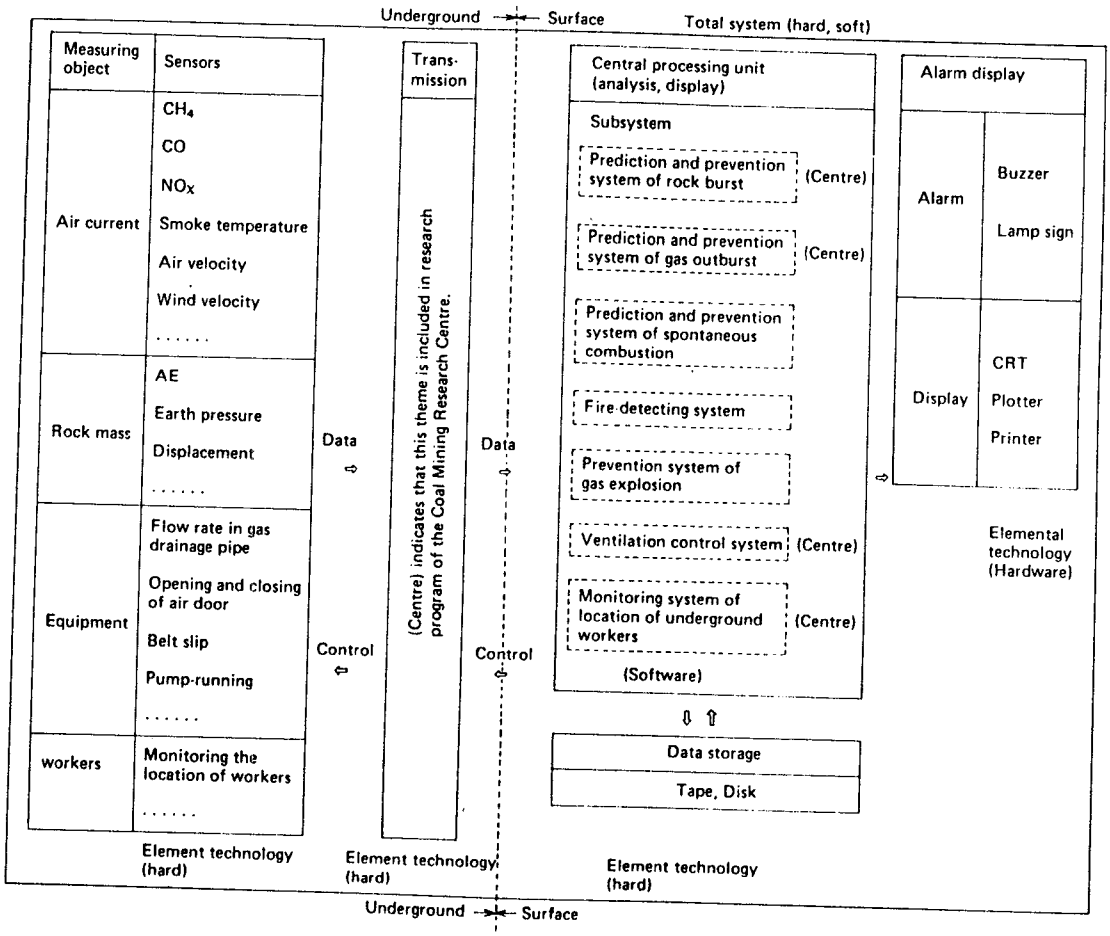


Fig. 1 Concept of comprehensive underground monitoring system

are 2,500 and maximum workers underground per shift are 1,000.

3. Workers movement

Two patterns of workers underground movement were considered.

a. Close to the station of man car

On the assumption that the number of 250 workers pass at a speed of 6km/h (1.67m/s) in colums of 4 at intervals of 0.75m, the traffic is about 9persons/s. In this case, the number of 133 personal transmitters emit the ID signal against the single antenna, so uncollision probability (Puc) of each ID signal of persanal transmitter is calculated as follows;

Number of transmitter

$$n = 133$$

Emission time

$$t = 2ms$$

Emitting probability

$$p = 0.25 \text{ (intervals of 4s on average)}$$

$$Puc = (1-2tp)^{n-1} = 0.875$$

When workers pass the check point (25m width by single antenna), it takes 15s. So that the ID signal is emitted 3.8 times on average. Then collision probability of all ID signal loss P is

$$P = (1-Puc)^{n-1} = 3.70 \times 10^{-4}$$

b. Close to the face

On the assumption that the number of 10 workers pass at a speed of 10km/h (2.8m/s) in colums of 1 at intervals of 3m, the traffic is

about 4persons/s. In this case the number of 10 personal transmitters emit the ID signal above mentioned, Then

$$Puc = (1-2tp)^{n-1} = 0.991$$

$$P = (1-Puc)^{n-1} = 5.619 \times 10^{-3}$$

SYSTEM CONFIGURATION

This system is composed of the following components;

1. Personal transmitter, molded by epoxy resins, fitted in the cover of miner's cap lamp carried by the workers underground, emitting randomly the ID signal of 2ms emission wave (personal code) by 50kbps at intervals of 4s on average, weighted 110g, sized 116mm(W) x 24mm(H) x 37mm(D).

2. Frequency converter, receiving the radio frequency signals of 400MHz emitted by the personal transmitters with an antenna, converting the frequency of these signals into 70MHz and transmitting them to the underground fixed station up to 500m in case of using 10D-2V coaxial cable.

3. Underground fixed station, furnishing 5 units of detector, taking out the digital ID code by receiving and demodulating the modulated signals of 70MHz through a single coaxial cable from each frequency converter, arranging the data and then performing the frequency shift keying (F.S.K.) and transmitting them to the surface mother station.

4. Surface mother station, demodulating the F.S.K. signals transmitted through telephone line at 1,200bps from the underground fixed station, taking out the ID codes from them, performing the collection of coding error, detection, compilation, abnormality check on transmission line, real-time data display on C.R.T. (cathode ray tubu) screen and printing out the output data.

5. Surface centralized monitoring and processing station, arranging the data after the integrared processing by the surface mother station and performing the necessary monitoring jobs on underground personnel such as graphic display and retrieval representation.

The block diagram of this system and the main specification of each component are shown in Fig. 2 and Table 1.

DATA PROCESS

Each personal transmitter had a own personal code emits the ID signals randomly at 4s intervals on average. At an underground check point, installed single antenna covered each direction of 12.5m of roadway. Also each detection unit in underground fixed station connected to frequency converter has own code number. A check point is made up of combination of single or plural number of anttenae, and the number of anttenae at each check point is determined by traffic of workers. The data of personal transmitter is composed of the data regarded as "IN" the check point and those as "OUT" of the check point. In order to leave out the overlapped data, they are duplicated like that

1. When several antennae receive the data emitting from a personal transmitter at same time, an order of priority is applied to them and the location data is limited to one of them by saving the data on the antenna with the highest priority and eliminating other data.

2. When the data of a personal transmitter is received from a same antenna in a certain time, they are eliminated.

In the surface cetralized monitoring and processing station, "IN" or "OUT" of check point is determined by comparing and processing the data of each antenna. The data transmitted from underground, such as the personal idetification codes, monitoring items of fixed station, circuit errors and others, are

Table 1 Specification of underground personal location monitoring system

Device	Specifications
Personal transmitter	Emission frequency; 400MHz band Emission output; -10dBm Emission time; 2ms (average; randomly 4s) Mode of modulation; Frequency modulation Emission speed; 50kbps
Frequency converter	Reception frequency; 400MHz band Converted output frequency; 70MHz band Amplifier gain; 33dB Antenna; 400MHz band
Underground fixed station	1. Reception unit Reception frequency; 70MHz band Sensitivity (Bit error ratio at a reception input of 5dBμV); less than 1×10^{-2} 2. Transmission unit (data processing unit) Code error control method; Parity check by B.H.C. (21, 12, 5) codes and reverse double transmission identification Mode of modulation; Frequency modulation Transmission speed; 1,200bps Encoding mode; NRZ isometric coding 3. Others Operating time of emergency battery; in less than 5H (also available without battery) Input power source; A.C. 100V ± 20%, single phase, 50/60Hz
Surface mother station	Input impedance; 600Ω ± 20% equivalent Code error control method; Parity check by B.H.C. (21, 12, 5) codes and reverse double transmission identification External computer input/output; According to RS232C Interface; 20mA current loop
Surface centralized monitoring and processing station	C.R.T., Key-board, Printer 16bit micro-processor (main memory; 1MB) Mini-floppy disk (1MB) Micro-disk (40MB) Graphic color disply (14inch, 960 × 720dots)

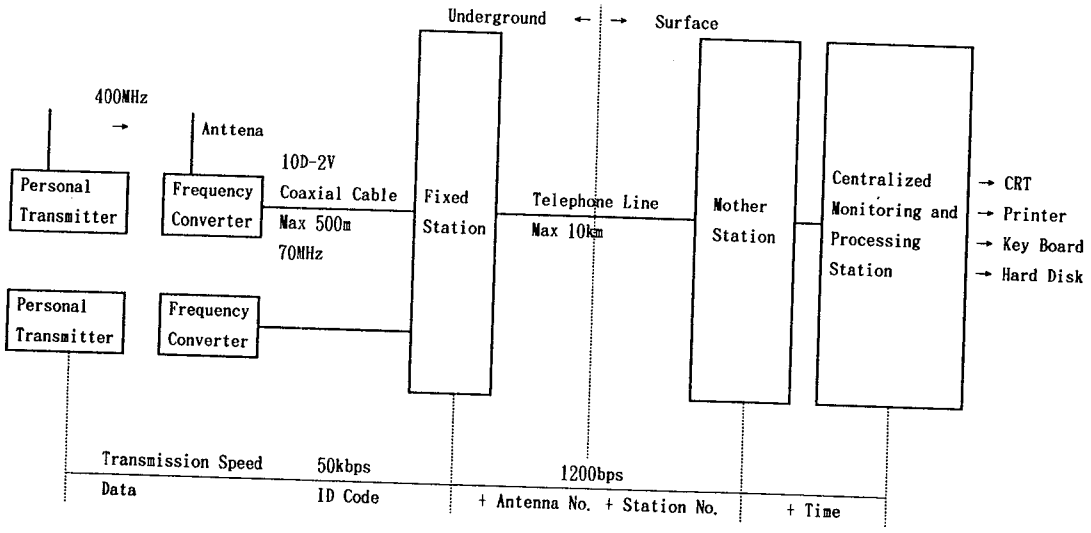


Fig. 2 Block diagram of underground personal location monitoring syatem

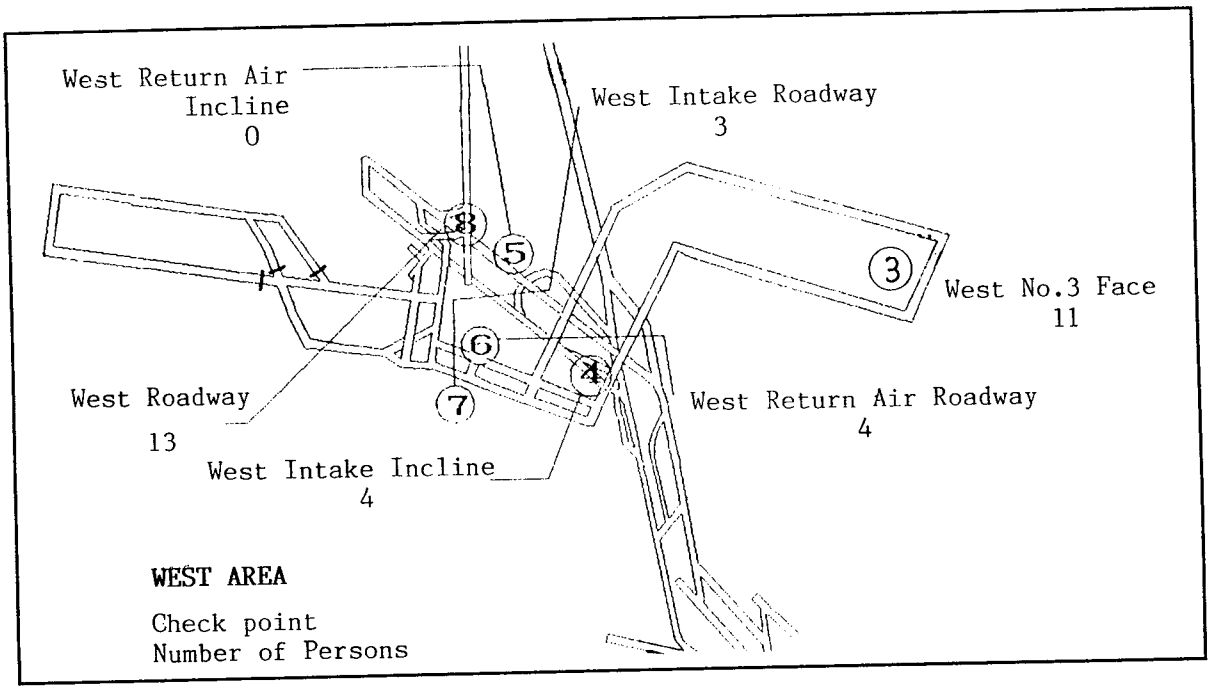


Fig. 3 Example of graphic display (map of west area check point)

No.	Check Point Name	IN
01	Pit Entrance	3
02	Hikishima Roadway	5
03	West 4 feet No.3 Face	11
04	West Intake Incline	4
05	West Return Air Incline	0
06	West Return Air Roadway	4
07	West Intake Roadway	3
08	West Roadway	13

Fig. 4 Example of graphic display (check point and number of persons)

statistically processed by a computer of the surface centralized monitoring and processing station. Moreover, screens of all kind can be displayed as occasion demand based on the data of underground personnel. Example of graphic display are shown in Fig. 3 and 4.

FIELD TEST

Field test has been demonstrated to carry on an investigation of efficiency, reliability, persistence of this system at Ikeshima colliery since November 1987.

Ikeshima colliery is located on Ikeshima island, west of Kyushu district, southern part of Japan. The geographical structure is rather simple and stable. There are 3 working coal seams with gentle dip at 1°-10°, namely 18 feet upper seam, 18 feet lower seam and 4 feet seam. Mining method is so called long-wall system combined with shield support and double ranging drum shearer.

Field test system was composed of 50 personal transmitters, 40 anttenae with frequency converters, 8 underground fixed stations, 1 surface mother station and

centralized monitoring and processing station. On the field test, check point was sellected and set up at pit entrance, intake and return air roadway, incline and thirl, face and so on. Underground fixed station is shown in Fig. 5, and example of graphic display of each check point is shown in Fig. 3 above mentioned.

CONCLUSION

As the result of field test are summarized as follows;

1. Each worker can be identified when he passes the check point without defect and error, even if plural number of workers pass in a group.

2. Personal transmitter and detection unit are of intrinsic safety explosion type and contains Ni-Cd batteries, so that this system can be operated at least 5 hours also in an emergency.

3. Personal transmitter is small-sized and light-weighted, and also fitted in the cover of reformed safety cap lamp, so that no trouble is caused during all works.

4. Movement direction is detected with arrangement of some antennae.

Now there is no production of coal without safety technology. Comprehensive monitoring of Air current (Ex. CH_4, CO, Air velocity), Rock mass(Ex. A.E., Earth pressure), Equipment (Ex. Fan, Pump, Conveyor) and etc. has been done in coal mine. Importance of monitoring the location of workers underground have a new appreciation by this system in view of mine safety. Moreover, not only workers but other moving object such as locomotive or coal tub can be detected applicatory by this system, so wide scope of utilization becomes possible.

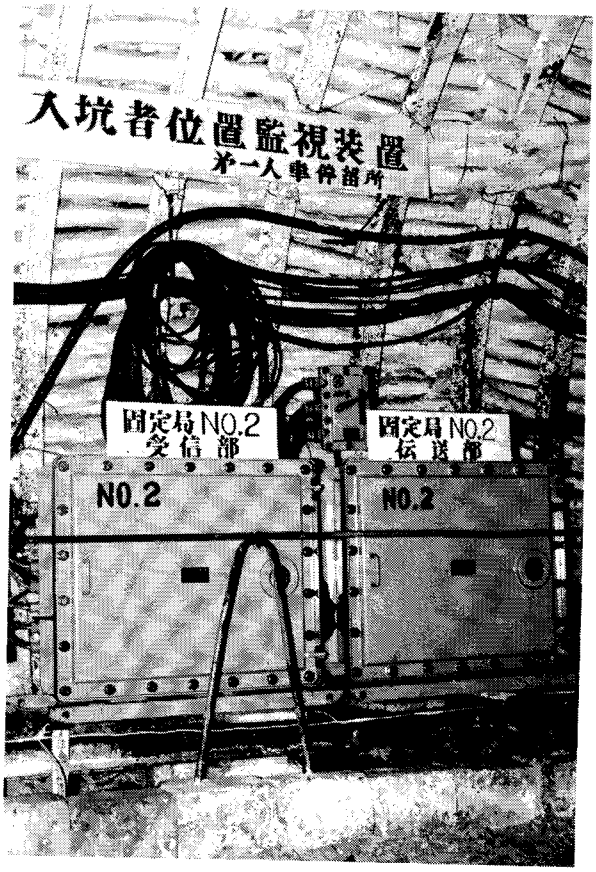


Fig. 5 Underground fixed station

ACKNOWLEDGEMENT

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DEVELOPPEMENT D'UN SYSTEME DE SURVEILLANCE DU PERSONNEL TRAVAILLANT SOUS LA TERRE

Par Shinya MATSUNAGA , Atsushi FUKUSHIMA , Katsuhiko SEO et Kouhei KUMAGAWA

L'objectif de cette recherche est de repérer continuellement l'emplacement des travailleurs miniers au moyen d'une station de surveillance et d'obtenir en cas d'urgence des informations utiles permettant de contrôler les désastres et de procéder au sauvetage des blessés.

Ce système comprend des émetteurs personnels, des antennes, des transformateurs de fréquence, des stations souterraines fixes et la station de surveillance en surface. L'émetteur personnel situé dans la lampe frontale émet en code ID à 70 MHz. Les antennes reçoivent ce code ID; celui-ci est converti en 400 MGHz par les transformateurs de fréquence, traité par la station fixe et transmis à la station de surveillance à la vitesse de 1200 bps. Toutes les données sont traitées synthétiquement par la station de surveillance puis affichées sur CRT, pour indiquer l'emplacement des travailleurs dans les galeries souterraines.

Un système permettant de détecter 50 personnes a été développé; sa démonstration a été faite à la Houillère de Ikeshima, sans produire de problèmes sérieux.

ENTWICKLUNG EINES ÜBERWACHUNGSSYSTEMS FÜR UNTERTAGE TÄTIGES PERSONAL.

Shinya Matsunaga, Dr. Atsushi Fukushima, Katsuhiko Seo, Kouhei Kumagawa

Ziel dieser Studie ist es, den Aufenthaltsort der Untertagearbeiter fortlaufend von einer zentralen Stelle aus zu überwachen und bei Unfällen brauchbare Informationen für die Rettungsmannschaften zu erhalten.

Das System besteht aus persönlichen Sendern, Antennen, Frequenzwandlern, ortsfesten Stationen untertage und der zentralen Überwachungsstation übertage. Der persönliche Sender befindet sich in der Grubenlampe und sendet die persönliche Kennung mit 70MHz aus. Das Signal wird von Antennen empfangen, von Frequenzwandlern auf 400MHz umgewandelt, in der ortsfesten Untertagestation aufgearbeitet und mit 1200 bits pro Sekunde an die Überwachungsstation übertragen. Die Daten werden in der Überwachungsstation aufgearbeitet und erscheinen auf einem Bildschirm. Damit ist Information über den Standort der Arbeiter untertage vorhanden.

Ein System zur Überwachung von 50 Arbeitern wurde entwickelt und wird ohne größere Probleme in der Ikeshima Grube demonstriert.

РАЗРАБОТКА СИСТЕМЫ НАБЛЮДЕНИЯ ЗА МЕСТОПОЛОЖЕНИЕМ ШАХТЕРОВ ПОД ЗЕМЛЕЙ.

Шинья Матсунага, Атсуши Фукушима, Катсухико Сео и Кухеи Кумагава

Цель этого исследования - иметь возможность постоянно наблюдать за местонахождением шахтеров под землей с наблюдательной станции, а также получать, в случае аварии, информацию, необходимую для их спасения.

Эта система состоит из индивидуальных передатчиков, антенн, преобразователей частоты, закрепленных станций под землей и надземной станции слежения. Индивидуальный передатчик находится в шахтерской лампе и передает личный код на частоте 70 мега-герц. Антенны получают личный сигнал, преобразователи частоты, находящиеся на закрепленных станциях под землей, преобразуют его на частоту 400 мега-герц и передают его на наблюдательную станцию со скоростью 1200 бит в секунду. Все данные обрабатываются синтетически на наблюдательной станции и воспроизводятся на экране, так что местоположение рабочих под землей может быть немедленно определено.

Завершена разработка системы наблюдения за местоположением 50 человек. Эта система была продемонстрирована на угольном месторождении Икешима, демонстрация прошла без всяких проблем.

在地下矿井工作人员侦察系统的发展

松长信也，福岛厚，濑尾胜彦， 及 能川小雨平 （均为译音）

这个研究项目的宗旨是从一个侦察站连续地确定地下矿井工作人员的位置，在控制灾害和拯救受难者的紧急情况下，这种情报便十分有用。

这个系统包括个人发送器，天线，变频器，地下的固定分站和地面的侦察站。个人发送器装在帽灯上，发送７０兆赫兹的标记信号。标记信号从天线收入，用变频器变成４００兆赫兹，由固定站整理好，以每秒１２００传送位数发送到侦察站。所有数据在侦察站上分析处理，然后显示在阴极射线管上，这样工人在地下矿井的位置的情报便可以得到。

现在可以侦察５０工作人员的系统已经发展成功，曾在池岛示范，没有重大困难。

COMPARISON OF VARIOUS APPLIANCES AND SYSTEMS FOR SELF-RESCUING AFTER FIRE AND EXPLOSION ACCIDENTS IN HARD COAL MINES; CONSEQUENCES FOR FURTHER DEVELOPMENTS

B. Prinz and H. Auberg
Ruhrkohle AG, FRG

CONTENTS

Two principals steps have to be taken after fire or explosion accidents and gas eruptions for self-rescueing:
- filtering the environmental air and converting harmful gases into innocuous gases by CO-self-rescuers;
- supplying pure oxygen from oxygen self-rescuers

Compared with oxygen self-rescuers, CO-self-rescuers have the disadvantage to offer protection only when the mine ventilation contains enough oxygen. If this proportion is sufficient, however, a long period of escape at low appliance weight is possible.

Oxygen self-rescuers which can be worn permanently, i. e. during the work as well, only have a short durability and have to be replaced in case of longer routes or periods of escape by bigger, more durable appliances.

In the hard coal mining industry, appliances and equipment have to meet different requirements depending on
- the amount of combustible materials,
- the layout of workings and
- the ventilation systems.

INTRODUCTION

The number of underground fires and explosions has steadily decreased in West German hard coal mines. Comparing the periods from 1970 to 1979 and from 1980 to 1988 shows that the total number of fires and gas and coal dust ignitions decreased from on the average 45,5/year to 28,5/year. Ignitions by mining equipment have particularly decreased (by obout 50 %), and fires by self-ignition could be reduced by about 41 per cent, whereas the number of gas ignitions remained unchanged.

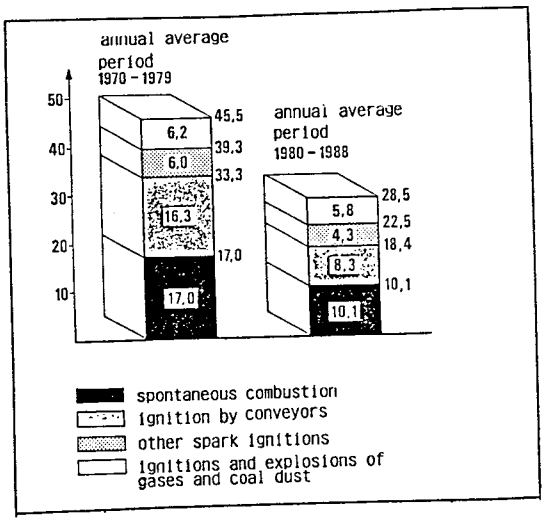


Figure 1
Fires, gas and coal dust ignitions in the hard coal mining industry of Northrhine Westfalia

Between 1956 and 1980, gas eruptions strongly increased. Due to specific research work and consequent application of preventive operational measures, their number could be essentially reduced, however. In 1986, 10 gas eruptions occurred in 5 mines, in 1987 there were 5 eruptions in 2 mines, but none was reported in 1988.

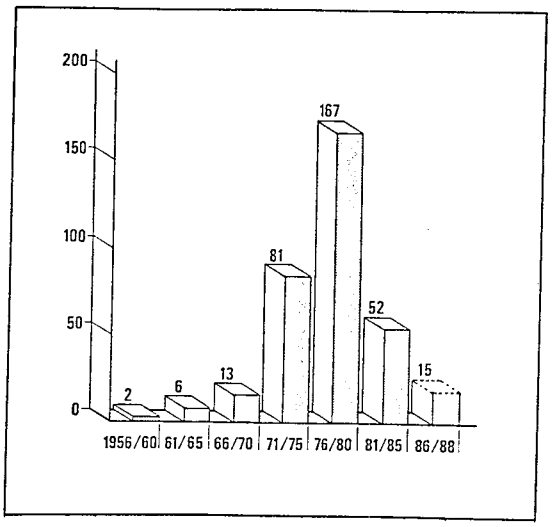


Figure 2
Gas eruptions in the hard coal mining industry of Northrhine Westfalia

This decline of greater mine combustions, explosions and gas eruptions in the hard coal mining industry of Northrhine Westfalia ist substantially based on the application of preventive measures reducing risks and restricting effects. Examples for these measures are:

- replacement of wood by fire-resistant materials;
- replacement of mineral oil by hardly combustible liquids;
- installation of conveyor belts with self extinguishing characteristics;
- dust-consolidation by solutions or pastes from hygroscopic salts;
- installation of water through explosion barriers in endangered mine positions;
- drilling of prospect holes or exploratory bores in headings and longwall systems endangered by gas eruptions.

At the same time, mines of the hard coal mining industry in Northrhine Westfalia were equipped with a dense network of continuously measuring CH_4 and CO measuring appliances. Data obtained by these appliances are delivered by remote-transmission to a permanently staffed central safety and control station, documented and mainly evaluated by process computers. This surveillance of mine ventilation currents and the systematic application of trend calculations yields a comprehensive and effective early recognition of risks.

Furthermore, all mines dispose of a dense network of telephones and loudspeakers in mining areas and haulage roads to warn employees as early as possible.

In spite of this good technical equipment of mines, a potential hazard for employees and mines by fires, gas and coal dust ignitions and gas eruptions always exists.

The global evaluation of accidents in mines shows that many miners who remained unhurt by the accident itself would have survived if they had been equipped with appropriate self-rescuers enabling them to get to the safety of the nearest unaffected ventilation current.

Each mine-fire produces CO_2 and toxic CO. In most cases, a still sufficient oxygen proportion > 17 vol % O_2 for respiration is present, however. This applies especially to mines continuous ventilation allover. The condition is, however, that the allover ventilation has not been interrupted by the accident itself.

Especially in long auxiliary ventilation systems with a high amount of combustible materials, the risk is extremely high. For example, diesel operated vehicles with rubber tyres present a high risk.

As shown by numerous explosion accidents, miners directly exposed to an explosion are in most cases not capable of rescuing themselves. Those miners, however, who only are exposed to explosion afterdamps have enough time to protect themselves by self-rescuers if they recognize immediately the indications of the explosion.

Investigating the physiological effect of carbon monoxide on human beings three influencing factors were found to determine the degree of contamination:
- CO proportion;
- respiration period volume;
- pressure reaction.

The degree of contamination is expressed by blood saturation in % CO Hb. A concentration of 500 ppm induces unconsciousness only after 90 min even in case of hard work and a respiration period volume of about 50 l/min. Higher concentrations than 2000 ppm also leave sufficient time to put on a rescuer and to get out of danger.

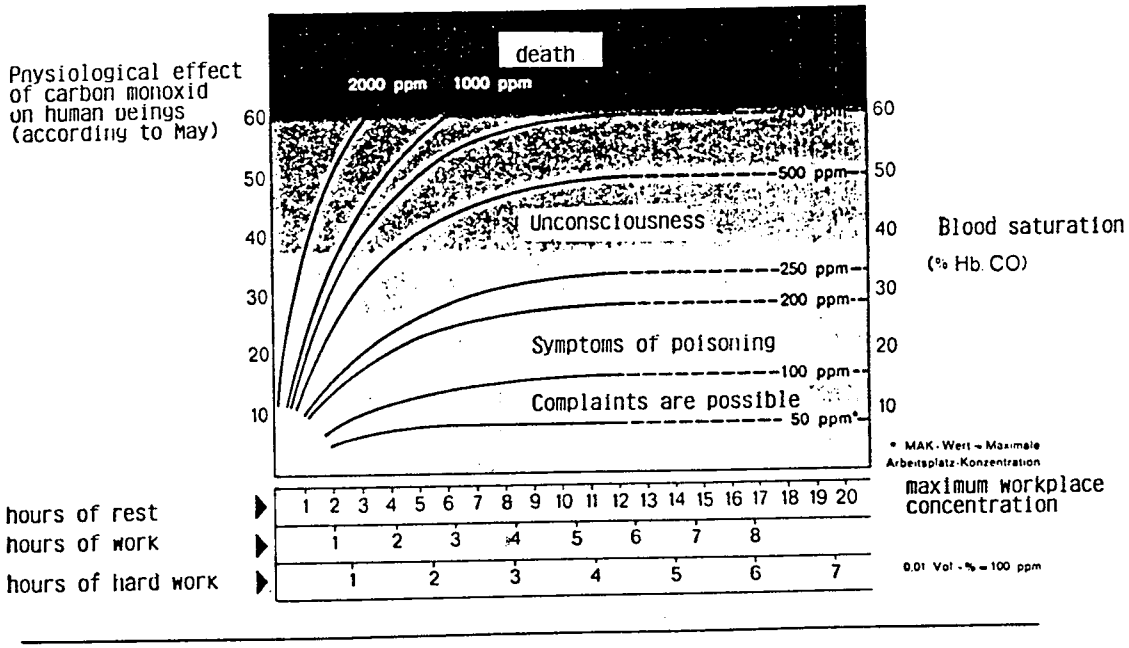


Figure 3
Physiological effect of carbon monoxide on human beings

However, sudden concentration increases above 10.000 ppm may cause an immediate paralysis of the breathing centre with fatal consequences within a few minutes. Such concentrations occur after heavy explosions and affect miners by undiluted explosion afterdamps.

Under normal conditions, as a small amount of combustible materials and a continuous ventilation, fires in mines produce cabon monoxyde concentration levels which leave enough time to use rescuers and to get out of danger. An excape may be impeded by optical limitation due to fumes, however.

In contrast, gas eruptions mostly do not produce toxic gases, but release high amounts of CO_2 or methane which is able to reduce the oxygen proportion in ventilation to an insufficient extent. In these cases, the immediate use of self-rescuers is obligatory.

Experiences with mine accidents show that the constant wearing of self-rescuers offers miners the chance to survive. Only under the condition that filters are kept ready to hand, miners are able to put them on in time. This requirement is increasingly considered in countries mining hard coal.

SELF-RESCUERS

Two types of self-rescuers have to be differentiated:

1. CO-self-rescuers
   This type converts produced CO into CO_2 by catalytic oxydation. For instance, a mixture of copper oxide and manganese peroxide (hopcalite) can serve as catalyst.
   This apparatus are required to keep specific threshold values on filtering harmful gases from environmental air. The limits are stipulated for the mining

industry in "Guidelines for construction, testing and evaluation of CO-self-rescuers for the mining industry". (Guidelines for CO-self-rescuers issued by the German Committee on mine rescue systems, on 3 June 1986)

New filters are required to keep the CO proportion of respiratory air below 200 ppm and the total CO throughput below 200 cm^3 at a test air with a CO proportion of 0,25 vol % = 2500 ppm, a water vapour proportion of 24,5 g/m^3 and breathing with a respiration period volume of 35 l/min within a minimum service life of 120 min (guarantee test).

At 1,5 vol % CO, the maximum inhalation temperature must not exceed 95° C.

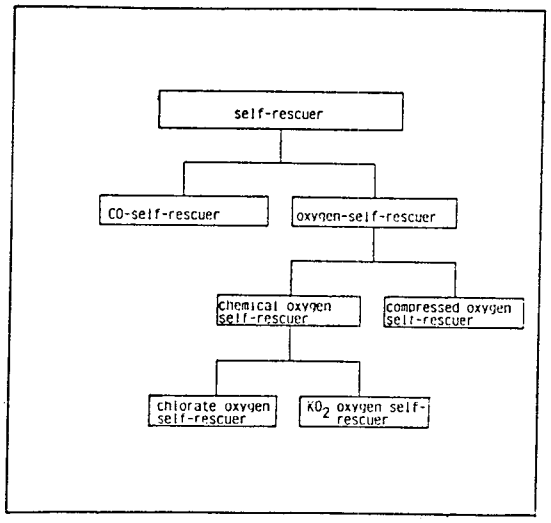


Figure 4
Self-rescuer types

2. Oxygen self-rescuers

Oxygen self-rescuers provide inhalable air irrespective of the environmental atmosphere. The oxygen reserve ist either gaseous or chemically bound.

Chemically bound oxygen reserves have hitherto applied potassium peroxide (KO_2) or chlorate ($NaClO_3$) as chemicals.

Chemical oxygen self rescuers on KO_2 basis operate on demand. Corresponding to the availability of CO_2 and water vapour in respiratory air, an equivalent amount of oxygen from the KO_2 cartridge is released.

According to "Guidelines for construction and testing of oxygen self-rescuers in the mining industry" (Guidelines for oxygen self-rescuers issued by the German committee on mine rescue systems on 15 June 1982), inhalation air has to contain 21 vol % oxygen, and the CO_2 proportion must not exceed 1,5 %. During the minimum durability, the inhalation temperature must not exceed 50° C. Slight excesses of this temperature limit are tolerated for KO_2 self-rescuers operated by dry inhalation air.

SELF-RESCUE SYSTEMS

The qualitative protective effect depends on the oxygen proportion.

- The protective effect of the CO-self-rescuer is sufficient as long as there is enough oxygen and the CO_2 proportion does not rise to a hazardous level. Moreover, the inhalation resistance must not unduely increase, e. g. by soot particles.

- An oxygen deficit requires the application of oxygen self-rescuers providing the user with O_2 irrespective of the environmental atmosphere.

The quantitative protective effect of self-rescuers demands a service life corresponding to the longest period of escape.

The system can also be designed in a way that permits the application of several instruments instead of one.

In the Federal Republic of Germany, it was felt that self-rescuers actually worn during operation should not exceed the

weight of 2 kg. In the Soviet Union for instance, a weight of 3 kg and dimensions of 250 x 135 x 135 mm are considered reasonable, however.

Parameters	DRAGER Oxyboks K Oxygen self-rescuer	AUER SSR 30/100 Oxygen self-rescuer	FENZY Biozell 1 Oxygen self-rescuer	RESPIRATOR SCH-SM1 Oxygen self-rescuer	AUER/DRAG W65/2/950 CO-self-rescuer
Dimensions					
height mm	190	180	185	170	140 | 135
width mm	205	190	180	150	110 | 110
depth mm	100	100	80	80	90 | 80
Weight					
closed kg	2,1	2,1	1,7	1,5	1,1-1,2
open kg	1,3	1,3	1,5	1,0	0,75-0,8
Service life					
35(l/min)min	33	27	29	30	> 90
50(l/min)min	20	20	20	20	
Dimensions					
weights kg open / closed	1,3 / 2,1	1,3 / 2,1	1,5 / 1,7	1,0 / 1,5	0,75 / 1,2

Figure 5
Dimensions and weights of portable self-rescuers to be worn at the belt

The decisive difference between oxygen self-rescuers and CO self-rescuers of the same size and the same weight is the durability. Service life of CO self-rescuers is 5 times higher than that of the other device. It also has to be taken into account that the protective effect of oxygen self-rescuers abruptly terminates, whereas that of CO self-rescuers gradually decreases.

The durability substantially depends on the respiration period volume of the wearer. After extensive tests perfomed by the of "Mine rescue centre" on the surface and underground, the average respiration period volume of escaping miners was determindes to be about 35 l/min for the FRG. A total time of escape of 90 min served as basis. In contrast, other industrial coumtries have calculated a respiratory period volume of 30 l/min only.

The required service life depends on underground escape routes and average velocities miners are able to obtain.

In the FRG the escape velocities of underground using CO self-rescuers also has been tested extensively by the "Mines Rescue Centre", and has been translated into standard values. These values are now preseripted by West-German Mine Authority.

In headings

dip gon	Intake roads upward m/min	Intake roads downwords m/min	return roads upwards m/min	return roads downwords m/min
0 - 5	55	55	55	55
>5 - 10	45	60	40	45
>10 - 15	35	55	30	35
>15 - 20	30	45	25	25
>20	25	30	20	20

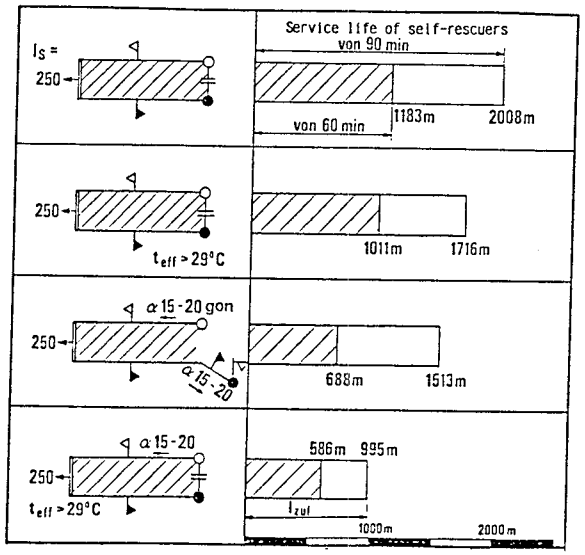
In faces

dip gon	thickness m	upwards and downwards m/min
0 - 20	≤1,00	8
	>1,00 - 1,40	10
	>1,40 - 1,40	15
	>1,80	25
>20 - 40	≤1,00 - >1,80	10

Figure 6
Escape velocities in the hard coal mining industry of FRG

The escape velocities decrease considerably when the effective temperature exceeds 29° C. The effective temperature is determined according to dry and humid temperature as well as ventilation velocities.

These parameters result in the maximum life of faces for a U-shape ventilated longwall mining:



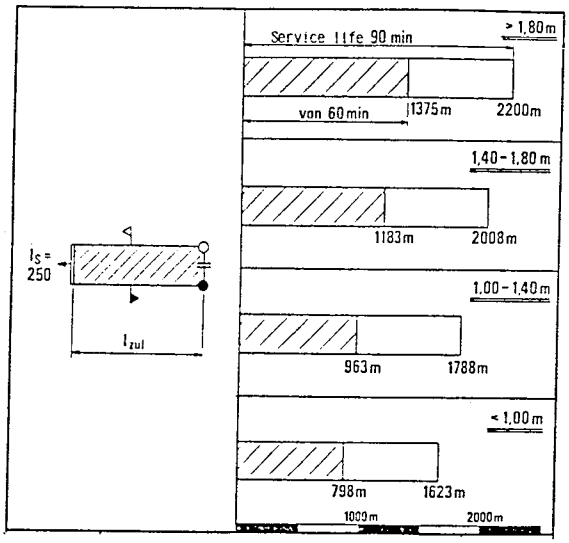


Figure 7
Dependence of the approved overall length on different escape route conditions at seam thickness of 1,40 to 1,80 m

1. As this figure shows, the durability of 60 min of a self-rescuer is only sufficient for a life of face of about 1200 m. A durability of 90 min extends the life of face to 2000 m.

2. An effective temperature in the return road of more than 29° C reduces the life of face by about 15 %.

3. If the working area has to be connected with the mine layout via a rock excavation with a length of 450 m, the approved overall lenghts decrease by about 25 % compared to the first example.

4. A dip of 15 - 20 gon and an effective temperature of less than 29° C reduces the life of face by about 50 % compared to flat seams.

Figure 8
Dependence of overall length on seam thickness

The approved life of face is also influenced by different seam thicknesses. While a thickness of more than 1,80 for a face of 250 m length requires 10 min a seam thickness below 1 m demands a period of 31 min. Under these conditions, escape attempts with shortterm oxygen self rescuers were found to demand higher respiration period volumes. Thus, a shortterm oxygen self-rescuer only covers a maximum escape period of 20 min.

Converting the ventilation system from U-shape to Y- or H-shape offers the possibility to substantially extend the approved life of face since the escape can be terminated earlier by the fresh ventilation.

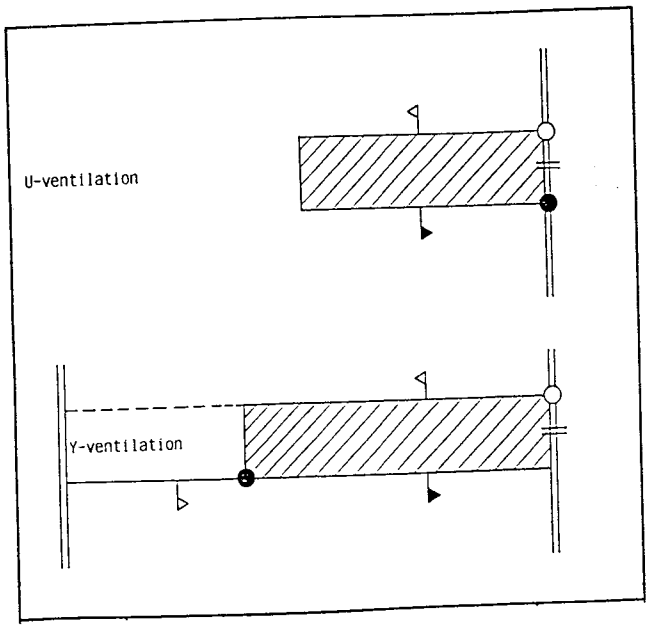


Figure 9
Extending the overall length by changing the ventilation system

Since West German hard coal mines frequently offer bad conditions due to
- a layout dependent on high rock pressure;
- difficult climate conditions;
- inclined seams;
- low thicknesses

service lives of self rescuers have to amount to at least 90 min,particularly when the personnel should be uniformly equipped with the same appliances.

This maximum permissible escape period of 90 min is reliably covered by the service life of CO-self-rescuers introduced in the West German hard coal mining industry.

Oxygen self rescuers, according to German standards light weighted enough to be worn permanently, only obtain a maximum service life of 30 min for the escape with a respiration period volume of 35 l/min. Efficient self-rescue with oxygen self-rescuers for longer escape periods than 30 min can be realized only with an extended rescue system, for instance by
- the provision of replacement stations storing additional apparatus and
- the erection of escape chambers sheltering miners until rescue.

These replacement stations or escape chambers should dispose of additional devices with the same or longer durabilities replacing used short-term oxygen self-rescuers. However, such systems require comprehensive logistics in order to ensure that replacement stations and escape chambers follow progressive mining. Furthermore, the number of stored self-rescuers has to be permanently adapted to the changing number of employees, and stored self-rescuers, have to be included in the maintenance system.

Apart from increased demands on complicated logistics, strong objections arise against the change of instruments in irrespirable atmospheres. The change of equipment does not only require miners to be familiar with the handling of these instruments - especially when using two types of instruments - but also a composed psychological state.

According to our experiences, such a change can only be performed without risk in an escape chamber provided for this purpose. It has to be questioned, however, whether panic-stricken escaping employees will determindly try to find these deposits, especially when fumes limit the visibility.

SUMMARY

The basis for a self-rescue system covering any area is the layout of underworkings and ventilation that corresponds to service lives of applied self-rescuers or the installation of an appropriate self-rescue system.

The division of the mine layout into ventilation sections by water trough barriers or other also efficient protective devices against explosions yielding many short sections is necessary, too. By these measures, the number of persons directly exposed to an explosion can be kept low, and the possibility for indirectly exposed employees to be rescued increases.

As already mentiond, self-rescuers and self rescue systems used in the hard coal mining industry of West Germany must have a service life of 90 min. Shorter periods of

escape only be obtained by reduced overall lengths or changed ventilation systems due to additional air supply.

At the present state of technique, CO-self-rescuers will continue the basic equipment for the German hard coal mining industry.
Mine layouts bearing the risk of gas eruptions or long auxiliary ventilation systems with a high amount of combustible materials, require oxygen self-rescuers. These devices should be stored on vehicles or in mobile escape chambers on the spot.

A strong formation of soot due to fire also demands the application of oxygen self-rescuers.

Therefore synthetic materials used in the West German hard coal mining industry are tested by the "mines rescue centre" before application.
They are studied to find out whether their burning behaviour reduces the protective effect of CO-self-rescuers.
Thus, many sinthetic materials are not approved for West German underground coal mining.

DEVELOPMENT TENDENCIES

The demand for increased safety yielded the following proposals for a further development and improvement of self-rescuers and self-rescue systems:

1. CO-self-rescuers
   - Development of improved training methods on opening and putting on CO-self-rescuers under conditions resembling emergencies;
   - Development of training devices for CO-self-rescuers simulating the effects by hot inhalation air;
   - Development of new CO-self-rescuers with lower temperatures of inhalation air and lower inhalation resistances.

2. Oxygen-self-rescuers
   - Improvement of self-rescuer handling, particularly opening, putting on and replacing different appliances;
   - Development of personal oxygen self-rescuers with lower weights to be worn during activities and longer service lives.

3. Self-rescuer systems
   - Development of mobile escape chambers and replacement stations with appropriate facilities for the storage and replacement of additional self-rescuers for special cases;
   - Improvement of training and instruction on the use of replacement stations and escape chambers;
   - Improvement of recognition and perception of escape chambers and replacement stations by optical and accoustic signals remaining efficient even in case of optical limitations;
   - Improvement of logistic systems for the positioning of escape chambers and their sufficient equipment with additional self-rescuers.

Regardless the intended improvement of self-rescuers and self-rescuer systems, the efforts to develop systems realizing a better, earlier and more distinct warning of employees have to be reinforced.

In the hard coal mining industry of the FRG, these development tendencies are determinedly pushed.

The mentioned objectives to protect the life and health of miners could be obtained faster if requirements to be met by self-rescuers and self-rescue systems were better coordinated.

VERGLEICH DER VERSCHIEDENEN GERATE UND SYSTEME ZUR SELBSTRETTUNG NACH BRAND ODER EXPLOSIONSUNGLÜCKEN IN STEINKOHLENBERGWERKEN UND DARAUS ZU ZIEHENDE FOLGERUNGEN FÜR WEITERE ENTWICKLUNGSARBEITEN

B. Prinz und H. Auberg
Ruhrkohle AG, FRG

INHALTSANGABE

Zur Selbstrettung nach Brand- oder Explosionsunglücken und Gasausbrüchen werden grundsätzlich zwei Wege eingeschlagen:
- Filtern der Umgebungsluft und Umwandlung von schädlichen Gasen in unschädliche Gase durch Filterselbstretter
- Zufuhr von reinem Sauerstoff aus Sauerstoffselbstrettern

Filterselbstretter haben im Vergleich zu Sauerstoffselbstrettern den Nachteil, daß sie nur bei Vorhandensein von ausreichendem Sauerstoffgehalt in den Grubenwettern Schutz bieten können. Bei genügend Sauerstoffgehalten haben sie aber den Vorteil, daß sie bei geringem Gewicht eine lange Fluchtzeit ermöglichen.

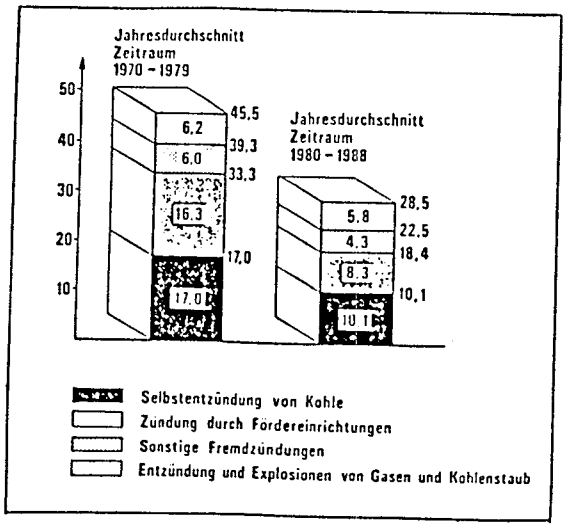
Sauerstoffselbstretter, die ständig - also auch während der Arbeit - am Körper getragen werden können, haben dagegen nur eine kurze Haltezeit und müssen bei längeren Fluchtwegen/Fluchtzeiten gegen größere Geräte mit längerer Haltezeit ausgewechselt werden.

Daraus ergeben sich für die Steinkohlenbergwerke
- je nach Brandlast und Risiko
- je nach Zuschnitt der Grubenbaue und
- je nach Zuschnitt der Wetterführung

unterschiedliche Forderungen an die Geräte und Systeme zur Selbstrettung.

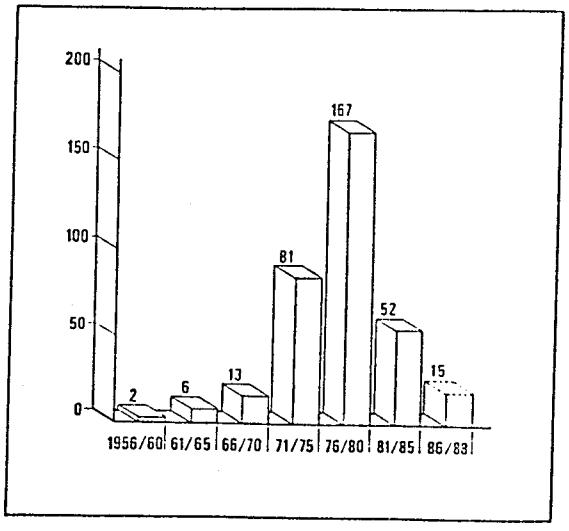
EINLEITUNG

Im westdeutschen Steinkohlenbergbau hat sich die Zahl der Grubenbrände und Explosionen ständig verringert. Eine Gegenüberstellung der Zeiträume von 1970 bis 1979 und von 1980 bis 1988 läßt erkennen, daß die Gesamtzahl der Brände und Entzündungen von Gasen und Kohlenstaub von durchschnittlich 45.5/Jahr auf 28,5/Jahr gesunken ist. An diesem Rückgang sind besonders stark die Zündungen durch Fördereinrichtungen (um 50 %) und die Grubenbrände durch Selbstentzündung (um 41 %) beteiligt, während die Entzündungen von Gasen nahezu unvermindert geblieben sind.



1. Grubenbrände sowie Entzündungen von Gasen und Kohlenstaub im nordrheinwestfälischen Steinkohlenbergbau

Die Gasausbrüche sind zunächst von 1956 bis 1980 stark angestiegen. Durch gezielte Forschung und konsequente Anwendung vorbeugender betrieblicher Maßnahmen konnte ihre Anzahl erheblich verringert werden. Im Jahre 1986 haben sich 10 Gasausbrüche auf 5 Bergwerken, 1987 noch 5 auf 2 Bergwerken und im Jahr 1988 hat sich kein Gasausbruch mehr ereignet.



2. Gasausbrüche im nordrheinwestfälischen Steinkohlenbergbau

Die rückläufige Entwicklung von größeren Grubenbränden, Explosionen und Gasausbrüchen im nordrheinwestfälischen Steinkohlenbergbau ist im wesentlichen auf die Anwendung vorbeugender Maßnahmen zurückzuführen, durch die das Risiko verringert und die Auswirkungen begrenzt worden sind. Beispielhaft seien genannt:

- der Ersatz von Grubenholz durch unbrennbare Stoffe
- der Ersatz von Mineralöl durch schwerentflammbare Flüssigkeiten
- die Einführung von Fördergurten mit verlöschender Qualität
- die Bindung von flugfähigem Kohlenstaub mit Lösungen oder Pasten aus hygroskopischen Salzen
- die Errichtung von Wassertrog-Explosionssperren an wichtigen Stellen im Grubengebäude
- das Herstellen von Erkundungs- bzw. Vorbohrlöchern in gasausbruchsgefährdeten Strecken und Langfrontbetrieben

Gleichzeitig sind die Bergwerke im nordrheinwestfälischen Steinkohlenbergbau mit einem dichten Netz aus kontinuierlich messenden CH_4- und CO-Meßgeräten ausgerüstet, deren Daten in ständig besetzte Gruben- und Sicherheitswarte fernübertragen werden und dort dokumentiert und überwiegend mit Prozeßrechnern ausgewertet werden. Diese Überwachung der Wetterströme und die systematische Anwendung von Trendberechnungen ermöglicht eine umfassende und sehr wirksame Früherkennung von gefahrbringenden Ereignissen.
Außerdem haben alle Bergwerke ein dichtes System von Telefonverbindungen und Lautsprecheranlagen in Abbaubetrieben und Förderstrecken, über die die Belegschaft sehr frühzeitig gewarnt werden kann.

Dennoch besteht trotz dieser guten technischen Ausrüstung der Bergwerke immer eine potentielle Gefahr für Menschen und Betriebe durch Brände sowie durch Entzündungen von Gasen und Kohlenstaub und durch Gasausbrüche.

Eine globale Auswertung von Grubenunglücken hat ergeben, daß viele Bergleute, die nach solchen Ereignissen zunächst unverletzt geblieben sind, überlebt hätten, wenn sie mit geeigneten Selbstrettergeräten ausgerüstet gewesen wären, die sie in die Lage versetzt hätten, sich zu Fuß in den nächsten unbelasteten Wetterstrom in Sicherheit zu bringen.

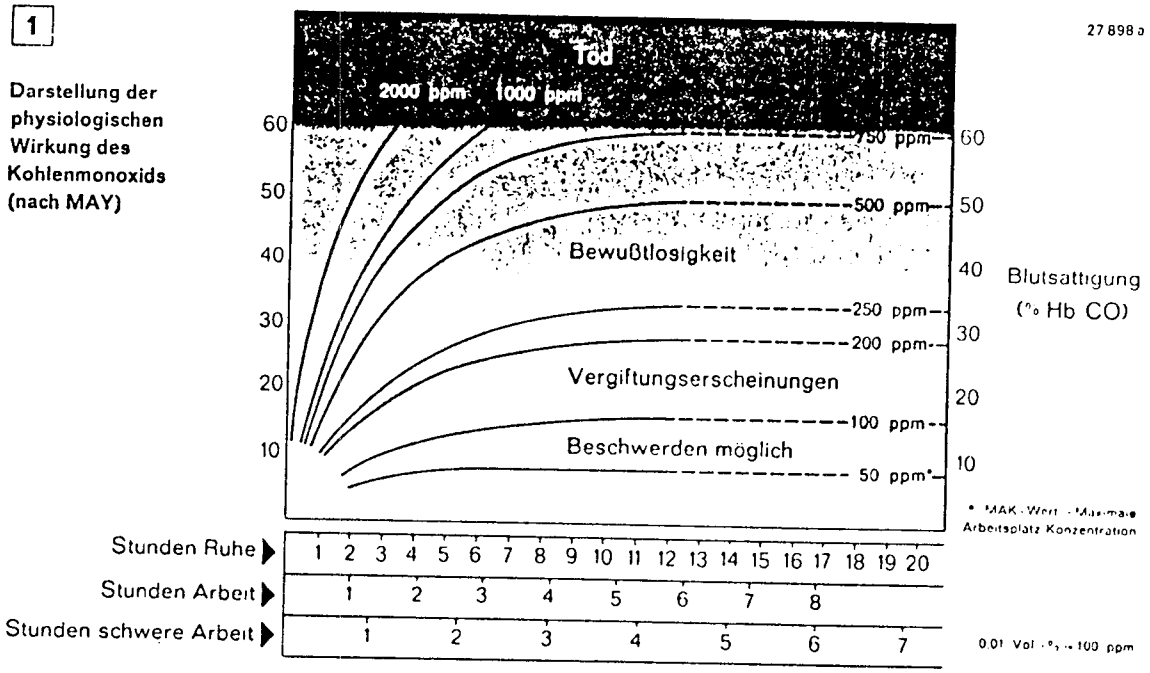
Bei jedem Grubenbrand entsteht CO_2 und das giftige CO. In den meisten Fällen ist aber noch ein ausreichender Sauerstoffgehalt von > 17 Vol % O_2 für die menschliche Atmung vorhanden. Das gilt insbesondere für durchgehend bewetterte Betriebe. Voraussetzung ist allerdings, daß die durchgehende Bewetterung durch das Ereignis selbst nicht unterbrochen worden ist.

In sonderbewetterten Betrieben ist bei einer hohen Brandlast das Risiko besonders groß. Solche Brandlasten liegen z. B. vor, wenn dieselbetriebene gummibereifte Gleislosfahrzeugen eingesetzt werden.

Die von einer Explosion unmittelbar betroffenen Bergleute sind, wie zahlreiche Explosionsunglücke in der Vergangenheit gezeigt haben, meist nicht mehr in der Lage, sich in Sicherheit zu bringen. Dagegen verbleibt denjenigen Bergleuten, die in den wettertechnisch nachgeschalteten Betrieben von den Explosionsschwaden erreicht werden, genügend Zeit, sich durch Anlegen von Selbstrettern zu schützen, sofern sie die Anzeichen der Explosion sofort erkennen.

Die Darstellung der physiologischen Wirkung des Kohlenstoffmonoxyds auf den Menschen zeigt, daß der Grad der Schädigung durch 3 Einflußfaktoren bestimmt wird:
- CO-Gehalt
- Atemzeitvolumen
- Druck der Einwirkung

Der Grad der Schädigung wird durch die Blutsättigung in % CO.Hb ausgedrückt. Bei einer Konzentration von 500 ppm tritt selbst bei schwerer Arbeit mit einem Atemzeitvolumen von rd. 50 l/min Bewußtlosigkeit erst nach 1 1/2 Stunden ein. Auch bei höheren Konzentrationen von mehr als 2000 ppm reicht die Zeit noch aus, Fluchtgeräte anzulegen und sich anschließend in Sicherheit zu begeben.



3. Physiologische Wirkung des Kohlenmonoxids auf den Menschen

Steigt die Konzentration dagegen plötzlich auf hohe Werte über 10.000 ppm an, so kann eine sofortige Lähmung des Atemzentrums eintreten, die innerhalb weniger Minuten zum Tod führt. Solche Konzentrationen treten nach heftigen Explosionen auf, wenn Bergleute von unverdünnten Explosionsschwaden überrascht werden.

Bei Grubenbränden entstehen unter normalen Bedingungen bei geringer Brandlast und durchgehender Wetterführung Kohlenstoffmonoxid-Konzentrationen, die ausreichend Zeit lassen, Selbstretter anzulegen und sich in Sicherheit zu begeben. Allerdings kann die Flucht mit Selbstrettern durch Sichtbehinderung infolge von Brandschwaden erschwert werden.

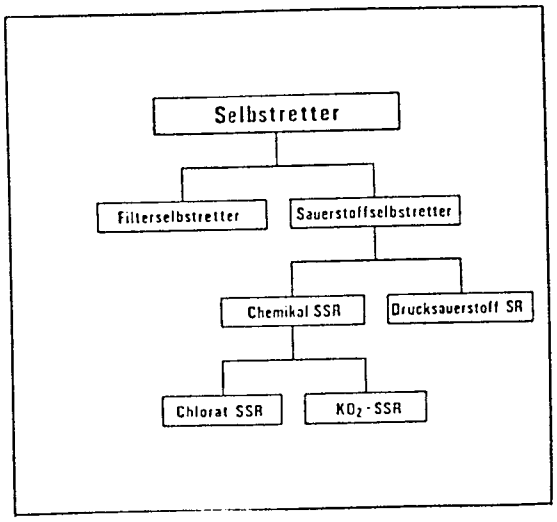
Im Unterschied dazu werden bei Gasausbrüchen meist keine giftigen Gase, sondern große Mengen von CO_2 oder Methan freigesetzt, die den Sauerstoffgehalt der Wetter soweit verringern können, daß er für eine Atmung nicht mehr ausreicht. Hier ist das sofortige Anlegen des Selbstretters zwingend erforderlich.

Erfahrungen bei vielen Grubenunglücken haben gezeigt, daß für die betroffenen Bergleute gute Chancen bestehen zu überleben, vorausgesetzt, sie tragen einen Selbstretter ständig bei sich. Nur dann, wenn das der Fall ist, kann der betroffene Bergmann den am Körper griffbereiten Filter auch rechtzeitig anlegen. Diese Forderung wird in allen Steinkohlenländern zunehmend beachtet.

SELBSTRETTER

Es sind 2 Arten von Selbstrettern zu unterscheiden:

1. Filterselbstretter
   In den Filterselbstrettern wird durch katalytische Oxydation das entstandene CO in CO_2 umgewandelt. Als Katalysator wird z. B. ein Gemisch aus Kupferoxyd und Manganperoxyd (Hopkalith) verwendet.
   Dabei müssen die Filterselbstretter beim Ausfiltern schädlicher Gase aus der Umgebungsluft bestimmte Grenzwerte einhalten. Diese Werte sind für den deutschen Bergbau in den "Richtlinien für den Bau, die Prüfung und die Bewertung von Filterselbstrettern für den Bergbau" festgelegt (FSR-Richtlinien des Deutschen Ausschusses für Grubenrettungswesen vom 03.06.1986).
   Sie verlangen, daß für neue Geräte (Garantieprüfung) bei einer Prüfluft mit einem CO-Gehalt von 0,25 Vol % = 2500 ppm und einem Wasserdampfgehalt von 24,5 g/m³ bei einer Beatmung mit einem Atemzeitvolumen von 35 l/min innerhalb einer Mindesthaltezeit von 120 min der CO-Gehalt der Einatemluft 200 ppm und der Gesamtdurchlaß an CO 200 cm³ nicht überschreiten darf.
   Die maximale Einatemtemperatur darf bei 1,5 Vol % CO nicht höher als 95° C sein.



4. **Arten von Selbstrettern**

2. Sauerstoffselbstretter
   In Sauerstoffselbstrettern wird atembare Luft unabhängig von der Umgebungsatmosphäre zur Verfügung gestellt. Dabei wird der Sauerstoffvorrat entweder in gasförmigem Zustand mitgeführt oder ist in chemisch gebundener Form vorhanden.
   Bei den chemisch gebundenen Sauerstoffvorräten sind bisher als Chemikalien Kaliumperoxyd (KO_2) oder Chlorat ($NaClO_3$) verwendet worden.

   Chemikalsauerstoffselbstretter auf KO_2-Basis arbeiten bedarfsabhängig. Dabei wird entsprechend dem Angebot von CO_2 und Wasserdampf aus der Atemluft eine äquivalente Menge an Sauerstoff aus der KO_2-Patrone freigesetzt.

Für Sauerstoffselbstretter wird in den "Richtlinien für den Bau und die Prüfung von Sauerstoffselbstrettern im Bergbau" (SSR Richtlinien des Deutschen Ausschusses für Grubenrettungswesen vom 15.06.1982) gefordert, daß die Einatemluft 21 Vol % Sauerstoff enthalten muß und der CO_2-Gehalt 1,5 % nicht überschreiten darf. Während der Mindesthaltezeit darf die Einatemtemperatur 50° C nicht überschreiten. Geringfügige Überschreitungen dieser Temperaturgrenze werden jedoch bei KO_2-Selbstrettern, die mit trockener Einatemluft arbeiten, toleriert.

SELBSTRETTUNGSSYSTEME

Der qualitative Schutzumfang der Selbstretter richtet sich nach dem Sauerstoffgehalt.

- Der Schutzumfang des Filterselbstretters reicht aus, solange genügend Sauerstoff vorhanden ist und der CO_2-Gehalt keine gefährliche Höhe erreicht; ferner darf der Einatemwiderstand sich nicht z. B. durch Rußpartikel unzulässig erhöhen.
- Bei Sauerstoffmangel sind Sauerstoffselbstretter erforderlich, die unabhängig von der Umgebungsatmosphäre den Geräteträger mit O_2 versorgen.

Der quantitative Schutzumfang der Selbstretter bedingt Haltezeiten, die die längste Fluchtzeit sicher überdecken.

Dabei kann das Rettungssystem auch so angelegt sein, daß nicht nur ein, sondern mehrere Selbstretter zum Einsatz kommen.

In der Bundesrepublik Deutschland gehen wir bisher davon aus, daß Selbstretter, die auch tatsächlich am Körper getragen werden, nicht schwerer als 2 kg sein sollten. In der Sowjetunion hält man z. B. ein Gewicht von 3 kg und Abmessungen von 250 x 135 x 135 mm noch für zumutbar.

Kennwerte	DRÄGER Oxyboks K SSR	AUER SSR 30/100 SSR	FENZY Biozell 1 SSR	RESPIRATOR SCH-SM1 SSR	AUER/DRÄG WG5'2/950 Filterselbstr.
Abmessungen:					
Höhe mm	190	180	185	170	140 | 135
Breite mm	205	190	180	150	110 | 110
Tiefe mm	100	100	80	80	90 | 80
Gewicht:					
geschlossen kg	2,1	2,1	1,7	1,5	1,1-1,2
geöffnet kg	1,3	1,3	1,5	1,0	0,75-0,8
Haltezeit:					
35(l/min)/min	33	27	29	30	> 90
50(l/min)/min	20	20	20	20	
Abmessungen					
Gewichte kg					
geöffnet	1,3	1,3	1,5	1,0	0,75
geschlossen	2,1	2,1	1,7	1,5	1,2

5. Abmessungen und Gewichte von tragbaren Selbstrettern, die am Gürtel getragen werden können

Der entscheidende Unterschied zwischen Sauerstoffselbstrettern und Filterselbstrettern besteht bei etwa gleicher Größe und gleichem Gewicht in der Haltezeit der Geräte, die bei Filterselbstrettern etwa fünfmal so groß ist. Zu beachten ist auch, daß die Schutzwirkung beim Sauerstoffselbstretter schlagartig beendet wird, dagegen beim Filterselbstretter nur allmählich abnimmt.

Die Haltezeit ist wesentlich abhängig vom Atemzeitvolumen des Gerätträgers. In der Bundesrepublik Deutschland ist - nach umfangreichen Versuchen der Hauptstelle für das Grubenrettungswesen über und unter Tage - das durchschnittliche Atemzeitvolumen flüchtender Bergleute mit etwa 35 l/min ermittelt worden. Dabei ist eine Gesamtfluchtzeit von 90 min zugrunde gelegt worden. Im Gegensatz dazu wird in einigen anderen Bergbauländer nur mit einem Atemzeitvolumen von 30 l/min gerechnet.

Die erforderliche Haltezeit ergibt sich aus der Fluchtwegsituation unter Tage und den dabei durchschnittlich zu erreichenden Fluchtgeschwindigkeiten. Die Fluchtgeschwindigkeiten mit Filterselbstrettern sind in der Bundesrepublik nach umfangreichen Versuchen, die ebenfalls von der Hauptstelle für das Grubenrettungswesen durchgeführt worden sind, als Richtgeschwindigkeit in bergbehördlichen Vorschriften übernommen worden.

In Strecken

Einfallen gon	Einziehstrecken aufwärts m/min	Einziehstrecken abwärts m/min	Ausziehstrecken aufwärts m/min	Ausziehstrecken abwärts m/min
0 - 5	55	55	55	55
>5 - 10	45	60	40	45
>10 - 15	35	55	30	35
>15 - 20	30	45	25	25
>20	25	30	20	20

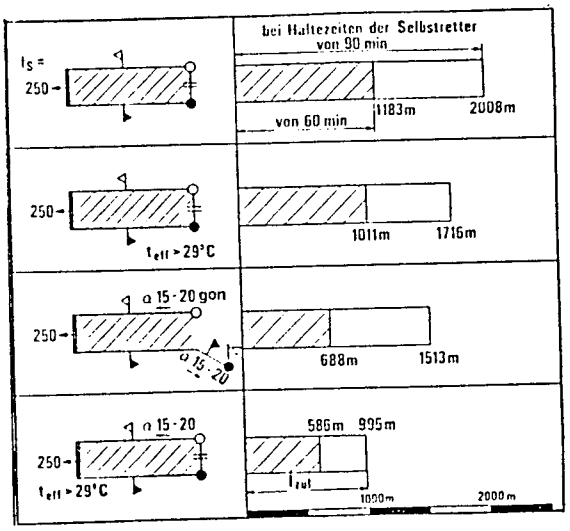
In Streben

Einfallen gon	Mächtigkeit m	auf- und abwärts m/min
0 - 20	≤1,00	8
	>1,00 - 1,40	10
	>1,40 - 1,80	15
	>1,80	25
>20 - 40	>1,00 - >1,80	10

6. Fluchtgeschwindigkeiten im westdeutschen Steinkohlenbergbau

Die gezeigten Fluchtgeschwindigkeiten verringern sich erheblich, wenn die Effektivtemperatur 29° C übersteigt. Die Effektivtemperatur wird dabei aus der Trocken- und Feuchttemperatur sowie der Wettergeschwindigkeit ermittelt.

Aus diesen Parametern ergeben sich die maximalen streichenden Baulängen für einen U-förmig bewetterten Strebbau:



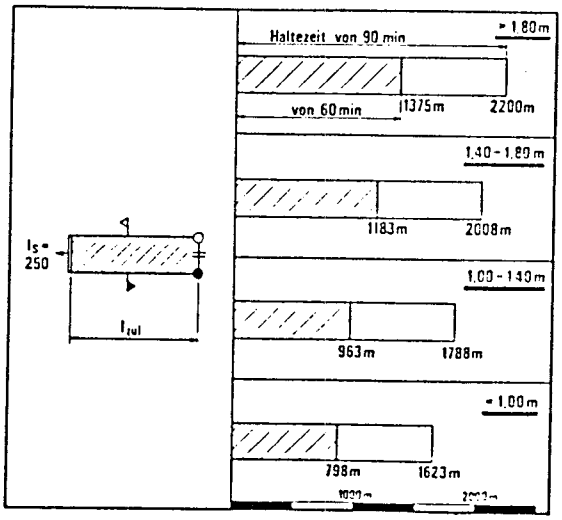
7. Die Abhängigkeit der zulässigen Baulänge von unterschiedlichen Fluchtwegbedingungen bei einer Flözmächtigkeit von 1,40 m bis 1,80 m

1. Wie aus diesem Bild zu entnehmen ist, reicht die Haltezeit eines Selbstretters von 60 Minuten nur für eine streichende Baulänge von rd. 1200 m. Bei einer Haltezeit des Selbstretters von 90 Minuten verlängert sich die streichende Baulänge auf rd. 2000 m.

2. Liegt die Effektivtemperatur in der Ausziehstrecke bei über 29° C, verkürzen sich die streichenden Baulängen um 15 %.

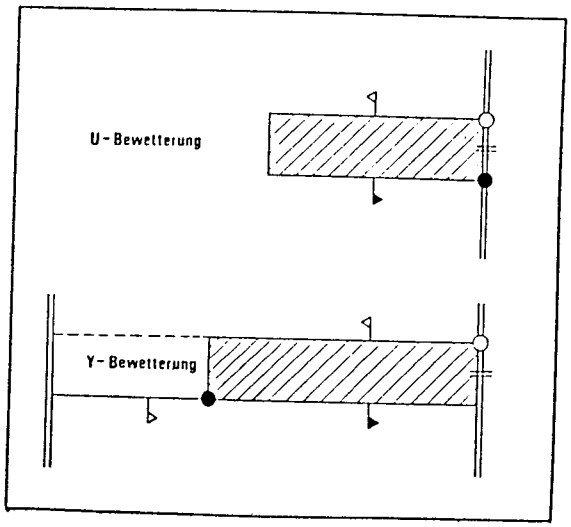
3. Muß der Abbaubetrieb z. B. über einen Gesteinsberg von 450 m Länge an das Grubengebäude angeschlossen werden, so verkürzen sich die zulässigen Baulängen gegenüber dem ersten Beispiel um rd. 25 %.

4. Bei einem Einfallen von 15-20 gon und weniger als 29° C Effektivtemperatur verkürzen sich die streichenden Baulängen auf rd. die Hälfte gegenüber den flach abgelagerten Flözbetrieben.



8. Die Abhängigkeit der Baulängen von der Flözmächtigkeit

Auch durch die unterschiedliche Flözmächtigkeit wird die zulässige Baulänge beeinflußt. Während in der Mächtigkeitsgruppe über 1,80 für einen 250 m langen Streb 10 min veranschlagt werden können, muß für eine Flözmächtigkeit unter 1 m mit einer Zeit von 31 min gerechnet werden. Bei Fluchtversuchen mit Kurzzeitsauerstoffselbstrettern wurden unter diesen Bedingungen noch höhere Atemzeitvolumen gefunden. Daher deckt der Kurzzeitsauerstoffselbstretter hier höchstens eine Fluchtzeit von 20 min ab.
Durch Umstellung des Bewetterungssystems von U- auf Y- oder H-Bewetterung kann die zulässige Baulänge allerdings entscheidend verlängert werden, weil durch den auffrischenden Wetterstrom die Flucht früher beendet werden kann.



9. Verlängerung der Baulänge durch Änderung des Bewetterungssystems

Da wir in westdeutschen Steinkohlenbergwerken häufig mit ungünstigen Bedingungen wie
- durch hohen Gebirgsdruck bedingten Zuschnitt
- erschwerten klimatischen Bedingungen
- geneigt gelagerten Flözen
- geringen Mächtigkeiten

zu tun haben, müssen die Haltezeiten für Selbstretter wenigstens 90 Minuten betragen, jedenfalls dann, wenn die Belegschaft einheitlich mit nur einem Selbstrettergerätetyp ausgerüstet werden soll.

Diese maximal zulässige Fluchtzeit von 90 min wird durch die Haltezeit der im westdeutschen Steinkohlenbergbau eingeführten Filterselbstretter sicher überdeckt.

Mit Sauerstoffselbstrettern, die nach unseren Maßstäben leicht genug sind, um ständig getragen werden zu können, kann nur eine Haltezeit von maximal 30 Minuten für die Flucht mit einem Atemzeitvolumen von 35 l/min erreicht werden. Wirksame Selbstrettung mit Sauerstoffselbstrettern für längere Fluchtzeiten als 30 Minuten kann nur möglich werden mit einem erweiterten Rettungssystem, z. B. durch

- Anlegung von Wechselstationen für zusätzliche Selbstretter und
- Errichtung von Fluchtkammern zum Aufenthalt bis zur Rettung.

In den Wechselstationen oder Fluchtkammern können zusätzliche Selbstretter mit gleicher oder längerer Haltezeit vorrätig gehalten werden, die gegen die verbrauchten Kurzzeitsauerstoffselbstretter gewechselt werden können. Solche Systeme erfordern jedoch eine umfangreiche Logistik, um die Wechselstationen und Fluchtkammern den fortschreitenden Abbaubetrieben entsprechend nachzuführen. Darüber hinaus muß die Anzahl der deponierten Selbstretter ständig der wechselnden Belegungsstärke in den Betrieben angepaßt und die vorrätig gehaltenen Geräte in das Selbstretter-Wartungssystem einbezogen werden.

Abgesehen von den erhöhten Anforderungen an eine komplizierte Logistik bestehen aber auch starke Bedenken gegen das Wechseln von Selbstrettergeräten in unatembarer Atmosphäre. Ein Gerätewechsel setzt nicht nur eine gute Übung mit den dafür erforderlichen Handgriffen - ganz besonders beim Umgang mit zwei Gerätesystemen - voraus, sondern auch eine ruhige Psyche.
Nach unseren Erfahrunge kann ein solcher Wechsel nur in einer dafür eingerichteten Fluchtkammer ohne Risiko durchgeführt werden.
Es ist aber zu fragen, ob eine in Panik flüchtende Belegschaft diese "Depots" zielstrebig aufsucht und findet, zumal wenn Rauchschwaden das Auffinden erschweren.

ZUSAMMENFASSUNG

Grundvoraussetzung für ein flächendeckendes Selbstrettungssystem ist die Planung des Zuschnitts der Grubenbaue und der Wetterführung entsprechend den Haltezeiten der eingesetzten Selbstretter bzw. die Einrichtung eines entsprechenden Selbstrettungssystems.
Dazu zählt auch die Unterteilung des Grubengebäudes in Wetterabteilungen durch Wassertrogsperren oder andere gleich wirksame Explosionsschutzsperren in möglichst viele kurze Abschnitte, damit die Zahl der durch eine Explosion unmittelbar Betroffenen möglichst klein gehalten wird und die Aussicht auf Rettung für die mittelbar Betroffenen steigt.

Für die Selbstretter und die Selbstrettungssysteme im westdeutschen Steinkohlenbergbau müssen - wie gezeigt wurde - Haltezeiten von 90 min verlangt werden. Kürzere Fluchtzeiten können nur erreicht werden, wenn die Baulängen verkürzt werden oder die Wetterführung durch zusätzliche Wettereinspeisungen verändert wird.

Im deutschen Steinkohlenbergbau dürfte - beim jetzigen Stand der technischen Entwicklung - der Filterselbstretter als Grundausrüstung auf absehbare Zeit unverzichtbar bleiben.
In Grubenbauen mit Gasausbruchsgefahr oder in sonderbewetterten Grubenbauen mit großer Brandlast, z. B. mit dieselbetriebenen gummibereiften Gleislosfahrzeugen, sind Sauerstoffselbstretter erforderlich. Diese sollen auf den Fahrzeugen und in mobilen Fluchtkammern im Vorortbereich gelagert werden.

Auch dort, wo durch Brände eine starke Rußbildung entsteht, müssen Sauerstoffselbstretter eingesetzt werden.

Die im westdeutschen Steinkohlenbergbau eingesetzten Kunststoffe werden deshalb vor ihrem Einsatz von der Hauptstelle für das Grubenrettungswesen auf ihr Brandverhalten hin untersucht, ob die Schutzwirkung der Filterselbstretter beeinträchtigt wird. Daher können viele Kunststoffe im westdeutschen Steinkohlenbergbau nicht verwendet werden.

ENTWICKLUNGSTENDENZEN

Aus der Forderung nach erhöhter Sicherheit ergeben sich folgende Vorschläge zur weiteren Entwicklung und Verbesserung von Selbstrettern und Selbstrettersystemen:

1. Filterselbstretter
   - Entwicklung von verbesserten Trainingsmethoden für das Öffnen und Anlegen von Filterselbstrettern unter ernstfallähnlichen Bedingungen
   - Entwicklung von Trainingsgeräten für Filterselbstretter, mit denen die Auswirkungen durch heiße Einatemluft simuliert werden können
   - Entwicklung neuer Filterselbstretter mit geringerer Temperatur der Einatemluft und niedrigerem Einatemwiderstand

2. Sauerstoffselbstretter
   - Verbesserung der Handhabung von Selbstrettern, besonders für das Öffnen, das Anlegen und den Wechsel von unterschiedlichen Geräten
   - Entwicklung von ständig am Mann tragbaren Sauerstoffselbstrettern mit geringerem Gewicht und längeren Haltezeiten

3. Selbstrettungssysteme
   - Entwicklung von ortsbeweglichen Fluchtkammern und Wechselstellen mit geeigneten Einrichtungen zum Aufbewahren und Wechseln zusätzlicher Selbstretter für Sonderfälle

- Verbesserung der Ausbildungs- und Informationsmaßnahmen zur Benutzung der Wechselstellen und Fluchtkammern
- Verbesserung der Erkennung und Wahrnehmung von Fluchtkammern und Wechselstellen durch optische und akustische Kennzeichen, die auch bei Sichtbehinderung wirksam bleiben
- Verbesserung logistischer Systeme zur Anordnung der Fluchtkammern und deren hinreichende Ausstattung mit zusätzlichen Selbstrettern

Unabhängig von der angestrebten Verbesserung der Selbstretter und Selbstrettungssysteme sind die Bemühungen zur Entwicklung von Systemen zu einer besseren frühzeitigen und eindeutigen Warnung der Belegschaft zu verstärken.

Im westdeutschen Steinkohlenbergbau werden diese Entwicklungstendenzen zielstrebig weiter verfolgt.
Die genannten Ziele könnten im Interesse von Leben und Gesundheit aller Bergleute schneller erreicht werden, wenn die Anforderungen an Selbstretter und Selbstrettungsystemen besser aufeinander abgestimmt werden würden.

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COMPARAISON DE DIFFERENTS DISPOSITIFS ET SYSTEMES D'AUTO-SAUVETAGE D'INCENDIES OU D'EXPLOSIONS. CONCLUSIONS CONCERNANT LES FUTURS TRAVAUX DE DEVELOPPEMENT.

Bernhard Prinz

On applique généralement deux méthodes d'auto-sauvetage:

1. Appareils à filtre: ils filtrent l'air ambiant et transforment les gaz nocifs en gaz non dangereux.
2. Appareils à oxygène: ils contiennent de l'oxygène pur.

Comparés aux appareils d'auto-sauvetage à oxygène, les appareils à filtre présentent l'inconvénient de ne protéger le mineur que lorsque la teneur en oxygène de l'air de la mine est suffisante. Mais ils sont légers et permettent un long temps de sauvetage.

D'un autre côté, les appareils d'auto-sauvetage à oxygène, que les mineurs peuvent conserver avec eux de manière permanente, c'est-à-dire également pendant leur travail, ne durent pas longtemps; ils doivent être remplacés par des appareils de taille plus importante, durant plus longtemps, si les voies/temps de sauvetage sont plus longs.

Pour des mines d'anthracite, les spécifications des appareils et des systèmes d'auto-sauvetage varient selon:

- le tracé des excavations et
- le plan de ventilation des mines.

La comparaison des divers appareils et systèmes de sauvetage sert à indiquer la direction des futurs travaux prioritaires de développement.

СРАВНЕНИЕ РАЗЛИЧНЫХ ПРИСПОСОБЛЕНИЙ И СИСТЕМ ДЛЯ САМОСПАСЕНИЯ ПОСЛЕ ПОЖАРА ИЛИ ВЗРЫВА И ВЫВОДЫ ДЛЯ ВУДУЩИХ ИССЛЕДОВАНИЙ

Бернхард Принц

В основном, два типа приборов применяются при самоспасении:

1. Фильтровые самоспасатели: фильтрование окружающего воздуха и превращение вредных газов в безвредные;
2. Кислородные самоспасатели: кислородные аппараты.

Фильтровые самоспасатели проигрывают при сравнении с кислородными, так как они могут защитить шахтера только в том случае, если содержание кислорода в рудничном воздухе достаточно высокое. В этом случае их преимущество заключается в том, что они предоставляют более длительную защиту, а также весят значительно меньше кислородных аппаратов.

В противоположность фильтровым, кислородные аппараты, которые шахтер всегда может носить при себе, функционируют короткое время и должны быть быстро заменены на приспособления с большей продолжительностью действия, если путь к спасению оказывается длиннее и дольше по времени.

Вследствие этого, в каменноугольных шахтах необходимо применение специфических приспособлений для самоспасения, которые зависят от расположения выработок и вентиляционного плана шахты.

Дается сравнение различных спасательных приспособлений и систем и показываются направления, в которых будут развиваться исследовательские работы в будущем.

火灾和爆炸时用的两种不同的自救办法和设备的比较，结论和以后研究方向

勃尔恩哈特　帕令兹

通常有两种不同的自救办法：

1．过滤式自救器：将周围的空气过滤，使有害气体变为无害

2．氧气式自救器：纯氧供应

与氧气式自救器比较，过滤式自救器的弊点是只有在矿里的氧气充足时，过滤式自救器才能保护矿工．假如氧气量充足，则过滤式自救器的利点是逃出矿井的时间较长，而且这种自救器比较轻．

氧气式自救器矿工可以经常携带．做工时亦然，但作用的时间较短，如果使用时间长则应换带大型的自救器．

在硬煤矿井里，自救器的设备和办法与下列因素有关：

- 挖掘布置情形
- 矿里通风设备

这里对不同的设备和办法做的比较是用来指出以后研究发展的方向．

Session VII

OUTBURSTS OF ROCK AND COAL, I

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INVESTIGATING THE ROLE PLAYED BY SOLID GAS-COAL SOLUTIONS IN OUTBURST HAZARD FORMATION

A. T. Airuni, I. V. Zverev, I. L. Ettinger, V. A. Sapunov, A. D. Alexeev, M. O. Dolgova, and A. V. Sinolitsky

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Reliable forecast of gas and coal sudden outbursts requires knowledge of the forms and conditions of stable existence of the "coal-gas" system at the atomic and molecular levels. Comprensive studies of gas-bearing coals (the electron microscopic analysis, electron and X-ray diffraction studies, nuclear magnetic resonance, electron paramagnetic resonance) /1-7/ enabled to establish the quasi-crystal coal base, determine characteristic sizes of coal macromolecules (in terms of the beats of the molecular density function) and establish the possibility of gas interaction with coal accompanied by formation of solid gas-coal solutions and generation of new vacancies. Average statistical atomic spacing within the range of the first three coordinating spheres have been established in terms of the function of electron density radial distribution. The role of gas dissolved in coal in the formation of the coal molecular structure has been identified. Principal crystallographic characteristics of the coal molecular structure have been established in terms of the interference curve decomposition.

Forecasting outburst hazard is of particular importance for coal mines in the Donets basin where over 85% of all outburst-prone seams in the USSR are worked. At the Donbass mines 93 to 95% of sudden outbursts of coal and methane and a hundred percent of all rock and gas outbursts occur. A total of 4689 outbursts occurred at the Donbass mines over the 1946 -1988 (6 months) period, including 2512 (57.0%) sudden

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outbursts proper (the remaining occurred during blasting operations and in machinery remote control), at 22 mines a total of 3719 outbursts of sandstone and methane occurred during blasting operations. At the USSR mines, outburst rate has been considerably reduced over the last 12-15 years and currently does not exceed 15 to 20 occurrences per year (Tabl.1,2). The total number of gas-dynamic phenomena remains great. For instance, in 1981-1988 (6 months) a total of 276 sudden outbursts of coal and methane occurred at the USSR mines, including:
256 sudden outbursts (92.8%) at the Donbass mines;
4 sudden outbursts (1.4%) at the Kuzbass mines;
5 sudden outbursts (1.8%) at the Karaganda mines;
11 sudden outbursts (4.0%) at the Vorkuta mines.

Over the last two years the total number of all gas-dynamic phenomena at the USSR mines accounted for 381 in 1986 and 351 in 1987, including 18 and 20 sudden outbursts respectively, or 4.7 and 5.6% of all gas-dynamic phenomena. Development of a reliable method to predict outburst hazard and techniques to alleviate outbursts is of top priority taking into account the nature of outburst hazard and high costs to eliminate damages caused by outbursts of coal and gas, rock and gas induced by shock blasting. It is well known that principle factors determining the outburst hazard of coal seams are as follows /1/: 1)characteristic"jump-wise" reversal distribution of rock pressure stresses in a gas-bearing coal seam; 2)energy of gas contained in coal in the free or sorbed state (adsorption, absorption); 3)coal seam physico-mechanical, physico-chemical and structural characteristics (parameters) peculiar for outburst-hazardous zones.

It has been proved /1,2/ that effects of the gas factor, physico-mechanical and structural properties of coal should be regarded only as a specific combination of strength, deformation, rheological, collecting and physico-chemical properties of the gas-bearing solid coal affecting its liability to outbursts in zones adjacent to mine workings. Changing of the nature and effect of one of the factors determining the liability to outbursts inevitably leads to relevant changes of the other factors in the system "stressed/strained coal-gas-coal micro- and macrostructure". The greater are the stresses

Table 1

Distribution of coal, gas and rock outbursts at the Donbass mines

Year	Outbursts in main workings and development headings			Sudden outbursts of coal and methane at coal faces
	outbursts of coal and methane		outbursts of sandstone and methane	
	sudden outbursts	in shock blasting		
1960	21	36	3	37
1965	24	87	109	32
1970	28	87	356	28
1975	15	139	26	9
1980	14	113	57	6
1985	11	195	13	2
1986	18	200	52	11
1987	9	225	46	3
1988 (6 months)	5	145	40	2

and the coal gas-bearing capacity, the lesser is the coal strength–the more drastic is the outburst hazard.

Gas pressure in coal leads to reduction or increase of full stresses and coal skeleton stresses by no more than 2%.

The microstructure of gas-bearing coal, one of the principal and most informative factors, has been the less investigated up to now, since the coal substance constitutents are fairly non-homogenous in structure and feature high presence of defects characterized by a great number of pores and fractures of different sizes - from microscopic to macroscopic. Gases contained in coal seams are considered to be distributed in the continuity defect volumes in the free state and sorbed on the pore and and fracture surfaces. Investigating sorption on coals enabled to establish that the majority of coals are homogenous microporous substances and there is a sharp peak in the 4-6 Å region on the differential curve of pore distribution for fossil coals. This result shows that porosity of fossil coals should be attributed to coal molecular characteristics. These

Table 2

Gas-dynamic phenomena at the Donbass coal mines

Year	Number of mines working outburst-prone seams	Number of outburst-prone seams being worked	Output from outburst-prone seams, t	Number of gas-dynamic phenomena: Total	including: due to blasting: rock & gas outbursts	including: due to blasting: in shock blasting	including: sudden coal squeezings accompanied by gas pressure increase	including: sudden outbursts in remote control *	including: sudden outbursts proper: total	outburst rate: per mln of coal output	outburst rate: per mln sq.m of extracted area
1950	34	21	1.18	168	-	74	28	-	66	56.0	42.1
1955	44	82	2.6	179	-	88	21	-	70	23.3	27.3
1960	59	115	5.0	153	3	36	39	17	58	11.6	13.6
1965	76	164	9.7	325	109	87	55	24	50	5.2	6.1
1970	84	172	20.0	619	396	96	51	29	56	2.8	3.6
1975	88	171	28.0	252	26	139	36	27	24	0.9	1.2
1980	96	207	29.5	255	57	113	42	23	20	0.7	0.6
1985	99	223	29.5	293	13	195	49	23	13	0.4	0.5
1986	98	235	29.8	399	52	222	66	32	29	0.6	0.8
1987	88	247	28.2	400	46	225	58	57	14	0.3	0.5
1988 (6 months)	104	245	15.0	282	40	145	13	17	7	0.3	0.5

* N o t e: At mines working thin steep coal seams (predominantly mines in the Central region of the Donbass)

characteristics of coal as a sorbate can be studied only if the density of the microporous substance (including pores) is known. Such studies were conducted employing X-ray diffration methods. Studuing coals with these methods is associated with the well-known problems since coals are not crystallic objects, i.e.they have no distant order in their structure. The object of the X-ray diffraction study for such systems is reduced to a solution of an integral equation, connecting the intensity of scattering X-rays and the electron density of the scatter object.

$$4\pi \cdot r^2 \cdot \rho_a(r) = 4\pi \cdot r^2 \rho_0 + 8\pi \cdot r \int_0^\infty S i(S) \sin(2\pi \cdot S \cdot r) dr$$

where r is atomic spacing; $\rho_a(r)$ are functions of atomic density distribution; $\rho_0(r)$ is average electron density;

$$S = \frac{2 \sin\theta}{\lambda}$$

θ is scattering angle;

λ is length of X-ray wave:

$$i(S) = \frac{\mathcal{J}(S) - \mathcal{J}_n(S)}{(\bar{f}_n)} - \frac{\bar{f}_n^2 - (\bar{f}_n)^2}{(\bar{f}_n)^2}$$

where $\mathcal{J}(S)$ and $\mathcal{J}_n(S)$ are experimental scattering intensity and intensity of noncoherent scattering, respectively; f_n is the average function of atomic scattering f_s for C, H, O, S and N whose content of coal was determined by chemical and spectrum analysis.

To reduce the row break effects the analysis procedure involved scattering measurement in the region of low angles on the low-angle diffractometer (according to Kratky) weth regard to collimating errors, while medium-angle measurements were carried out on the X-ray diffractometer (according to Bragg-Brentano) with vaccuum collimators and using copper and molybdenum monochromatized by the first bundle of rays. According to measurement results (Fig. 1) the diffraction function of scattering was synthesized. This function is intended for calculating the function of electron density distribution. The electron density function was additionally expanded by a series of its dome-like functions, after which variants of atomic distribution were evaluated.

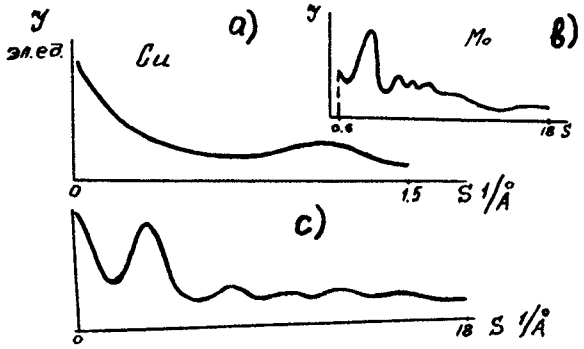


Fig.1 Synthesis of X-ray photograph according to the resulting scattering on low (a) and medium (b) angles; c-resulting X-ray photograph.

Ascertaining the structure of the vacant volume was carried out in terms of the results of the coal study involving diffusion process for introduction of "heavy" atoms of rubidium and caesium. The investigation results enabled to construct fragments of coal macromolecules, based on the concept of the mean static substructural component containing the polyarene nucleus (3-8 cycles) conjugate to aliphatic hydroaromatic structures and functional (predominantly oxygen-containing) groups. Depending on the degree of metamorphism and petrographic composition, qualitative ratios of the components are changing.

Electron donor and electron associative interactions, as well as complexes with the charge transfer play a significant part in stabilization of macromolecular formations.

The principal conclusion of X-ray diffraction studies lies in the fact that impurity atoms or molecules distributed in the coal substance by the diffusion mechanism lead to changes of the configuration coal enthropy and, thus, change its chemical potential.

System complexety predetermines its low volumetric density and fairly high elasticity. X-ray diffraction study has shown that there can be a variety of alternative spacial carbon distribution in coal associated with formation of both graphite-like (turbostratic) elements and carburan (polyhedron) structures.

Analysis of various alternative structures of coal macromolecule fragments has shown that the majority of such structures have extremely narrow interval of thermodynamic stability. The studies allowed to establish that coals in a virgin coal seam have intermolecular links differing from laboratory samples. This has been proved by additional studies of coals subjected to quasihydrostatic compression in the pressure interval of 1 to 720 MPa. (Fig. 2).

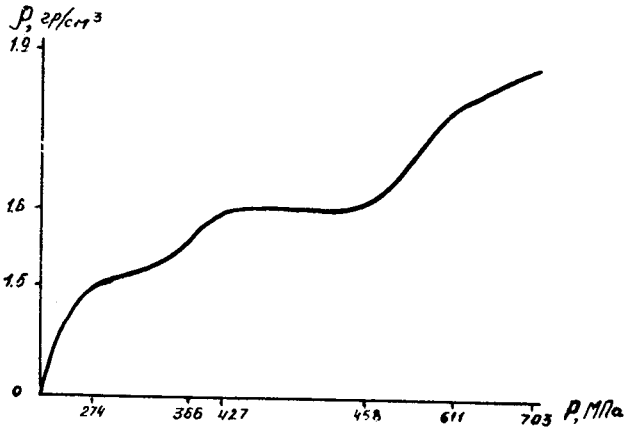


Fig. 2 Dependence of Coal density on external pressure

Drastic increase of density and structural parameters at pressures up to 80 MPa shows that in-situ coal practically does not contain macrodefects of continuity. Thus, there are no gases in the free state in the in-situ coal seam, while mass transfer is determined, mainly, by solid-state diffusion. Since in in-situ coals individual molecules and pores are not distinguished, coal is the solvent of gases being generated in metamorphism and migrated from deep levels.

Electron microscopic mass spectrometry studies have established that dissolved gas and coals form a homogenous system which has collecting properties. It was observed that thermodestruction of such regions takes course at temperatures of 130 to 180°C. Destruction results in release of molecule elements containing up to 50 % (in terms of mass) of interstitial methane. Some amount of interstitial gas in intramolecular vacancies weakens bonds in the coal molecular structure due to conformating transformations thus transferring the coal structure into the metastable state. Investigating formation of secondary solid solutions due to gas interstice from the circumpore space into the intermolecular space allowed to establish, for the first time, a new property of the coal substance, i.e. the ability to form supersaturated solid solutions of abnormal properties in the circumpore space.

The authors have developed the Kaskad physical system intended for studying structural and electron properties of gas-coal solid solutions, as well as for investigating mechanoemission processes taking course in coals under destruction. Comprehensive investigations conducted on the Kaskad set-up have established kinetic peculiarities of solid solution disintegration and coal destruction during gas emission. Gas transfer vs. electromagnetic emission relationship has been established.

The process of homogenation taking course similarly to the solid gas-coal solution in coal saturation with methane was studied on the NMR (nuclear magnetic resonance) spectrometers. It has been established that the NMR spectrum of methane-saturated coal contains two lines: a) ΔH_1=0.1-0.5 oersted depending on saturation pressure (corresponds to adsorbed methane) and b) with width ΔH_2 =5-6 oersted provided by hydrogen in the coal organics.

In long coal saturation with methane, ΔH_1 quickly stabilizes whereas ΔH_2 amplitude slowly (2-3 months) grows till reversal equilibrium is established due to solid-state diffusion. Increase of ΔH_2 amplitude shows that the homogenation takes course in the mode of solid solution in the system "coal-methane", i.e. when methane pressure in coal reaches $P = P_{cr}$, drastic methane absorption takes place accompanied by the formation of the intercallating solid gas-coal solution.

Structures of different coals, in terms of metamorphism degree, - from anthracite to bituminous coal (volatile matter content is from 4 to 30 %) were investigated in order to check the formation of solid methane -coal solutions on the NMR-spectrometer of high resolving power on the isotope ^{13}C. It has been established that: 1)the greatest maximum of absorption is in the region of 126 milliDarcy which is associated with the presence of heteroaromatic and aromatic carbons; 2)the line in the absorption region of 72 milliDarcy is referred to absorption of carbon associated with heteroatoms of oxygen, nitogen and sulphur bridge bonds; 3) only the absorption region of 18 and 20 milliDarcy is attributed to the carbon atoms of methyl groups; the intensity of absorption in the region of 18 and 20 milliDarcy increases with lower degree of metamorphism; 4)maximums in the region of 176-200 milliDarcy are referred to absorption of thioketons, carbonyl and amide groups.

In-situ coal seams are proximate to the equilibrium state in relation to some variables (e.g. temperature and pressure). Proceeding from the existence of methane-coal solid solution in the seam, it means that the equilibrium-state system consists of only one phase. Practically, all its disturbances - geological or technological leading to a break in continuity - bring the system to a non-equilibrium state with the appearance of free gas filling the vacancies formed under pressure proximate to critical, gas pressure in the existing vacancies also changes.

Though in the thermodynamics of adsorption processes the adsorbent's potential remains constant, the part played by the adsorbent is only to create the field of sorptive forces, in other words, Gibbs adsorption equation is satisfied. In case of dissolution,

the principal Gibbs-Dügem equation is satisfied:

$$m_1 d\mu_1 + m_2 d\mu_2 = 0$$

where m_1 is sorbent mass;

μ_1 is sorbent chemical potential; m_2, μ_2 are sorbate chemical potential and mass, respectively.

According to Gibbs-Dügem equation, the sorbent chemical potential changes in dissolution. The latter statement is fundamental for all processes of mass transfer and gas emission from coal seams. This implies that chemical potentials and, hence, different concentrations of gases dissolved in the coal substance vary over the seam area.

Thus, the following has been established:

1. The coal organic substance being in natural conditions of a coal seam not disturbed by mining operations is a solid gas-coal solution. Physical and physico-chemical properties of coal and gas-coal solid solution differ substantially.

2. Solid gas-coal solution is metastable and disintegrates under pressure changes leading to the evolution of gaseous phases.

3. There is a possibility of forming secondary solid solutions with higher concentrations of methane or carbon dioxide in the regions of abutment rock pressure.

4. The amount of gas in the coal seam is determined by the portion of gas dissolved in coal and not by the free volume of pores.

5. Sudden outburst of coal and gas at the stage of its origination is to be regarded as a phase transition in the solid gas-coal solution accompanied by solution disintegration, formation of continuity macrodefects and spontaneous release of gaseous products.

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ИССЛЕДОВАНИЕ РОЛИ ТВЕРДЫХ ГАЗОУГОЛЬНЫХ РАСТВОРОВ В ФОРМИРОВАНИИ ВЫБРОСООПАСНОСТИ

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Для надежного прогноза опасности внезапных выбросов газа и угля необходимо знать формы и условия стабильного существования системы "уголь-газ" на атомном и молекулярном уровнях. Комплексными исследованиями газоносных углей (электронная микроскопия, электроннография, X -ray анализ, ядерномагнитный резонанс, электронно-парамагнитный резонанс) /1-7/ установлена квазикристаллическая основа углей,определены характерные размеры макромолекул угля (по биениям функции распределения молекулярной плотности) и установлена возможность взаимодействия газов с образованием твердых газоугольных растворов и генерации новых вакансий. По функциям радиального распределения электронной плотности определены среднестатистические межатомные расстояния в пределах первых трех координационных сфер. Установлена роль растворенного в угле газа в формировании молекулярной структуры угля. По разложению интерференционной кривой установлены основные кристаллографические характеристики молекулярного строения угля.

Особенно большое значение проблема прогноза выбросоопасности имеет для Донецкого бассейна, в шахтах которого разрабатывается свыше 85% всех опасных угольных пластов СССР, происходит 93 - 95% внезапных выбросов угля и метана и практически 100 % всех выбросов породы и газа. Всего на шахтах Донбасса за период 1946-1988 гг.(6 мес.) произошло 4689 выбросов угля и метана, в том числе 2512 (57 %) собственно внезапный выброс (остальные произошли при взрывных работах и при дистанционном управлении машинами) в 22 шахтах при взрывных работах произошло 3719 выбросов песчаника и метана Частота внезапных выбросов в шахтах СССР за последние 12-15 лет значительно снизилась и в настоящее время не превышает 15-20 случаев в год (табл.1,2). В то же время общее число газодинамических явлений продолжает оставаться высоким. Так в 1981-1988 годах (6 мес.)на шахтах СССР произошло 276 внезапных выбросов угля и метана, в том числе: - на шахтах Донбасса - 256 внезапных выбросов (92,8%);
- на шахтах Кузбасса - 4 внезапных выброса (1,4 %);
- на шахтах Караганды - 5 внезапных выбросов (1,8%);
- на шахтах Воркуты - 11 внезапных выбросов (4,0 %):

В последние 2 года на шахтах СССР общее число всех газодинамических явлений составило: в 1986 г.- 388 и в 1987 г.- 351, в том числе внезапных выбросов соответственно 18 и 20 случаев или 4,7% и 5,6% от всех газодинамических явлений. Учитывая опасность внезапных выбросов и высокие затраты на ликвидацию последствий спровоцированных сотрясательным взрыванием выбросов угля и газа и породы и газа, большое значение имеет создание достоверного метода прогноза опас-

Таблица 1

Распределение выбросов метана, породы и угля на шахтах Донбасса

Год	Выбросы в капитальных и подготовительных выработках			Внезапные выбросы угля и метана в лавах
	выбросы угля и метана		выбросы песчаника и метана	
	внезапные	при сотрясательном взрывании		
1960	21	36	3	37
1965	24	87	109	32
1970	28	87	356	28
1975	15	139	26	9
1980	14	113	57	6
1985	11	195	13	2
1986	18	200	52	11
1987	9	225	46	3
1988 (6 мес.)	5	145	40	2

ности.

Основными факторами, определяющими выбросоопасность угольных пластов, являются /1/: 1) характерное "скачкообразное" знакопеременное распределение напряжений горного давления на газоносный угольный пласт; 2) энергия газа, содержащегося в угле в свободном и сорбированном (адсорбция,абсорбция) состояниях; 3) характерные для выбросоопасных зон отличия природных физико-механических, физико-химических и структурных характеристик (параметров) угольного пласта.

Доказано /1, 2/, что влияние газового фактора,физико-механических и структурных свойств угля следует учитывать только во взаимосвязи, как особое сочетание прочностных, деформационных, реологических, коллекторских и физико-химических свойств газоносного угольного массива, определяющих его выбросоопасность в прилегающих к горным выработкам зонах. Изменение характера и степени воздействия одного из определяющих выбросоопасность фактора неизменно приводит к соответствующим изменениям остальных в системе "напряженно-деформируемый уголь-газ-микро и макроструктура угля". Выбросоопасность массива тем выше, чем выше напряженность и газоносность угля и чем менее прочен уголь.

Газовое давление в угле приводит к снижению или увеличению полных напряжений и напряжений скелета угля не более, чем на 2 %.

Наименее изученным до настоящего времени был один из основных и наиболее информативных факторов - микроструктура газоносного угля, так как составляющие угольного вещества неоднородны по своей структуре и обладают высокой степенью дефектности, характеризуемой большим числом пор и трещин различных размеров - от микроскопических, до макроскопических. Считают, что газы, находящиеся в угольных пластах, распределены в объемах дефектов сплошности в свободном состоянии и сорбированы на поверхностях пор и трещин. По результатам исследования сорбции на углях было установлено, что в подавляющем большинстве уг-

Таблица 2

Газодинамические явления в угольных шахтах Донбасса

Год	Число шахт, разрабатывающих выбросоопасные пласты	Число разрабатываемых выбросоопасных шахтопластов	Добыча угля из опасных пластов, млн. т	Количество газодинамических явлений: всего	в том числе: в результате взрывных работ выбросов породы и газа	в том числе: при сотрясательном взрывании	в том числе: внезапных выдавливаний угля с повышением газовыделения	в том числе: внезапных выбросов при дистанционном управлении *	собственно внезапных выбросов: всего	собственно внезапных выбросов: частота выбросов на 1 млн.т добычи угля	собственно внезапных выбросов: частота выбросов на 1 млн. кв.м отработанной площади
1950	34	21	1,18	168	–	74	28	–	66	56,0	42,1
1955	44	82	2,6	179	–	88	21	–	70	23,3	27,3
1960	59	115	5,0	153	3	36	39	17	58	11,6	13,6
1965	76	164	9,7	325	109	87	55	24	50	5,2	6,1
1970	84	172	20,0	619	396	96	51	29	56	2,8	3,6
1975	88	171	28,0	252	26	139	36	27	24	0,9	1,2
1980	96	207	29,5	255	57	113	42	23	20	0,7	0,6
1985	99	223	29,5	293	13	195	49	23	13	0,4	0,5
1986	98	235	29,8	399	52	222	66	32	29	0,6	0,8
1987	88	247	28,2	400	46	225	58	57	14	0,3	0,5
1988 (6 мес.)	104	245	15,0	282	40	145	13	17	7	0,3	0,5

* Примечание: на шахтах, разрабатывающих точкие крутые угольные пласты (в основном шахты Центрального района Донбасса)

ли являются однородными микропористыми веществами и на дифференциальной кривой распределения пор для ископаемых углей имеется острый пик в области 4-6 Å. Этот результат свидетельствует о том, что пористость ископаемых углей следует относить к молекулярным характеристикам угля. К этим характеристикам угля как сорбата можно подойти лишь имея сведения о плотности микропористого вещества (включая поры). Изучение углей этими методами рентгеноструктурного анализа сопряжено с известными трудностями тем, что угли не являются кристаллическими объектами, то есть не имеют дальнего порядка в своей структуре. Задача рентгеноструктурного анализа для таких систем сводится к решению интегрального уравнения, связывающего интенсивность рассеяния рентгеновских лучей и электронную плотность рассеивающего объекта"

$$4\pi \cdot r^2 \rho_a(r) = 4\pi \cdot r^2 \rho_0 + 8\pi \cdot r \int_0^\infty S i(S) \sin(2\pi \cdot S \cdot r)\, dr$$

где r - межатомное расстояние; $\rho_a(r)$ - функции распределения атомной плотности; $\rho_0(r)$ - средняя величина электронной плотности; $S = \frac{2\sin\Theta}{\lambda}$, Θ - угол рассеяния; λ - длина волны рентгеновского излучения;

$$i(S) = \frac{J(S) - J_n(S)}{(\bar{f}_n)^2} - \frac{\bar{f}_n^2 - (\bar{f}_n)^2}{(\bar{f}_n)^2}$$

где $J(S)$ и $J_n(S)$ - соответственно экспериментальная интенсивность рассеяния и интенсивность некогерентного рассеяния; $\bar{f}_n$ средняя функция атомного рассеяния f_s для C, H, O, S и N, содержание которых в угле определялось химическим и спектральным анализом.

Для уменьшения влияния обрыва ряда методика анализа включала в себя измерения рассеяния в области малых углов на малоугловом вакуумированном рентгендифрактометре (по Кратки) с учетом коллимационных погрешностей, а на средних углах - на рентгендифрактометре (по Брегту-Брентано) с вакуумными коллиматорами и с использованием монохроматизированных по первичному пучку излучений меди и молибдена. По результатам измерений (рис.1) синтезировалась дифракционная функция рассеяния, по которой проводится расчет функции распределения электронной плотности. Функция электронной плотности дополнительно раскладывалась в ряд по куполообразным функциям, после чего оценивались варианты атомного распределения.

Уточнение структуры вакантного объема проводилось по результатам исследования угля, в который по диффузионной технологии вводились "тяжелые" атомы рубидия и цезия. По результатам исследований построены фрагменты макромолекул угля, в основу которых положена концепция среднестатистической субструктурной составляющей, содержащей полиареновое ядро (3-8 циклов), сопряженное с алифатическими, гидроароматическими структурами и функциональными (преимущественно кислородосодержащими) группами.

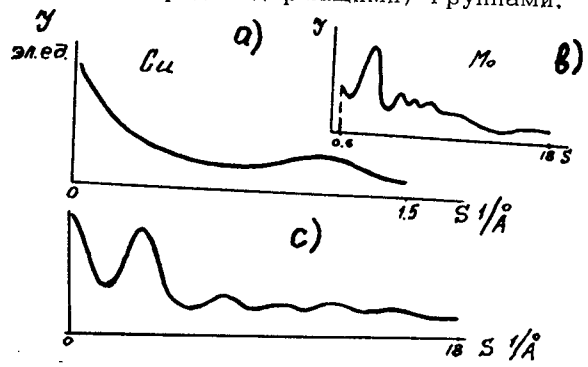


Рис.1 Синтез рентгенограммы по результирующему рассеянию на малых (а) и средних (в) углах; с -результирующая рентгенограмма.

В зависимости от степени метаморфизма и петрографической принадлежности угля, количественные соотношения составляющих компонентов меняются. Большую роль в стабилизации макромолекулярных образований играют электронно-донорные и электронноакцепторные ассоциативные взаимодействия, а так же комплексы с переносом заряда.

Основной вывод рентгено-структурных исследований состоит в том, что примесные атомы или молекулы, распределяющиеся в угольном веществе по диффузионному механизму, приводят к изменениям конфигурационной этропии углей и изменяют его химический потенциал. Сложность системы предопределяет ее низкую объемную плотность и достаточно высокую упругость. Рентгеноструктурный анализ показал, что в углях возможны различные варианты пространственного распределения углерода с образованием как графитоподобных (турбостратных) элементов, так и карборановых (полиэд-

рических) структур.

Анализ различных вариантов строения фрагментов макромолекул угля показал, что интервал термодинамической устойчивости большинства из этих структур чрезвычайно узок. В результате этих исследований установлено, что угли, находящиеся в ненарушенном горными работами угольном пласте, имеют отличные от лабораторно изучаемых образцов межмолекулярные связи. Это подтверждено исследованиями углей, подвергнутых квазигидростатическому сжатию в интервале давлений от 1 до 720 МПа (рис.2).

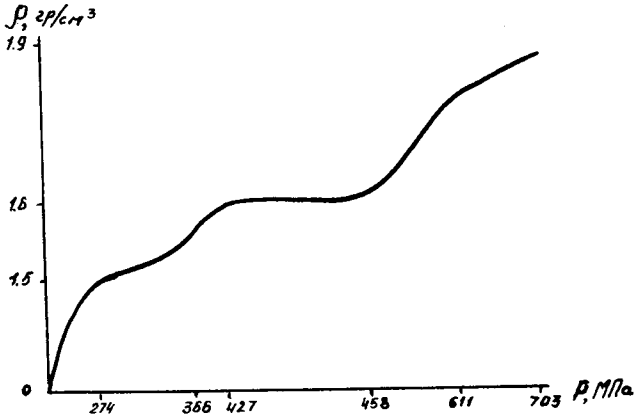


Рис.2 Зависимость плотности угля от внешнего давления

Резкое увеличение плотности и структурных параметров при давлениях до 80 МПа показывают, что в естественных условиях залегания уголь практически не содержит макродефектов сплошности. Таким образом, в угольном пласте **in Situ** практически отсутствуют газы в свободном состоянии, а массоперенос определяется в основном твердотельной диффузией. Так как в углях **in Situ** отдельные молекулы и отдельные поры не выделяются, то уголь в этих условиях является растворителем газов, генерирующихся при метаморфизме и мигрирующих с больших глубин.

Методами электронномикроскопической масспектроскопии установлено, что растворенный газ образует с углями гомогенную систему, обладающую коллективными свойствами. Наблюдалось, что термодеструкция таких областей протекает при температурах 130-180°С. В результате деструкции выделяются элементы макромолекул, содержащие до 50% (по массе) внедренного метана. Некоторое количество внедрившегося во внутримолекулярные вакансии газа ослабляет связи в молекулярной структуре угля за счет конформационных преобразований, что переводит структуру угля в метастабильное состояние. Изучением явления образования вторичных твердых растворов за счет внедрения газа из околопорового пространства пор в межмолекулярное пространство впервые установлено новое свойство угольного вещества образовывать в околопоровом пространстве пересыщенные твердые растворы с аномальными свойствами.

Для изучения структурных и электронных свойств газоугольных твердых растворов, а также для исследования механоэмиссионных процессов, протекающих в углях при разрушении, авторами разработана физическая установка "Каскад". Проведенными на установке "Каскад" комплексными исследованиями установлены кинетические особенности распада твердых растворов и деструкция угля при газовыделении. Установлены взаимосвязи между газопереносом и эмиссией электромагнитного излучения.

Процесс гомогенизации по типу твердого газоугольного раствора при насыщении угля метаном исследовался на спектрометрах ЯМР (ядерно-магнитного резонанса). Установлено, что спектр ЯМР насыщенного метана угля содержит две линии: а) шириной ΔH_1 =0,1-0,5 эрстед в зависимости от давления насыщения (соответствует адсорбированному метану) и б) шириной ΔH_2 = 5-6 эрстед, которую дает водород в органике угля. При длительном насыщении угля метаном быстро стабилизируется амплитуда ΔH_1 и медленно (2-3 месяца) растет амплитуда с шириной линии ΔH_2 - до установления сорбционного равновесного равновесия за счет твердотельной диффузии. Увеличение амплитуды ΔH_2 свидетельствует о гомогенизации по типу твердого раствора в природной системе "уголь-метан", то-есть при достижении в угле давления метана $P = P_{кр}$ наступает резкое поглощение метана с образованием интеркаляционного твердого газоугольного раствора. Для проверки образования твердых метано-угольных растворов на ЯМР - спектрометре высокого разряжения на изотопе 13_C исследовали структуры различных по степени метаморфизма углей - от антрацита до битуминозного угля (выход летучих веществ от 4 до 30 %). Установлено, что: 1) наибольший максимум поглощения

приходится на область 126 миллидарси, что связано с наличием гетероароматических и ароматических углеводородов; 2) линия в области поглощения 72 миллидарси относится к поглощению углерода, связанного с гете[illegible]ами кислорода О, азота и серными мос[illegible]ковыми связям[illegible]; 3) лишь области поглощения 18 и 20 [illegible]лидарси принадлежат атомам углерода метильных групп; интенсивность поглощения в области 18 и 20 миллидарси увеличивается при снижении степени метаморфизма; 4) максимумы в области 176-200 миллидарси относятся к поглощению тиокетонов,карбонильных групп альдегидов и амидных групп.

Угольные пласты in Situ по отношению к некоторым переменным (например, к температуре и давлению) близки к равновесному состоянию. Из существования в пласте метаноугольного твердого раствора следует, что равновесная система состоит только из одной фазы. Практически любые ее возмущения - геологические или технологические, вызывающие нарушение сплошности, приводят ее в неравновесное состояние, при этом возникает свободный газ, заполняющий под близким к критическому давлению возникшие пустоты, изме-[illegible]няется так же и давление газа в ранее существовавших пустотах.

Если в термодинамике адсорбционных процессов химический потенциал адсорбента остается постоянным, а роль адсорбента сводится лишь к созданию поля сорбционных сил, то удовлетворяется адсорбционное уравнение Гиббса. В случае же растворения газа, выполняется основное уравнение Гиббса-Дюгема:

$$m_1 d\mu_1 + m_2 d\mu_2 = 0$$

где m_1 - масса сорбента; μ_1 - химический потенциал сорбента; m_2, μ_2 - соответственно химический потенциал и масса сорбата.

Из уравнения Гиббса-Дюгема следует, что при растворении происходит изменение химического потенциала сорбента. Последнее утверждение является фундаментальным для всех процессов массопереноса и газоотдачи из угольных пластов. Из него следует, что химические потенциалы, а, следовательно, и различные концентрации растворенных газов в угольном веществе, по площади пластов различны.

Таким образом установлено:

1.Органическое вещество угля, находящееся в естественных условиях нена - рушенного горными работами угольного пласта, представляет собой твердый газоугольный раствор. Физические и физико-химические свойства угля и газоугольного твердого раствора существенно различны.

2. Твердый газоугольный раствор метастабилен и распадается при изменении давления и температуры с выделением газообразной фазы.

3. Существует возможность формиро вания вторичных твердых растворов с по вышенной концентрацией метана или углекислого газа в областях опорного горного давления.

4. Количество газа в угольном пласте определяется не свободным объемом пор, а величиной растворенного газа в угле.

5. Внезапный выброс угля и газа на стадии его зарождения следует рассматривать как фазовый переход в твердом газоугольном растворе, сопровождающийся распадом раствора,образованием макродефектов сплошности и спонтанным выделением газообразных продуктов.

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INVESTIGATION DU ROLE JOUE PAR LES SOLUTIONS SOLIDES DE GAZ-CHARBON DANS LES RISQUES DE DEGAGEMENTS

A.T. Airuni, I.V. Zverev, I.L. Ettinger, V.A. Sapunov, A.D. Alexeev, M.O. Dolgova et A.V. Sinolitsky

Afin de pouvoir déterminer correctement l'apparition des dégagements soudains de gaz et de charbon, il est nécessaire de savoir quelles sont les formes et les conditions de l'existence stable du système "charbon-gaz" aux niveaux moléculaire et atomique. Des recherches approfondies des charbons gazéifères (analyse des électrons au microscope, études par diffraction des rayons X, résonance magnétique nucléaire et résonance paramagnétique des électrons) /1-7/ ont permis d'établir la base quasi cristalline du charbon, de déterminer les tailles caractéristiques des macromolécules de charbon (en termes de battements de la fonction de densité moléculaire) et d'établir l'interaction des gaz et du charbon qui s'accompagne de la formation de solutions solides de gaz-charbon et de l'apparition de nouveaux vides. Nous avons déterminé par statistiques l'espacement atomique moyen des trois premières sphères de coordination en termes de la fonction de la répartition radiale de la densité des électrons. Nous avons établi le rôle joué par le gaz dissous dans le charbon dans la formation de la structure moléculaire de celui-ci. Nous avons également établi les principales caractéristiques cristallographiques de la structure moléculaire du charbon en termes de l'interférence de la courbe de décroissance.

ERFORSCHUNG DES EINFLUSSES VON FESTEN GAS-KOHLE-LÖSUNGEN AUF AUSBRUCH-GEFÄHRDETE FORMATIONEN

A. T. Airuni, I. V. Zwerev, I. L. Ettinger, V. A. Sapunov, A. D. Alexejev, M. O. Dolgova, A. V. Sinolitsky

Eine zuverlässige Vorhersage plötzlicher Gas- und Kohleausbrüche erfordert die Kenntnis der Formen und Bedingungen der stabilen Existenz des 'Kohle-Gas'-Systems auf atomarem und molekularem Niveau. Umfassende Studien der gashaltigen Kohlen (Elektronen-mikroskopische Analyse, Elektronen- und Röntgenstrahlen-Diffraktionsstudien, Analysen der Kernmagnet-Resonanz und der Elektronen-paramagnetischen Resonanz) /1-7/ ermöglichten es, die quasi-kristalline Basis der Kohle zu erklären, die charakteristische Größe der Kohle-Macromoleküle (ausgedrückt durch die Funktion der Molekulardichte) zu bestimmen, die Möglichkeiten einer Wechselbeziehung zwischen Gas und Kohle in Zusammenhang mit der Bildung von festen Gas-Kohle-Lösungen zu ergründen, sowie die Bildung von neuen freien Bindungsstellen zu erklären. Die durchschnittliche statistische Atomverteilung im Bereich der ersten drei Sphären wurde als Ausdruck der Funktion der radialen Elektronendichteverteilung bestimmt. Die Rolle der Gaslösung in Kohle in der Bildung der Kohle-Molekularstruktur konnte identifiziert werden. Die grundlegenden kristallographischen Charakteristiken der Molekularstruktur von Kohle wurden als Ausdruck der Interferenzkurven-Zerlegung dargestellt.

固态气体煤碳溶体在突出危害中的作用

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I。L。艾丁格， V。A。沙布诺夫，
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要对气体和煤的突出作出可靠的预测，必须要对“煤与气体”系统在原子分子等级下的形态及稳定性有相当的了结。含有气体煤碳的详细研究（电子显微镜分析，电子及X光衍射分析，核磁共振，电子顺磁共振）／1－7／能帮助建立准晶体的煤基，测定煤的大分子（用分子密度函数的频差），及建立气体及煤相互作用的可能性。这种相互作用常伴有固态气体煤碳溶体的形成与新空位的产生。在头三个配合球范围内平均统计的原子间距曾由电子密度径向分布的函数决定。气体溶于煤而形成煤分子结构的作用已经辨识。煤分子结构的主要晶体特性亦经干涉曲线分解而检定。

THE ROCK MECHANICAL BASIS OF ROCK AND GAS OUTBURSTS

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ABSTRACT

A physical model was created differing the previous ones in several respects for revealing the causes of rock and gas outbursts. The basis of this model is load-bearing the rock and gas system, the mechanical interactions of two elements of this system is placed into centre of our discussion.

The deterioration process of the rock containing gas, under special cirsumstances, takes place as a chain-reaction. The rock and gas outburst is nothing else, than a chain reaction like deterioration process of the rock face (drift, pit bottom, working).

Conditions of outbursts:
- presence of gas containing rock under pore gas pressure ahead of the face
- stress causing decrease of rock volume
- deterioration of the rock solid matrix
- the appropriate distance of the deterioration place from the rock face

Outburst occurs if all these conditions are fulfilled.

INTRODUCTION

In the category of rock and gas outbursts belongs the phenomenon when great quantity of rock and gas enters simultaneously and suddenly into the openings of the mine. Gas can be CH_4, CO_2, N_2, H_2, the gas-containing rock can be coal, salt, sandstone. In Hungary the only minework endangered by rock and gas outbursts is the Mecsek region, where metallurgical coal is mined (coal-CH_4).

GENERAL FEATURES

Outbursts are dynamic phenomena lasting to some senconds or some minutes and accompanied by strong crackling, rustling, roaring noises. In coal mining outbursts have outbursts some indications. These are: sudden increase of rock pressure, greater load of supports, srackling, rustling, sudden increase of the air's gas content; crushing out of the coal face, decrepitating, peeling of the rock; various noises.

Outbursts occure in differenct drifts, in shaft feet, in wide and narrow workings. It is a common characteristic of outbursts that they occur always in advancing,, in "fresh" heading. After the outbursts remain hollows, ducts, which may be empty or filled up and are detectable as loosened rock.

Intensity of the outbursts is characterized by the quantity

of the ejected rock (t/case, m^3/case) or by the quantity of deliberated gas (m /case). In Hungary in the Pécs region on the average intensity of the outbursts between 170-430 m depth is 10-40 m^3 rock/case or 10^3 - 4.10^3 m^3 gas/case, increasing with depth. Between 400-500 m depth the average intensity increases dramatically, in 590 m depth it reaches a value of 350 m^3 rock/case or $3,5.10^4$ m^3 gas/case. In the Pécs region the specific quantity of gas changes between 60-150 m^3 of gas/m^3 of rock (abt. 30-70 m^3/t), slowly increasing with depth. This specific quantity is about three times higher than in the Donietsk-basin.

In the case of salt-gas and sandstone-gas outbursts the specific quantity of gas is much less than at coal-gas outbursts. In the South-Harz region in the potassium salt mines deliberated specific quantity of gas is about 3-5 m^3/t, in the Werra mined abt. 8 m^3/t. In the Ruhr region the gas quantity is abt. 10 m^3/t in case of sandstone-gas outbursts. However, it must be mentioned that sandstone and salt is twice as dense as coal is.

Experiences show that outbursts are bounding to less stable sites, layers and in accordance with this, outbursts are more frequent by aprroaching disturbed zones (throws, foldings). In the case of faster gaseous rocks outbursts can occure only at higher gas-pressure, in less stable rocks a pressure of several bars is satisfactory for initiating outburst, as experienced.

According to observations of coal mining the gas-content of the layers (cm^3/g) does not determine itself the hazard of outburst, in many cases the gas-content is less, but outbursts are more frequent, furthermore in the contrary more gaseous layers are less or are not hazardous.

In salt mines it was established that in the rock salt active to burst is less internal strength, which is accompanied by larger internal friction angle. In salt mines in the active zones the gas content of the salt is 3-20 cm^3/g, which is much more than at other sites.

CHARACTERISTICS OF THE GAS-STORAGE

In gas-containing, highly gaseous rock and cracks, bonded to the walls of the pores or freely in them. Major part of the gas is bonded by physical absorption on the surface of the subcapillar gaps, since these pores have a very large specific surface. This specific surface of the supercapillar pores is incomparably less than that of the subcapillar pores, so in these pores gas exists freely.

Equilibrum of the gas bonden on the pore-walls and the free gas in the pores is determined by the pore gas pressure (p) and temperature (t). Process of absorption can be followed by means of the Langmuir monomolecular absorption theory, according to which absorption lasts until full monomolecular saturation is reached on the surface.

According to the Langmuir monomolecular absorption theory the maximally absorbed

gas (V_{max}) at a theoretic $p = \infty$ pressure is independent from the temperature, its value is determined mainly by the specific surface and the free surface energy of the solid phase relating to the given gas.

In the case of sandstone and salt gas-carriers the absorptively bonded gas plays a less important role because of the much smaller specific surface. While in the case of coal the quantity of the stored gas is great at relatively low pressures, in sandstone and is salt it is possible that great quantity of gas is stored at great pressures.

The equilibrum of the free an adsorptively bonded gas in the rock is determined definitely by the available volume , the pore gas pressure and the temperature. If only one of these parameters is altered, than occurs the altering of the conditions of the free gas and of the bonded and free gaseous phase, that is thermodynamic and physico-chemical alterations take place.

By the elastic deformation of the gas containing rock - as deformation preceeds mainly to the expense of pores and crack-systems - volume of the free gas is altered, that is this mechanical alteration (deformation) initiates thermodynamical and physico-chemical alterations in the rock-gas system.

MECHANICAL CHARACTERISTICS OF THE GASEOUS ROCK (ROCK-GAS SYSTEM RICH IN GAS)

If in the pores or in the crack-systems gas or liquid pressure rises, then the two phases are regarded as a system and the three kind of stresses are consequently differentiated. Stress conditions are discussed as pore-stress rising in the pores of neutral stress (p), as effective stress (σ) rising in the solide structure of the rock, or as cumulated or total stress ($\sigma_{\ddot{o}} = \sigma_t$).

Mechanical regularities of the rocks are valid to the effective stresses, these are

$$\sigma_x = (\sigma_x)_t - kp; \quad \sigma_y = (\sigma_y)_t - kp;$$

$$\sigma_z = (\sigma_z)_t - kp;$$

$$\tau_{x,z}; \qquad \tau_{x,y}; \qquad \tau_{x,z}$$

Aside from coalesced rocks of great stability, generally k = 1 value can be considered. Slipping stresses are not influenced by pore gas pressure.

Deformation of the rock in the presence of pore gas pressure is exclusively a functions of the effective stresses, in elastic state in the Hook's rule determining deformations are effective stresses.

Pore gas pressure considerably influences the characteristics of the rock, mainly the strength. Inside the rock strength is fatigued by effective stresses; from the viewpoint of loadability it is important what pressure can endure the rock. Therefore the strength referring to the effective stresses and referring to the accumulated (total) stresses is equally important at solving these problems.

Substituting $\sigma = \sigma_t - p$ effec-

tive stress into the linear failure curve

$$\tau = c + \sigma \mathrm{tg}\varnothing = c + (\sigma_t - p)\mathrm{tg}\varnothing$$

where c is cohesion, $\varnothing$ is internal angle of friction.

The pore gas pressure thus has effect on the strength through the internal friction mechanism in the range of the compressive loads, neutralizing a part of the accumulated normal stresses, so the internal resistance becomes less. The greater is the internal friction angle the more sensitively influences the pore gas pressure the strength.

Uniaxial compression strength as total limit-stress is

$$(\sigma_c')_t = \sigma_c - p\,(B_\varnothing - 1)$$

Uniaxial compression strength as effective limit-stress is:

$$\sigma_c' = (\sigma_c)_t - p = \sigma_c - pB_\varnothing$$

where σ_c is uniaxial compressive strength,

$$B_\varnothing = \mathrm{tg}^2\,(45^o + \frac{\varnothing}{2}).$$

That is, if the limit-curve of deterioration is linear, then by increasing the pore gas pressure the uniaxial pressure strength linearly decreases relating to both the total and effective stresses.

On the basis of 154 laboratory experiments accomplished on coal-samples from the Pécs region

$$(\sigma_c')_t = 11{,}06 - 0{,}64\ p;\quad \mathrm{MPa};\ p/\mathrm{bar}$$

where the correlation coefficient is 98,3 %.

In the natural state of the gas containing rock, when the gas can not leave or leak in the rate of deformation or volume alterations, the pressure of the gas alters together with the altering of volume and so influences the strength. Consequently in the case of gas containing rock the strength of the rock is in connection with the alteration of volume. This provides quite different nature of deterioration compared with deterioration of rock without gas pore pressure.

In rocks without pore gas pressure the process of deterioration under the effect of quasi-static load preceeds chematically as follows. As the load acting on the rock increases, in a given moment occurs deterioration at the weakest site of the reock, where loads are favourable. In this case loads are shifted to the surroundings, so deterioration occures in stronger sites, too, afterwards more load is shifted to the sound points of the rock, so on this sites also occurs deterioration. Ultimately the strongest neighbouring points are also deteriorated, so the process of deterioration comes to an end.

In the case of pore gas pressure this process is more complex. In the case of load bearing rock-gas system deterioration two kinds of precesses can be differentiated depending on it that during deterioration specific volume of the rock decreases ($\Delta V/V > 0$) or increases ($\Delta V/V < 0$). It is mentioned here that as accepted in the mechanics of rock, reduction of volume has a positive sign (compressing loads and shortening

deformations have a negative sign).

Deterioration type "A" is when the load causes reduction of the specific volume ($\Delta V/V > 0$) and the deterioration commences under such conditions. As the strong structure as load bearing element is gradually eliminated, load is shifted upon the pore gas pressure, so the later increases. Increase of the pore gas pressure results in decreasing the strength, so the process of deterioration accelerates. So deterioration of the strong structure is explosion-like whilst pore gas pressure increases.

Deterioration type "B" is when the load causes increase of the specific volume ($\Delta V/V < 0$) and deterioration commences under such conditions. As the strong structure as load bearing element is gradually eliminated, pore gas pressure decreases. Decrease of the pore gas pressure results in increasing strength, so deterioration becomes slow. Under such conditions deterioration of the rock structure takes place slowly, it is not explosion-like, meanwhile pore gas pressure is decreasing. Influence of pore gas pressure upon the velocity of deterioration is demonstrated by laboratory tests.

In the mining practice there are examples of loads accompanied by both increasing and decreasing specific volumes. On the face (in the working place, shaft foot and often in the workface, too) deterioration as a result of load occurs accompanied by reduction of volume. On the contrary deterioration of plains of the specific volume. Experiences show that outbursts occur always on the face or in the very proximity of it, at the working ends, at shaft foots, in the workface, but never on the plains of drifts and shafts, so it is obvious that reduction of the specific volume ($\Delta V/V > 0$) is an essential condition of outbursting.

OCCURENCE CONDITIONS OF OUTBURSTING

Outburst is a deterioration process of the load bearing rock-gas system, which takes place like an explosion if $\Delta V/V > 0$ condition is fulfilled.

To the occurence of outburst simultaneous satisfaction of several conditions is necessary:

1. Pore gas pressure in the gas containing rock.
2. Stress causing reduction of volume in the region in front of the face.
3. Commence of deterioration in the region in front of the face.
4. Deterioration of the rock would be not on the face but is some distance from it in a closed region.

According to the abovementioned the mechanism of outbursts is essentially as follows. The face of the cavern under drawing and the rock in front of it is exposed to overload because of the loads transferred from the cevern. Therefore the load of the region in front of the face is greater than at the distant regions and it is less favourable on the other hand. Under this overload the region in front of the face suffers a reduction of volume. Because

of the reduction of volume pore gas pressure increases in some degree. As a result of the presence of the pore gas pressure the original strength of the rock reduces. If the face in advance reaches a point in the gas containing rock where deterioration is in progress, it proceeds as follows.

At the weakest points of the region in front of the face deterioration commences or proceeds, so these points do not bear load, therefore the sound points of the rock bear greater loads, in consequence compression and pore gas pressure increase. As aresult of the greater load on the neithbourly points and increase of pore gas pressure these points are deteriorated, too, compression and pore gas pressure increase, rock strength reduces and so on. In the rock-gas system a chain-reaction-like deterioration on one hand and this deterioration is spread suddenly in a wide region. At the end of the process the result is a rock without cohesion and with a high pore gas pressure. This increased pore gas pressure expels the impermeable rock, outburst occurs during which gas stream carries a plenty of earlier deteriorated, shopped fine grains of rock into the mine area. After the outburst is commenced as a result of reduction of the original pressure - in the case of coal-gas outburst - great quantity of absorbed gas is deliberated which hinders the reduction of gas-pressure during the outburst, that is maintains the process. Adsorption of gas is provoked by vibrations occuring during deterioration of rigid rocks.

Essence of the outburst mechanism is the self-excitating, accelerating, chain-reaction-like deterioration of the load-bearing rock. Such a chain-reaction-like process is a characteristic of the majority of dynamic, natural phenomena.

Foregoings can be tracked analytically as in references (3-10).

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BASE DE LA MECANIQUE DES ROCHES DES DEGAGEMENTS ROCHEUX ET GAZEUX

Z. Somosvari

Un modèle physique des dégagements rocheux et gazeux qui a été développé diffère sur plusieurs points des modèles précédents. Il est basé sur la charge portante du système rocheux et gazeux; ce sont les interactions mécaniques de deux éléments de ce système qui sont l'objet principal de notre discussion.

Dans certaines circonstances, la détérioration des roches gazéifères est une réaction en chaîne. Les dégagements rocheux et gazeux deviennent un processus de détérioration du front de taille rocheux (galerie, fond de la fosse, excavation) semblable à une chaîne.

Circonstances des dégagements :

- présence de roches gazéifères au-dessous des pressions interstitielles des gaz à l'avant de la taille;
- contraintes provoquant une diminution du volume des roches;
- detérioration de la matrice des roches solides;
- distance appropriée entre le lieu de détérioration et le front de taille.

Un dégagement se produit lorsque toutes ces conditions sont présentes.

DIE GEBIRGSMECHANISCHEN GRUNDLAGEN DER GESTEINS- UND GASAUSBRÜCHE

Z. Somosvari

Ein physikalisches Modell für Gesteins- und Gasausbrüche wurde entwickelt, das sich in vielerlei Hinsicht von früheren Modellen unterscheidet. Die Grundlage für dieses Modell ist das lasttragende Gesteins- und Gassystem; die mechanischen Wechselbeziehungen von zwei Elementen dieses Systems stellen den Mittelpunkt unserer Diskussion dar.

Unter gewissen Umständen ist der Zerfall des gasführenden Gesteins eine Kettenreaktion. Der Gesteins- und Gasausbruch entwickelt sich zu einem kettenähnlichen Zerfallsprozeß der Gesteinsfront (Strecke, Streb, Liegendes).

Voraussetzungen für einen Ausbruch:

- Vorhandensein von gasführendem Gestein mit Porengasdruck vor dem Streb
- Spannungserzeugende Volumenverringerung des Gebirges
- Zerstörung des festen Gebirgsgefüges
- Angemessene Entfernung der Zerstörungszone von der Gesteinsfront.

Ein Ausbruch findet statt, wenn alle diese Bedingungen erfüllt sind.

РОЛЬ ГОРНО-МЕХАНИЧЕСКИХ СВОЙСТВ В ОБРАЗОВАНИИ ВЫБРОСОВ ПОРОДЫ И ГАЗА

З. Сомосвари

Несколько аспектов отличают предлагаемую физическую модель от предыдущих. Модель основана на системе "несущая порода-газ"; механическое взаимодействие между двумя элементами этой системы является предметом обсуждения. В некоторых обстоятельствах разрушение газосодержащих пород представляет собой цепную реакцию. Выбросы породы и газа становятся как бы цепным процессом разрушения стенки горной породы (в штреках, на дне карьера, в забоях). Условия выбросов: наличие на дне забоя газосодержащих пород с повышенным давлением газа в порах; уменьшение объема породы, вызываемое повышением давления; разрушение основной массы твердой горной породы; соответствующее расстояние от участка разрушения до забоя. Выброс происходит при наличии всех этих условий.

岩石与气体突出的岩石力学基础

Z。索莫士伐瑞

我们发展出一个岩石和气体突出与从前不同的物理模型．这模型的根据是承受载荷的岩石与气体的系统；这系统的两元件相互的机械作用是下面讨论的中心问题．在特殊情况下，含有气体的岩石的恶化是一种连锁反应．岩石与气体突出变为岩石表面（横坑，矿底，开采处）的连锁恶化．突出的条件如下：

- 在工作面前面有受到孔隙压力含有气体的岩石．
- 应力引起岩石体积减小．
- 固体岩石基体的恶化．
- 恶化处与岩石表面达到适当的距离．

如果上述条件全部满足，突出便会发生．

THE FIGHT AGAINST DYNAMIC PHENOMENA: FRENCH COAL MINES EXPERIENCE

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ABSTRACT

The term "dynamic phenomena" covers sudden and violent failures of the rock mass, which occur in the vicinity of mining excavations.

In the French coal mines, these phenomena occured in different forms :

- rockbursts, associated with longwall mining and resulting in a Sudden expulsion of the coal seam,
- rock outbursts, projections of rock with considerable emission of gas, which affect some roadways driven in Sandstones,
- coal outbursts and other various kinds of phenomena, combining breakage of coal with release of gas.

The incidence of these phenomena on the safety of the workings and the evolution of the phenomena observed in the course of recent years have induced an important research programme.

In each case, these studies have enabled us to define the natural and the mining factors that appear to have a determining influence on the initiation of these phenomena : tectonic structure of the sector, natural stresses and stresses induced by mining operations, nature and characteristics of the strata, behaviour of the strata...

The paper proposes a methodology of investigations, which can be adapted for the different phenomena. The aim of these investigations is to UNDERSTAND, FORECAST and PREVENT these occurrences ; for each of these aspects, the paper presents the most important results obtained, and also states future lines of investigation to be followed or to be initiated, in this field.

INTRODUCTION

Because of the diversity and the difficulties presented by natural conditions in French coal measures, a whole range of dynamic phenomena have been encountered and have had to be controlled, during mining. Two types of characteristic phenomena were observed at a very early stage and form the subject of special specifications in the "Réglement Général d'Exploitation des Mines" (General Regulations for Mining) : outbursts and rockbursts.

In the case of outbursts, the Cevennes coalfield has long held the world record for having had one of the first phenomena noted (1st April 1879), and more than 6300 recorded, with more than 1 million tonnes thrown out.

Rockbursts are a characteristic feature of the Provence measure, with a first descriptive report dated 8 March 1892 and with a frequency rising to 50 per year, in the nineteen-fifties.

Since then, other types of phenomena have occured, such as the recent rock outbursts and other seams, spared so far, have been affected. At present, we can say that 25 to 30 % of the national coal extraction comes from sectors prone to dynamic phenomena during mining.

1 - THE PHENOMENA OBSERVED

The dynamic phenomena manifest themselves by sudden failures in the rock mass around mining excavations. In comparison with traditional failures frequently observed in underground workings, these phenomena are characterised by their suddenness, their violence and, in a general way, by the extent of the affected zone. In coal mines, we can group these phenomena in three categories :

- rockbursts, that affect the coal seams worked by longwall faces,
- coal outbursts of methane or carbon dioxide, still localised in the coal workings, but where the gas is the principal motive element,
- rock outbursts, which affect roadways driven with blasting in sandstones.

1.1 - Rockbursts

Rockbursts appear like a sudden expulsion of the coal seam (coal-bump), accompanied by violent tremor, which may be felt as far as the surface, without much gas emission (Josien et al., 1982). The volume thrown out varies considerably, from some m^3 to 300 m^3 in the case of the greatest. The roof, apparently, remains intact during the phenomenon and roof closure is often limited to a few centimeters. Its surface is striated with brownish marks, signs of the friction that has occured during the expulsion. The effects may be confined to the face/gates intersection, or may, on the contrary, extend into the roadways, where long distances may be affected (more than 100 m). In this case, the phenomenon is accompagnied by a sudden and violent heavage (which may reach 1 m) and the support is blown down.

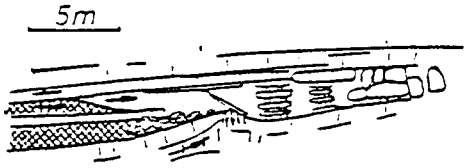
At present, these phenomena have occured in very varied types of longwall mining situations (fig. 1) : in caved level seams, in inclined filled or caved seams, and in steep filled seams (Revalor, 1987).

The diversity of these conditions and the numerous cases encountered make it possible to determine the conditions necessary for the occurrence of these phenomena (Josien et al., 1987) :

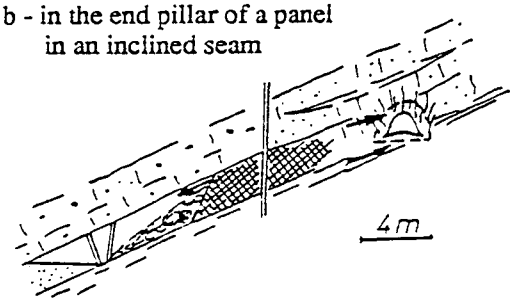
- an abnormally higt stress condition in the zone resulting from the influence of old mined panels or from natural stresses,
- a mined width sufficient to involve the overlying roof, the behaviour of which is also associated with the large faults of the sector,
- thick, hard and strong roof strata in contact with the coal seam,
- a fairly strong coal seam, often heterogeneous in structure (hard layer in the middle).

Two failure mechanisms seem to be at the origin of these phenomena (Josien et al., 1987 ; Revalor, 1987) :

a - in a level seam



b - in the end pillar of a panel in an inclined seam



c - in a steep seam

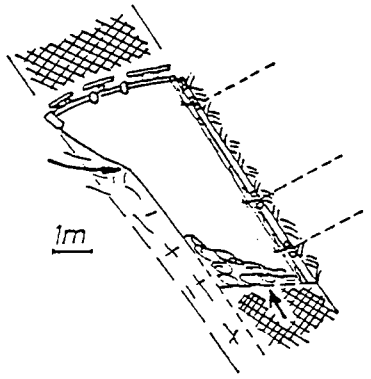


Fig. 1 : Schematic description of different types of rockburst

- a local mechanism, where the rockburst appears like a break in the coal seam, caused by the effect of the excessive stresses to which it is subjected (coal bump stricto sensu)

- a mechanism at great scale, related to the overall behaviour of the strata around longwall faces, and in which the effects observed at the level of the working appear as the consequence of breaks starting in the overlying layer.

1.2 - Coal outbursts

Allowing for the gradual abandonment of the very prone coal measures, such as the Cévennes coalfield, typical coal outbursts at present involve only a marginal proportion of the coal extraction (1 %), mainly a steep measure, with many faults, and with very irregular thick seams (average thickness 6 m). The mining method used is coal drawing with different variants (see fig. 7) and the phenomena occur during the excavation of headings in the seam or when the drawing ducts enter in the seam.

Three factors appear to be, simultaneously, necessary for the occurence of a typical outburst :

- a pattern of abnormal stresses due to the structure of the formation (natural stresses) and/or due to the layout of the mining operations,

- a type of coal that is often not very strong, with many micro-cracks and with a high concentration of desorbable gas and a high rate of desorption,

- a high gas pressure gradient ahead of the working face.

When one of these factors is absent or not completely developed, various phenomena may occur, resulting from different breaking mechanisms.

Unlike the typical outbursts, these various phenomena affect a larger number of mines, because of the increase in the depth and in the concentration of firedamp.

We may mention :

- the rockfalls, with gas emission, in steep formations with a thick seam, where the gravity is the principal motive element, and which one can combat by suitable support for the coal crown without degassing ;

- the discontinuous and sudden emissions of firedamp, which have occured in a vein of level formations, O.5 to 3 m thick, mined at a depth of 1000 m. These phenomena have consisted in a sudden projection of the standing coal, of 1 to 2 m, accompagnied by a dull noise and an emission of firedamp which may exceed

1500 m^3. These phenomena have been related to :

- overstressing related with old mined panels, located more than 300 m above the seam,

- an hard and strong roof : presence of a layer of sandstone in the roof, at a distance of less than 5 m from the seam.

- a coal seam with a high concentration of desorbable gas (16 m^3/t), but (by chance) without a lot of microcracks (fissuration index $\Delta Ps = 23$ (*), less than the danger limit, fixed at 30 for this type of coal). The less intense fissuration of the coal explains why these phenomena have not taken the same form as the traditional outbursts : the breaking mechanism is more similar to a coal bump, a sudden destressing in the seam associated with a readjustment of the roof, occurring in a very gassy vein.

(*) Upper limit risk 5 % of ΔP index (see de Vergeron, 1966).

1.3 - Rock outbursts

These are dynamic phenomena affecting roadways driven with blasting in sandstones (Josien and Revalor, 1983). They occur just after a blast, projecting rock into the roadway (the volume may be as great as several hundred m^3), with much fissuration of the standing rock immediately ahead of the face and with the release of a large volume of gas (several thousands of m^3 in the case of the largest ; fig. 2).

Since 1978, the date of the occurrence of the 1st phenomenon, 26 such incidents have been noted. The sectors concerned have the following common features :

- they are situated in virgin aeras, in "contorted" zones of the measure : folds in the seams, closing of anticlinals,

- the sandstones are not very resistant (40 MPa, when compressed, compared with the usual 100 MPa) and are very porous (6 to 11 % compared with 1 to 2 %),

- the gas is a free gas, contained under pressure in the pores of the rock. This pressure is in the region of 3 to 4 MPa.

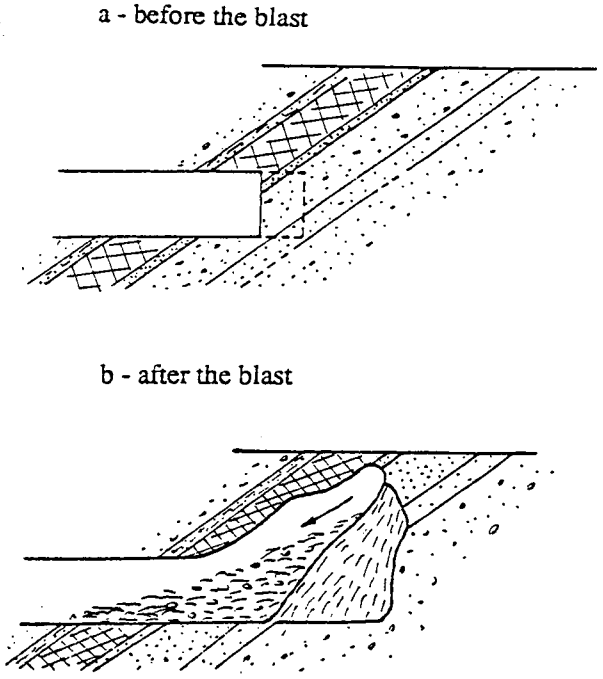


Fig. 2 : Schematic description of a rock outburst

The mechanism of failure involves both mechanical stresses and gas pressure (Revalor et al., 1985). Also, it is certain that the method of coalwinning (with blasting) used in these workings plays an important part in triggering the phenomenon. In passing through zones liable to rock outbursts, inducer shotfiring is recommended, in order to avoid any accidental phenomenon occuring when persons are present in the working.

1.4 - Characteristics common to the different phenomena

The great variety of the dynamic phenomena observed in the French coal measures is related, above all, to the diversity of the natural and the mining conditions encountered.

However, at the origin of every phenomenon, we encounter :

- **a motive element,** stresses or gas pressure or both, and the need for a build-up of energy, explaining the violence of the phenomenon : blockage of deformations and tensile failure in the case of rockbursts, low permeability and high gas pressure gradient in the case of sudden outbursts,

- particular strata, rather rigid and resistant in the case of rockbursts, often not very resistant in the case of sudden outbursts.

Understanding the failure mechanisms at the origin of each type of phenomena is important with regard to adapting the means of fight. At this stage, we have a common line of approach to all the phenomena. It aims :

- to classify the mining sectors in order to mark out the prone zones right at the planning stage,

- to detect a danger while mining operations are proceeding in a prone sector,

- to eliminate this danger, while mining, by suitable methods of treatment.

While not claiming to be exhaustive, the following presentation lists some of the things that have been done in this field.

2 - MONITORING OF THE DEPOSITS

In districts where it is not possible to exclude the risk of a dynamic phenomena, monitoring is undertaken. It is generally based on the measurement of indexes giving the conditions, of the strata (stresses, gas, etc..) in holes drilled from the mine working. As the mechanisms at the origin of the dynamic phenomena are complex, the effectiveness of monitoring is improved by combining several tests that represent the influence of different factors. In particular, seismoacoustic methods enable us to supplement the measurements made in the working, by including information on what is happening in the surrounding strata.

2.1 - Monitoring of the layout workings in coal measures subject to sudden outbursts

For the risk of sudden outbursts, desorption measurements on samples

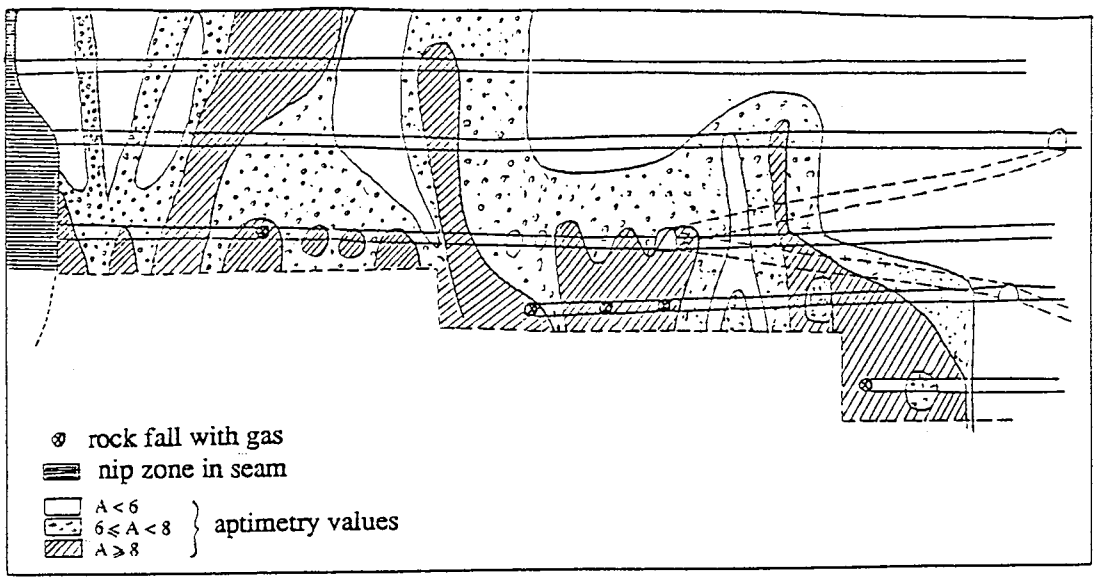


Fig. 3 : Messeix Mine Iso-aptimetry curves
(projection of workings on a vertical panel)

taken at every meter up to 4 to 6 m ahead of the face, enables us to integrate both the residual gas pressure and the tendancy of the coal to release it rapidly.

For the rapid measurement of the desorption rate of the CO_2, an apparatus known as an "aptimeter" has been developed. The method consists in measuring the increase of pressure in a reservoir of 750 cm^3 produced by the desorption of 10 g of drilling cuttings in 60 seconds.

Fig. 3 shows the plan for the aptimeter measurements made during the drifting in the district being mined.

It shows :

- the increase in the values when one approaches the nip limiting the district to the South and as the depth increases,

- the correlation between the high aptimeter values (A) and the occurrence of rock falls with gas emission.

2.2 - Assessment of the risk of rock outburst by analysing core discing

One of the most widely used methods of assessing, in the working, the risk of dynamic phenomena, consists in analysing the reaction of a hole drilled in the standing rock. In the case of rockbursts, the test hole is drilled into the seam, and the analysis is carried out on the volume of cuttings collected and, from this procedure, danger assessment standards have gradually been adapted to suit our conditions, (Josien, 1981 ; Revalor, 1987). We proceed by cored boreholes in the case of the sandstones subject to rock outbursts and, in this case, the danger is shown by the appearance of core discing in the boreholes (diameter = 76 mm). The correlation between discing in boreholes and rock outbursts is excellent : in dangerous zones, the cores of boreholes show a stack of disks with meniscus-like surface ("like a watchglass"), an average thickness of some centimeters, showing, on the external surface, numerous incipient breaks (fig. 4). The application of the method consists, for every gallery excavated in a prone area, of drilling a cored borehole in the heading, measuring 100 m in length, starting from the front of the gallery. It is assumed that the danger is great, if, in the borehole, one finds disked zones with a minimum length of 3 m and when the total number of disks (including incipient ones) is greater than 15. In our deposits, this standard makes it possible to detect the phenomena projecting more than 40 tonnes of rocks and emitting more than 200 m^3 of gas.

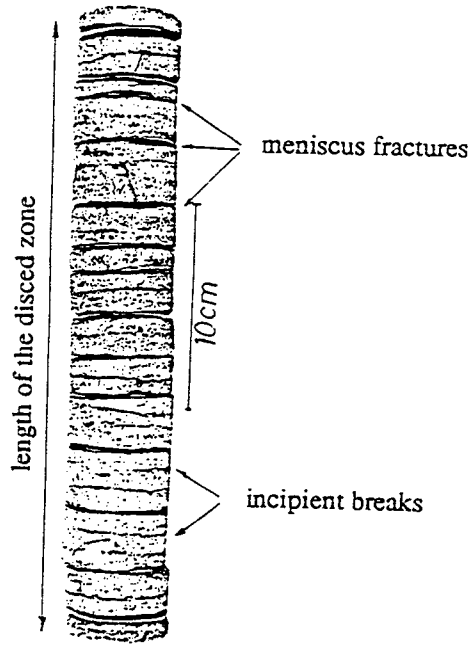


Fig. 4 : Vouters Mine - Core discing in areas prone to rock outbursts

2.3 - Seismoacoustic monitoring

The use of seismoacoustic as a supplement to the specific measurements taken in the working was introduced for the classification of zones according to their liability to sudden outbursts. Thus, in the Cevennes coalfield, the classification was based on the measurement of 2 parameters when the first headings were being driven in the sector (De Vergeron and Belin, 1966).

- the degree of fissuration of the coal seam concerned (Δ Ps index, upper limit at 5 % of the distribution of the Δ P).

- the index S measuring the seismic disturbance after blasting in preparatory workings in the sector (S is the proportion of blast followed by seismic disturbance). This index gives an idea of the pattern of stresses in the preparatory working concerned.

The figure 5 represents the characterisation of different preparatory workings in the coal field, affected or not affected by sudden outbursts, in relation to these two parameters. It is clear that these two parameters evolve conjointly. The triggering of a sudden outburst depends on the existence of considerable stresses (S high) in a coal with very few fissures (Δ Ps low) and vice versa. Also, allowance for these two parameters correctly distinguishes the two categories of preparatory workings : it thus makes it possible to define, locally, the proneness of the coal seam, specially with a view to the future workings of the panels.

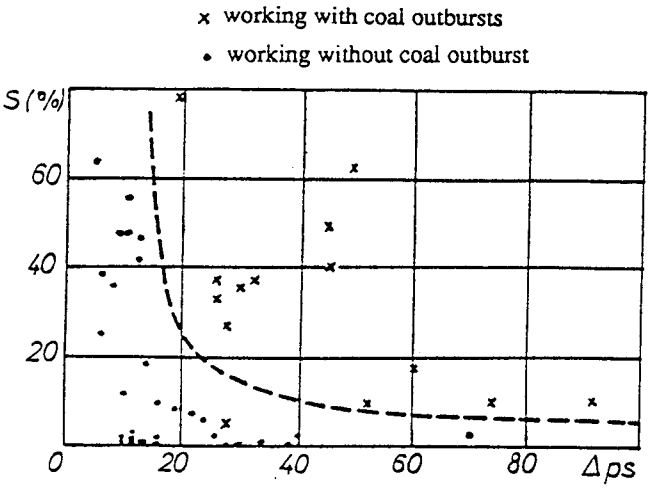


Fig. 5 : Cevennes Mine - Seismoacoustic index for classification of panels with regard to the risk of coal outbursts

In the Provence coalfield, the faces being mined are monitored daily as to the risk of rockbursts. The test-hole method is used, supplemented by seismoacoustic monitoring. Four transducers are used, installed in the gates, ahead of the face and moved periodically as the face advances. The large number of seismic events recorded (more than 100 each day) and the need for having available information quickly, made it necessary to ensure the acquisition and the automatic processing of these data. This was done with a micro-computer, (Déchelette et al., 1982 ; Revalor et al., 1986).

Observation of more than ten panels have made it possible to define a criterion enabling us to detect, when the working reaches a dangerous zone (Revalor et al., 1988). This criterion quantifies the way in which the seismoacoustic energy is released and it is assumed that a zone becomes dangerous when more than 35 % of the energy is released in violent shocks (E > 1 MJ : fig. 6). It is important to emphasise that such a criterion is implemented in situ, where it is examined every day, so as to be integrated in the mining decisions.

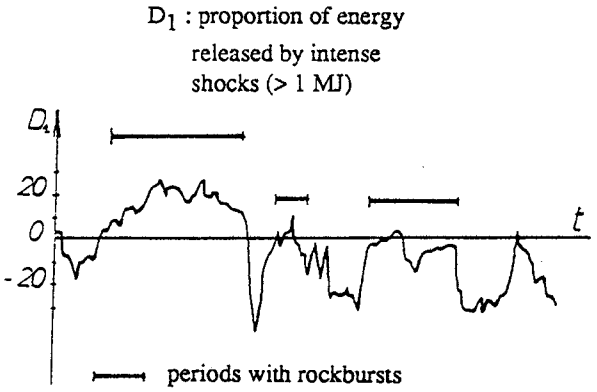


Fig. 6 : Provence Mine - Seismoacoustic detection of zones prone to rockbursts

3 - PREVENTION OF THE PHENOMENA

The procedures for combatting dynamic phenomena involve destressing the strata both with regard to coal outbursts, in which case the permability of the strata must be increased, and also with regard to rockbursts, in which case the mechanical stresses must be reduced.

In densely coal fields containing several seams, this intention to destress the strata

must be allowed for when the mining operations are being planned, selecting the first seam being mined (protective or destressing seam) the least prone seam in the group. However, at this level, there is a great number of constraints, which means that it is not always possible to apply this principle correctly. It is thus necessary to have available, suitable mining or destressing methods, implemented in the working.

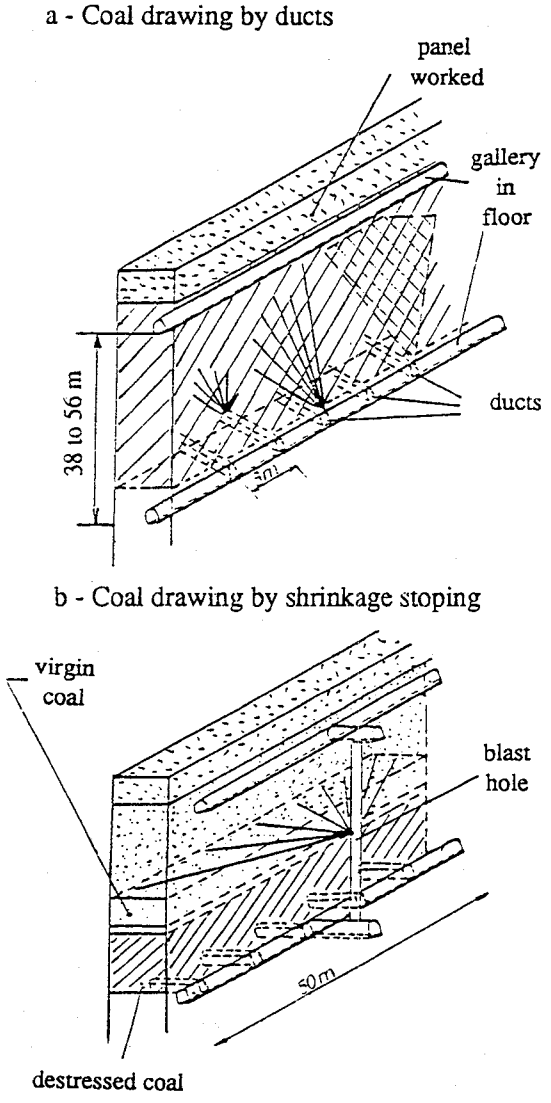


Fig. 7 : Dauphine Mine - Coal drawing in sectors very prone to coal outbursts

3.1 - Adaptation of mining methods in steep seams with coal outbursts

In the Dauphiné coalfield, with thick and steep seams, mining methods with coal drawing are perfectly fitted with the characteristics of mined coal ; moreover, they allow a good recovery of the coal reserves.

In the prone sectors of the deposit, the basic method consists in excavating a gallery in the floor of the seam to be worked and, from this gallery, reaching the seam via horizontal ducts, 8 m apart (fig. 7a). Coal drawing is started from these ducts, by long shotholes and operated using a tyre-mounted Diesel operated/conveyor ; the coal drawing is carried on at a height between 38 m and 56 m according to the sector.

In sectors very susceptible to sudden outbursts, two variations to this method have been devised, in order to ensure safety, while increasing productivity in the workings :

a) the first variation consists in loading the heap of debris with the aid of a remote-controlled gathering arm loader. The control point is in a recess in the gallery in the floor, about a hundred meters from the working, upwind of the ventilation. The key points in the operation are viewed by means of a television circuit. The control point is equipped with devices for supplementary safety control, informing the operator of the development of certain parameters during the coal drawing : monitoring of the atmosphere, noises from the rock mass... Any intervention at the actual site of the coal drawing is very limited, which means that the persons are protected almost continuously throughout the operation.

b) based on the same system (gallery in the floor and ducts), the second variant (shrinkage stoping) consists in making a big hole every 50 m, in the floor of the seam (fig. 7b). This makes it possible to have a fanshaped shotfiring arrangement in the seam, consisting of successive horizontal layers. With each blast, one extracts, through the ducts, only the tonnage corresponding to the

increased volume of material produced by the blast, which avoids the creation of large empty spaces in the chamber. On completion of this phase, the drawing can proceed safely, via the ducts at the base, the coal being completely destressed and degassed.

3.2 - Destressing by drilling large holes

Destressing by drilling large diameter holes in the coal seam has been successfully applied in different conditions in French mines.

In the Cevennes coalfield, it was an effective method of prevention, applied in different situations : headings, face, passage through the seam. The length of the holes was 20 m, their diameter varying between 115 and 140 mm and they were between 2.5 and 4 m apart. The drilling frequency depended on the rate at which the working advanced ; a small protective band, estimated at 5 m, had to be left ahead of the face. In this case, the effectiveness of the method was due to its combined destressing-degassing effect in the coal seam.

In the Provence coalfield, this method has, in recent years, made it possible to control the rockbursts that affect longwall mining.

The method is applied from gates over a length of up to 70 m ahead of the face.

The length of the holes is 25 m, their diameter 100 mm and they are spaced by 2 to 3 m (Josien, 1981).

Allowing for the considerable reactivity of coal in this deposit, the method is extremely effective : volumes of cuttings exceeding several hundred liter per meter drilled are frequently noted in the overstressed zones.

The drilling has been mechanised : a servo-driller machine, to avoid the traditional blockage of the bits, has been devised and, in one shift, 3 holes, 25 m in length, with 2 workers, can be drilled. In these rapidly-advancing faces, this mechanisation makes it possible to guarantee, at less cost, a constant treated zone ahead of the face.

More generally, the various advantages associated with this method are no longer worthwhile when one can no longer drill the destressing holes from the gates. In inclined formations in the Lorraine coalfield, the application of this method from the coal face failed : difficulties were experienced when applying it in inclined seams and, in particular, the cost was high, due to the stoppages imposed on the working by the drilling.

In concluding this survey, we may point out that the method is not applicable in zones subject to rock outbursts : sandstones are too resistant and their permeability is too low to allow effective degassing of the dangerous zones.

4 - CONCLUSION

From the experience acquired in the course of a century of mining in variable deposits subject to dynamic phenomena, we can conclude that it is necessary, constantly, to adapt the control methods to the requirements of the deposit.

The developments of appropriate and effective means of control is based on investigation of the natural and mining conditions which will reveal the mechanisms of occurrence of the phenomena. This becomes all the more important with the increase in "mixed phenomena" producing by both a state of stress in the strata and the gas contained in the strata.

The present and future research, linked with the recent developments observed, concern :

- measurements of natural stresses which, in the zones in question, produce a very great horizontal stress (Revalor et al., 1985) ; these measurements are indispensable for the application of suitable methods of calculation.

- large-scale monitoring of the seismic activity induced by the mining, which makes it possible to take into account the overlying strata as a whole and to

demonstrate interactions at great distances between mining operations.

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L'EXPERIENCE DES MINES DE CHARBON FRANCAISES EN MATIERE DE LUTTE CONTRE LES PHENOMENES DYNAMIQUES

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RESUME

On regroupe sous le terme de "phénomène dynamique" des ruptures soudaines et violentes du massif rocheux qui surviennent à proximité des excavations minières.

A Charbonnages de France, ces phénomènes se présentent sous différentes formes :

- les coups de terrains, associés à des configurations d'exploitation par longue taille qui se manifestent par une expulsion brutale de la couche de charbon,
- les "surtirs", projections de roches avec émission importante de gaz qui affectent certains creusements de galeries au rocher,
- les dégagements instantanés et autres phénomènes mixtes associant la rupture du charbon et le dégagement de gaz.

L'incidence de ces phénomènes sur la sécurité des chantiers et l'évolution des phénomènes observés au cours des dernières années ont justifié la poursuite d'un important programme d'études.

Ces études ont permis, dans chaque cas, de préciser les facteurs naturels et d'exploitation qui paraissent avoir une influence déterminante sur le déclenchement de ces phénomènes : structure tectonique du secteur, contraintes naturelles et d'exploitation, nature et caractéristiques des terrains, comportement des terrains...

L'article propose une démarche d'études générale pour les différents phénomènes. Cette démarche vise à COMPRENDRE, PREVOIR et PREVENIR ces manifestations ; pour chacun de ces aspects, l'article présente les résultats essentiels obtenus ainsi que les axes futurs d'études qui sont à poursuivre ou à lancer dans ce domaine.

INTRODUCTION

Du fait de la diversité et des difficultés des conditions naturelles des gisements de charbon français, toute une gamme de phénomènes dynamiques ont été rencontrés et ont dû être maîtrisés lors de l'exploitation des mines. Deux types de phénomènes caractéristiques ont été observés très tôt et font l'objet de spécifications spéciales dans le Réglement Général d'Exploitation des Mines : les dégagements instantanés et les coups de toit.

Pour les dégagements instantanés, le bassin des Cévennes a détenu longtemps le record mondial avec un des premiers phénomènes notés (1er avril 1879) et plus de 6300 enregistrés, totalisant plus de 1 million de tonnes projetées.

Pour les coups de terrains, il s'agit d'une manifestation caractéristique du gisement de Provence, avec un premier rapport descriptif du 8 mars 1892 et une fréquence atteignant 50 par an, dans les années 1950.

Depuis d'autres types de phénomènes sont apparus, tels les "surtirs" récemment, et d'autres gisements, jusqu'à présent épargnés, ont été touchés. Actuellement, on peut avancer que 25 à 30 % de la production nationale de charbon sont issus de secteurs susceptibles de présenter des phénomènes dynamiques en cours d'exploitation.

1 - LES PHENOMENES OBSERVES

Les phénomènes dynamiques se manifestent par des ruptures brutales du massif rocheux autour des excavations minières. Par rapport aux phénomènes de ruptures plus classiques et fréquemment observés dans les travaux souterrains, ces manifestations se caractérisent par leur soudaineté, leur violence et, d'une manière générale, par l'étendue de la zone affectée. Dans les mines de charbon, on peut regrouper ces phénomènes en trois catégories :
- les coups de terrains qui affectent les couches de charbon au voisinage des chantiers exploités par longues tailles,
- les dégagements instantanés de méthane ou de gaz carbonique, localisés là encore dans des ouvrages au charbon, mais où le gaz joue le rôle d'élément moteur principal,
- les "surtirs" qui affectent les creusements de galeries au rocher.

1.1 - Les coups de terrains

Les coups de terrains se manifestent par une expulsion brutale de la couche de charbon (coup de couche) accompagnée d'un ébranlement violent qui peut être ressenti jusqu'en surface, sans émission importante de gaz (Josien et al., 1982). Le volume projeté est très variable, de quelques m^3 à 300 m^3 pour les plus importants. Le toit reste apparemment intact au cours de la manifestation et sa descente est souvent limitée à quelques centimètres. La surface est striée de marques brûnatres, témoins du frottement qui a lieu lors de l'expulsion. Les effets peuvent se localiser au carrefour taille-voie, ou au contraire s'étendre dans les voies, où de grandes longueurs peuvent être affectées (supérieures à 100 m). Dans ce cas, le phénomène s'accompagne d'un soufflage brutal et important du mur (qui peut atteindre 1 m) et le soutènement porteur est chassé.

Actuellement, ces phénomènes se sont manifestés dans des gisements très variés (figure 1) : en plateures foudroyés, en semi-dressants remblayés ou foudroyés, en dressants remblayés (Revalor, 1987).

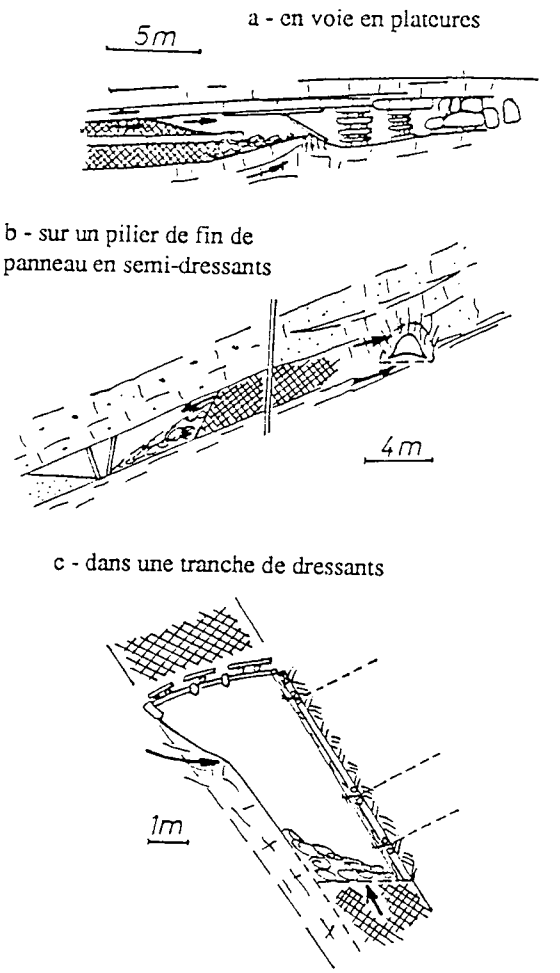


Fig. 1 : Description schématique des différents types de coups de terrains

La diversité de ces conditions et les nombreux cas rencontrés permettent de cerner les conditions nécessaires à l'apparition de ces phénomènes (Josien et al, 1987) :
- un état de contrainte anormalement élevé sur la zone, conséquence de l'influence d'exploitations passées ou de contraintes naturelles,
- une largeur exploitée suffisante pour solliciter le haut-toit des exploitations, en association avec les grandes failles du secteur
- des bancs du toit épais, raides et résistants, au contact de la veine,
- une couche de charbon plutôt résistante et souvent à structure hétérogène (banc raide au milieu).

Deux mécanismes de rupture semblent être à l'origine de ces manifestations (Josien et al, 1987 ; Revalor, 1987).
- un mécanisme local, où le coup de terrain apparaît comme une rupture de la couche, sous l'effet des contraintes excessives auxquelles elle est soumise,
- un mécanisme plus global, faisant appel au comportement d'ensemble des terrains autour d'un chantier exploité par longues tailles et dans lequel les effets observés au niveau du chantier apparaissent comme la conséquence de ruptures qui s'initialisent dans le recouvrement.

1.2 - Les dégagements instantanés

Compte tenu de l'abandon progressif des gisements très suceptibles, comme le gisement des Cévennes, les phénomènes typiques de dégagement instantanés n'intéressent à l'heure actuelle qu'une portion marginale de la production (1 %), essentiellement un gisement de dressants, très faillé, à veines puissantes (épaisseur moyenne de 6 m), mais très irrégulier. La méthode d'exploitation mise en oeuvre est le soutirage avec différentes variantes (voir figure 7) et les phénomènes surviennent au cours du creusement des traçages de base en couche ou lors de l'entrée en couche des gaines de soutirage.

Trois facteurs apparaissent simultanément nécessaires à l'apparition d'un dégagement instantané typique :
- un régime de contraintes anormales du fait de la structure du gisement (contraintes naturelles) et/ou de la configuration d'exploitation,
- un charbon souvent peu résistant, très microfissuré, présentant une concentration en gaz désorbable élevée et une forte vitesse de désorption,
- un gradient de pression de gaz élevé, en avant du front d'exploitation.

Lorsqu'un de ces trois facteurs est absent ou incomplètement réalisé, des phénomènes mixtes peuvent survenir, qui résultent de mécanismes de rupture différents.

Contrairement aux dégagements instantanés typiques, ces phénomènes mixtes concernent un plus grand nombre d'exploitations du fait de l'augmentation de la profondeur et de la concentration en grisou.

On peut citer :
- les éboulements gazés des gisements de dressants à couche puissante, où c'est la pesanteur qui joue l'élément moteur principal et contre lesquels on peut lutter par un soutènement adapté de la couronne de charbon, sans dégazage.
- les dégagements discontinus de grisou, qui sont survenus dans une veine en plateures, de O,5 à 3 m d'ouverture, exploités à 1000 m de profondeur. Ces phénomènes ont consisté en une avancée brutale du massif de charbon, de 1 à 2 m, accompagnée d'un bruit sourd et d'un dégagement de grisou pouvant dépasser 1500 m^3. On a relié ces phénomènes :
- à des surcontraintes dues à des limites d'exploitations, pouvant être situées à plus de 300 m au-dessus de la veine,
- à un toit raide et résistant : présence d'un banc de grés dans le toit à moins de 5 m de la veine,
- à un charbon à concentration en gaz désorbable élevée (16 m^3/t), mais (par chance) peu microfissuré (indice de fissuration ΔPs = 23, inférieur au seuil de danger fixé à 30 pour ce type de veine). La moins grande fissuration du charbon explique que ces phénomènes n'aient pas pris la même forme que des dégagements instantanés classiques : le mécanisme de rupture s'apparente plutôt à un coup de couche, détente brutale de la couche en liaison avec un réajustement du toit, se produisant dans une veine très grisouteuse.

1.3 - Les surtirs

Les surtirs sont des phénomènes dynamiques qui affectent des galeries creusées à l'explosif dans des grès (Josien et Revalor, 1983). Ils se manifestent juste après un tir par des projections de roches dans la galerie (dont le volume peut atteindre plusieurs centaines de m3), une fissuration importante du massif juste en avant du front et l'émission d'un volume de gaz important (plusieurs milliers de m^3 pour les plus importants ; figure 2).

Depuis 1978, date d'apparition du 1er phénomène, 26 surtirs ont été notés. Les secteurs concernés présentent les caractéristiques communes suivantes :
- ils sont situés en vierge, dans des zones "tourmentées" du gisement : flexure des couches, fermeture d'anticlinaux,
- les grés sont peu résistants (40 MPa contre 100 MPa habituellement) et ont une porosité élevée (6 à 11 % contre 1 à 2 %).

- le gaz est un gaz libre, contenu sous pression dans les pores de la roche. Cette pression est voisine de 3 à 4 MPa.

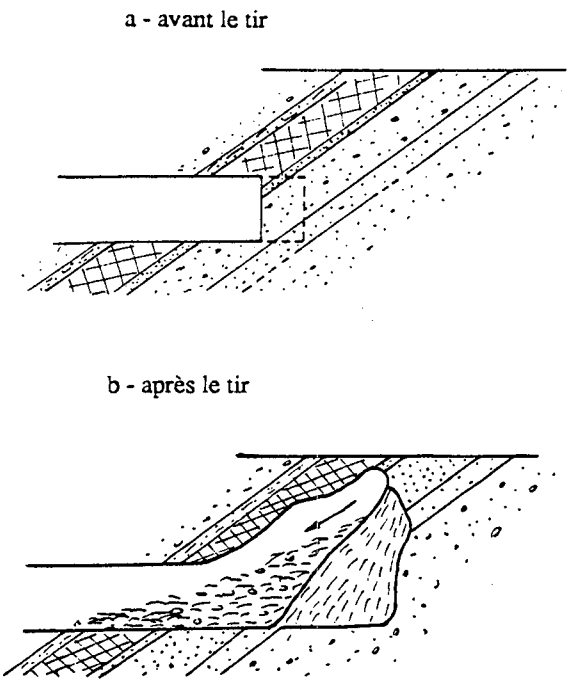


Fig. 2 : Description schématique d'un surtir

Le mécanisme de rupture fait intervenir à la fois les contraintes mécaniques et la pression de gaz (Revalor et al., 1985). Par ailleurs, il est certain que le mode d'abattage à l'explosif pratiqué dans ces chantiers joue un rôle essentiel dans le déclenchement. Dans la conduite à tenir dans la traversée de zones à surtirs, ce caractère provocateur du tir est recommandé pour éviter tout phénomène intempestif, survenant en présence de personnel au chantier.

1.4 - Caractéristiques communes aux différents phénomènes

La grande variété des phénomènes dynamiques observés dans les gisements français est liée avant tout à la diversité des conditions naturelles et d'exploitation rencontrées. Toutefois, à l'origine de chaque phénomène, on rencontre :

- *un élément moteur*, contraintes ou pression de gaz, ou les deux, et la nécessité d'une accumulation d'énergie expliquant la violence du phénomène : blocage des déformations et rupture par traction dans le cas des coups de terrains, faible perméabilité et gradient élevé de pression de gaz pour les dégagements instantanés.

- *des terrains particuliers*, plutôt raides et résistants dans le cas des coups de terrains, souvent peu résistants dans le cas des dégagements instantanés.

La compréhension des mécanismes de rupture à l'origine de chaque type de phénomène est importante pour adapter les moyens de lutte. A ce niveau, on retrouve néanmoins une démarche commune pour tous les phénomènes qui vise :
- à classer les secteurs d'exploitations pour repérer les zones susceptibles dès la planification,
- à détecter un danger en cours d'exploitation d'un secteur susceptible,
- à supprimer ce danger en cours d'exploitation par des méthodes de traitement adaptées.

Sans se prétendre exhaustive, la présentation suivante rassemble quelques réalisations dans ce domaine.

2 - SURVEILLANCE DES GISEMENTS

Dans les quartiers pour lesquels il n'est pas possible d'exclure le risque de phénomène dynamique, des mesures de surveillance sont entreprises. Elles reposent en général sur la mesure d'indices d'état des terrains (contraintes, gaz, etc...) dans des trous forés à partir de l'ouvrage minier. Les mécanismes à l'origine des phénomènes dynamiques étant complexes, l'efficacité de la surveillance est améliorée en combinant plusieurs tests représentatifs de l'influence de facteurs différents. L'écoute sismoacoustique permet en particulier de compléter les mesures faites au chantier en englobant le comportement des terrains environnants.

2.1 - Surveillance des traçages dans les gisements à Dégagements Instantanés

Pour le risque de dégagements instantanés, la désorbométrie effectuée sur des échantillons prélevés tous les mètres jusqu'à 4 à 6 m en avant du front, permet d'intégrer à la fois la pression résiduelle de gaz et l'aptitude du charbon à le libérer rapidement.

Pour la mesure rapide de la vitesse de désorption du CO_2, un appareillage appelé

"aptimètre" a été développé. La mesure consiste à mesurer l'augmentation de pression dans un réservoir de 750 cm^3 provoquée par la désorption de 10 g de fines de foration en 60 secondes.

La figure 3 donne le plan des mesures d'aptimétrie effectuées lors du creusement des traçages dans le quartier en exploitation. Il apparaît :

- l'augmentation des valeurs lorsque l'on se rapproche de la serrée limitant le quartier au Sud et avec l'approfondissement.
- la corrélation entre les valeurs fortes d'aptimétrie (A) et l'apparition des éboulements gazés.

sondages carottés dans le cas des grés sujets à surtirs et, dans ce cas, l'indicateur de danger est l'apparition du disquage dans les sondages (diamètre = 76 mm). La corrélation entre disquage en sondages et surtirs est excellente : dans les zones à risque, les carottes de sondages présentent un empilement de disques à surface ménisquée ("en verre de montre"), d'épaisseur moyenne centimétrique, présentant sur leur face externe de nombreuses amorces de rupture (figure 4). L'application de la méthode prévoit, pour chaque galerie creusée en zone susceptible, de forer un sondage carotté à l'avancement, d'une longueur de 100 m, à partir du front de la galerie. On admet qu'il y a risque important de surtir au cours du creusement, si on trouve dans le sondage des

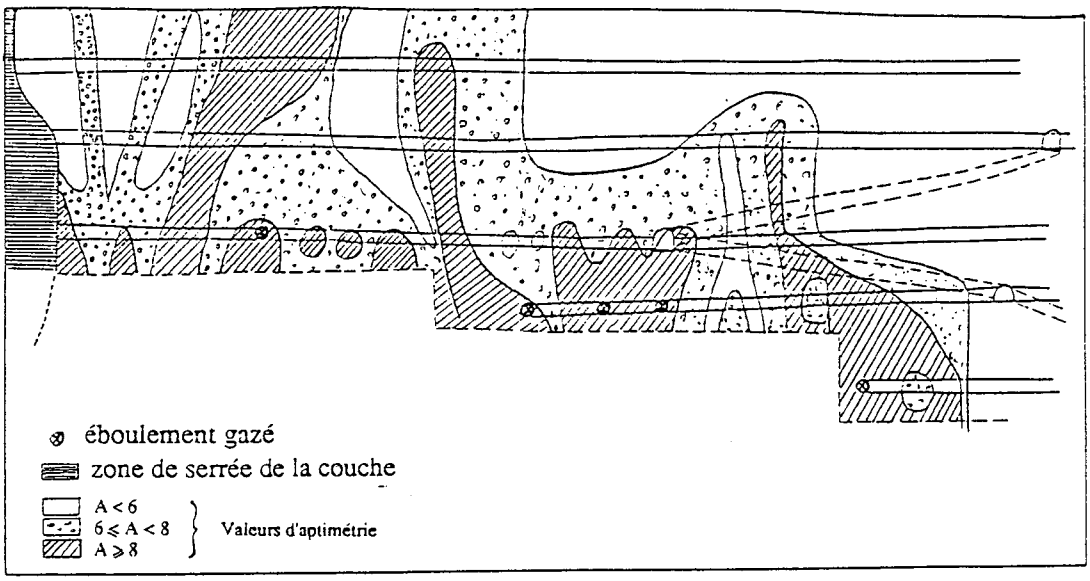


Fig. 3 : Exploitation de Messeix
Courbes d'isoaptimétrie
(projection des travaux sur un panneau vertical)

2.2 - Appréciation du risque de surtir par analyse du disquage en sondages

Une des méthodes les plus répandues pour apprécier au chantier le risque de phénomènes dynamiques consiste à analyser la réaction d'un trou foré dans le massif rocheux. Dans le cas des coups de terrains, le trou-test est foré en couche en destructif, et l'analyse porte sur le volume de débris de foration recueillis, à partir duquel des normes de danger ont été progressivement adaptées à nos gisements (Josien, 1981 ; Revalor, 1987). On procède par zones disquées d'une longueur minimale de 3 m et dont le nombre total de disques (amorces comprises) est supérieur à 15. Dans nos gisements, cette norme permet de repérer des phénomènes produisant plus de 40 tonnes de roches projetées et plus de 200 m^3 de gaz émis.

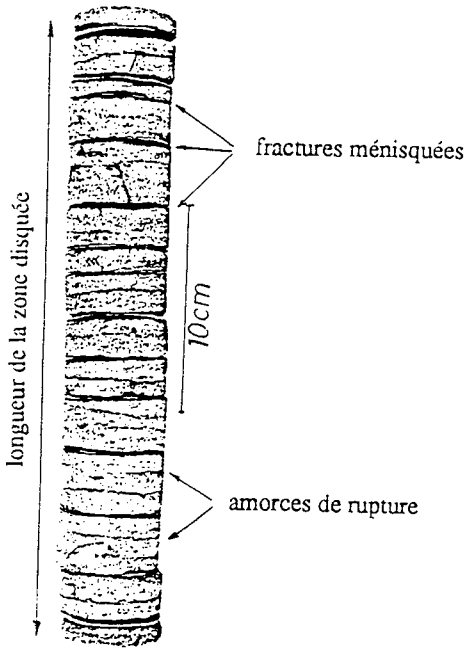


Fig. 4 : Exploitation de Vouters
Phénomène de disquage en sondages

2.3 - Ecoute sismoacoustique

L'utilisation de l'écoute sismoacoustique en complément des mesures spécifiques au chantier a été introduite pour la classification des zones vis-à-vis de leur susceptibilité aux dégagements instantanés. Ainsi, dans le bassin des Cévennes, la classification des quartiers reposait sur la mesure de 2 paramètres lors des premiers traçages dans le secteur, (De Vergeron et Belin, 1966).

- le degré de fissuration de la couche de charbon concernée (indice ΔPs, limite supérieure à 5 % de la distribution des ΔP).

- l'indice S mesurant l'agitation sismique après tir dans les traçages du secteur (S est la proportion de tirs suivis d'agitation sismique). Cet indice donne une image du régime de contraintes appliqué sur le traçage concerné.

La figure 5 représente la caractérisation de différents traçages du bassin, affectés ou non par des dégagements instantanés, vis-à-vis de ces deux paramètres. Il est clair que ces deux paramètres évoluent conjointement. Le déclenchement d'un dégagement instantané nécessite des contraintes élevées (S important dans un charbon peut fissuré (ΔPs faible) et inversement. En outre, la prise en compte de ces deux paramètres discrimine correctement les deux catégories de traçages : elle permet donc de préciser localement la susceptibilité de la couche de charbon, surtout en vue de l'exploitation future des panneaux.

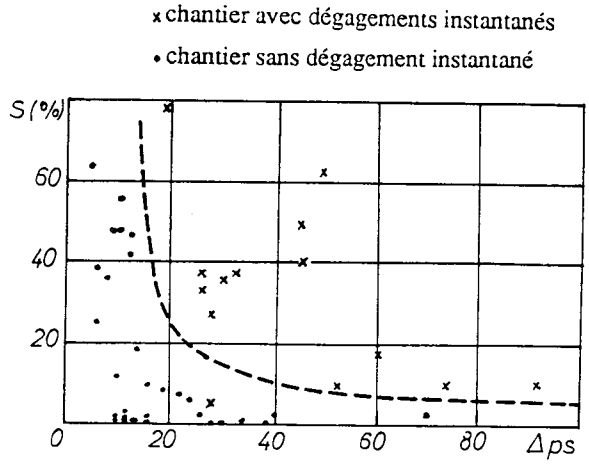


Fig. 5 : Exploitation des Cévennes
Indice sismoacoustique pour la classification des panneaux vis à vis du risque de dégagement instantané

Dans le gisement de Provence, la surveillance des tailles en exploitation, vis-à-vis du danger de coups de terrains, est réalisée journalièrement par la méthode des trous-tests, complétée par une écoute sismoacoustique. L'écoute est réalisée par quatre capteurs, implantés dans les voies, en avant du front de taille, et déplacés périodiquement avec l'avancement de ce dernier. Le nombre important d'événements sismiques enregistrés (plus de 100 par jour) et la nécessité de disposer d'informations permettant une prise de décision rapide, ont rendu nécessaire l'acquisition et le traitement automatique de ces données, réalisés par micro-ordinateur (Déchelette et al, 1982 ; Revalor et al., 1986).

Le suivi de plus d'une dizaine de panneaux a permis de définir un critère permettant de repérer l'arrivée du chantier dans une zone dangereuse (Revalor et al., 1988). Ce critère quantifie la manière avec laquelle se libère l'énergie sismoacoustique et on admet qu'une zone devient dangereuse lorsque plus de 35 % de l'énergie se libère à travers des fortes secousses (E > 1 MJ : figure 6). Il est important de souligner qu'un tel critère est implanté sur le site où il est examiné tous les jours pour être intégré aux décisions d'exploitation.

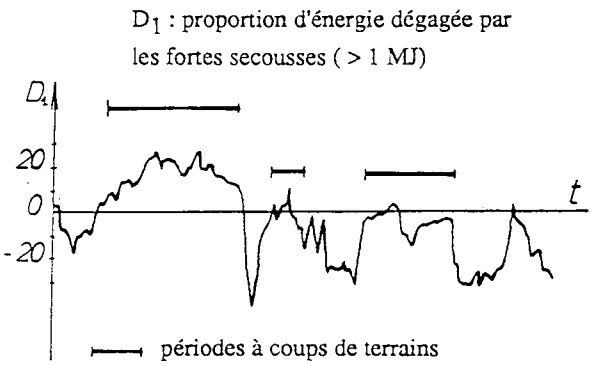


Fig. 6 : Exploitation de Provence
Détection des zones dangereuses de coups de terrains par écoute sismoacoustique

3 - LA PREVENTION DES PHENOMENES

Les procédés de lutte contre les phénomènes dynamiques recherchent la détente des terrains tant pour les dégagements instantanés, où il faut augmenter la perméabilité des terrains, que pour les coups de terrains, où il faut diminuer les contraintes mécaniques.

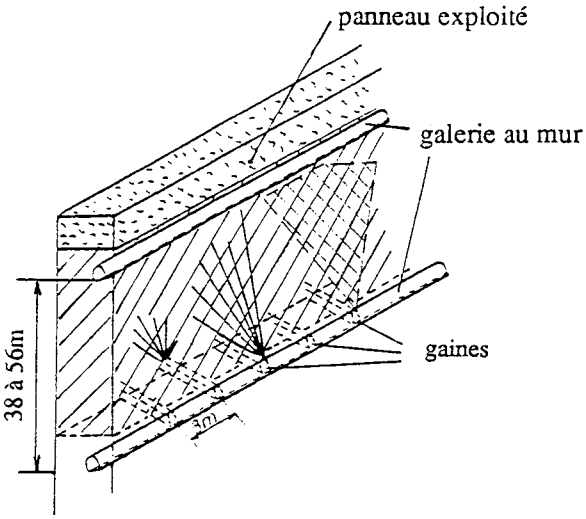
Dans les gisements denses multicouche, cet objectif de détendre des terrains doit être pris en compte dès la planification de l'exploitation, en choisissant comme veine égide la veine la moins susceptible du faisceau. Toutefois, à ce niveau, les contraintes sont multiples, ce qui ne permet pas toujours d'appliquer correctement ce principe. Il en résulte la nécessité de disposer de méthodes d'exploitation adaptées ou de méthodes de détente, applicables au chantier.

3.1 - Adaptation des méthodes d'exploitation des gisements pentés à dégagements instantanés

Dans le gisement penté à couches épaisses du Dauphiné, les méthodes d'exploitation par soutirage sont apparues très tôt comme parfaitement adaptées à la nature du charbon exploité, tout en permettant une récupération correcte du gisement.
Dans les secteurs susceptibles du gisement, la méthode de base consiste à creuser une galerie au mur de la couche à exploiter, à partir de laquelle on accède à la couche par des gaines horizontales, espacées de 8 m (figure 7a). Le soutirage est amorcé à partir des gaines par des longs trous à l'explosif, et exploité par chargeur transporteur diésel sur pneus ; la hauteur de soutirage varie entre 38 m et 56 m suivant les secteurs.

Dans les secteurs très susceptibles de dégagements instantanés, deux variantes à cette méthode ont été trouvées, afin de garantir la sécurité tout en améliorant la productivité des chantiers :

a - Soutirage par gaines



b - Soutirage par chambres magasin

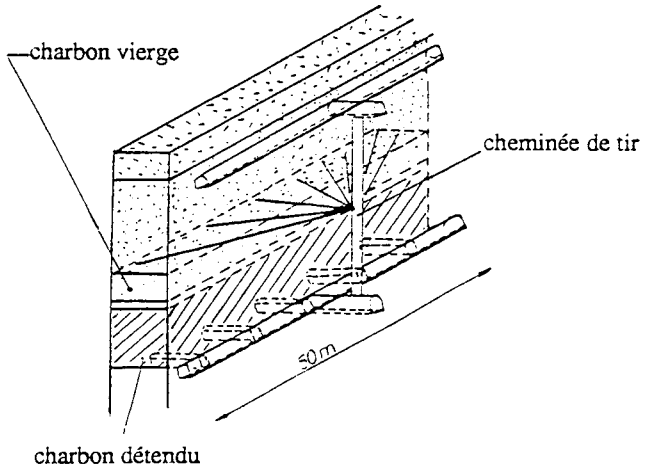


Fig. 7 : Exploitation du Dauphiné
Soutirage du charbon dans les secteurs très susceptibles de dégagements instantanés

a) la première consiste à charger le tas à l'aide d'une chargeuse à pinces télécommandée. Le poste de commande est situé dans une niche de la galerie au mur, à une centaine de mètres du chantier, en amont aérage. Les points clefs de la manoeuvre sont visionnés par un circuit de télévision. Le poste de commande est muni de dispositifs de contrôle annexes de sécurité

informant l'opérateur de l'évolution de certains paramètres en cours de soutirage : surveillance de l'atmosphère, bruits émis par le massif,... Les interventions à front de soutirage sont très limitées, ce qui permet aux personnes d'être protégées de manière quasi permanente au cours de l'opération.

b) à partir du même schéma de base (galerie au mur et gaines) la deuxième variante (chambres magasin) consiste à réaliser tous les 50 m une cheminée au pendage dans le mur de la couche (figure 7b). Cette cheminée permet d'effectuer des éventails de tir en couche, par nappes horizontales successives. A chaque tir, on ne soutire par les gaines que le tonnage correspondant au foisonnement du tir, ce qui évite toute création de vides importants dans la chambre. A la fin de cette phase, le soutirage peut s'opérer sans danger par les gaines à la base, le charbon étant complètement détendu et dégazé.

3.2 - Détente par foration de gros trous

La détente par foration en destructif dans la couche de charbon de trous de gros diamètre a été appliquée avec succès dans différentes conditions dans les gisements français.

Dans le bassin des Cévennes, c'était une méthode de prévention efficace, appliquée dans différentes configurations : traçage en veine, taille, traversée de couche. La longueur des trous était de 20 m, leur diamètre variant entre 115 et 140 mm et leur espacement entre 2,5 et 4 m. La fréquence de foration dépendait de l'avancement du chantier, une frette de protection minimale estimée à 5 m devait être respectée en avant du front. Dans ce cas, l'efficacité de la méthode était due à son effet mixte détente-dégazage de la couche de charbon.

Dans le gisement de Provence, cette méthode a permis, ces dernières années, de maîtriser les coups de terrains qui affectent les chantiers par longues tailles.

La méthode est mise en oeuvre à partir des voies d'accompagnement de tailles sur une longueur en avant du front pouvant atteindre 70 m. La longueur des trous est de 25 m, leur diamètre de 100 mm et leur espacement peut varier de 2 à 3 m (Josien, 1981).

Compte tenu de la forte réactivité de charbon dans le gisement, la méthode est extrêmement efficace : des volumes de débris supérieurs à plusieurs centaines de litres par mètre de foration sont fréquemment notés dans les zones chargées.

La foration a été mécanisée : une sondeuse asservie, pour éviter le phénomène classique de blocage des mèches, a été conçue et permet, en un poste, de forer 3 trous de 25 m de longueur avec 3 servants. Dans ces tailles à avancement rapide, cette mécanisation permet de garantir une longueur traitée constante en avant du front, à moindre coût.

Plus généralement, les divers avantages liés à cette méthode ne valent plus dès que l'on ne peut plus forer les trous de détente à partir des voies de taille. Dans les gisements de semi-dressants du bassin de Lorraine, l'application de cette méthode à partir du front de taille s'est soldée par un demi-échec : difficultés de mise en oeuvre en gisements pentés et surtout coût élevé, dû aux arrêts imposés au chantier par la foration.

Pour clore ce panorama, signalons que la méthode est inapplicable pour le traitement des zones sujettes à surtirs : les grès sont des roches trop résistantes pour se détendre en masse et leur perméabilité est trop faible pour permettre un dégazage efficace des zones dangereuses.

4 - CONCLUSION

De l'expérience acquise au cours d'un siècle d'exploitation dans des gisements variés sujets à des phénomènes dynamiques, il est possible de conclure à la nécessité d'adapter constamment les méthodes de lutte aux exigences du gisement.

Le développement de moyens de lutte appropriés et efficaces repose sur une reconnaissance des conditions naturelles et d'exploitation, qui permettra d'élucider le mécanisme d'apparition du phénomène. Ceci est d'autant plus important avec l'augmentation des "phénomènes mixtes", faisant intervenir à la fois l'état de contrainte des terrains et le gaz qu'ils contiennent.

Les recherches actuelles et futures, liées aux évolutions récentes constatées, concernent :
- les mesures de contraintes naturelles

qui présentent dans les zones concernées une valeur élevée de la contrainte horizontale (Revalor et al, 1985) et qui sont indispensables à la mise en oeuvre de méthodes de calcul appropriées
- l'écoute à grande échelle de la sismicité induite par l'exploitation, qui permet de prendre en compte l'ensemble du recouvrement, et de mettre en évidence des interactions à grande distance entre exploitations.

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ERFAHRUNGEN IN FRANKREICH MIT DEM 'DYNAMISCHEN PHÄNOMEN' VON GESTEIN IN KOHLEGRUBEN.

J.P. Josien, R. Revalor et al.

Das 'dynamische Phänomen' ist ein plötzlicher und gewaltsamer Ausbruch von Gesteinsmassen in der Nähe von Grubenbauen.

Dieses Phänomen ist in Charbonnage de France in verschiedenen Formen aufgetreten:

Gebirgsschläge, die mit der Anordnung der Streben in Zusammenhang standen und zu einem plötzlichen Ausbruch der Kohle in dem Streb führten;

"Overshots", die in übergroßem Ausmaß durch Schießarbeiten in einigen Strecken hervorgerufen wurden und zu bedeutenden Ausgasungen führten;

Kohleausbrüche oder andere ähnliche Phänomene, die Kohle- und Gasausbrüche einschließen (CO_2 oder CH_4).

Der Einfluß dieser Phänomene auf die Sicherheit unter Tage und ihre in den letzten Jahren steigende Anzahl haben zur Einleitung eines wichtigen Forschungsprogrammes geführt.

Die ermittelten Ergebnisse zeigen den Einfluß von einigen geologischen und bergbaulichen Faktoren auf die Auslösung der Vorkommen: tektonische, natürliche Gebirgsspannung und Belastungen, die durch Abbau und Streckenfahren hervorgerufen werden, Art und mechanische Eigenschaften des Gebirges und das zeitabhängige Verhalten des Gebirges.

Der Vortrag enthält Vorschläge für eine Vorgehensweise, die auf die verschiedenen Phänomene adaptiert werden kann. Ziel ist es zu verstehen, vorauszusagen, und zu schützen. Nach Beschreibung der Hauptergebnisse werden zukünftige Wege der Forschung in diesem Gebiet vorgeschlagen.

ФРАНЦУЗСКИЙ ОПЫТ "ДИНАМИЧЕСКОГО ФЕНОМЕНА" ПОРОД НА УГОЛЬНЫХ МЕСТОРОЖДЕНИЯХ

Ж.П. Жозьен, Р. Ревалор и др.

"Динамические феномены" - это внезапные и очень интенсивные выбросы породы на горных выработках. На французских угольных месторождениях эти феномены проявляются в различных формах:

- отслоения, которые относятся к определенному плану выработки длинного забоя и которые заканчиваются внезапным выбросом угольного пласта;
- "овершоты", которые являются неожиданно сильным результатом взрывания шпуров в некоторых штольнях в породе с высокой газовой эмиссией ;
- выброс угля и другие взаимосвязанные явления, которые включают выброс угля и газа (CO_2 или CH_4).

Влияние этих явлений на безопасность труда на угольных месторождениях и возрастание их количества в последние годы стимулировали начало важной исследовательской программы. Полученные результаты показали влияние таких геологических и горных факторов на возникновение "динамических феноменов" как тектонические стрессы и стрессы, вызванные горными разработками; тип слоя и его механические свойства и стратиграфия пород. Доклад предлагает методологию, которая может быть применена для различных динамических феноменов. Цель этой методологии - их понять, предсказать и предотвратить. После отчета о главных результатах будут предложены некоторые направления для будущих исследований.

法国对煤矿里岩石的 "动态现象" 的经验

J．P 爵希恩，R． 热伐罗 等

"动态现象" 是在开采区附近岩石体的突然和剧烈的喷溅现象．在法国的煤矿中，这种现象有如下几种爆发形式，

* 与长壁工作面一些部位有关的突出，造成煤层的突然喷溅；

* "过爆" 是某些有严重气体散发的岩石坑道里爆破时造成爆破面积过大的情形；

* 煤矿石的突出或其他有关的现象，包括矿石和气体 （二氧化炭与甲烷）的喷溅．

这些现象对安全的威胁和近年来这些现象的发生比较频繁，促成了一个重要的研究方案．

所得的研究结果指出如下一些地质和采矿因素会触发这些现象的发生：地壳结构，自然应力与采矿作业造成的应力，地层的类别和力学性质，及地层的时间迁变过程．

本文提出一种可适用于各种不同现象的法则．此法则能用于了解，预测和防[1]护此类现象．主要结果叙述完后，本文还指出了一些今后的研究方向．

FIGHT AGAINST ROCKBURSTS IN THE CZECHOSLOVAK MINING INDUSTRY

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ABSTRACT

The authors divide the rockburst problem into prediction and prevention. In practice, prevention is mine-specific. Prediction is based primarily on the seismoacoustic determination of the stress and strain within the rock mass. Stress is observed to occur due to abandoned coal pillars in overlying mined coal seams; full pillar extraction is recommended where possible. Other preventive measures are shot firing, water infusion, and destress drilling. In general, examples are taken from the Ostrava Karvina Coalfield, known to be rockburst-prone since 1918. Ultimately, automation is recommended to fight rockburst hazards.

INTRODUCTION

The fight against rockbursts has a long tradition in Czechoslovakia, because the first rockburst occurred in the Kladno Coal Field as early as in 1872, while the first rockburst in the Ostrava Karvina Coal Field was recorded in 1918. In both cases, bituminous coal fields were involved. The problems of rockburst prevention in our mining conditions still persist, particularly in the Ostrava Karvina Coal Field and to a minor extent also in the Kladno Coal Field and in the Pribram Deposit, where uranium ore is mined.

The reason for the persisting actuality of the rockburst prevention endeavours is the fact that mining operations are carried out in continually changing conditions. Depths at which mine workings are operated continue to increase, the technology of mining operations varies, etc. Specifically new problems from viewpoints of the rockburst prevention appeared in the Ostrava Karvina Coal Field when coal winning operations started in the seam group No. 37, where the seam height exceeds 5 m and overlying strata consist of very strong and compact rocks, mainly sandstones and conglomerates, which attain the height of 180 m.

THE ROCKBURST PREVENTION SCHEME

Coal mining operations in areas where the occurrence of rockburst is imminent require numerous special activities which can be principally divided into the prediction of rockburst occurrences and their prevention.

The objective of the prediction is to determine the predisposition of the rockmass to the occurrence of rockbursts in planned, designed, and operated mine workings and to select correctly means for rockburst prevention, including the control of their effect, while the main objective of prevention is to eliminate the occurrence of rockbursts or at least to reduce their consequences.

Practical application of the rockburst prevention scheme is realized in conjunction with the valid safety standards. Principal provisions relating to the rockburst prevention scheme are contained in the Safety regulations which apply to all deposits exploited by means

of subsurface mining. However, conditions which allow potential release of rockbursts are not identical in every deposit. Rockbursts also do not occur in every deposit. Therefore, the respective provisions have been elaborated in more detail in orders and instructions where both natural and technical conditions are considered. For example, the Decree of the District Mining Inspectorate in Ostrava "Rockburst prevention scheme in the Ostrava Karvina Coal Field" was issued on 1 June 1980, for conditions prevailing in mines in this coal field. Technical data which are necessary to realize preventive measures against rockbursts and treat the provisions of the above decree in more detail have been summarized in the Instruction to the Decree of the District Mining Inspectorate in Ostrava issued by the Scientific Coal Research Institute in Ostrava - Radvanice.

The system of legislative standards regulating the rockburst prevention scheme is not finally completed. Acquisition of new knowledge and experience, as well as results obtained in tests of new technical means, are implemented for the completion, amendment, and more precise specification of legislative standards. These changes are usually realized at intervals of several years.

Particularly in the Ostrava Karvina Coal Field, systematic application of the rockburst prevention scheme is carried out by means of an already established organisatory structure, comprising particularly research and development institutions, specialized organizations for approvals, controls, and practical implementation of preventative means and measures, and departments of geomechanics established at every coal mine which falls under the direct competence of the chief engineer of the respective mine.

The chief engineer - who is the first deputy director - relies in this particular activity upon the support of the Rockburst Prevention committee organized at every coal mine which is also his consultant. In addition, the main rockburst prevention committee is active in the Ostrava Karvina Coal Field as the consultant of the General Director of the Ostrava Karvina Coal Mines regarding rockburst prevention. This committee solves principal questions relating to concepts and methodologies of enhancement and practical applications of the rockburst prevention scheme.

PREDICTION OF ROCKBURSTS

The prediction of rockburst occurrence is the activity which determines the tendency of the rock mass to occurrence of rockbursts, aiming at specification of categories of risk levels due to potential rockburst occurrence in the planned, designed, and operated mine workings. The final objective is a correct selection of preventive measures against rockbursts, which is considered to be a foundation stone in the rockburst prevention scheme. Its level and reliability in recognizing of the imminent risk due to a potential rockburst occurrence affects both the safety of personnel and course of the mining processes, as well as the economy of mining.

The prediction of rockburst occurrence can be principally divided in the following way:

- regional prediction of the rockburst occurrence
- local prediction of the rockburst occurrence.

Regional prediction of the rockburst occurrence is an activity aimed at the determination of the predisposition of the rock mass to the rockburst release. It is carried out during the period of deposit exploration and its accuracy is enhanced during periods of planning the layout of mine working and their operation.

Regional prediction relies principally upon evaluations of:

- sedimentary and tectonic conditions of the respective site such as: depth under the surface, existence of fold structures and faults, as well as lithologic properties of the rock mass;

- properties of the coal seam which include: seam height, petrography, strength and strain properties, and tectonics;

- properties of adjacent rocks such as: petrography, mechanical properties, as well as sedimentary and lithological characteristics;

- data which can be found out as analogies, relying particularly upon information on rockbursts which have already occurred in the respective area.

Regional prediction activities would be organized so as to allow that before the start of coal mining activities in the first coal panel, the respective mine workings can be classified in one of the three existing categories of risk due to the occurrence of rockbursts. This activity is carried out during periods of planning, designing, and operation of mine workings.

The principal method upon which the local prediction continues to rely during the mine working is the method of drill tests carried out by means of drilling bits with diameter 42 mm, with the objective to find out whether the so-called protective zone has developed in the immediate vicinity of the mine workings. In some cases where technical reasons prevent drilling with 42 mm bits, other diameters of the drilling bit can be used. In our conditions, mainly drillholes with diameters 115 mm or 200 mm are used.

In some cases, drill tests do not indicate reliably the risk of potential rockburst. The reasons may be multiple.

One possible reason is that the variation of the state of stress within the coal seam is abrupt, due to an instantaneous fracture in the overlying rock strata which cannot be detected by drill tests. Results of seismoacoustic and seismologic observations confirm this opinion. Prior to the occurrence of some rockbursts, the seismoacoustic activity monitored in coal seams was low and; on the contrary, after a released rockburst, it was comparatively high and remained at this level for long periods. A steep increase of the seismoacoustic activity relates directly to events located outside the affected coal seam, usually in the overlying strata, as indicated by seismologic observations carried out simultaneously with seismoacoustic measurements. In the Ostrava Karvina Coal Field, release of such rockbursts was experienced during coal mining operations in seams where the height exceeded 5 m and fracturing of strong beds of compact rocks in the overlying strata occurred.

Insufficient reliability of drill tests in certain circumstances draws attention to other promising methods. Due attention is given to applications of mine surveying methods for the investigation of deformations within the rock mass. Movements of the rock mass, particularly the vertical ones, are monitored both in subsurface mine workings and on the surface as well. The ratio of subsidence to the height of the extracted coal seam, with due respect to the method of ground control, represents in a certain measure the state in the rockmass. If the subsidence of overlying strata is close to expected "final" subsidence values, the state within the rock mass is more stabilized and usually also more safe.

Similar information for evaluations of the state within a rock mass are expected from results of convergence measurements and eventually also from stress measurements in the stowing material in entirely

backfilled longwalls. There is also a tendency to implement convergence and extensometric measurements carried out in zones in front of longwalls, etc. Considerable attention is also given to the application of geophysical methods in the rockburst prevention scheme, particularly to seismoacoustic and seismologic measurements.

The principle of both methods are determinations of stress and strain variations in zones affected by mining activities. During the brittle fracture of the rock mass, which is a phenomenon accompanying these variations, occurrence of seismic events of different character and intensity is experienced. Both methods, however, differ because of predominant frequency ranges which are monitored - frequency range from approximately 60 Hz to 1.5 kHz in seismoacoustic measurements, as opposed to frequency range from 2 to 20 Hz in seismologic measurements - and focus dimensions - single digits and tens of meters in seismoacoustic, contrary to tens and hundreds of meters in seismologic measurements.

Seismoacoustic measurements are indispensable at present in the prevention of rockbursts in the Ostrava Karvina Coal Field and this position will persist in the future. They are applied in the local prediction of potential rockburst occurrence and to specialized observations of stress-and-strain variations within the rock mass in the area of interest. Results of such observations serve for operative applications of active and passive measures against rockbursts, as well as to check their effectiveness in the routine rockburst prevention mode.

Objectives of seismoacoustic observations, carried out as a daily routine, are the following:

- continuous acquisition of data related to the development and variations of dynamic frequency and foci of seismoacoustic pulses generated in the monitored area of interest in relation to technological activities (even in periods of production stops) and to the assessment of safety in the area of interest;

- acquisition of information on the development of dynamic frequency both during coal winning and roadways drivage, by production shotfiring, and following important seismic events which are required particularly to determine a safe break before renewal of mining operations;

- assessment of effectiveness of applied preventive measures against rockbursts, particularly inducement shotfiring (volley shotfiring) carried out with the objective to release the accumulated seismic energy and to investigate the reverberation sequences;

- acquisition of reliable data for evaluations and analyses of rockburst events which have already occurred and investigations of extraordinary events.

Continuous seismoacoustic measurements in the Ostrava Karvina Coal Field are carried out and used in harmony with the Methodologic Instructions. Daily geomechanic routine indicates that these measurements contribute particularly to:

- demarcation of seismically active areas, i.e. areas where development of seismic foci of dynamic events of different intensity (even rockbursts) occurs during mining operations;

- investigations of development trends of seismic activity both in time and space which allow to determine variations of stress and strain states within the rock mass in particular parts of the mine field e.g. in a tectonic block;

- local prediction when individual mine workings are classified or re-classified in appropriate categories of risk due to

potential occurrence of rockbursts, with the aim of selecting and applying active and passive means of the rockburst prevention, and checking their effectiveness; this applies particularly to shotfiring;

- complex assessment of dynamic events (rockbursts) which have already occurred.

Monitoring of seismic activities in the eastern part of the Ostrava Karvina Coal Field is achieved either by means of the local seismologic network, comprising both subsurface and surface stations and stations of the regional diagnostic polygone; seismometers used in this system are located in shallow drillholes. The dislocation of individual stations and their instrumentation is shown on Figure 1.

Valuable information on geomechanical processes taking place in areas with imminent risk of rockbursts are obtained in models of equivalent materials. It allows not only the simulations of events which already occurred and were measured and observed, but also presents experimental evidence of measures designed to allow future exploitation in difficult conditions particularly in terms of suitable operation of mine workings both in time and space, artificial alteration of rock properties as a result of shotfiring, etc. Such model experiments are usually scaled 1 : 100 or 1 : 250 and carried out in testing frames with dimensions up to 7.5 x 5 m.

Dynamic events and their consequences, i.e. devastations of either main gate or return gate which take place in the rockburst occurrence during mining of thick coal seams under very thick beds of very strong overlying rock in the Ostrava Karvina Coal Field have already been modelled in linear scale 1 : 50. Part of the rock mass comprising the respective mine working was exposed to dynamic loading by means of weight falls on the model. This procedure simulated shock loading in the vicinity of the respective mine working as a consequence of fracture in the main strata overlying the coal seam. Results of the model experiments have shown that the characteristics of deformation processes in the immediate environments of the mine working, as well as the deformation of the respective mine working, are very similar to actual in situ effects as indicated in Figure 2.

Dynamic loading in models is achieved by means of the weight fall. Really, the reason of this dynamic loading is ascribed to a rapid and important redistribution of stress due to fracturing of either the overlying or underlying rock strata of the respective mine working. Deformations or destructions of mine workings are, therefore, the consequence of critical states of the rock strength, particularly the coal seam in the immediate vicinity of the mine working resulting from shock loading due to fracturing of rocks in the high and strong overlying rock strata complexes.

In addition to scale modelling, mathematical models are used to solve complex tasks.

ROCKBURST PREVENTION

The objective of preventive measures is to eliminate the occurrence of rockbursts, or at least, to reduce their consequences. The most efficient means of the rockburst prevention are applications of strategic principles which are aimed at the prevention of rockburst occurrence, both in time and space, because they result in development of favorable stress and strain states within the rock mass. They are applied in the planning and design of mine workings and have been principally dealt with in the respective legislative standards.

In analyses of the rockburst occurrences in the Ostrava Karvina Coal Field it is evident that their occurrence always coincides with

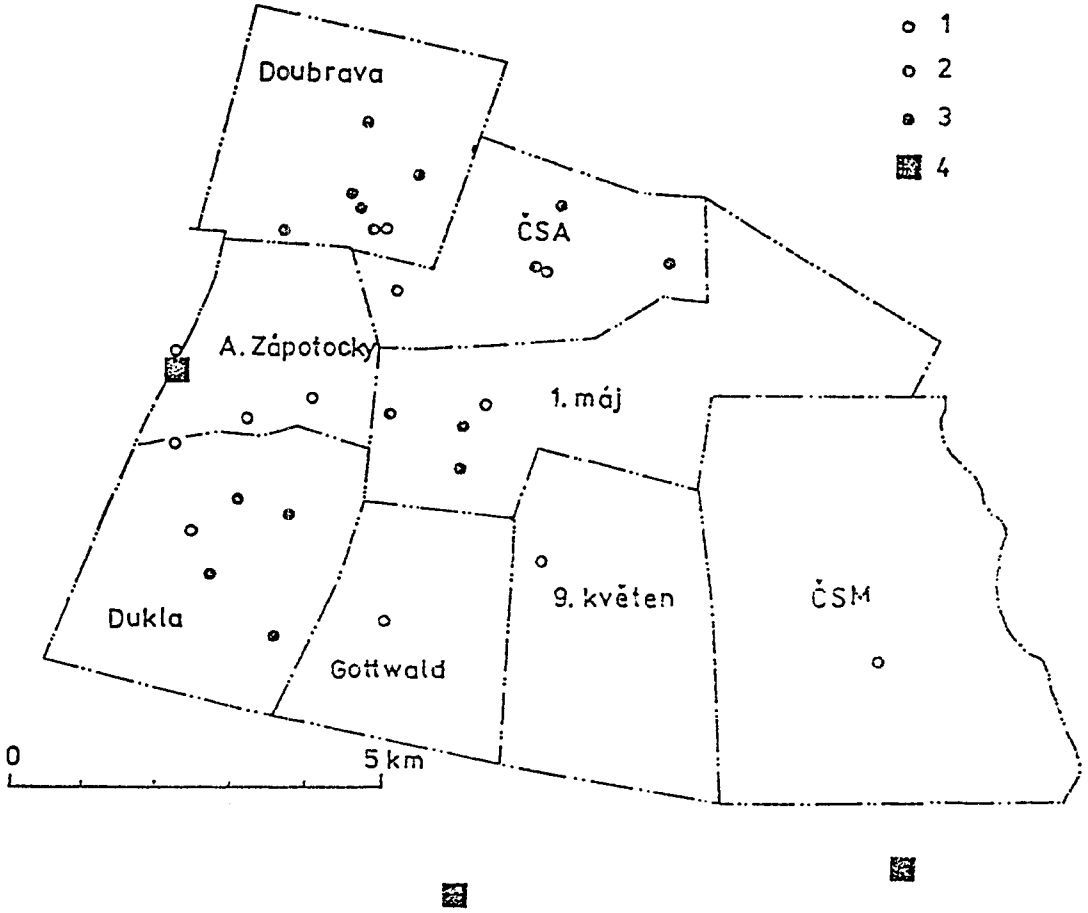


Fig. 1 Scheme of seismologic stations dislocations in the Ostrava Karvina Coal Field

1 - surface station, seismographs ELSMO

2 - subsurface station, galvanometric registration and apparatus DSLA-1

3 - subsurface station, apparatus DSLA-1 developed in the Scientific Coal Research Institute, Ostrava-Radvanice

4 - System RDP - Lennartz Electronics

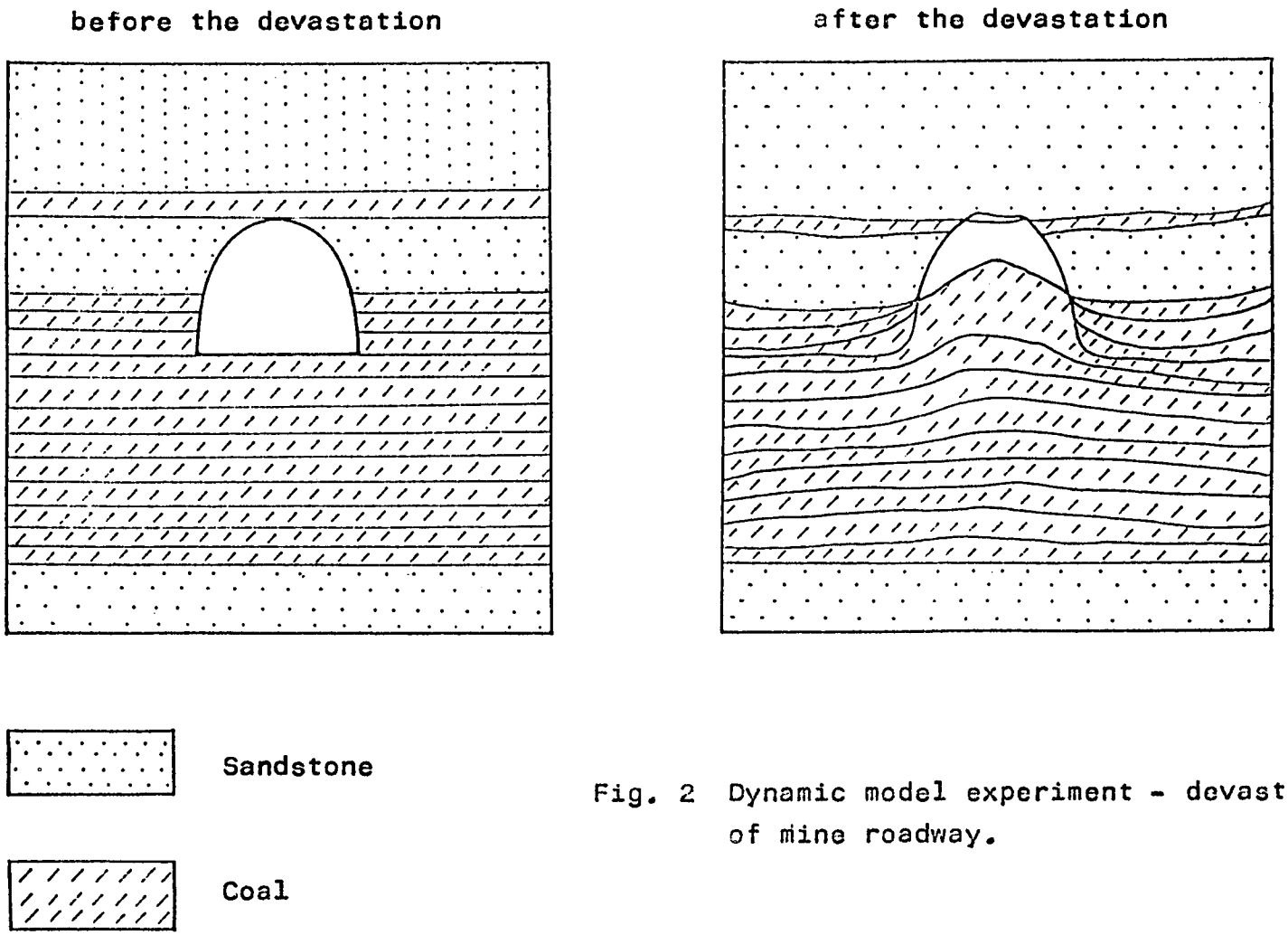


Fig. 2 Dynamic model experiment - devastation of mine roadway.

zones exposed to effects of concentrated stress due to abandoned coal pillars in overlying seams. Therefore, the most important strategic principle is the operation of mine workings in protected zones, that is usually under the mined out part of the overlying coal seam, and full extraction of coal seams without unnecessary remnant pillars.

However, this requirement cannot always be complied with. It is frequently in contradiction with the regular configuration and maximum possible length of the respective coal panel, which is important in implementing heavy duty mining machinery complexes requiring costly investments. However, this contributes to abandoning unmined coal pillars, particularly in the vicinity of tectonic faults. Consequently, attention is given to efficient mining of these pillars simultaneously with the retreat of the adjoining longwall or eventually making them harmless by means of other available methods. However, mining out of remnant pillars and efforts to minimize their dimensions cannot be eliminated completely. Mining activities are carried out in natural conditions characterized, among other things, by complicated tectonics, variable areal development of coal seams, seam height variations, etc., which do not allow complete mining out of the respective coal seam. Numerous remnant pillars represent the heritage from the past and their mining is unreal because of economy and safety. It is impossible to eliminate some unmined pillars because of other reasons, such as protection of shafts and other access mine workings, safety pillars for undermining water bearing horizons, etc.

Therefore, solutions of problems related to rockbursts cannot only rely upon the "full extraction" of coal seams and application of strategic principles. It is necessary to implement also strategic and routine means of the rockburst prevention scheme.

Mining activities are carried out at present, and so will be in the future, in conditions affected by the existence of unmined pillars and limits of mined out areas in overlying coal seams. Numerous examples indicate that safety of mining operations can be achieved also in areas with increased risk of imminent rockburst occurrence. Active means of rockburst prevention play a positive role in such circumstances and the most important ones are:

- special shotfiring technologies both in the coal seam and adjacent rocks,
- water infusion in the coal seam and adjacent rocks,
- destress drilling in the coal seam.

However, also passive means of the rockburst prevention scheme are applied, particularly to reduce consequences of rockbursts. Among others, these means comprise:

- artificial release of rockbursts by special blasting technologies in the absence of personnel;
- reduction of personnel employed in exposed mine workings or, eventually, prevention of access of personnel in such mine workings;
- remote control of mining machinery,
- appropriate supports of mine workings;
- suitable installations of mining machinery in mine workings.

Microelectronics, electro-hydraulics, and computer assisted systems, automation and, in the future, also robotisation elements offer new possibilities in reduction of rockburst consequences, particularly from viewpoints of reduction of the personnel exposed to risk in longwalls during coal winning periods. In this way, novel

means which substantially contribute to the reduction of consequences of eventual rockbursts in the most complicated conditions are at our disposal and their applications are, therefore, planned in the future.

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LUTTE CONTRE LES DEGAGEMENTS ROCHEUX DANS L'INDUSTRIE MINIERE TCHECOSLOVAQUE

A. Skrabis et P. Janas

Les auteurs divisent le problème des dégagements rocheux en deux : détermination et prévention. En pratique, la prévention est spécifique à chaque mine. La détermination est basée principalement sur l'évaluation sismoacoustique des contraintes et des déformations à l'intérieur de la masse rocheuse. On a observé que ces contraintes se produisent à cause des piliers de charbon abandonnés dans les couches de charbon sus-jacentes; il est recommandé d'effectuer une extraction totale lorsque cela est possible. D'autres mesures préventives comprennent les tirages, les infusions d'eau et les forages qui atténuent les contraintes. Les exemples que nous donnons ont été généralement tirés de la mine de charbon d'Ostrava Karvina qui est sujette à ce genre de risques depuis 1918. Enfin, nous recommandons l'usage de l'automatisation pour combattre les risques de soudains dégagements rocheux.

DER KAMPF GEGEN DEN GESTEINSAUSBRUCH IN DER TSCHECHOSLOWAKISCHEN KOHLEINDUSTRIE

A. Skrabis, P. Janas

Die Autoren unterteilen die Behandlung des Problems der Gesteinsausbrüche in Vorhersage und Verhütung. Die Vorhersage stützt sich dabei hauptsächlich auf die seismo-akustische Bestimmung der Gebirgsspannungen. Es wurde beobachtet, daß Spannungen durch aufgegebene Kohlepfeiler in höher liegenden, abgebauten Flözen verursacht wurden; der vollständige Abbau der Pfeiler wird, wo immer möglich, empfohlen. Andere Verhütungsmaßnahmen sind das Entspannungsschießen, Tränken mit Wasser und Entspannungsbohren. Im allgemeinen wurden Beispiele aus dem Ostrava Karvina Revier benutzt, welches seit 1918 als besonders ausbruchsgefährdet bekannt ist. Abschließend wird Automatisierung als Mittel gegen die Ausbruchsgefahr empfohlen.

ПРЕДОТВРАЩЕНИЕ СТИХИЙНЫХ ВЫБРОСОВ ПОРОД В ЧЕХОСЛОВАЦКОЙ ГОРНОЙ ПРОМЫШЛЕННОСТИ

А. Скрабис и П. Янас

Авторы анализируют проблемы выбросов пород в двух аспектах - прогнозирование и предупреждение. На практике проблема предотвращения выбросов является специфической для каждой шахты. Прогнозирование базируется первоначально на сейсмоакустическом определении давления и напряжения внутри угольного массива. Давление изучается с целью обнаружения заброшенных угольных целиков в соседних выработанных угольных пластах; рекомендуется полное извлечение целика, где возможно. Другими мерами предотвращения выбросов могут служить взрывные работы, нагнетание воды и аварийное бурение. Примеры, приведенные в докладе, взяты из Острава-Карвинского угольного бассейна, известного своей выбросоопасностью с 1918 года. Автоматизация рекомендуется в качестве лучшего способа борьбы с взрывоопасностью.

捷克煤矿工业岩石突出的防止

A．司葛瑞华士 及 P．加拿士

作者将岩石突出问题分成预测与预防两部份讨论．预测是与矿的具体情况有关预防则主要根据用地震声波方法测定岩石体里的应力与应变．在覆盖的开采过的煤层里被废弃的煤柱中发现常有应力存在；如果可能的话，应当尽量除去煤柱．其他的预防方法包括引爆，注水及应力消除的钻孔．大体上我们用奥士屈拉伐卡维那煤矿的例子．这煤矿自１９１８年开始常有岩石突出现象。我们建议最终应该用自动化办法来处理岩石突出的危害。

A HYDROFRACTURING PROGRAM FOR OUTBURST PREVENTION IN THE SYDNEY COALFIELD, NOVA SCOTIA

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ABSTRACT

Between 1977 and 1984, a total of 37 rock/gas outbursts occurred in the workings of the No. 26 Colliery, Sydney Coalfield, Nova Scotia. All the events occurred below a depth of 700 m and in areas where paleolithic overbank sandstone channels came within 2.7 m of the top of the coal seam. Blasting appears to have triggered 36 of the events and in 23 cases abnormal quantities of methane gas were recorded in the outburst vicinity.

After a brief introduction to the outburst phenomena experienced and the concept of using hydrofracturing for their prevention, the paper presents a detailed examination of the geotechnical, drilling and equipment requirements necessary for an underground hydrofracturing demonstration program.

INTRODUCTION

A total of 37 outburst events were recorded between 1977 and 1984 in the workings of No. 26 Colliery, Sydney Coalfield, Nova Scotia. All the events occurred below a depth of 700 m and in areas where overbank sandstone deposits came within 2.7 m of the top of the coal seam. Each event exhibited a characteristic 'onion-skin' foliation, pulverisation of the sandstone material and appeared to increase in severity with increasing depth, Figure 1. In 36 cases, the outbursts were triggered by blasting, with the exception occurring while workmen were preparing to blast. In 23 cases, abnormal quantities of methane were recorded in the vicinity of the outburst (Forgeron, 1984; Aston and Cain, 1985).

At the time of the outburst occurrences, No. 26 Colliery was operating two conventional advancing single entry longwalls, each 210 m wide and at a depth in excess of 700 m, Figure 2. The longwall panels were

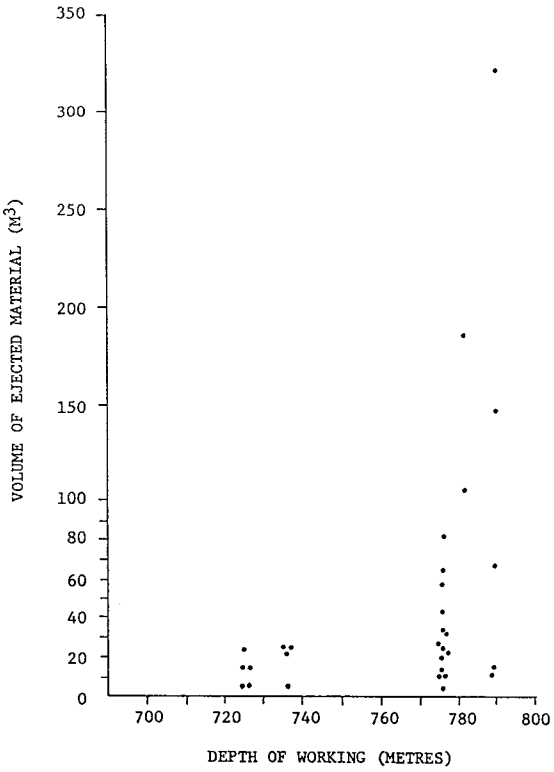


Figure 1 - Volume of ejected material against increased depth of working

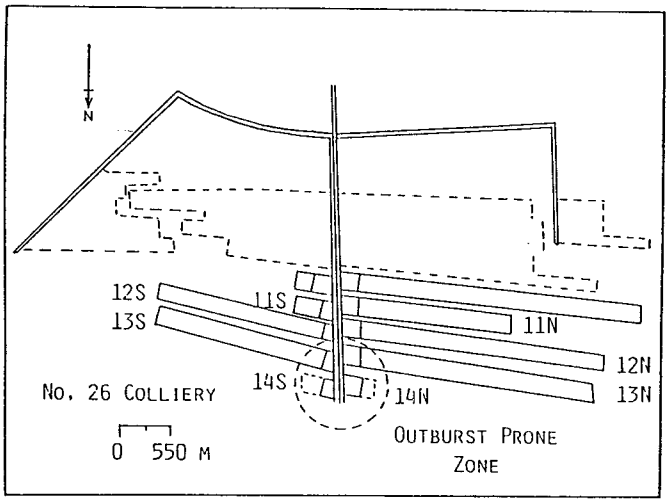


Figure 2 - Plan of No. 26 Colliery

worked along the seam strike, while the main 'deeps' were developed along the seam dip. All development was undertaken using conventional drill and blast methods, with the base of the seam forming the floor of the entry. Steel arches set on 0.9 m intervals, strutted and lagged with timber constituted the support system, although the arch type and size depended on the nature of the roadway. Most of the outburst events occurred outside the influence of adjacent mine workings, although three small events were recorded in the return gateroad of the 13 North panel. Several events were also inferred as having occurred in the longwall waste area, because of tremors on the faceline associated with sudden elevated methane levels in the return roadway.

In April 1984, a fire in the main intake roadways resulted in the closure of the No. 26 Colliery and although the outburst problem disappeared, the phenomena was closely associated with overbank sandstones located in paleolithic river channels which traverse the Sydney Coalfield (Forgeron, 1986). The down dip projection of the adjacent Lingan Mine workings suggests that the same paleolithic river channels (encountered in No. 26 Colliery) will be intersected below 700 m and possibly result in similar outburst occurrences within the next 5 years.

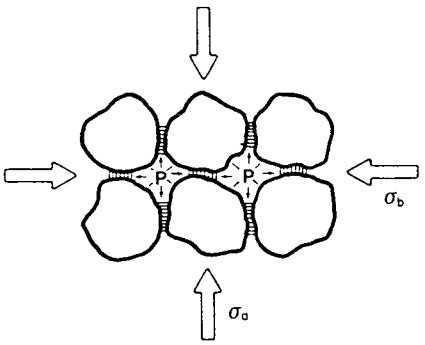
In 1984, the Cape Breton Coal Research Laboratory initiated a research program to examine the feasibility of using hydrofracturing techniques to reduce the internal pore gas pressure of the overbank sandstone deposits, prior to mining, such that the combination of pore gas pressure, blasting and the decrease in confining stress due to mining does not exceed the tensile strength of the sandstone material (AMCL, 1985).

After a brief introduction to the concept of using hydrofracturing techniques for the prevention of gas/rock outbursts, the paper presents a detailed examination of the geotechnical, drilling and hydrofracturing equipment requirements for an underground demonstration program.

Probable Outburst Mechanism

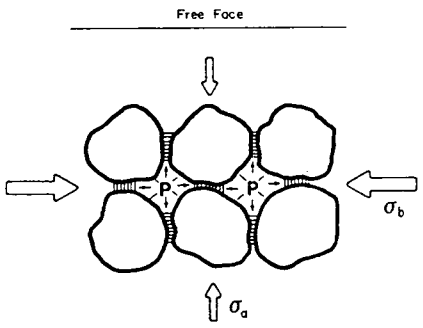
To account for the observed phenomena, an outburst mechanism is postulated based on an internal pore gas pressure which overcomes the tensile strength of the rock mass when the triaxial stress field is altered due to mining. The tensile strength of the immediate roof strata at No. 26 Colliery varied between 8 and 12 MPa (Aston and Cain, 1985).

A simplified analogy is given in Figure 3, which shows a two-dimensional representation of the rock mass grains and the interstitial pore spaces containing methane at a pressure P. Initially, Figure 3 (A), the field stresses are assumed to be equal in all directions and unaffected by mining ($\sigma_a = \sigma_b = \sigma_c$). However, as the roadway approaches, Figure 3 (B), σ_a reduces and the pore pressure approaches the tensile strength of the grain bonds ($\sigma_a < \sigma_b = \sigma_c$). If blasting now occurs in close proximity, the resultant 'P wave' will set up an oscillating longitudinal wave with compressive and tensile cycles. The tensile component of the 'P wave' will further reduce σ_a and result in the failure of the grain bonds in the σ_b/σ_c plane, Figure 3 (C). The bond strength in the other two planes is held in compression, σ_b and σ_c, and do not failure. This mechanism can be used to explain the



A) In situ - steady state

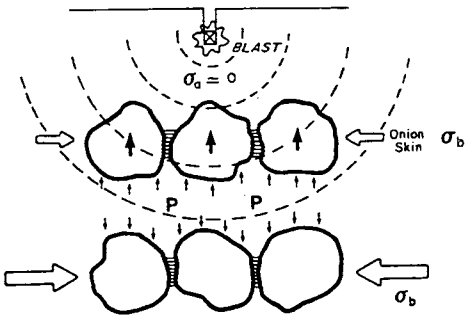
$\sigma_a \approx \sigma_b \approx \sigma_c$



B) Effect on stress configuration caused by new roadway

σ_a reduced

$\therefore \quad \sigma_a < \sigma_b \approx \sigma_c$



C) Onion Skin Exfoliation and Outburst Phenomenon

σ_a approaches 0

σ_b and σ_c reduced

Figure 3 - Conceptual outburst mechanism

'onion-skin' foliation characteristic of the outburst sandstone (AMCL, 1985).

The object of hydrofracturing would be to reduce the sandstone pore pressure prior to the alteration of the stress field due to mining. Total pore gas extraction is not required, although it must be reduced below the point where the combination of pore pressure, blasting and the decrease in confining stress due to mining does not exceed the tensile strength of the sandstone.

HYDROFRACTURING TECHNIQUES FOR OUTBURST PREVENTION

Hydrofracturing techniques have been used since 1947 by the petroleum industry to increase the production capacity of oil and gas wells, by extending their fluid gathering capability beyond the immediate vicinity of the borehole wall. Essentially, the technique involves the creation of a fracture within the target formation by pressurizing a fluid located within the borehole. Fracturing occurs when the induced fluid pressure overcomes both the natural in situ stress regime of the strata and the tensile strength of the rock.

Hydrofracturing techniques have also been extensively used by the mining industry for the surface drainage of methane from coal seams prior to mining (Lambert et al., 1980).

Conceptual Program

The use of hydrofracturing techniques from within the mine workings, for the prevention of outburst occurrences would require two distinct program phases: 1) an exploratory drilling phase to locate the outburst prone sandstones and 2) a hydrofracturing and drainage phase to reduce the internal pore gas pressure.

Conceptually, a borehole would be drilled from within and parallel to the axis of the development roadway.

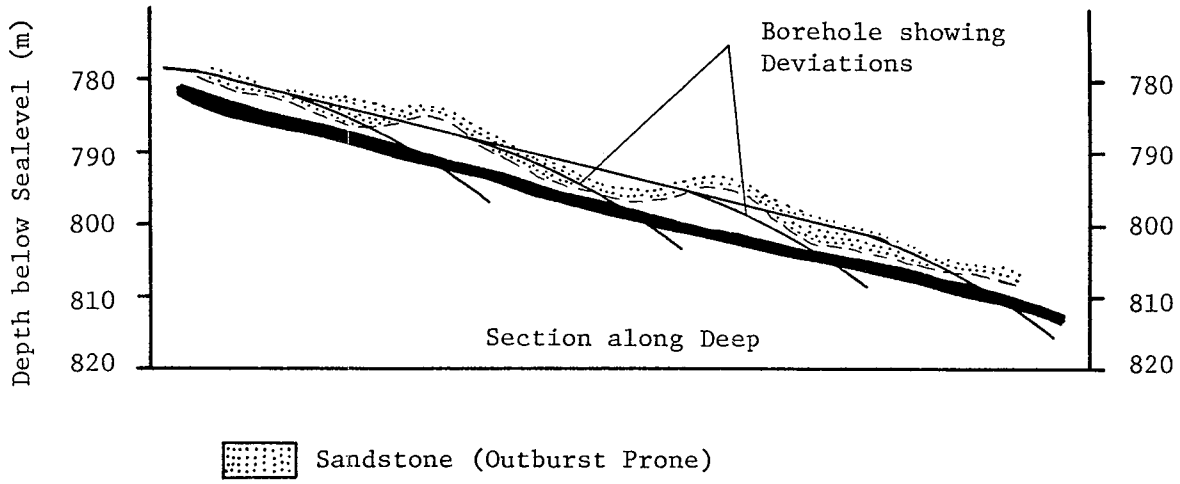


Figure 4 - Conceptual representation of borehole requirements for the location of outburst prone sandstones

In the case of the Sydney Coalfield mines, it would be drilled down dip and preferably located between 3 and 10 m above the coal seam, Figure 4. To assist with horizon control, the borehole would be deflected downwards every 50 m to intersect the coal seam. Once intersected, the deflected portion of the borehole would be grouted, the seam gradient/location calculated and the borehole orientation corrected to maintain the required seam clearance. The horizon control procedure would be repeated every 50 m until either 400 m of borehole were drilled or potential outburst prone sandstone channels encountered. Once a targeted unit was intersected, the hydrofracturing process would be initiated and the liberated gas drained off into the existing methane drainage system.

DRILLING REQUIREMENTS

Drilling Equipment

The ideal borehole size for an underground hydrofracturing demonstration program is considered to be 75 mm diameter (NW). Coring would not necessarily be required, therefore allowing the use of tungsten carbide, diamond drag or tricone bits depending on the expected rock hardness.

Two methods of drilling could be employed: 1) Conventional Rotary or 2) Downhole Motor. The principal advantage of the downhole motor techniques is that the required borehole deflection could be made continuous, therefore preventing 'dog legs' which often occur with the use of 'whipstocks'. Drilling speeds are also much faster since the drill string does not rotate and this causes reduced frictional losses against the borehole sidewall. In addition, more power is transmitted to the bit resulting in increased penetration rates. The downhole motor technique is capable of deflecting up to 10 degrees per 30 m of travel when applied to non-coring applications.

The primary disadvantages of downhole motors is the increased operating cost, although the final unit cost per metre may be similar when compared to conventional rotary techniques because of the greater penetration rate. A high drilling fluid volume is required to power the downhole motor. Up to 6 l/sec may be required, which would necessitate the use of BX rods for an NW size borehole in order to

reduce the fluid friction losses between the drill rods and borehole sidewall. The necessary mud pumps and sump boxes would also require appropriate sizing. Finally, the motor length for a 75 mm diameter hole is 5 m, which would require a drilling location capable of accommodating the one piece motor.

Borehole Deflection

If the downhole motor drilling technique was adopted, borehole deflection could be undertaken using a spring loaded deflection shoe. Essentially, the shoe acts as an eccentric stabiliser which forces the motor off centre at the bit end, thereby causing the bit to deflect from the borehole axis. The amount of deflection is controlled by one or more 'Belville' type springs inserted into the shoe as well as the hardness of the formation being penetrated. A shoe can also be attached at the rod end of the motor and this is orientated 180 degrees from the first shoe, to provide additional deflection force.

In small diameter boreholes, deflections of between 8 and 10 degrees per 30 m of hole can be achieved, although this is reduced to about 5 degrees per 30 m when a wireline core barrel is used. The deflection direction is controlled by turning and locking the drill rod into the derived location at the collar. Straight holes are drilled by removing the deflection shoes and installing concentric stabilisers.

Boreholes drilled using conventional drilling techniques are deflected using 'whipstock plugs'.

Borehole orientation can be checked using a variety of techniques. However, the preferred instrument is considered to be a wireline 'one shot Eastman Whipstock camera', which utilises a magnetic compass to measure azimuth. The instrument is used in conjunction with 6 to 9 m of non-magnetic drill rods to prevent distortion of the magnetic field and must be attached to the rod end of the downhole motor. To compliment the directional survey results, a wireline gamma detector sonde should also be used to identify and differentiate between the different rock types penetrated. Since logging would occur from within the drill string, a detailed calibration program would be required on the known rock types.

HYDROFRACTURING EQUIPMENT REQUIREMENTS

A variety of sophisticated computer models exist for the design of hydrofracturing treatment processes for both oil/gas reservoirs and the drainage of methane from coal seams. The models can be applied to the design of hydrofracturing treatments for outburst control provided a detailed understanding of the reservoir geometry is available as well as an understanding of the relationship between in situ stress, pore pressure, permeability and rock mass strength.

A detailed analysis of the available data from the No. 26 Colliery outburst phenomena, recommends the following treatment process (AMCL, 1985):

Volume of Water	- 6.3 m^3
Weight of Sand	- 2 tonnes (20/40 mesh)
Sand/Water Mixture	- 960 $kg.m^3$
Pumping Rate	- 0.64 m^3/min
Pump Pressure	- 28 MPa
Fracturing Pressure	- 24 MPa
Fracturing Fluid	- Water gelled with hydroxypropyl guar gum

Packers

Optimum packer selection is essential for any hydrofracturing operation. Historically, previous work hydrofracturing coal seams has been done in vertical boreholes with diameters greater than 100 mm. Under these conditions, the packers are of sufficient size to allow over 1 m^3/min flow of fracturing fluid. However, if a 75 mm diameter hole is used, smaller packers would be necessary and these may have to be specially constructed.

If the hydrofracturing process is designed for a pressure of 27.6 MPa, a minimum packer element with an outside diameter of 60 mm would be required. A 60 mm packer has an internal diameter of 19.1 mm and would account for an estimated pressure drop of 40 kPa. Since the anticipated pressure drop through the tubing is estimated at 3.2 kPa/m, the pressure drop through the packer will be comparatively small when compared with that through the tubing.

A recoverable packer is recommended with replacement elements, since the wear factor is expected to be high when used in horizontal boreholes. Alternatively, the use of straddle packers, which comprise two connected packers, might be more convenient since this would allow isolation of the fracture zone within the straddle interval. However, this type of packer may not be feasible because their internal complexity restricts the mandrel diameter to a point where the pressure loss becomes too large to perform the fracturing treatment at an acceptable pumping rate.

Pumps

To meet the proposed hydrofracturing treatment specifications an SA-644-5 well service pump is recommended. This type of pump has a maximum capacity of 31.7 MPa at 0.7 m^3/m and an estimated power requirement of 300 kW. In addition, the pump should be skid mounted for use underground.

Sand Mixing

An SAS-63 slurry averaging system is suggested for the sand/water mixing process. This unit was originally developed for the mixing of cement slurries via a manual loading system and contains a densometer for the accurate determination of sand/water ratios. Ideally, the slurry should be continually mixed during the hydrofracturing operation.

The unit does not directly pump the mixture into the borehole and a 75 kW drive unit is required to operate the mixing pump and activate the pressure operated check values for feeding the high pressure pump. Normally, the unit is diesel powered, but modifications with a flameproof or hydraulic motor would be required for use underground.

Tankage

A pumping rate of 0.7 m^3 and the requirement for gelled water to enhance the sand transportation characteristics, necessitates the provision of an 8 m^3 onsite storage tank(s). An ambient mine temperature of greater than 10°C means that no preheating of the gel would be required, as is normally the case in the oil industry. Also, the use of inflatable bladders for water storage maybe an attractive alternative to the use of metal tanks underground.

DISCUSSION

The drilling program associated with the proposed hydrofracturing demonstration program would serve two purposes: 1) exploration and 2) access to the formations to be fractured. The No. 26 Colliery outburst site was located beneath the Atlantic Ocean, as are all the current Sydney Coalfield mine workings. Few vertical boreholes have therefore been drilled and this has resulted in the poor delineation of both the coal seam structure and the paleolithic sandstone channels thought to be associated with the outburst phenomena. Drilling horizontal boreholes from within the mine workings would allow a much better delineation of the sandstone channel structures as well as access to them.

Once intersected, the relative position of the potential outburst prone unit and the coal seam could be calculated. If hydrofracturing were deemed necessary, no further drilling would be required provided the proper sequencing of the operations was undertaken. To locate outburst prone units further down dip, either a second borehole could be drilled or the first one

extended prior to undertaking the hydrofracturing operations. The use of straddle packers would alleviate the need for multiple boreholes, but unfortunately the internal diameter of the currently proposed packers is too small to achieve the desired pumping rates.

In addition to the acquisition, modification and certification of the equipment necessary for the demonstration program, continuing research should also be undertaken in the areas of outburst prone unit identification, geotechnical/reservoir testing, equipment modification and operational feasibility. Detailed information on both the geotechnical and reservoir properties of the outburst prone unit would be required (in situ stress conditions, rock strength, pore pressure and permeability) in order to optimise the hydrofracturing process and reduce the internal pore gas pressure to a 'safe limit' within a minimum time period.

The equipment and techniques proposed for the demonstration program are currently available and its initiation is not dependent on any new developments. However, some modifications would be required particularly concerning the equipment motors in order to make them permissible for use in underground coal mines. In addition, some new equipment development may be necessary, in order to reduce costs, especially if the technique was to be integrated into the regular mining cycle. However, these modifications are not recommended until a demonstration program has been completed and fully evaluated.

Finally, the demonstration site must be carefully choosen to ensure that the actual hydrofractured location is intersected by the mine workings. This would provide valuable information on crack propagation and the effectiveness of the treatment process and allow adjustments to be made to the simulator model for more accurate predictions.

CONCLUSION

Outburst phenomena experienced at No. 26 Colliery appear to be related to the internal pore gas pressure of the outburst prone sandstone unit, blasting and the alteration of stress conditions around the mine workings. Hydrofracturing techniques offer an attractive method for reducing the internal pore gas pressure of the sandstone unit and decreasing the risk of an outburst occurrence.

The equipment and techniques necessary to conduct a hydrofracturing demonstration are currently available and its initiation is not dependent on new developments. Some modifications would be necessary, primarily regarding the motors, in order to make them permissible for use in underground coal mines. Finally, some new developments may be necessary if the hydrofracturing process were to be integrated into the regular mining cycle. However, the initiation of these developments is not recommended until the demonstration program has been completed and fully evaluated.

ACKNOWLEDGEMENTS

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PROGRAMME D'UTILISATION D'HYDROFRACTURES POUR LA PREVENTION DE DEGAGEMENTS DANS LES MINES DE CHARBON DE SYDNEY, NOUVELLE-ECOSSE

T.R.C. Aston, H.U. Bielenstein, H.A. Anderson

Entre 1977 et 1984, 37 dégagements de roches/gaz se sont produits au total dans les chantiers de la Houillère No. 26 des Mines de Charbon de Sydney, en Nouvelle-Ecosse. Tous se sont produits à plus de 700 m de profondeur, dans des zones où les couches de grès paléolithiques étaient situées à 2,7 m maximum du sommet des couches de charbon. Les opérations de tirage auraient provoqué 36 de ces dégagements; dans 23 cas, des quantités anormales de gaz de méthane ont été relevées au voisinage des dégagements.

Après une brève introduction sur les phénomènes de dégagement qui se sont produits et sur le concept de l'utilisation d'hydrofractures pour leur prévention, notre rapport présente une étude détaillée des éléments géotechniques, de forage et d'équipement nécessaires au développement d'un programme de démonstration d'hydrofractures souterraines.

HYDRAULISCHES AUFLOCKERUNGSPROGRAMM ZUR VERHÜTUNG VON GASAUBRÜCHEN.

T. R. C. Aston, H. U. Bielenstein, H. A. Anderson

Zwischen 1977 und 1984 kam es insgesamt zu 37 Gesteins- und Gasausbrüchen in der Braunkohle-Grube # 26, Sydney Coalfield, Nova Scotia. Alle Ausbrüche fanden in Tiefen von mehr als 700 Metern und in Gebieten, in denen eine paläozoische Sandsteinbank näher als 2,70 m im Hangenden an den Kohleflöz heranreicht, statt. Schießarbeiten scheinen 36 Ausbrüche ausgelöst zu haben, und in 23 Fällen wurden abnormale Methangaskonzentrationen in der Umgebung der Ausbrüche gemessen.

Nach einer kurzen Einführung in das Ausbruchphänomen und das Konzept der hydraulischen Gebirgslockerung zur Verhütung von Ausbrüchen werden im Einzelnen die für ein tieferdiges Auflockerungsverfahren notwendigen geotechnischen Voraussetzungen, sowie Bohrarbeiten und Ausrüstungen beschrieben.

ИСКУССТВЕННЫЙ РАЗРЫВ ПЛАСТА ГИДРАВЛИЧЕСКИМ МЕТОДОМ ДЛЯ ПРЕДОТВРАЩЕНИЯ ВНЕЗАПНОГО ВЫБРОСА ГАЗА НА СИДНЕЙСКОМ УГОЛЬНОМ МЕСТОРОЖДЕНИИ В НОВОЙ СКОТИИ, КАНАДА.

Т.Р.К. Астон, Х.Ю. Биленстейн и Х.А. Андерсон

На 26-й шахте Сиднейского угольного района в Новой Скотии, между 1977 и 1984 годами произошло всего 37 выбросов угля и газа. Все выбросы произошли на глубине ниже 700 м и в местах, где палеорусла, выполненные песчаником, проходили на расстоянии 2.7 м от вершины угольного пласта. Взрывные работы послужили наиболее вероятной причиной 36-ти выбросов, а в 23-х случаях в районе выбросов было зарегистрировано повышенное количество метана. Доклад включает краткое введение, описывающее выбросы газа и породы, а также концепцию использования метода гидравлического разрыва пластов для предотвращения выбросов. Далее детально описываются геотехнические и буровые условия и оборудование, необходимое для демонстрационной программы с использованием метода гидравлического разрыва пластов.

新斯科舍省悉尼煤矿场防止突出的液力加压开裂方案

T．R．C．艾士顿
H．U．比仑士丹
H．A． 安德生

在1977年与1984间，新斯科舍省悉尼煤矿场第26号矿井一共发生了37起岩石和气体的突出事故．所有这些事故都发生在深度为700米和离开煤层顶部2．7米处的旧石器时代的砂石工作面区域内．其中有36起是由爆破触发的，在23起中突出区周围发现大量甲烷．

本文首先简单描述所经历突出现象和使用液力加压开裂方法防止突出的发生，然后用一地下液力加压开裂例子详细介绍对地质构造条件，钻井技术和设备的要求．

STUDY ON THE MECHANISM OF PREVENTING GAS OUTBURSTS BY INFUSING WATER THROUGH SHORT WALLHOLES AND THE CONTROL AREA IN YANGQUAN COAL MINE

B. Qi and F. Zhang
The People's Republic of China

SYNOPSIS

The mechanism of preventing gas outbursts by infusing water through short wallholes and control area are discussed .The investigation in fully mechanized longwall faces of the gassy coalbed "3" reveals that the infused water is used to drive gas out of coalmassif , decrease the initial velocity of gas emission of the coal seam , and move the peak of abutment pressure in front of the working face forward . As a result , when the infusion pressure of an individual hole in coalwall is move than 7.85 Mpa and the length of holes is 2.5 to 3 metres , the control area is 1.5 metres in length ahead of the working face .

INTRODUCTION

The outbursts of coal and gas imperilled mine safety seriously . According to incomplete statistics, No.1 and Beitouzhui pits of the coalbed "3" in Yangquan mining area occurred 466 outbursts from 1982 to 1986 . Among them there were 80 percent on working faces , especially frequent on fully mechanized faces , 76.5 percent of outbursts on the working faces Then the intensity of the outbursts is annually increasing (See Fig.1). Coal magnitude of maximum outburst was 525 ton , Gas ejection with outburst was 18750 cubic metres . Two gas explosion accidents occurred and 5 miners killed because of the outbursts , therefore it is extremely urgent to research the technique of preventing outbursts for fully mechahized longwall faces . Coal seam "3" is anthracite at the top position of Shanxi measure of Permian . The thickness of the coal seam is 1.4 to 2.2 metres, stable seam and simple tectonic condition . It is a simple tilted as whole area . The angle of dip of the seam is 4 to 8 degrees . The coal seam is compact tissue and low permeability . The quantity of gas emission during working was up to 25 to 35 cubic metres / ton . The purpose of the present

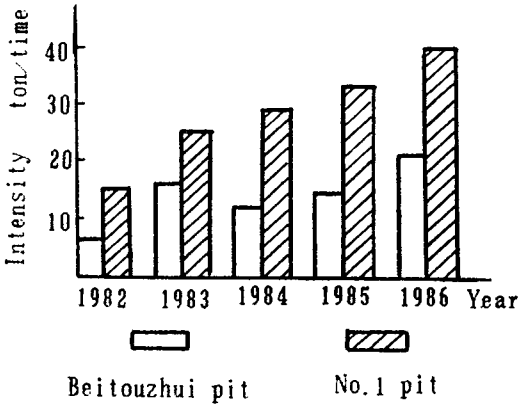


Figure 1 The outburst intensity of the fully mechanized faces .

paper is just to study on the mechanism of preventing gas outbursts by infusing water through short wallholes and the control area on the longwall face "808" at No.1 pit based upon the initial analysis of reasons of outbursts of gas and coal on the working face .

GAS OUTBURSTS ON THE WORKING FACE

Gas outbursts of the coalbed "3" was caused by various factors such as coal seam structure , gas pressure and vertical pressure of overburden. Coalbed "3" could roughly divide into three natural layers .

Upper and middle layers are harder , and low layer is softer (i.e value of "f" ,protodyakonov coefficient of rock strength is about 1) . Hard coal , strong and brittle . Soft coal (tectonic disturbed coal) was lower in strength and larger in tenacity .The initial velocity of gas emission was abrupt .The gas pressure in coal seam is 1.23 Mpa . The coal wall of longwall faces is subjected to abutment pressure of the working face and gas pressure. In general, the peak of abutment pressure in front of the coalseam "3" working faces is 10 to 12 metres ahead of the face . Coal massif which was subjected to compressed pressure in this area moved to working face , but it kept stable by the resistance of the large coal pillar.The potential energy of elastic deformation increased with the thickness of the soft coal and cutting speed of the machine . As a result , it disturbed the equilibrium of perssure of the working face, for that moving the peak of abutment pressure in front of the face forward (See Fig 2) . The peak of abutment pressure which measured on the face "906" of Beitouzhui pit was only 7 metres ahead of the face when outburst occurred , therefore it caused the hard coal layers appeared plastic deformation . And then soft coal layer could sud-

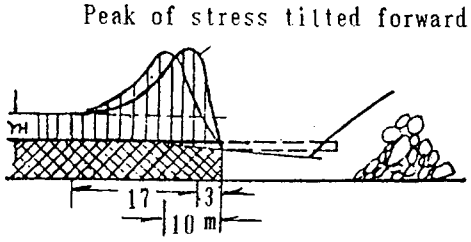


Figure 2 The distribution of the abutment pressure of the working face .

denly release the elastic potential energy , coalwall was thrown out toward the working face . In the meanwhile the vertical cleats appeared at 3 to 5 metres of length inside of coal massif in front of the face . Gas in the soft coal seam degased rapidly and emitted out through vertical cleats and fissure separation between the roof rock and coal at a high speed .Roof fractured and dramatically sound of coal damaging during the period of outbursts . The outburst coal was non-separability. This dynamical phenomenon is called pressing out of the coal . Both stratum pressure of the working face and gas pressure are the main forces .

The outburst can divide into three stages depend upon the process of development , i.e Deliberate on , break out , and declining stage (See Fig.3).The characteristic of each stage as following,

Deliberate on , Means of the movement of the roof commenced , and the overburden load on wall increased.The gas emission changed with time and increased as a whole .At this time, the factors which affecting the change of vertical pressure of the working face came from cutting of machine and moving of supports . The rate of outbursts increased with the high cutting speed and dramatical change of vertical pressure of the working face . Unsupported area of the roof enlarged during

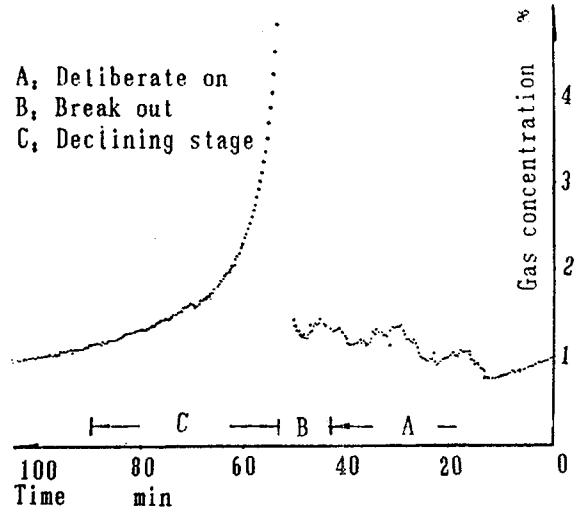


Figure 3 The curve of gas concentration in return air on the face"808" when an outburst occurred (a recording of the gas remote-automatic continuous monitoring meter).

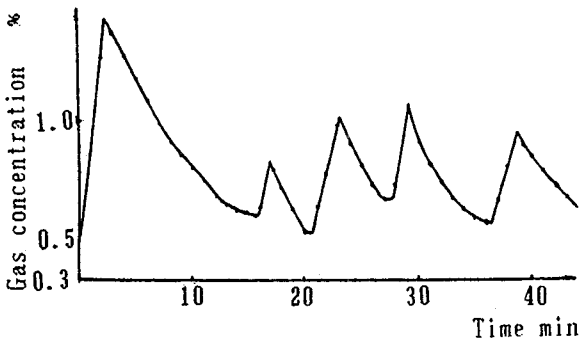


Figure 4 The affecting curve of gas concentration in return air due to support moving.

the process of moving supports, and abutment pressure on wall intensively increased;moreover , the gas emission rose (See Fig. 4).

Break out, Means of the stratum pressure extremely changed . Coalwall was fractured and thrown toward the working space , large quantity of gas emitted from top separation of coalseam at once . The sustained time of ejection from only several seconds to several minutes . Fig. 3 is a typical curve of gas concentration of return air of the longwall face when the outburst occurred . On the face "602" of Beitouzhui pit , the gas concentration of return air was up to more than 30 percent .

Declining stage, Potential energy in coal massif basically released . Gas of tectonic disturbed coal zone largely emitted out , output of residual gas declined with time . Gas emission appeared the "Wave of output" because of the roof moving .

The variation regularity of gas emission while breaking out and declining stages with time , which follows the empirical equations,

Breaking out, $q_1 = q_o e^{0.4 t^{1.7}}$

Declining stage, $q_2 = q_m e^{-0.415 t^{0.4}}$

Where, q_1 --- Gas output of "t" time during process of outburst , cubic metres per minute;

q_2 --- Gas output of "t" time during declining stage , cubic metres per minute;

q_o --- Initial gas output of outbursts, cubic metres per minute;

q_m --- Maximum quantity of the outburst gas , cubic metres per minute;

t --- Time , minute;

e --- 2.7183 .

The relation of qm to the area of tectonic disturbed coal is direct ratio .

We believe that the deliberate on is control lable stage of gas outbursts by the analysis of three stages of outburst process associate with practice experiences. Minners stopped the operation when the roof moving began , such as coal cutting and

support moving . In this way , they could release the pressure of wall and roof, and avoid the outburst accidents or decrease the intensity of outbursts . Miners found a set of techniques to prevent gas outburst , for example , cutting speed is not more than 3 metres/minute , to stop cutting at once when heard the roof sound , the distance of coal cutting and support moving is more than 20 metres , it is particularly to avoid some supports moving at same time , and no residual coal massif at two ends of the working face .

The gas outbursts is impassable at breaking out stage . Automatic power-off firedamp dectator could disconnect the electricity power of machines and equipment located at face and return system when the gas concentration rapidly rose .Using this method , we could eliminate accidents of the gas outbursts . This is important to mine safety production .

THE MECHANISM OF PREVENTING OUTBURSTS BY WATER INFUSION

Infusing water through short wallholes took advantages of abundance of regenerating cleats within destress zone and high permeability of water to meet the hydraulic fracturing .Infusing boreholes which against coalwall and single row distributed along the working face.The length of an individual hole was 2.5 to 3 metres . The diametre of holes was 45 milimetres . The interval among holes was 2.5 to 3 metres . Using self-sealing type apparatus . The water penetrated length of the hole was 1.5 to 2 metres .The experience has proven that the infusion pressure of the coalbed "3" should be more than 7.85 Mpa. the quantity of water infusion estimated by the "perspiration" appeared on coalwall or water out from the nearby holes. The infusion pressure of an individual hole on fully mechanized longwall face was from 7.85 to 11.77 Mpa . Average time of infusion was 4.7 minutes .The water quantity for each hole was 0.21 cubic metres .

By investigation , water infusion through short wallholes produced the following physical changes to coal massif and gas emission;

1. The water infusion is used to drive gas out of the coalseam .On the fully mechanized face "808" , the quantity of gas emission in return air was 4.4 cubic metres / minute before infusion . It rose up to 5.63 cubic metres/minute during the period of infusion. The differential value was 1.23 cubic metres /minute . Fig.5 is the variation curves of gas concentration in return air before and after infusion .

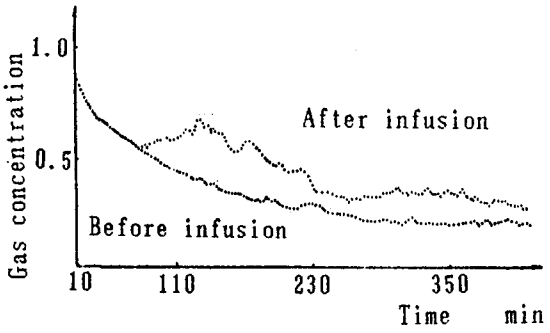


Figure 5 The variation curves of the gas concentration in return air before and after infusion .

Incremental gas while infusion came from fractures of coalwall and boreholes . The variation regularity of velocity of the gas emission from the nearby boreholes (See Fig. 6) . The gas emission from just finished borehole was rapid , the speed might be up to 1.1 metres/minute , then it declined with time . Its change is ralated to variation of pressure on the roof . The rate of gas emission rose rapidly while the nearby holes

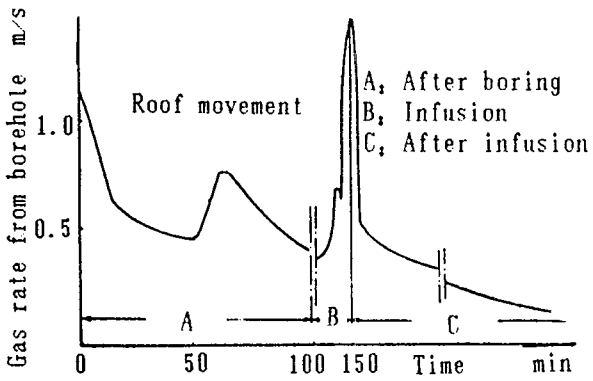


Figure 6 The gas velocity curves during the boring and infusing .

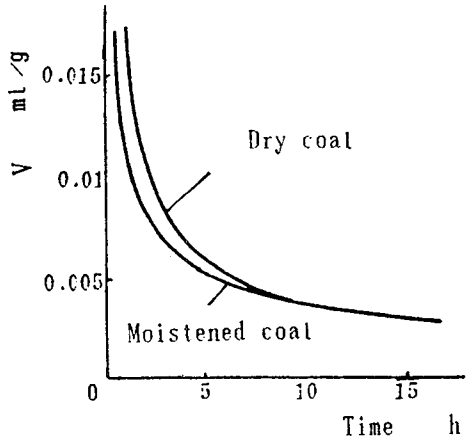


Figure 7 The velocity curves of gas diffusing .

were infusing , and when water maximum out of fracturing wallholes , the rate was 1.4 cubic metres/minute . After then it declined abruptly to output of gas before infusion . According to statistic data , the quantity of gas out from each hole is up to 7.8 cubic metres .

2. water infusion could moisten the coal massif, all of which reduced the initial velocity of gas emission . On the fully mechanized face "808" , the moisture content of the coal changed from 1.6 to 2.45 percent after infusion . while the moisture content on the face "1210" of Beitouzhui pit increased by 1.23 percent . The diffusing experiments of the moistened coal sample in laboratory has shown that the initial velocity of moistened coal decreased by 69.63 percent. Fig. 7 shows the gas emission curves of the coal sample of the coalbed "3" .

Water infusion is designed to break up the coal massif under the impact of high pressure water infusion. It changed the mechanic state of gas within coalseam and had the gas emission while cutting of machine reduced obviously .In the same condition of the production and cutting cycle , the gas emission of a cut on the face "808" reduced 25.07 percent . It reduced 28.6 percent after infusion on the face "1210" , see table 1 .

Table 1 Gas emission on the fully mechanized face "1210" .

Circumstances	Gas emission (m^3/min)	Specific gas (m^3/ t)
Before infusion	15.65	17.12
After infusion	11.17	11.73

3. Aluod sound appeared during the process of the infusion , the water wedge with the high pressure broke up coalmassif and removed toward the working space, sometimes the coalwall collapsed . The measurements on the working faces of East panel of the " Смоляниновский " seam in USSR. have shown that the maximum displacing of the coal seam was 52 metres when coal fractured by hydraulic force (1) . In this case , the destress zone in front of the working face widened , support power to

roof decreased and the peak of abutment pressure ahead of the working space were moved forward . and widened tecnotic disturbed zone relieved the accumulation of abutment pressure .

As mentioned above , waterinfusion changed the physical and mechanical property, decreased the stratum pressure of the working face and retention of methane, and alleviated the accumulation of potential stress on the wall. so eliminated the hazard of outbursts .

DISCUSSION OF THE CONTROL AREA OF INFUSION

The effect of preventing gas outbursts by infusing water has obviously proven . 22 outbursts occurred among ten days before infusion on the face "808" , and only 2 outbursts occurred in moistened coal zone after infusion . The number of outbursts of 1000 ton of coal production reduced 82.9 percent, and the outbursts on the face "1210" of Beitouzhui pit reduced 81.82 percent after infusion .In 1987, we popularized this technique of preventing outbursts by water infusion to six fully mechanized faces liable to outbursts .The outbursts and ejections on these faces decreased 72.2 percent as compared with 1986. The average intensity of an outburst decreased 36.5 percent .

To probe into the significance of the control area to safety production , we carried on the monographic studies .

1. The interval among holes depends upon the moistened radius . When the sealling length of a hole is 1 metre and operating two cuts of coal after infusion (advanced 0.5 m per cut), we measured the moisture content of the coal sample , see table 2 .If the incremental water content was 1 percent (effect of controlling gas outbursts is obvious when the incremental content is 1 percent) , the moistened radius was 2.4 metres . Therefore we believe that the 3 metres of interval among holes is enough to pressing fracture even and moistening the coalmassif .

Table 2 The measurements of moisture content along the infusion hole .

Distance from a hole collar (m)	0.75	1.9	3.05
Moisture content (%) (Initial value is 1.6)	3.24	2.9	2.225

2. The control depth of preventing outbursts. The axial control range of the short wallholes depends upon the length of a hole , sealling length and infusion presuure . The practical method of estimating the range in-situ was that how many cuts were operated before next outburst . According to the observation on the face "808" , in general, first cut was operated after 4 hours of maintenance of equipment on the face , whether water infution or not , the chance of outburst was very low (See Fig. 8) .

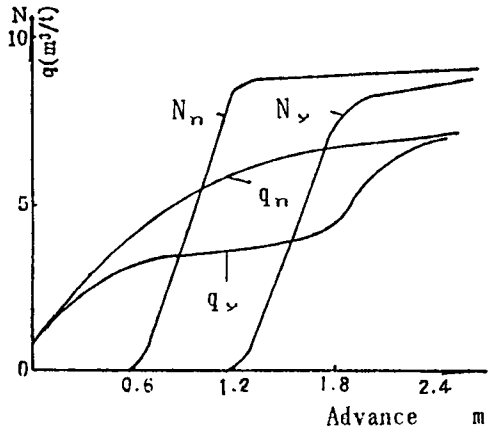


Figure 8 The correlation curves of water infusion against the gas emission and outbursts .

Where, N_n --- The outburst number of 10000 ton of coal production without infusion;

N_y --- The outburst number of 10000 ton after infusion;

q_n --- Gas emission before infusion;

q_y --- Gas emission after infusion;

The outburst chance of both fourth cut of coal after infusion (advanced 2 metres) and second cut without infusion (advanced 1 metre) was 8.45 times per 10000 ton of coal production, so that we believe that the controlling depth is three cuts of coal (advanced about 1.5 metres).

We measured the moisture content of the coal sample along the length of the hole, see table 3. When the incremental value of first three cuts of coal was 1.38 percent, the effect of preventing gas outbursts was obvious. In Fig.8, qn and qy are gas emission curves during the period of cutting before and after infusion. Gas sufficiently released within three cuts of coal, the ratio was low.

By the analysis of the ratio of outbursts, the incremental water content and the gas releasing, we believe that the three cuts of coal (advanced 1.5 m) as the controlling depth is quitable.

On the other hand, Infusing water enables to widen the fractured zone of the coalwall, we estimated that the value of the widened fracture zone is control depth of outburst by water infusion. This value was determined by the distance between the stirring point of outburst and coalwall. The stirring point of outbursts is beginning point of pressing out of the coal. In the common case, there were vertical cleats conducting the dividing line of virgin and disturbed coalmassif. The latter was abundance of fracture and the gas released sufficiently. The gas in virgin coal seam hadn't released. We drilled the boreholes within the outburst zone after the outburst, gas of virgin coal seam beyond the stirring point emitted into boreholes. We determined the outburst area by measuring the velocity of gas emission. Fig. 9 is correlation curve of outburst area versus the velocity of gas emission of the hole without infusion. The rate of gas at the

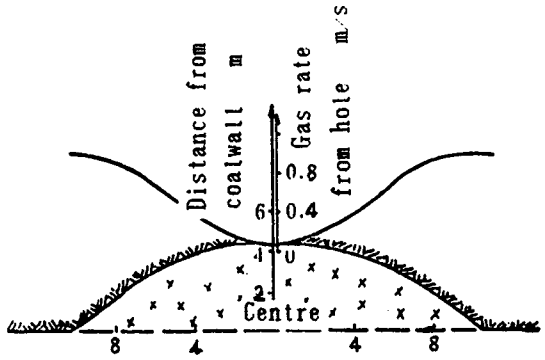


Figure 9 The relation curve of the outburst range versus the gas velocity.

point of 4.5 metres inside of wall was 0.09 m/s. In the same may, the velocity of gas at the point of 6 metres inside of wall in outburst zone after infusion was 0.11 m/s, therefore we realized that the different

Table 3 The measurements of moisture content around the hole.

Length of hole (m)	0.5	1.0	1.5	2.0	2.5
Moisture content (%) (Initial value is 1.6)	2.1	2.8	2.98	2.6	2.2

value of the stirring point of outburst before and after infusion is 1.5 metres and inferred the peak of abutment pressure in front of the working face 1.5 metres forward . It has proven the analysis described above .

It should be particularly pointed out that the present 3 metres of a hole is less than the distance between the peak point of abutment pressure and wall , so it couldn't completely prevent the outbursts .

CONCLUSION

The outbursts of coal and gas in the coalbed "3" is belong to pressing out type . Both the stratum pressure of working space and gas pressure are main forces of stirring outbursts , it occurred on the working faces frequently .

Infusing water by short wallholes could change physical and mechanical property of coalseam , reduce the accumulation of the potential energy . It has an obvious effect of preventing outbursts associate with controlling the cutting speed and moving supports . By present infusion parametres , the controlling depth is 1.5 metres , but outburst couldn't completely eliminate .

To ensure safety of production , exert the ability of the fully mechanized equipment and eliminate the hazard of outbursts , we should use the length of the hole corresponded with the distance between the peak point of abutment pressure and coalwall, increase the sealling length of holes and raise the infusion pressure . If we utilized the crisscross boreholes , destressed the high pressure zone during the process of working operation and enlarged safety range of working , we would get an ideal effect of preventing outbursts by infusing water .

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阳泉煤矿浅孔煤壁注水防突机理与有效范围研究

(Study on the Mechanism of Preventing Gas Outbursts by Infusing Water through Short Wallholes and the Control Area in Yangquan coal Mine)

[中 国] 邵 兵 印 张 福 喜

提 要

本文对浅孔煤壁注水防突机理和有效范围进行论述。通过在3号煤层综采工作面试验研究表明：注水有驱赶煤壁瓦斯的作用，还可降低煤层瓦斯放散速度和使煤层应力集中峰向煤层深部推移。在综采工作面单孔注水压力达7．85 Mpa以上，钻孔深度为2．5～3m时，有效防突范围深1．5m。

前 言

煤与瓦斯突出严重威胁矿井安全生产。阳泉煤矿开采3#煤层的一号井和北头咀井从1982年至1986年共发生突出446次，其中回采工作面的突出次数占80%。综采工作面突出频繁，占回采工作面突出次数的76．5%，而且突出强度逐年增大（见图1），最大一次突出煤炭525吨，喷出瓦斯17850m^3。有两次突出酿成瓦斯爆炸，死亡5人。探讨既能防止突出又能适应综采发展的防突技术是十分急迫的。

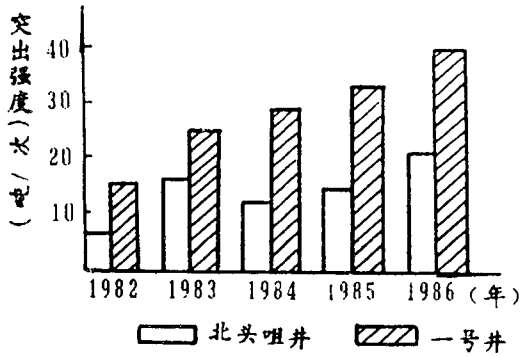


图1 综采面突出强度

3#煤层为二迭纪山西组上部无烟煤层，厚1．4～2．2m，赋存稳定，构造简单，全区呈波状起伏的单斜构造，倾角4°～8°。煤层致密，透气性低。采煤过程中瓦斯涌出量达到25～35m^3/t。本文在初步分析回采工作面煤与瓦斯突出原因的基础上，重点对一号井808综采工作面浅孔煤壁注水防突机理及有效范围进行探讨。

一、回采工作面瓦斯突出原因

3＃煤层瓦斯突出是受到煤结构、瓦斯压力、地应力的综合作用结果。3＃煤层大致可分为三个自然分层，上、中分层较硬，下分层软（f值为1左右）。硬煤强度高，性脆；软煤强度低，韧性大，瓦斯放散初速度快。煤层瓦斯压力为 1．23Mpa。采煤过程中工作面煤壁受到采场应力和瓦斯压力的作用。一般3＃煤层工作面超前集中应力峰顶距煤壁10～12m，该区域煤体承受压缩应力，有可能向采煤工作空间一侧移动，但较大的煤柱阻力使煤壁保持稳定。软煤较厚时，煤层弹性变形潜能增大，机组割煤速度快，瞬间破坏了采场应力平衡，使工作面超前集中应力峰前移（参阅图2），北头咀井906工作面实测发生突出时峰顶距煤壁只有7m。超前集中应力峰前移造成硬煤呈塑性破坏，软煤弹性潜能突然释放，煤壁被推出。同时在煤体内3～5m处出现竖直裂缝，软煤瓦斯很快解吸经由竖直裂缝和顶部离层口高速喷出。突出过程顶板断裂，煤体破坏声响剧烈。突出的煤没有分选性，这种动力现象属煤的突然压出型。采场应力和瓦斯压力是激发突出的主要动力。

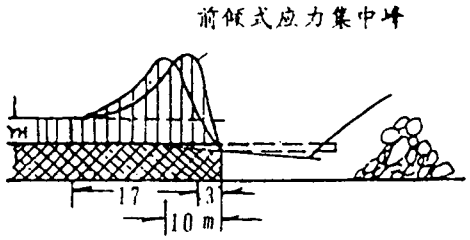


图2　采场应力分布示意

突出按其发展过程可分为潜伏期、发动期和衰减期三个阶段（见图3），各阶段特征如下：

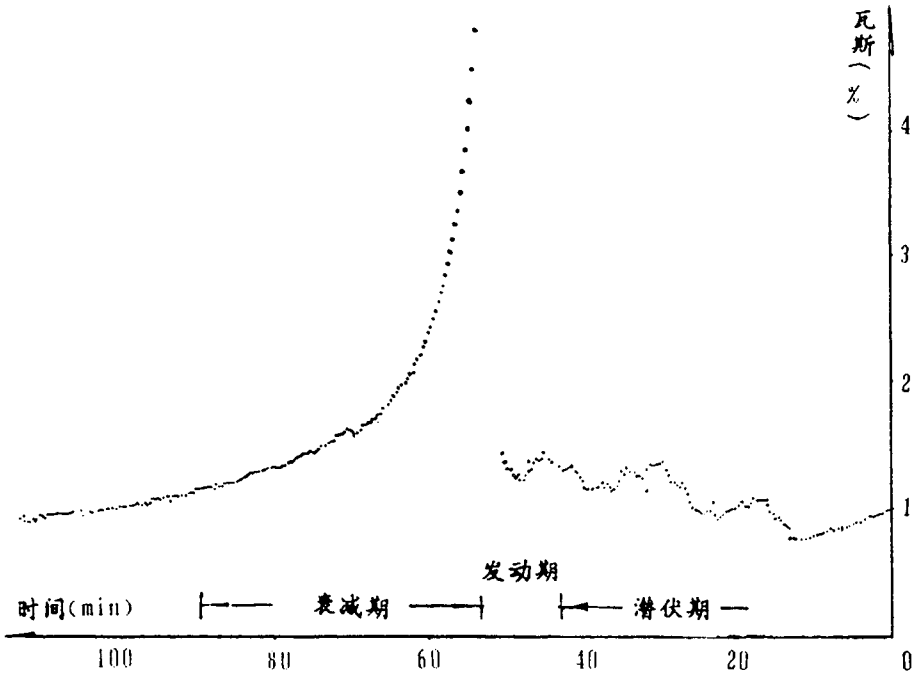


图3　一号井808综采面突出时回风瓦斯浓度变化曲线（瓦斯遥测仪记录）

潜伏期：顶板开始活动，煤壁承载增加，可听到“隆隆”声响。释放瓦斯忽大忽小，总的为上升趋势影响采场应力变化的因素来自机组割煤和推移支架。机组割煤速度快，采场应力变化剧烈，突出机率增高推移支架过程空顶面积增大，煤壁应力急增而破碎，瓦斯涌出量增加(见图4)。

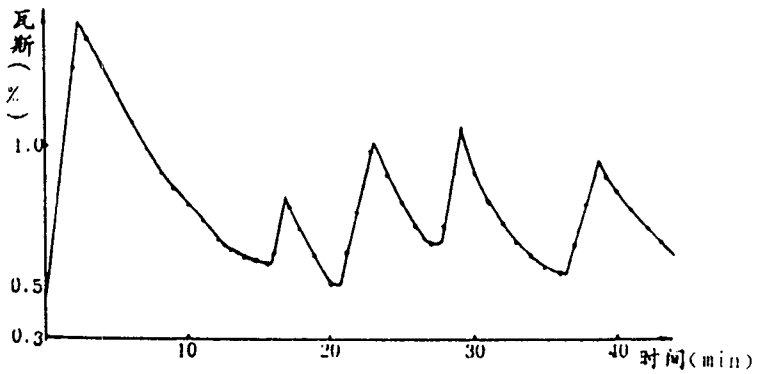


图4　工作面推移支架时引起的煤炮声对回风流瓦斯浓度的影响曲线

发动期：地压变化剧烈，煤壁压裂发出雷鸣般的撅炮声，在煤壁被推出同时，顶部喷出大量瓦斯，时间只有几秒至几分钟。北头咀井602工作面突出时回风瓦斯浓度达30%以上。

衰减期：煤体潜能基本释放，破坏煤体内卸压瓦斯大部喷出，残余瓦斯涌出量随时间下降。因顶板应力释放后采场尚有短时间余震，瓦斯涌出呈微小波动。

发动期和衰减期瓦斯涌出量随时间变化规律符合指数方程：

发动期　$q_t = q_0 e^{0.4 t^{1.7}}$

衰减期　$q_t = q_m e^{-0.415 t^{0.4}}$

式中：q_t———突出过程t时瓦斯量，m^3/min；

q_t———衰减过程t时瓦斯量，m^3/min；

q_0———突出初始瓦斯量，m^3/min；

q_m———突出高峰点瓦斯量，m^3/min；

t——— 时间，min；

e——— 自然对数底。

q_m与突出破坏煤体范围成正比关系。

通过对瓦斯突出过程三个阶段的分析，并结合现场实践经验，认为潜伏期是瓦斯突出的可控阶段。当顶板开始移动发出撅炮声，立即停止破坏采场应力平衡的作业，如机组割煤、移架等，而使顶板和煤壁应力缓慢释放，就能避免一次突出或消弱突出的强度。采煤工在实践中摸索出一套控制操作工艺防止突出的经验，诸如割煤速度不超过3m/min，割煤时听到顶板响声立即停机；割煤和移架距离大于20m，特别要避免几台支架同时推移；割煤过程中不留隅角，不斜采造成三角煤出现应力集中

进入发动期瓦斯突出已不可避免，但当瓦斯浓度迅速上升时，可利用瓦斯超限自动断电仪切断工作面和回风系统电气设备电源，杜绝发生瓦斯事故，这一点对煤矿安全生产极为重要。

二、浅孔煤壁注水防突机理

浅孔煤壁注水利用了工作面煤壁卸压带再生裂隙发育，透水性能提高的特点，达到水力破裂煤层的目的。注水钻孔垂直煤壁呈单排眼布置，深2．5～3m，孔径为45mm，孔间距2．5～3m。采用橡胶钢丝网自动膨胀封孔器封孔，钻孔透水长1．5～2m。实践证明，3#煤层注水压力不能低于7.85Mpa。注水质量以煤壁“挂汗”或相邻钻孔出水为标准。综采面单孔注水压力在7．85～11．77Mpa，平均注水时间4．7min，注水量为0．21m^3。

通过考察，浅孔注水对煤体和瓦斯涌出发生以下物理变化：

1、浅孔注水驱赶煤层瓦斯释放。808综采面注水前回风瓦斯量为4．4m^3 min，注水过程中增加到5．63m^3/min，比注水前增加了1．23m^3/min。图5为工作面注水前后回风瓦斯浓度变化曲线。

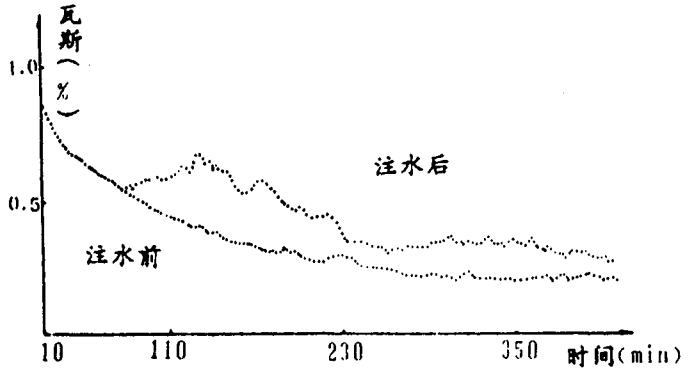


图 5　工作面注水前后回风流瓦斯浓度变化曲线

注水期间所增加的瓦斯量，一方面来自压裂的煤壁，另一方面是钻孔涌出的。注水过程中相邻钻孔瓦斯涌出流速变化规律见图6。钻孔刚打完孔内瓦斯涌出较快，流速可达1．1m/s，以后随时间衰减，受顶板压力变化而波动。邻孔注水时，瓦斯涌出速度迅速上升，当煤壁压裂钻孔出水达最大值1．4m/s，以后急速衰减接近注水前瓦斯涌出速度。据统计每个钻孔可涌出瓦斯7．8m^3。

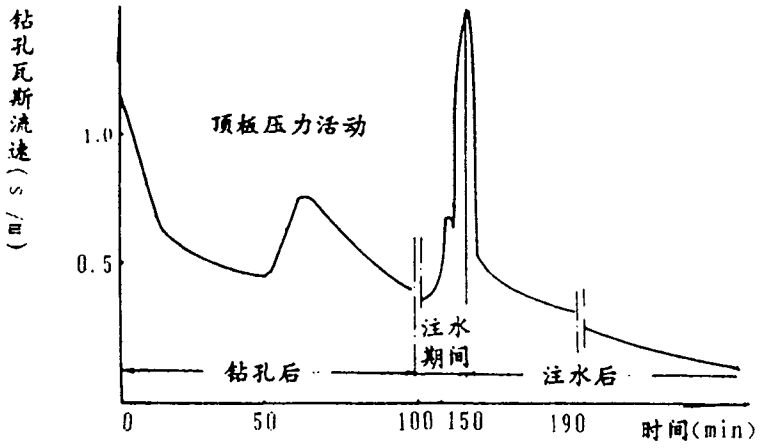
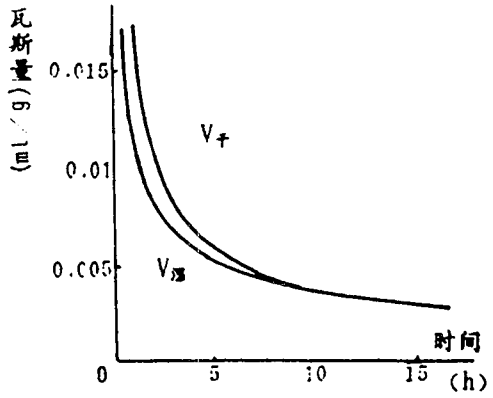


图 6　打钻注水期间钻孔瓦斯流速曲线

2、注水湿润煤体降低煤层瓦斯放散速度。808综采面注水后煤层水份由1.6%平均增至2.75%。北头咀井1210综采面水份净增1.23%。实验室进行湿煤样瓦斯扩散试验表明，湿煤瓦斯扩散速度较干煤最大降低69.63%。图7为3#煤层煤样瓦斯扩散速度曲线。



煤样粒度：10～15 mm
瓦斯压力：1.51 Mpa
干煤：V干 = 0.2299t −0.6345
湿煤：V湿 = 0.1007t −0.5124

图 7　瓦斯扩散速度曲线

浅孔注水压裂煤体驱赶瓦斯和湿润煤体抑制瓦斯结果，改变了瓦斯在煤体内的力学状态，使机组割煤时瓦斯涌出量明显减少。在产量和割煤循环时间接近条件下，808综采面割一刀煤瓦斯涌出量减少25.07%。从表1看出1210综采面湿润煤体后瓦斯涌出量降低28.6%。

1210 综采面瓦斯涌出量　　表 1

注水情况	瓦斯涌出量 m³/min	相对瓦斯量 m³/t	对比 %
未注水	15.65	17.12	
注水后	11.17	11.73	−28.6

3、浅孔注水过程中煤壁发出连续“叭、叭”的声响，高压水楔将煤壁进一步压裂，煤体向工作空间推移，有时出现片邦。据苏联高尔基矿井“斯姆良尼诺夫”煤层东翼工作面测定，进行水力破裂时煤层最大移动值可达52mm。工作面前方煤壁破碎卸压带加宽，对顶板支承力降低，而使超前集中应力峰向煤体深部移动。加宽的破碎卸压带阻抗住集中应力对煤壁的挤压。

综上所述，浅孔煤壁注水改变了煤层物理力学性质，降低了采场应力和瓦斯赋存状态，缓和了煤壁应力潜能的聚积，从而消除突出的危险性。

三、浅孔煤壁注水有效防突范围的探讨

浅孔注水防突效果通过实践证明是显著的。808综采面未注水的10天中发生突出22次；注水后在湿润区仅发生2次，千吨突出次数减少82.9%。北头咀井1210综采面注水后突出次数降低81.82%。1987年在两对突出矿井的6个突出综采面推广浅孔注水防突技术，突出和喷出次数比1986年减少72.2%。综采面平均突出强度也降低36.5%。

探讨注一次水的有效防突范围对综采安全生产有现实意义，对此进行了专题考察。

1、单孔注水湿润半径是合理确定钻孔间距的依据。当封孔长度为1m时，注水后机组割完两刀煤（每刀进度0.5m）测定煤样水分，结果见表2。以煤层水分提高1%为基础（实践经验水分提高1%

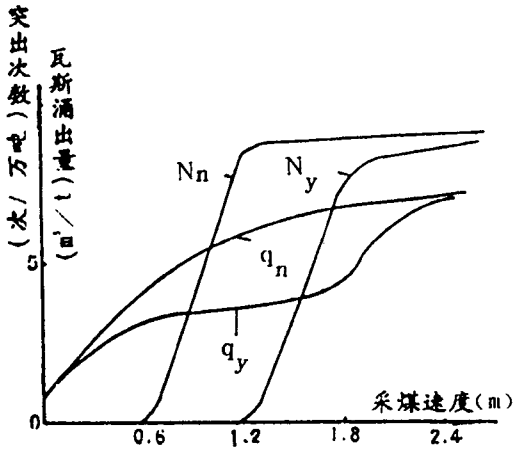
处理瓦斯效果较明显），则湿润半径为2．4 m，因而确定3 m钻孔间距对均匀压裂和湿润煤体是充分的

注水孔径向水分测定结果　　表 2

与钻孔距离 m	0．75	1．9	3．05
水分值 % （原始水分1.6%）	3．24	2，9	2．225

2、浅孔注水有效防突深度。

浅孔注水轴向有效防突深度与钻孔长度、封孔深度和注水压力有直接关系。现场衡量有效深度最实际的办法就是考察注水后机组割第几刀煤时开始发生突出。据808综采面考察，一般在工作面检修四小时后割第一刀煤，不论注水还是不注水，突出机率都是很低的（参阅图8），注水后机组割第四刀煤（进度2 m）和不注水割第二刀煤（进度1 m）突出机率都在8．45次／万吨左右，可以认为有效防突深度为三刀煤（进度1．5 m）。



Nn：　未注水万吨突出次数
Ny：　注水后万吨突出次数
qn：　未注水瓦斯涌出量
qy：　注水后瓦斯涌出量

图 8　注水对工作面瓦斯涌出量和万吨突出次数影响关系曲线

沿钻孔不同深度（钻孔两侧1．5 m）取煤样测定水份（结果见表3），第三刀煤水份增加1．38%．防突效果较明显。

注水孔轴向水份测定结果　　表 3

沿钻孔深度 m	0．5	1．0	1．5	2．0	2．5
水分值 % （原始水分1．6%）	2．1	2．3	2．98	2．6	2．2

图 8 q_n与q_y为注水前后机组割煤过程中瓦斯涌出曲线，圈内面积为注水压裂煤体放出的瓦斯量（包括湿润煤体抑制涌出的瓦斯量）。前三刀煤（进度1．5 m）瓦斯得到较充分释放，万吨突出机率也低。

通过对突出机率、水分增长和瓦斯释放的综合分析，认为浅孔注水有效防突深度为三刀煤（进度为

1．5 m）的理由是充分的。

另外，浅孔注水使煤壁破碎带加宽，可以认为破碎带加宽的数值就是浅孔注水有效防突深度，通过分析注水与不注水时突出策动点位置与煤壁距离来确定此值。突出策动点也即是发生突出煤体被压出的始点往往在策动点有竖直裂缝作为原始煤体和压碎煤体的分界。压碎煤体裂隙发育瓦斯得到充分释放；原始煤体内瓦斯尚未释放。突出后沿突出区打钻孔，策动点以里原始煤体应向钻孔涌入瓦斯，用热球式风速探头测定钻孔内不同深度瓦斯流速来确定突出范围。图 9 为未注水区突出范围与钻孔瓦斯流速关系曲线。煤壁以里4．5 m地方瓦斯流速为0．09 m／s，同样测定注水后突出区煤壁以里6 m地方钻孔瓦斯流速为0．11 m／s。由此可以认为浅孔注水前后突出策动点相差1．5 m，也可推断工作面超前集中应力峰向煤体深部推移1．5 m，这和前面分析的有效防突深度是一致的。

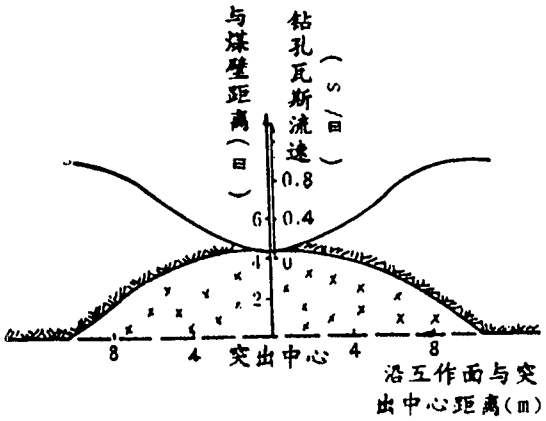


图 9　突出范围与钻孔瓦斯流速关系曲线

应该指出，目前采用的3 m长注水钻孔小于激发突出的超前集中应力峰与煤壁的距离，因而还不能完全防止煤与瓦斯突出。

四、结　　论

3＃煤层煤与瓦斯突出属压出型，采场应力与瓦斯压力是激发突出的主要动力，其特点是回采工作面突出频繁。

浅孔煤壁注水能改变煤层物理力学性质，减小聚积势能，与机组割煤速度、拉架相配合，有明显的防突效果。现用注水参数有效防突深度为1．5 m，而且还不能完全防止突出。

为确保综采面安全生产，并充分发挥综采设备能力，消除突出的危险性，应使注水钻孔长度与工作面超前集中应力峰距煤壁尺寸相一致，并增加封孔深度和提高注水压力。采用交错布孔注水，消除采煤过程中煤层高应力区，扩大安全回采范围，取得浅孔注水防突最佳效果。

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ETUDE DES MOYENS DE PREVENTION DES DEGAGEMENTS GAZEUX PAR INFUSION D'EAU DANS DE COURTS TROUS DE PAROIS ET DANS LA ZONE DE CONTROLE DE LA MINE DE CHARBON DE YANGQUAN.

Qi Bingyin, Zhang Fuxi

Nous examinons les moyens de prévention des dégagements gazeux par infusion d'eau dans de courts trous de parois et dans la zone de contrôle. L'étude en profondeur des fronts de taille de la couche de charbon 3 révèle que l'eau infusée fait sortir les gaz du massif de charbon, diminue la vélocité initiale de l'émission gazeuse de la couche de charbon et déplace vers l'avant la pression de pointe de la charge de culée devant le front de taille. En résultat, lorsque la pression d'infusion d'un trou individuel d'une paroi de charbon est supérieure à 7845 KPa et lorsque la longueur du trou est comprise entre 2,5 et 3 mètres, la zone de contrôle se situe à 1,5 mètres à l'avant du front de taille.

UNTERSUCHUNG ZUR VERHÜTUNG VON GASAUSBRÜCHEN DURCH TRÄNKEN DURCH KURZE BOHRLÖCHER UND DURCH DIE KONTROLLZONE IN DER YANGQUAN GRUBE.

Qi Bingyin, Zhang Fuxi

Der Vortrag behandelt Methoden zur Verhütung von Gasausbrüchen durch Tränken durch kurze Bohrlöcher und durch eine Kontrollzone. Die Untersuchung in dem umfassend mechanisierten Streb in Flöz 3 zeigt, daß das injizierte Wasser das Gas aus der Kohle heraustreibt, wobei es die anfängliche Geschwindigkeit der Gasausströmung herabsetzt und die Spitze des Abbaudruckes weiter in Vertriebsrichtung verlagert. Als Ergebnis zeigte sich, daß die Kontrollzone dem Abbau um 1,5 m voreilt, wenn der Injektionsdruck in den Bohrlöchern größer als 7845 KPa ist, und die Länge der Bohrlöcher 2,5 bis 3 m beträgt.

ИЗУЧЕНИЕ МЕХАНИЗМА ПРЕДОТВРАЩЕНИЯ ВНЕЗАПНЫХ ВЫБРОСОВ ГАЗА ПОСРЕДСТВОМ НАГНЕТАНИЯ ВОДЫ ЧЕРЕЗ НЕГЛУБОКИЕ СКВАЖИНЫ В СТЕНКЕ ЗАБОЯ И ЗОНА ВЛИЯНИЯ ЭТОГО ПРОЦЕССА В УГОЛЬНОЙ ШАХТЕ ЯНГКУАН.

Ки Бингьин и Жанг Фукси

Обсуждается способ предотвращения выбросов газа посредством нагнетания воды через неглубокие скважины в стенке забоя и зона влияния этого процесса. Комплексная механизация забоя угольного пласта #3 показала, что нагнетенная вода удаляет газ из угольных пластов, уменьшает начальную скорость газовой эмиссии из индивидуального пласта и перемещает максимум опорного давления впереди забоя. В результате этого, когда нагнетательное давление в отдельной скважине в стенке забоя достигает 7845 КПа и скважина имеет глубину от 2,5 до 3 метров, зона воздействия достигает полутора метров в длину впереди забоя.

COAL AND GAS OUTBURSTS IN THE OSTRAVA-KARVINA COALFIELD

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ABSTRACT

Describes the two main formations of Ostrava-Karvina Coalfield which is part of the Upper Silesian Basin. Reviews history of outbursts and their causes. Especially frequent highly carbonized friable seams. Role of tectonic deformation is discussed. Methods of prediction and prevention are described. Prevention can be active involving methods of pressure relief (blasting, drilling. water injection, gas drainage) or passive (induces shotfiring in evacuated mine, monitoring and alarming, two-way communication, adequate postdisaster protection).

BRIEF GEOLOGICAL CHARACTERISTICS OF THE OSTRAVA-KARVINA COALFIELD

The Ostrava-Karvina Coalfield is an integral part of the Upper Silesian Hardcoal Basin occupying a total area of approximately 7,000 km^2 and extending largely on the territory of Poland. A smaller part of about 1,600 km^2 area, protruding into the territory of Czechoslovakia, is referred to as the Ostrava-Karvina Coalfield (OKR).

The Upper Carboniferous coal-bearing OKR sediments can be classified into two main formations: the Ostrava Formation and the Karvina Formation.

The Ostrava Formation is stratigraphically older and is developed throughout the coalfield. Its thickness is varied, ranging from 600 to 3,000 m. The coal density is relatively low and is unevenly distributed. The mean thickness of workable coal seams is about 80 cm, the seam development is very irregular, characterized by variable thickness, splitting and balks. Mining operations in the formation are currently performed at depths of 600 to 1,300 m. The seams of Petrkovice Member and Hrusov Member in this formation are characterized by the occurrence of coal and gas outbursts.

The Karvina Formation is younger and continuous coal bearing areas are developed only in the Karvina part of the coalfield. Its average total thickness amounts to 800 m and is characterized by very dense coal with seams of mean thickness of 228 cm.

Seam thickness ranges from 1 m to 8 m. When mining in this formation, the strata are liable to rock burst occurrence and spontaneous ignition of coal.

The structural and tectonic patterns of the Ostrava and Karvina parts in the OKR differ considerably. While the Ostrava part is characterized by steep and overturned folds and complex tectonics, the Karvina part has a typically germanotype pattern.

HISTORY OF COAL AND GAS OUTBURSTS IN THE OKR

The first historical record of an outburst dates back to 1894 at the former Ignat Colliery. Before 1965, outbursts had occurred only at this mine (except for several cases at the existing Vitezny unor Colliery).

In connection with the development of mines in the southern part of the coalfield, outbursts have been occurring at the Paskov Colliery since 1965 and at the Staric Colliery since 1970. In 1975 the Rudy Rijen Colliery was classed as a mine liable to outbursts, due to an outburst occurrence. The total number of outbursts registered in the OKR by the end of 1988 was 468, of which 180 at the Jan Sverma Colliery, 136 at the Paskov Colliery, 132 at the Staric Colliery, 13 at the Vitezny unor Colliery and 7 at the Rudy Rijen Colliery.

Apart from these coal and gas outbursts, five cases of rock and gas outbursts have been recorded at the Staric Colliery, in contact of the sheet formation with Carboniferous.

The most intensive rock and gas outburst recorded so far occurred in 1981 with about 1,200 tons of rock and 100,000 m^3 of methane.

Elsewhere, OKR outbursts accounted for 10 to 100 tons of coal. The greatest coal outburst recorded occurred at the Jan Sverma Colliery on 13 June 1983, where 580 tons of coal were emitted. The quantity of gas released during outbursts ranges from some hundreds to several thousands of cubic meters.

Outbursts of methane prevail, with the exception of the Jan Sverma Colliery where there is an occurrence of mixed outbursts, i.e. CH_4 + CO_2 in a ratio of about 1:2. Outbursts occur in development workings with the exception of shaft sinking, in roadways driven in seams and also in longwall mining. A major part of the outbursts is initiated by blasting during the technological cycle or while taking prevention measures.

NATURAL CONDITIONS OF OUTBURST OCCURRENCE

Analyses indicate that the main factors leading to an outburst occurrence in the OKR include particular sedimentation conditions, the degree of seam carbonization, the intensity and nature of tectonic deformation, and finally nature of gas migration processes.

In the OKR, outbursts recorded have been 345 outbursts in the Petrkovice Member and 123 outbursts in the Hrusov Member.

Certain differences occur even inside the deposit, for instance at the Staric, J. Sverma and Vitezny unor Collieries, a majority of outbursts occurred in the Petrkovice Member seams while at the Paskov Colliery more outbursts occurred in the Hrusov Member.

The largest number of outbursts in the OKR, i.e. 18 cases, has been recorded in the Petrkovice seam No. 090.

Outbursts are recorded especially in seams with a high degree of carbonization which can be characterized by the content of volatile combustible matter in the pure combustible in the range of 14 to 32%, by low strength, increased coal friability, and an increased sorption capacity of coal.

Tectonic deformation is a decisive natural factor of outburst predisposition. Practically in all outburst occurrences, investigation has proved the presence or the vicinity of a tectonic deformation or of seam development irregularities. The impact of tectonic deformation radically reduces coal strength; changes in the coal structure and in sorption properties can be observed. In most cases the extent of tectonic fault is of no great importance, because outbursts occurred even at amplitudes as small as 0.1 m.

Through the influence of migration processes, the gas generated by carbonization is unevenly distributed in the deposit. A pronounced zone with outburst predisposition can be observed in seam planes also in the vertical direction. The increase of such

zones is significantly influenced by the morphology of the carboniferous surface and important tectonic faults.

The contact of the Carboniferous with sheet is generally of an impermeable boundary nature, and tectonic faults function as significant migration paths. Therefore, a decline in the frequency of outbursts can be observed with increasing distance from the sheet.

By investigating the effects of other natural conditions it has been concluded that, for example, seam thickness and seam dip have no immediate influence on outburst occurrence. However, if the predisposition to the outburst is determined by other factors, in the specific area, with increasing seam thickness and seam dip, the likelihood of outburst increases.

The natural and geologic predisposition of the deposit is one of the most important conditions for the occurrence of outbursts. Another important category of factors is represented by mining and technological conditions by which unfavourable natural conditions can to a certain degree, be eliminated, and thus the risk of outburst occurrence can be reduced. Therefore the application of suitable technologies, as for instance working by narrow webs on a long and straight longwall face, plough winning, etc. becomes a part of outburst control.

OUTBURST CONTROL SYSTEM

Outburst control represents an extensive set of activities divided into prognosis and prevention. Prediction can be regional, local or continuous, and prevention can be active or passive.

Regional prediction is based on the assessment of natural conditions inside the mining field and considers the outburst occurrence in tectonically confined areas in individual seams. Based on this evaluation, all the seams at outburst mines are classified by the State Mine Inspection as seams without or with an outburst risk.

Seams with outburst risk are then divided according to the degree of risk into dangerous and very dangerous.

Local prediction is based on a set of prediction measurements with the aim of defining the zones with anomalous gas content, and the areas with an increased tectonic deformation in the immediate surroundings of mine in operation. The result of the prediction is the classification of the mine into the risk categories.

Continuous prediction is carried out continuously during working of the mine. Its objective is the determination of zones with an increased probability of outburst occurrence. This procedure comprises the following basic prognostic parameters:

- Gas pressure in prognostic boreholes of 42 mm diameter which must overlap the planned advance by a minimum of 3 meters. The pressure is measured with a pressure gauge after sealing the prognostic borehole with a special probe (so-called Engler pin).

- Gas desorption rate of a sample of coal cuttings obtained by drilling of prognostic boreholes. It is measured by isochoric or isobaric desorption meters.

- Seam thickness variability rate based on the measurements of the in-situ seam section.

For each of the above mentioned parameters the so-called critical values are determined and on reaching them blasting with an increased seismic effect and more stringent safety measures must be performed (shotfiring with fresh air flow, prolonged waiting time).

Active prevention consists in creating areas of reduced gas

pressure, either in the surroundings of an active mine or over a wider expanse

For this purpose the following methods are applied:

- working of protecting seam
- relief blasting
- relief drilling
- water injection of the coal seams
- underground gas drainage.

Working out the neighbouring protecting seams is considered as a most efficient prevention method because a larger of reduced gas pressure is created, depending on the worked out area in the protecting seam. In view of outburst prevention undermining at a distance of 70 m, or overmining at a maximum distance of 40 m (at dips below 70°) appear to be efficient means. The area of a protected zone is defined by critical angles of efficient effect. There is a standard procedure for the determination of the area extent. Application of this approach is limited due to the complex tectonic and geological conditions found in the OKR.

Relief blasting is applied to create an area of reduced gas pressure in the forefield of a mine working. It consists in shotfiring of a predetermined amount of explosive (usually 0.6 - 1 kg) in boreholes preceding the planned advance by minimum 3 m. Cracks caused by non-production blasting enable a continuous degassing. This method is applied mainly during development of long mine roadways.

Relief drilling consists in drilling boreholes of 85 to 200 mm diameter in the coal seam in such a way that they precede the planned advance by minimum 5 m. The number of relief boreholes depends on seam thickness. It also is applied mainly to long roadway development, but it is not as widely used as relief blasting.

In some cases both methods are applied at the same time in zones of increased outburst risk.

Coal seam injection by pressurized water is the most widely used prevention method for longwall faces. The water injection is usually carried out by means of long boreholes drilled from the entries into the face pillar in advance of the advancing longwall face. Face sections beyond the reach of injection from entries are re-injected perpendicularly from the longwall face. The difficulty in this technique is the control of injection medium passage through coal seam.

Gas drainage ensures elimination of a certain amount of gas during mining operations which cannot then contribute to the outburst occurrence. In the OKR some of the described prevention methods must be used parallelly with gas drainage.

Passive prevention represents a whole range of means to reduce outburst effects and their unfavourable impact on operational safety, as well as on the safety of underground workers. First of all, it includes so-called inducer shotfiring, provoking outbursts at a time when the crew are at a safe distance from the face and electrical equipment is switched off. It is applied in cases where the prognosis is signalling an increasing outburst risk. The inducer effect is achieved by increasing the amount of explosive in overlapping boreholes at the shortest delay interval.

Increased safety measures taken in case of an actual outburst event include the following:

- equipping all face workers with self-rescuer apparatus
- availability of a permanently ready rescue team at every colliery with outburst risk
- monitoring of firedamp content at working points with outburst risk

by means of sensors with automatic power blocking when exceeding the limit values of 1.5% methane content

- availability of free access escapeways and adequate lighting of outburst working points

- installation of a unified alarm system

- availability of reliable two-way communications between underground working points and the control room on the surface.

CONCLUSION

The present level of the outburst control system introduced in the OKR are the achievements of long research work and experience. Although some of the problems have been successfully solved in the past years, particularly increased prognosis reliability, there are still a number of unsolved questions. Teams engaged in science, research and mine operation continue to seek new approaches to secure the maximum safety of personnel and underground operation.

DEGAGEMENTS DE CHARBON ET DE GAZ DANS LA MINE DE CHARBON D'OSTRAVA KARVINA

D. Pavelek, A. Cervenka et G. Takla

Notre rapport examine les deux formations principales de la mine de charbon d'Ostrava Karvina qui fait partie du Bassin de la Haute Silésie. Il donne l'historique et les causes des dégagements soudains; ceux-ci se produisent principalement dans les couches friables à haute teneur en carbone. Nous examinons le rôle de la déformation tectonique et présentons des méthodes de détermination et de prévention. La prévention peut être active; elle emploie alors des méthodes de soulagement de pression (tirages, forages, injection d'eau, drainage des gaz) ou passive (tirages dans les mines évacuées, surveillance et alarme, liaisons bilatérales, protection adéquate après les désastres).

KOHLE- UND GASAUSBRÜCHE IM OSTRAVA KARVINA REVIER

D. Pavelek, A. Cervenka, G. Takla

Die Arbeit beschreibt die beiden Hauptformationen des Ostrava Karvina Reviers, welches zu dem 'Oberschlesischen Becken' gehört, und gibt einen Rückblick auf die Geschichte der Ausbrüche und ihrer Ursachen. Die Rolle der tektonischen Verformung wird diskutiert, und Methoden zur Vorhersage und Verhütung werden vorgestellt. Die Methoden der Verhütung können aktiv sein (Entspannungsschießen, Bohren, Tränken und Gasabsaugung) oder passiv (z.B. Lockerungsschießen in abgebauten Feldern, Überwachung und Alarmierung, wechselseitige Kommunikation und angemessener Schutz nach einem Unglück).

РЕШЕНИЕ ПРОБЛЕМЫ ВНЕЗАПНЫХ ВЫБРОСОВ УГЛЯ И ГАЗА НА ОСТРАВА-КАРВИНСКОМ УГОЛЬНОМ МЕСТОРОЖДЕНИИ

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Описываются две главные геологические формации Острава-Карвинского угольного района, являющегося частью Верхне-Силезского бассейна. Дается обзор истории выбросов и их причин. Выбросы происходили главным образом в хрупких пластах с высоким содержанием углерода. Обсуждается роль тектонических нарушений. Описываются методы прогнозирования и предупреждения. Предупреждение может быть активным, включающим методы снятия давления (взрывы, бурение, впрыскивание воды, откачка газа) или пассивным (стимулирование взрывания шпуров в эвакуированной шахте, наблюдение и аварийная сигнализация, двусторонняя связь, адекватные послеаварийные меры защиты).

奥士屈拉伐卡维那煤矿的煤与气体突出问题

D。帕弗拉克，A。塞尔凡卡 及 G。塔克拉

本文叙述了奥士屈拉伐卡维那煤矿的煤与气体突出两个主要的地层结构。这煤矿是席来西上部盆地的一部份，并且讨论了突出的历史及其原因，特别是常常有高度碳化的松散煤层。地壳的形成亦在文中讨论到。预防及预测的方法亦有所叙述。预防方法有积极的减低压力的方法（如爆破，钻孔，注水，排气，等），及消极的办法（包括在疏空的矿中引爆，监测与报警，双向联络，及适当的善后处理。

NOVEL METHOD OF IMPROVING THE EFFICIENCY OF OUTBURST ALLEVIATING PROTECTION OF COAL SEAMS

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The most reliable method of alleviating sudden outbursts of coal, rock and gas is advance in space and time mining of adjacent protective seams resulting in a change of the stressed/strained and gas-dynamic conditions of hazardous coal seams (pressure and gas content of coal decrease, while gas permeability and emission rate from the seam increase). However, the efficiency of advance mining of a protective seam depends on the distance between a protective seam and a hazardous seam under protection, and with increase of this distance the efficiency of protection decreases. In the majority of cases, according to geological conditions thickness of the interseam strata separating the protective and hazardous seams exceeds the safe distance established by practice for employing reliable protection. Hence, a novel method developed by the authors and tested on a pilot basis for artificical increase of the protective zone in advance mining of protective seams is of particular importance.

In recent years in the USSR top priority is given to the problem of forecasting and alleviating sudden outbursts of coal and gas at coal mines and reducing the number of coal and methane outbursts provoked by blasting (mainly by shock blasting) due to greater mining depths and growth of natural methane content of coal to 30-45 m^3/t. By 1989, maximum mining depth reached 1325 m in the Donbass (at 40 mines mining depth exceeded 1,000 m), 750 m in the Karaganda basin, 1020 m in the Vorkuta basin and 920 m in the Far East.

Over the 1980-1988 period the number of the USSR mines working outburst-hazardous coal seams increased from 139 to 162 (Table 1), while the total number of development headings and coal faces in outburst-prone seams increased from 800 to 1515. During the same period depth of mining gas-bearing seams increased on the average by 100-110 m. Coal output from outburst-hazardous seams expanded from 116 mln. t in 1980 to 141 mln. t in 1987, this resulted in greater hazard and number of dynamic emissions of mine gases. For example, at the Donbass mines featuring the greatest outburst hazard (93-95 % of all outbursts in the country occur there) every year 270 to 400 gas-dynamic phenomena are recorded, the major part of which resulting from blasting operations, i.e. at the presence of personnel. However, with rapid reduction of specific rate of sudden outbursts (per million tons of coal output) - from 23-56 in 1950-1955 to 0.3-0.6 in 1985-1988, the total number of sudden outbursts remains at the level of 16-20 occurrences per year, despite the tendency towards decrease (Table 2). Besides, 25-57 outbursts occur in machine remote control. Greater outburst hazard can be explained by rapid increase of the number of working seams-prone to outbursts - from 512 to 1063 over the recent decade. In the last 5-8 years increased number of outbursts during shock blasting was recorded (by 15 %) and in remote control of coal winning and drilling machines (by 30 %).

In 1946-1988 a total of 9757 gas-dynamic phenomena were recorded at the USSR coal mines, including 7225 (74 %) outbursts of coal, rock and gas during blasting operations. Over the same period at the Donbass mines a total of 8309 gas-dynamic phenomena occurred including 4689 coal and methane outbursts of which 1987 (42.3 %) were coal and methane outbursts proper and 3620 were outbursts of sandstone and methane (in blasting only). The USSR major sudden outbursts occurred at the Donbass mines (Table 3).

Coal seam hazard, in terms of sudden outbursts leading to the necessity of employing special methods of outburst forecast and prevention, deteriorates mine technical and economic indices. For instance, average daily output per coal face is 10 to 30 % lower in outburst-prone seams, OMS is 15 to 20 % lower, while coal

Table 1

Number of the USSR mines working seams prone to sudden outbursts of coal and gas

Coal basin	Year								
	1980	1981	1982	1983	1984	1985	1986	1987	1988
Donets	96	96	97	98	98	99	98	99	104
Kuznetsk	14	15	15	17	19	20	21	22	23
Karaganda	18	19	19	19	19	19	20	21	21
Pechora	7	7	7	7	7	7	7	7	7
Other*	4	4	4	4	5	6	6	6	7
T o t a l	139	141	142	145	148	151	152	155	162

* Note: Coal deposits in the Far East, Urals, Sakhaline Island

Table 2

Number of sudden outbursts of coal and gas at mines in the USSR coal basins in 1946-1988

Period, years	Number of sudden outbursts	including coal basins				
		Donets	Kuznetsk	Karaganda	Pechora	Deposits in the Urals, Far East, Sakhaline Island
1946-1950	231	132	26	-	2	42
1951-1955	450	272	27	-	17	134
1956-1960	274	219	23	5	3	24
1961-1965	417	320	11	14	67	5
1966-1970	392	330	19	10	17	16
1971-1975	252	223	12	8	6	3
1976-1980	149	131	8	4	6	-
1981-1985	240	227	1	4	8	-
1986-1988	107	102	5	-	1	-
T o t a l	2512	1985	132	45	127	224

Table 3

Major outbursts of coal and methane at the mines in the Donets basin

Depth, m	Year	Outburst power	
		volume of methane released, m^3	amount of coal released, t
865	1961	90,000	4,370
710	1969	600,000	14,500
740	1973	70,000	4,000
980	1975	72,000	4,650
750	1981	60,000	3,000
815	1981	170,000	1,020

production costs are 15 % higher than in non-hazardous seams.

At the USSR mines use is made of current local and regional methods of forecast. Outburst forecast is carried out in 170 to 190 development headings, including forecast in terms of the seam seismoacoustic activity in 105 to 120 headings and forecast in terms of initial gas emission rate in 50 to 80 headings. At coal faces use is made of the seismoacoustic method of forecast (19-20%), as well as forecast in terms of gas emission rate (14-15 % of faces).

To alleviate sudden outbursts use is made of shock blasting (30 % of all development headings and 10 % of coal faces and local techniques - relief/draining boreholes, slots and grooves, hydroloosening and hydrospalling, torpedoing, water-infusion into hazardous coal seams, etc. Local techniques are employed at 170-180 development headings and coal faces in hazardous seams.

Outburst-alleviating regional techniques involve preliminary gas drainage of hazardous seams through boreholes (25-35 coal faces in the Karaganda basin and Kuzbass, 5-7 faces in the Donbass) and advance mining of protective seams. At the Donbass mines the scope of protection has been steadily growing , the number of working outburst-prone seams increases at the same time (Table 4).

Increase of the number of

Table 4

Indice	Period			
	on 01.01.75	on 01.01.80	on 01.01.86	on 01.01.88
Number of outburst hazardous and prone seams	251	264	273	291
Number of coal faces in outburst-hazardous	1049	1040	1063	1078
number of non-hazardous seams mined with outburst forecast	253	242	237	233
number of coal faces in outburst-prone seams working under full protection	135	224	246	249

protected seams and faces is a promising trend in the Central region of the Donbass where mines work contiguous thin (0.5 to 1.6 m thick) steep (60-70°) coal seams. At the mines in this region only 30 to 33 % of outburst-prone seams lie at the distances from protective seams that allow for advance protection of the seam. At the mines in this region 140 to 145 seams are outburst-hazardous, the total number of seams is 260-270, and only 110 to 150 coal faces of the total number of faces in outburst-prone seams (160 to 170 faces) operate under protection.

It has been established previously /1-3/ that during under- and overworking of flat (Fig. 1a) and steep (Fig. 1b) coal seams their gas-dynamic conditions undergo changes and, first of all, gas pressure reduces to residual characterized by aerodynamic resistance of the interseam strata to gas flow from a hazardous seam to mine workings. Later on, this pronciple was proved in Poland /4/, China /5, 6/ and West Germany /7/.

It has been established that gas drainage boreholes drilled down to under- or overworked seams and intersecting

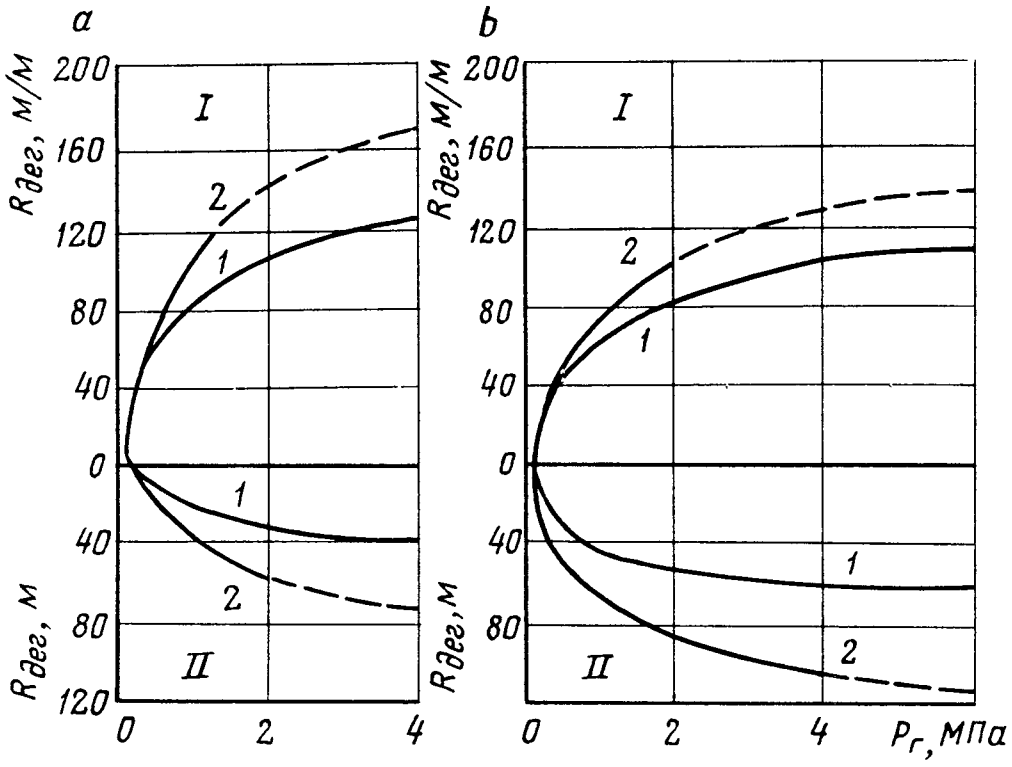


Fig. 1. Dependence of residual gas pressure in under- and overworked flat (a) and steep (b) coal seams on the radius of gas drainage.

the under- or overworked seam at the distance of 10 to 15 m from the point of gas pressure measurement lead to greater gas pressure reduction than without boreholes. Such a change is particularily effective in under- or overworking of seams lying at greater distances from the protective seam.

Studies of the borehole gas drainage effects on the remote seam being gas drained /8/ enabled to establish that under conditions of minor disturbances of the interseam strata by fissures due to mining (in case of very thick interseam strata) the gas draining effect of boreholes grows, whereas at greater distances from the gas drainage borehole its gas draining role diminishes (Fig. 2). With an increase of thickness of the interseam strata, the effect of gas drainage on the methane pressure reduction in the overworked seam increases (Fig. 3). By comparing the results of determining gas pressure changes in overworked steep coal seams with

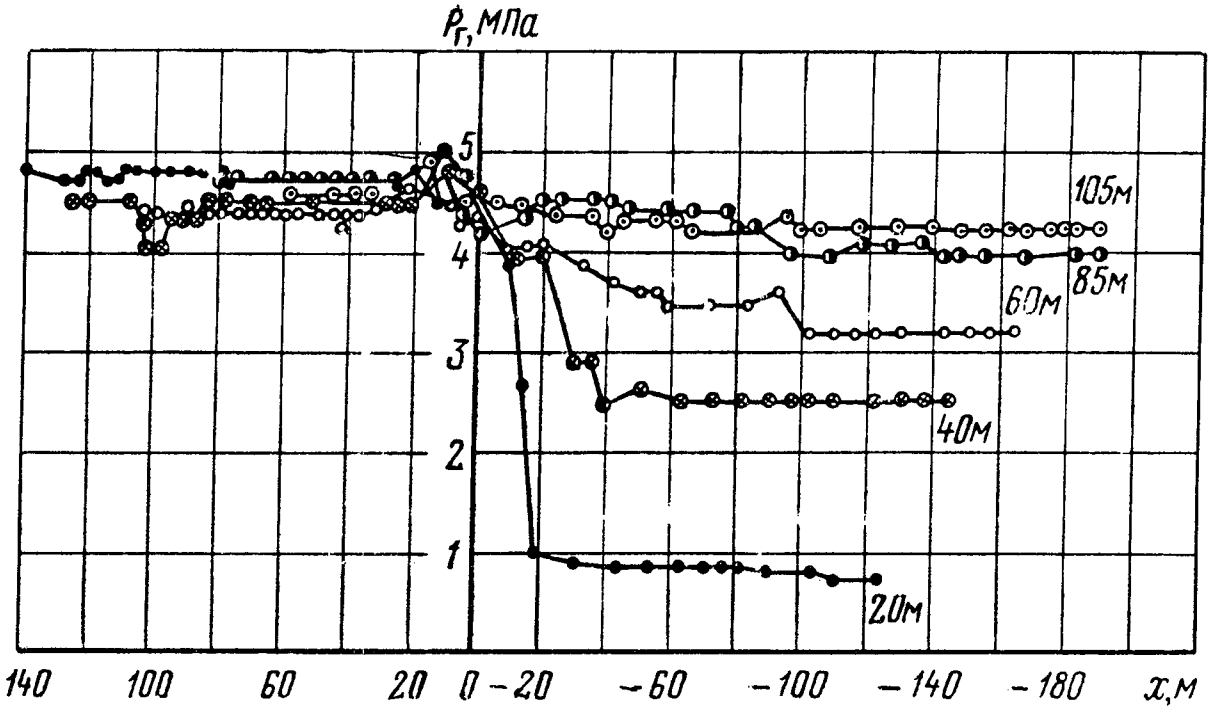


Fig. 2. Gas pressure pattern in artificial gas drainage of a steep coal seam depending on the distance to the coal face (Donbass, Komsomolets Mine, 740/850 level, interseam strata thickness is 60 m)

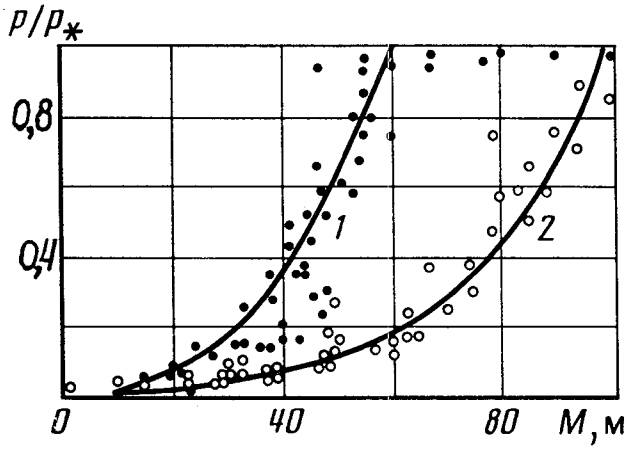


Fig. 3. Relative gas pressure distribution in underworking steep outburst-prone seams depending on interseam thickness (M),m

the statistic data on outbursts occurred at 23 mines in the region over the 24-year period, it has been established that methane pressure distribution and pressure reduction in overworked seams, depending on the interseam strata thickness and distance in the seam to a gas drainage borehole, are one of the principal criteria of outburst hazard. Results of comparative analysis showed that overworking of steep outburst-prone seams (Fig. 4, 5) at interseam strata thickness of

10 to 25 m use of artificial gas drainage of a hazardous

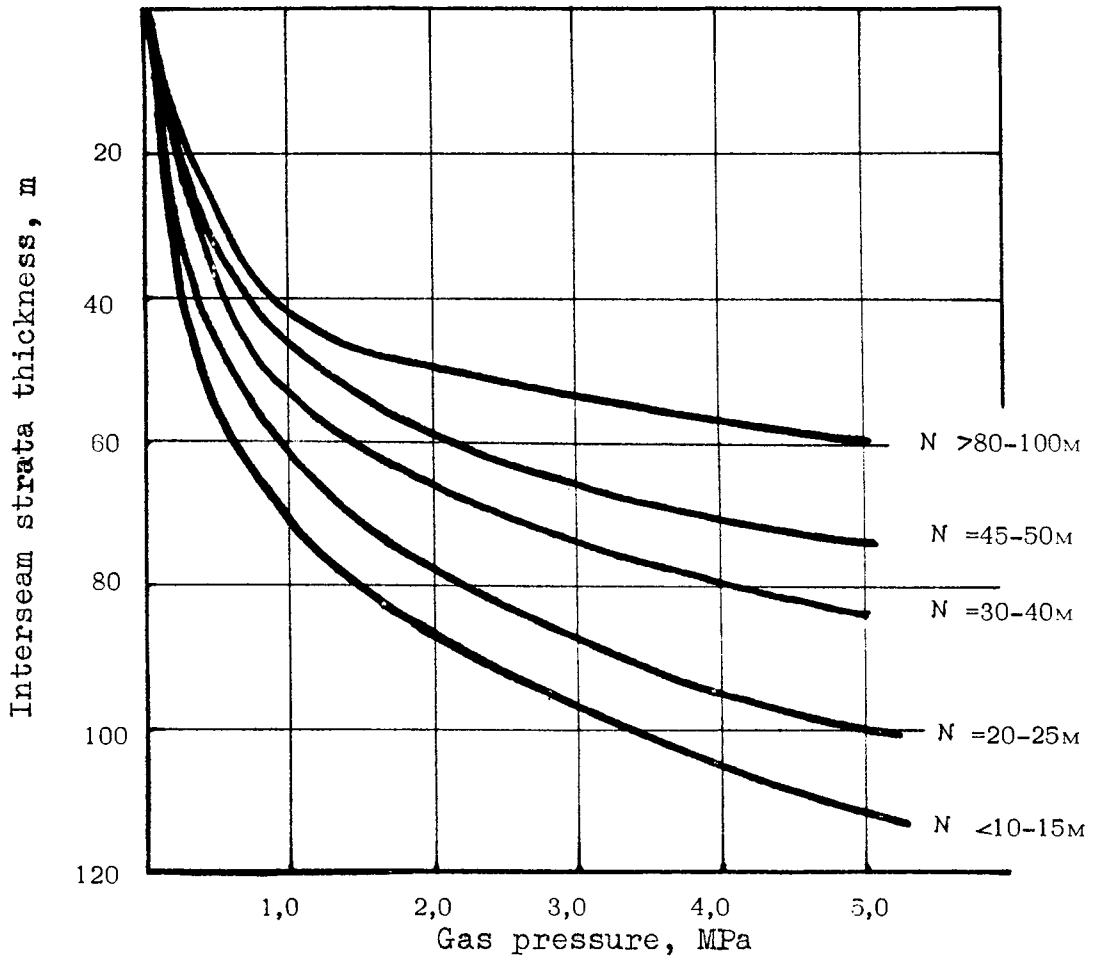


Fig. 4. Residual gas pressure in steep overworked seams in artificial gas drainage through boreholes drilled at different intervals (N).

seam to prevent sudden outbursts is not necessary;

25 to 40 m gas drainage of a hazardous seam is necessary only in unfavourable conditions (inclusion of thick layer of hard rocks in the interseam strata, low thickness and a small advance of protective seam mining) and for accelerating the effect of the protection action;

40 to 50 m gas drainage of an overworked outburst-prone seam through boreholes is necessary for improving reliability and accelerating the effect of the protection action;

50 to 80 m artificial gas drainage of the seam is necessary to prevent sudden outbursts in overworking steep hazardous seams with borehole spacing not exceeding 20-25 m along the seam strike and the level height.

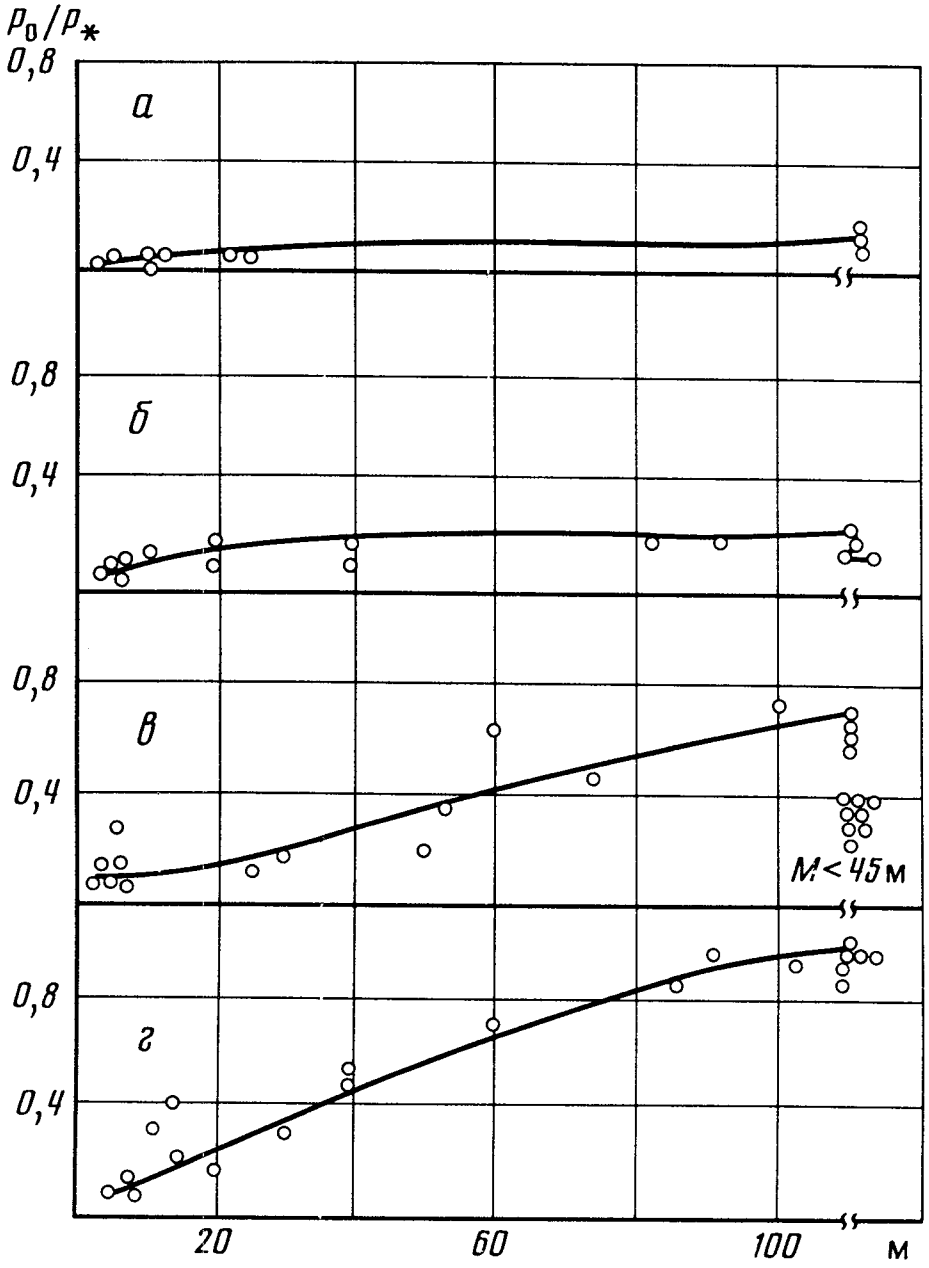


Fig. 5. Relative gas distribution in the bedding plane of the overworked seam depending on the distance to the gas drainage borehole.

CONCLUSIONS

1. Studies of gas pressure pattern in steep overworked outburst-prone coal seams and underground pilot tests have proved that protective overworking in combination with artificial gas drainage of hazardous seams allows for the sizes of the effective protection zone to be enlarged 1.4-1.6-fold depending on the interseam strata thickness and lithological composition.

2. Relative methane pressure determined by the relationship of natural vs. residual methane pressure in an outburst hazardous seam is a reliable criterion for evaluating the effectiveness of protective overworking of steep seams employed in combination with artificial gas drainage of the outburst-prone seam through boreholes.

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НОВЫЙ МЕТОД УВЕЛИЧЕНИЯ ЭФФЕКТИВНОСТИ ПРОТИВОВЫБРОСНОЙ ЗАЩИТЫ УГОЛЬНЫХ ПЛАСТОВ

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Наиболее надежным методом предотвращения внезапных выбросов угля, породы и газа является опережающая в пространстве и во времени выемка соседних защитных пластов, в результате которой происходит изменение напряженно-деформируемого и газодинамического (снижается давление и содержание газа в угле, повышается его газопроницаемость и скорость газоотдачи) состояний опасных угольных пластов. Однако эффективность опережающей выемки защитного пласта зависит от расстояния между защитным и защищаемым опасным пластами и по мере его увеличения снижается эффективность защиты. По геологическим условиям в большинстве случаев мощность породного междупластья, разделяющего защитный и опасный пласты, превышает установленное опытом безопасное расстояние для применения надежной защиты. Поэтому важное значение приобретает созданный и прошедший опытно-промышленную проверку метод искусственного увеличения зоны защитного действия при опережающей выемке защитных пластов.

В последние годы проблема прогноза и предотвращения внезапных выбросов угля и газа в угольных шахтах и сокращение числа спровоцированных взрывными работами (в основном сотрясательным взрыванием) выбросов угля и метана приобрела в СССР особенно большое значение в связи с быстрым углублением горных работ и ростом природного содержания метана в угле до 30-45 м³/т. Так к началу 1989 г. максимальная глубина разработки угольных пластов достигла: в Донбассе 1329 м (в 40 шахтах глубина работ превысила 1000 м), в Караганде-750 м, в Воркуте-1020 м и на Дальнем Востоке - 920 м.

Число шахт СССР, разрабатывающих выбросоопасные угольные пласты,за период 1980-1988 г.г.увеличилось со 139 до 162 (табл.1), а общее число подгото-

Таблица 1

Число угольных шахт СССР, разрабатывающих опасные по внезапным выбросам угля и газа пласты

Угольный бассейн	Год								
	1980	1981	1982	1983	1984	1985	1986	1987	1988
Донецкий	96	96	97	98	98	99	98	99	104
Кузнецкий	14	15	15	17	19	20	21	22	23
Караганда	18	19	19	19	19	19	20	21	21
Печорский	7	7	7	7	7	7	7	7	7
Остальные*)	4	4	4	4	5	6	6	6	7
Итого:	139	141	142	145	148	151	152	155	162

*) Примечание: угольные месторождения Дальнего Востока, Урала, о-ва Сахалин

вительных и очистных забоев на выбросоопасных пластах увеличилось с 890 до 1515. За этот период рост глубины разработки газоносных пластов в среднем составил 100-110 м. Увеличение количества добываемого из выбросоопасных пластов угля - со 116 млн.т в 1980 г. до 141 млн.т в 1987 г. способствовало повышению опасности и росту числа динамических форм выделений рудничных газов. Так на шахтах Донбасса, которые характеризуются наибольшей выбросоопасностью (происходит 93-95% всех внезапных выбросов в шахтах СССР) ежегодно регистрируется 270-400 газодинамических явлений, основная часть которых происходит в результате взрывных работ, то есть при отсутствии людей. Однако при быстром сокращении удельной частоты внезапных выбросов (на 1 млн.т добытого угля) с 23-56 в 1950-1955 гг. до 0,3-0,6 в 1985-1988 гг.общее число внезапных выбросов, несмотря на тенденцию к уменьшению, остается на уровне 16-20 случаев в год (табл.2). Кроме того, происходит 25-57 выбросов при дистанционном управлении машинами. Рост выбросоопасности объясняется быстрым увеличением числа разрабатываемых опасных шахтопластов с 512 до 1063 за последние 10 лет. В последние 5-6 лет отмечено увеличение числа выбросов при сотрясательном взрывании (на 15%) и при дистанционном управлении выемочными и буровыми машинами (на 30%).

В 1946-1988 гг. на угольных шахтах СССР было зарегистрировано 9757 газодинамических явлений, в том числе 7225 (74%) выбросов угля,породы и газа при взрывных работах. В этот же период на шахтах Донбасса из общего числа газодинамических явлений 8309 произошло 4689 выбросов угля и метана, в том числе 1987 (42,3%) собственно внезапных выбросов угля и метана и 3620 выбросов песчаника и метана (только при взрывных работах). Крупнейшие в СССР внезапные выбросы произошли на шахтах Донбасса (табл.3).

Опасность угольных пластов по внезапным выбросам, вызывающая необходимость применения специальных методов прогноза и предотвращения выбросов, ухудшает технико-экономические показатели работы шахт. Так на выбросоопасных пластах средняя нагрузка на очистной забой на 10-30% ниже, производительность труда на 15-20% ниже, а себестоимость добычи угля на 15% выше, чем на неопасных пластах.

На шахтах СССР применяется текущий локальный и региональный методы прогноза. Прогноз выбросоопасности осуществляется в 170-190 подготовительных выработках, в том числе по сейсмоакустической активности пластов в 105-120 забоях и по начальной скорости их газоотдачи в 50-80 забоях. В очистных забоях применяется сейсмоакустический метод прогноза (19-20%) и по начальной скорос-

Таблица 2

Число внезапных выбросов угля и газа в шахтах угольных бассейнов СССР за период 1946-1988 гг.

Период, годы	Число внезапных выбросов	в том числе в угольном бассейне				
		Донецком	Кузнецком	Карагандинском	Печорском	на месторождениях Урала, Дальнего Востока и о-ва Сахалин
1946-1950	231	131	26	-	2	42
1951-1955	450	272	27	-	17	134
1956-1960	274	219	23	5	3	24
1961-1965	417	320	11	14	67	5
1966-1970	392	330	19	10	17	16
1971-1975	252	223	12	8	6	3
1976-1980	149	131	8	4	6	-
1981-1985	240	227	1	4	8	-
1986-1988	107	102	5	-	1	-
Итого:	2512	1985	132	45	127	224

Таблица 3

Крупнейшие выбросы угля и метана на шахтах Донецкого бассейна СССР

Глубина, м	Год	сила выброса	
		объем выброшенного метана, $м^3$	объем выброшенного угля, т
865	1961	90.000	4.370
710	1969	600.000	14.500
740	1973	70.000	4.000
980	1975	72.000	4.650
750	1981	60.000	3.000
815	1981	170.000	1.020

ти газоотдачи (14-15% забоев).

Для предотвращения внезапных выбросов применяется сотрясательное взрывание (30% всех подготовительных и 10% очистных забоев) и локальные мероприятия - разгружающе-дегазирующие скважины, щели и пазы, гидрорыхление и гидроотжим, торпедирование, увлажнение опасного угольного массива и т.п. Локальные мероприятия применяются в 170--180 подготовительных и очистных забоев на опасных пластах.

В качестве противовыбросных региональных мероприятий применяется предварительная дегазация опасных пластов скважинами (25-35 очистных забоев в Караганде и Кузбассе и 5-7 забоев в Донбассе) и опережающая отработка защитных пластов. На шахтах Донбасса объем применения защиты непрерывно растет при одновременном росте числа разрабатываемых выбросоопасных шахтопластов (табл.4).

Таблица 4

Показатель	Период			
	на 01.01.75	на 01.01.80	на 01.01.86	на 01.01.88
выбросоопасных и угрожаемых шахтопластов:	251	264	273	291
очистных забоев на выбросоопасных пластах:	1049	1040	1063	1078
неопасных шахтопластов, разрабатываемых с прогнозом выбросоопасности:	253	242	237	233
очистных забоев на выбросоопасных пластах, работающих под полной защитой:	135	224	246	249

Увеличение числа защищенных пластов и забоев имеет перспективу в Центральном районе Донбасса, где 23 глубокие (800-1200 м) шахты разрабатывают свиты сближенных тонких (0,5-1,6 м) крутых (60-70°) угольных пластов. В шахтах района только 30-33% выбросоопасных пластов залегают от защитных на расстояниях, при которых опережающая выемка защитного пласта обеспечивает защиту. Так из 260-270 разрабатываемых в шахтах района шахтопластов 140-145 являются выбросоопасными, причем из 160-170 очистных забоев на выбросоопасных пластах только 110 - 150 работают под полной защитой.

Ранее было установлено /1-3/, что при подработке и надработке пологих

(рис.1[а]) и крутых (рис.1[б]) угольных пластов изменяется их газодинамическое состояние, и в первую очередь, снижается величина газового давления до остаточного, характеризуемого аэродинамическим сопротивлением междупластовой породной толщи движению газа из опасного пласта в выработки защитного. Позднее эта закономерность была подтверждена в ПНР /4/,КНР /5, 6/,ФРГ /7/.

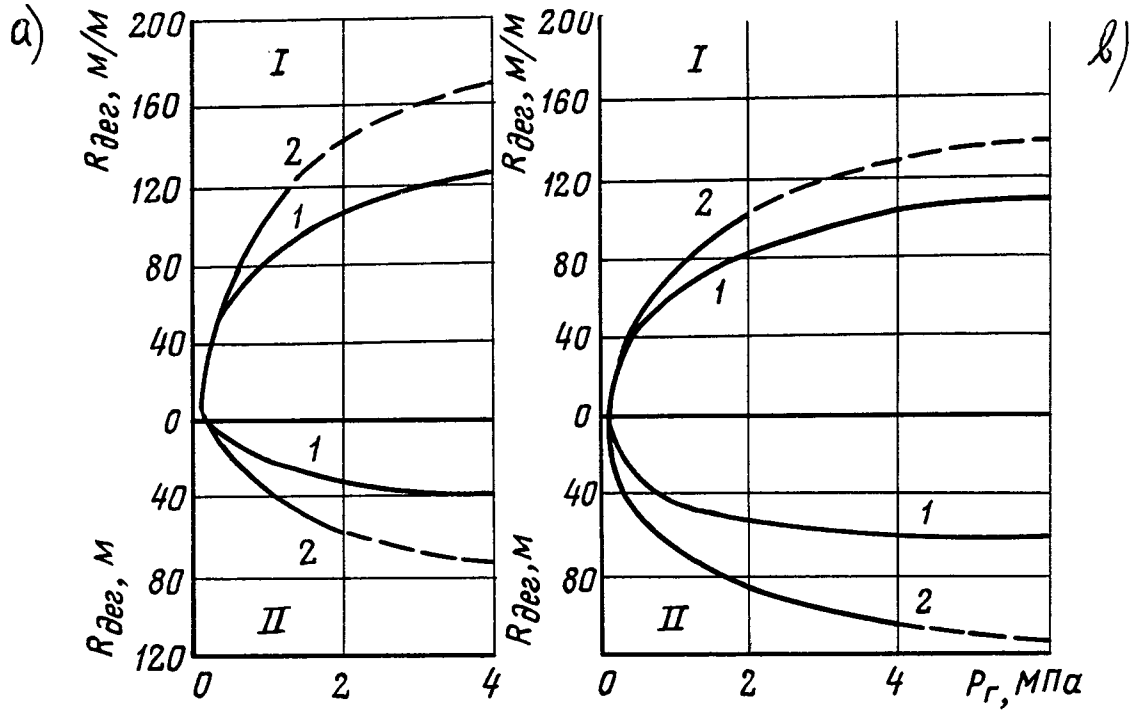


Рис.1 Зависимость величины остаточного давления газа в под- и надработанных пологих (а) и крутых (б) угольных пластах от радиуса дегазации.

Установлено, что дегазационные скважины,пробуренные до под- и надрабатываемого пласта и пересекающие под- или надрабатываемый пласт в 10-15 м от места замера давления газа, обуславливают снижение давления газа в пласте на большую величину, чем без применения скважин.Особенно это изменение сказывается при под- и надработке пластов, удаленных от разрабатываемого защитного пласта.

Исследованиями дегазирующего влияния скважины по удаленному дегазируемому пласту установлено /8/,что в условиях незначительного нарушения междупластовой породной толщи эксплуатационными трещинами (при большой мощности междупластья) дегазирующая роль скважин растет, в то время как по мере удаления по пласту от дегазационной скважины ее дегазирующая роль уменьшается (рис.2). При увеличении мощности междупластья, влияние дегазации на снижение давления метана надработанного пласта увеличивается (рис.3). Сопоставление результатов определения динамики газового давления в надрабатываемых крутых угольных пластах со статистическими данными о происшедших на 23 шахтах района за 24-летний период внезапных выбросов установлено, что распределение давления метана и величина снижения давления в надрабатываемых пластах, зависящие от мощности разделяющей пласты междупластовой породной толщи и расстояния по пласту до дегазационной скважины, является одним из основных критериев выбросоопасности. По результатам сопоставления выявлено, что надработка крутых выбросоопасных пластов (рис.4, 5) при:

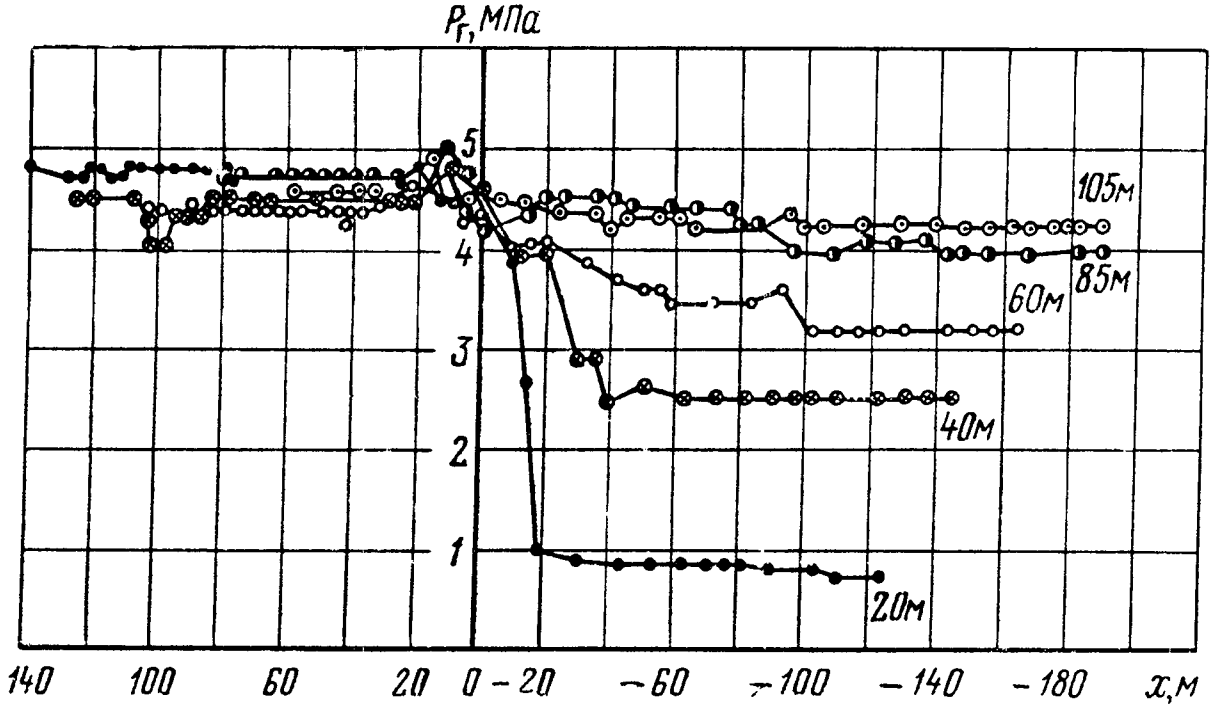


Рис.2 Динамика давления газа при искусственной дегазации крутого угольного пласта в зависимости от расстояния до очистного забоя (Донбасс, шахта "Комсомолец", горизонт 740/850, мощность междупластья 60 м)

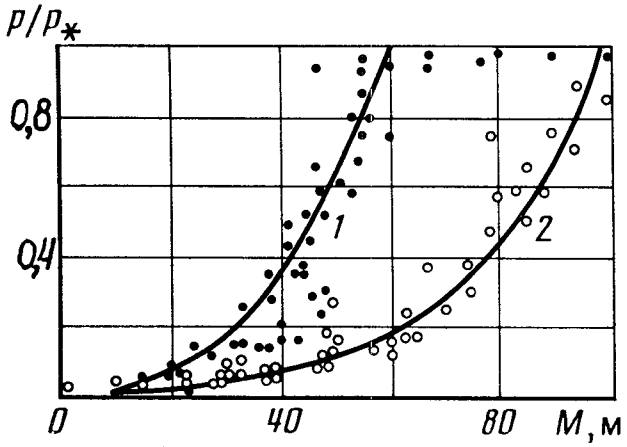


Рис.3 Распределение относительного давления газа при надработке крутых выбросоопасных пластов в зависимости от мощности междупластья (М),м

-мощности междупластья 10-25 м применения искусственной дегазации опасного пласта для предотвращения внезапных выбросов не требуется;

- мощности междупластья 25-40 м дегазация опасного пласта необходима лишь в неблагоприятных условиях (залегания в составе междупластовой породной толщи мощных прочных пород, малая мощность и небольшое опережение выемки защитного пласта) и для ускорения эффекта защитного действия;

- мощности междупластья 40-50 м дегазация надрабатываемого выбросоопасного пласта скважинами для повышения надежности и ускорения эффекта защитного действия;

- мощности междупластья 50-80 м для предотвращения внезапных выбросов при надработке крутых опасных пластов необходимо применять искусственную дегазацию пласта с расстояниями между скважинами по простиранию пластов и по высоте этажа не свыше 20-25 м.

В Ы В О Д Ы

1. Исследованиями газовой динамики крутых надрабатываемых выбросоопасных угольных пластов и опытно-промышленной проверкой в шахтах доказано, что защитная надработка в комплексе с ис-

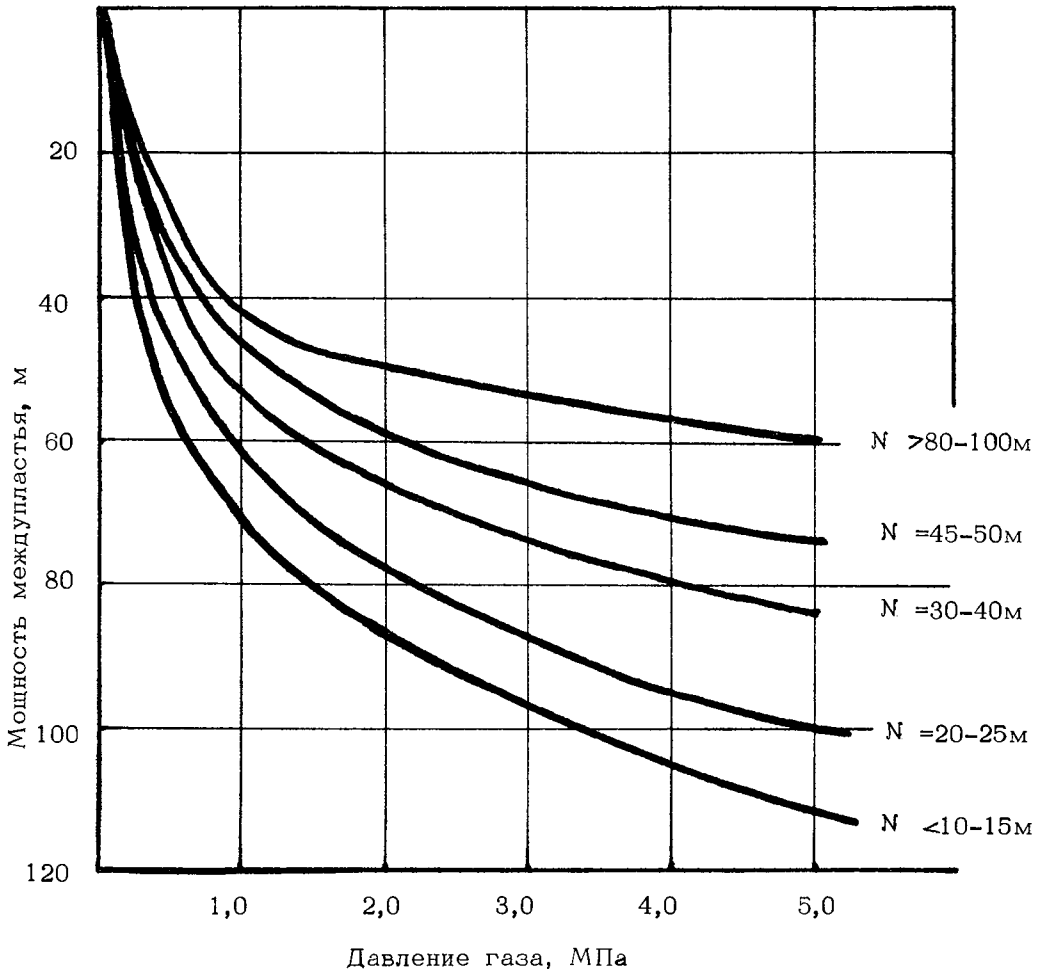


Рис.4 Величина остаточного давления газа в крутых надработанных пластах при искусственной дегазации скважинами, пробуренными с различными интервалами (N)

кусственной дегазацией опасного пласта увеличивает в зависимости от величины междупластовой породной толщи и ее литологического состава в 1,4-1,6 раза размеры зоны эффективной защиты.

2. Надежным критерием оценки эффективности защитной надработки крутых пластов, применяемой в комплексе с искусственной дегазацией выбросоопасного пласта скважинами, является величина относительного давления метана, определяемая соотношением природного к остаточному давлению метана в опасном пласте.

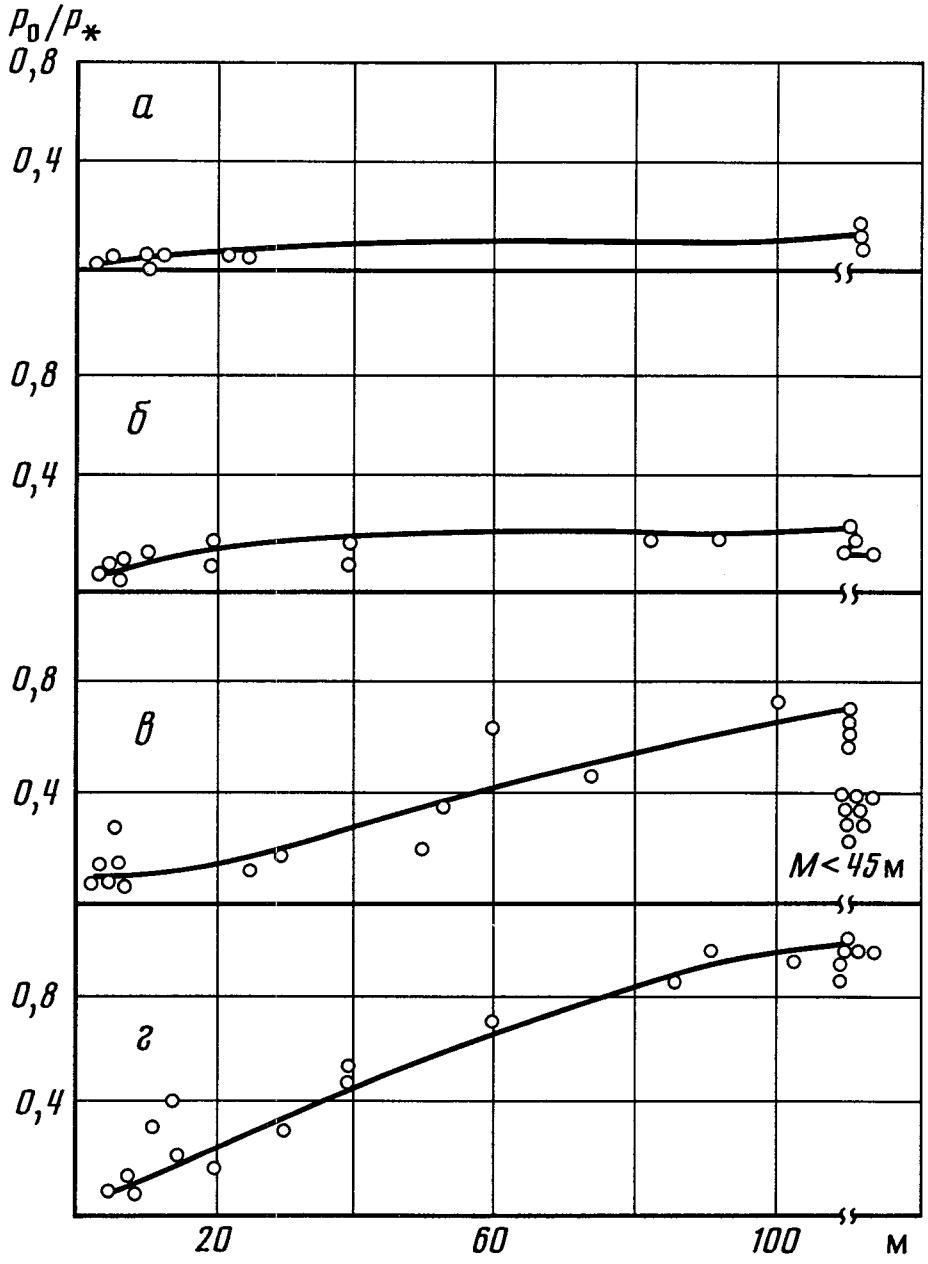


Рис.5 Распределение относительного давления газа в плоскости напластования надрабатываемого пласта в зависимости от расстояния от дегазационной скважины

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NOUVELLE METHODE VISANT A AMELIORER L'EFFICACITE DES PROTECTIONS CONTRE LES DEGAGEMENTS DANS LES GISEMENTS DE CHARBON

Y.P. Smorchkov, A.T. Airuni et L.M. Zenkovich

Pour atténuer les dégagements soudains de charbon, de roches et de gaz, la méthode la plus fiable consiste à avancer, dans l'espace et dans le temps, dans les couches protectives adjacentes, ce qui produit des modifications des états de contraintes/déformations et des états gazeux/dynamiques des couches de charbon sujettes aux risques (il y a une diminution de la pression et de la teneur en gaz du charbon en même temps qu'une augmentation de la perméabilité aux gaz et du taux d'émission des couches de charbon). Toutefois, l'efficacité de l'avancement minier dans une couche protectrice est fonction de la distance entre une couche protectrice et une couche sujette aux risques, mais protégée; à mesure que cette distance augmente, l'efficacité de la protection diminue. Dans la majorité des cas, selon les conditions géologiques, l'épaisseur des strates séparant les couches protectrices des couches à risque est supérieure à la distance considérée comme sûre pour l'emploi de protections fiables. C'est pourquoi une nouvelle méthode, d'une importance particulière, a été développée par les auteurs; elle est utilisée pour effectuer des essais pilotes afin d'augmenter artificiellement la zone protégée des couches protectrices des avancées minières.

NEUE METHODEN ZUR VERBESSERUNG DER EFFEKTIVITÄT DES SCHUTZES GEGEN ENTSPANNUNGSAUSBRÜCHE IN KOHLEFLÖZEN

Y. P. Smorchkov, A. T. Airuni, L. M. Zenkovich

Die zuverlässigste Methode der Abschwächung von plötzlichen Kohle-, Stein- und Gasausbrüchen ist ein räumlicher und zeitlicher Vorbau in angrenzenden Schutzflözen, welcher dazu führt, daß sich die Spannungen in den gefährdeten Flözen abbauen, da sich der Gasdruck und Gasgehalt verringert, während die Gasdurchlässigkeit und Ausgasungsrate ansteigen. Allerdings verringert sich die Effektivität dieser Methode mit steigender Entfernung zwischen dem zu schützenden Flöz und dem zum Schutz abgebauten Flöz. In den meisten Fällen übersteigt die Entfernung zwischen dem zu schützenden Flöz und dem zum Schutz abgebauten Flöz den in der Praxis bestimmten Wert, da die Gesteinsschichten zwischen den Flözen zu mächtig sind. Daher wurde eine neuartige Methode von den Autoren entwickelt und in einem Pilotprojekt getestet. Sie basiert auf einer künstlichen Vergrößerung der Schutzzonen im Vorbau.

减轻煤层突出办法的效率改进新办法

Y。P。司莫尔契可夫， A。T。阿鲁尼 及L。M。蝉可维奇

减轻煤岩石和气体突出最可靠的办法是在邻近保护煤层前进开采，使有危害的煤层改变它应力应变及气体动力状态，（煤碳内压力及气体含量减低，而煤层内气体渗透率及发射率增高）。但保护煤层的前进开采效率与保护煤层及危害煤层之间的距离有关。故这距离愈增加则保护的效率渐渐减少。在大多数的情形下，从地质的观点而言保护煤层及危害煤层之间的厚度超过实行保护措施所订的安全距离。因此作者们发展的新方法经过试验增加保护区域的距离，很有价值及其重要性。

Session VIII

OUTBURSTS OF ROCK AND COAL, II

АВТОМАТИЗИРОВАННЫЕ СИСТЕМЫ ПРОГНОЗА И КОНТРОЛЯ ЭФФЕКТИВНОСТИ ВЫПОЛНЕНИЯ МЕРОПРИЯТИЙ ПО ПРЕДОТВРАЩЕНИЮ ВНЕЗАПНЫХ ВЫБРОСОВ УГЛЯ И ГАЗА

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С переходом горных работ на более глубокие горизонты усиливается вредное влияние горного давления, увеличивается газоносность угольных пластов, и, как следствие, повышается опасность возникновения внезапных выбросов угля и газа. Отработка выбросоопасных пластов требует постоянного применения противовыбросных мероприятий, что приводит к материальным затратам, снижает темпы проведения горных выработок и нагрузку на очистной забой.

Известно, что выбросоопасность имеет локальный характер, т.е. на выбросоопасном пласте можно выделить опасные и неопасные зоны, для определения которых применяются специально разработанные методы прогноза. Использование методов прогноза позволяет отменить противовыбросные мероприятия в неопасных зонах и применять их только при входе выработки в выбросоопасные зоны, которые занимают по площади, как правило, 4-6% шахтного поля. Однако на отдельных, особо опасных по выбросам шахтопластах, они могут составлять 30% и более шахтного поля.

Макеевский научно-исследовательский институт по безопасности работ в горной промышленности еще в 1951 г. начал работы по созданию способа прогноза зон, опасных по выбросам угля и газа, основанного на изучении и анализе сейсмоакустической активности пласта. В дальнейшем сейсмоакустический прогноз динамических явлений был усовершенствован совместно с Институтом горного дела им. А.А.Скочинского и Донецким угольным институтом и применяется более чем в 150 очистных и подготовительных забоях. Метод основан на наблюдениях за шумностью угольного массива с помощью специальной сейсмоакустической аппаратуры типа ЗУА. Сейсмоакустические импульсы являются результатом локального разрушения угольного массива.

Интенсивность трещинообразования в массиве достаточно надежно характеризуется статистическим средним значением шумности, вычисленным на 30-часовом интервале усреднения. Постоянство этого значения свидетельствует об устойчивом режиме трещинообразования в угольном пласте. Монотонное снижение средней шумности указывает на упрочнение призабойной части массива или на снижение действующих в ней напряжений. Снижение прочности угля или повышение напряжений, в результате увеличения горного давления или давления газа, вызывает рост трещинообразования и повышение сейсмоакустической активности.

Исследованиями установлено, что прогностическим признаком входа забоя в опасную по динамическим явлениям зону является монотонное повышение средней шумности не менее чем за двое суток подряд или появление хотя бы одного значения часовой шумности, превышающего среднее ее значение в 4 ра-

за и более. В 97% случаев прогноз дается не менее чем за I м по простиранию до входа забоя в опасную зону; в 3% случаев динамическое явление может возникать при отработке первого же метра пласта.
Длина опасных зон для разных шахтопластов находится в пределах от 6-I0 м до I50-300 м по простиранию пласта.
Сейсмоакустический метод оказался довольно чувствительным методом исследования изменяющегося поля напряжений горного массива. Его применение на шахтах Центрального района Донбасса позволило в 3 раза сократить объем применения способов предотвращения выбросов и уменьшить за последние I0 лет количество газодинамических явлений в 4 раза.
Дальнейшее развитие этого метода прогноза идет по пути автоматизации распознавания сейсмоакустических импульсов с использованием средств вычислительной техники, исключающей субъективные характеристики оператора.
Развитие сейсмоакустического метода прогноза позволило разработать способ оперативного управления выемкой угля и породы, заключающийся в снижении скорости подачи выемочной машины при повышении акустической эмиссии вплоть до остановки при достижении критической величины сейсмоакустической активности пласта. Оперативное управление процессом выемки угля получило широкое применение на шахтах Донбасса и позволило осуществлять добычу угля в опасных зонах без применения специальных способов предотвращения выбросов.
По внезапным выбросам угля и газа зоны геологических нарушений пласта являются наиболее опасными. Статистическая обработка данных по свойствам и состоянию массива в местах проявления выбросоопасности показала, что до 90% всех выбросов угля и газа приурочено к зонам структурной нарушенности угля, в том числе и к геологическим нарушениям.
Детальная геологическая разведка не дает достаточно полного представления о зонах нарушенности в угольных пластах. Специальными исследованиями было установлено, что применение этого метода в Донбассе позволяет выявить лишь I0-I5% нарушений в угольных пластах, а остальные нарушения вскрываются в результате ведения горных работ. Поэтому возникла необходимость в разработке такого метода прогноза, который бы заблаговременно (за 20-30 м) позволял бы определять подход выработки к зоне структурной нарушенности угля.
Зоны структурной нарушенности характеризуются изменением следующих параметров угля:
I. Электро-физических, проявляющихся в увеличении концентрации парамагнитных центров (число разорванных химических связей) и обуславливающих изменение диэлектрических свойств угля.
2. Структурных, сопровождающихся повышенной пористостью.
3. Газодинамических - увеличением скорости газоотдачи.
4. Геодинамических - в выбросоопасных зонах практически отсутствует конвергенция вмещающих пород.
Совокупное и однонаправленное изменение указанных свойств и состояния массива в зонах склонных к выбросам угля и газа, позволили сформулировать и реализовать способ контроля структурной нарушенности угольных пластов, базирующийся на измерении радиационной температуры забоя, определяемой по интенсивности его инфракрасного излучения, причем первые два параметра определяют изменение излучательной способности, а два последних - абсолютную температуру поверхности угольного пласта. В общем случае радиационная температура забоя в выбросоопасных зонах определяется следующим выражением:

$$T_{рад} = \sqrt[4]{\varepsilon_o - \Delta\varepsilon}\,(T_o - \Delta T_{десор} + \Delta T_{мех}),$$

где $T_{рад}$ - радиационная температура забоя угольного пласта, К;

ε_o - излучательная способность ненарушенного угля;

T_o - абсолютная температура поверхности угольного пласта, К;

$\Delta\varepsilon$ - изменение излучательной способности угля в зонах структурной нарушенности;

$\Delta T_{десорб}$ - изменение абсолютной температуры поверхности забоя за счет десорбции газа;

$\Delta T_{мех}$ - изменение абсолютной температуры поверхности забоя в результате механического деформирования и разрушения массива в зоне влияния выработки.

Оценка изменения радиационной температуры массива в нарушенных зонах показала, что оно может достигать 20% по сравнению с ненарушенными участками. Поскольку переход от ненарушенных участков массива к максимально нарушенным происходит непрерывно, то это позволяет осуществлять и прогнозирование зон структурной нарушенности по градиенту изменения радиационной температуры массива. Указанное направление реализуется на выбросоопасных пластах с помощью специального радиационного пирометра в рудничном исполнении. Масса прибора не превышает 2 кг. Питание автономное. Точность измерения радиационной температуры забоя 0,5 к.

Применение данного способа позволяет впервые в практике горного дела количественно оценивать уровень структурной нарушенности газоносных угольных пластов, осуществлять прогнозирование указанных участков массива без непосредственного механического контакта с углем путем непрерывного измерения радиационной температуры массива с расстояния I,5 м от забоя (рис. I).

Изменение радиационной температуры по длине очистного забоя при приближении его к зоне структурной нарушенности приведено на рис. 2.

Применение данного способа не требует больших материальных и временных затрат и с использованием ЭВМ может быть автоматизированным.

Выполненные в последнее время МакНИИ эксперименты позволили исследовать общие закономерности изменения газопроницаемости и газоносности пласта в различных горно-геологических условиях, а также выявить причины, обуславливающие сохранение выбросоопасности впереди забоя выработок. В выбросоопасных зонах, где угольный пласт перемят, под воздействием горного давления либо упругого восстановления пород почвы происходит "защемление" его краевой части, снижение газопроницаемости и сохранение высокой газоносности и газового давления вблизи от кромки пласта. В неопасной зоне, с высокими прочностными характеристиками пласта, влияние горного давления ведет к увеличению трещиноватости и, как следствие, к улучшению дегазации его краевой части (рис. 3). Принцип различного газонасыщения угольного пласта в опасных и неопасных зонах был положен в основу нового прогноза по определению удельного газовыделения из отбитого угля.

При выполнении способа производятся замеры концентрации метана в зоне выемки угля, рас-

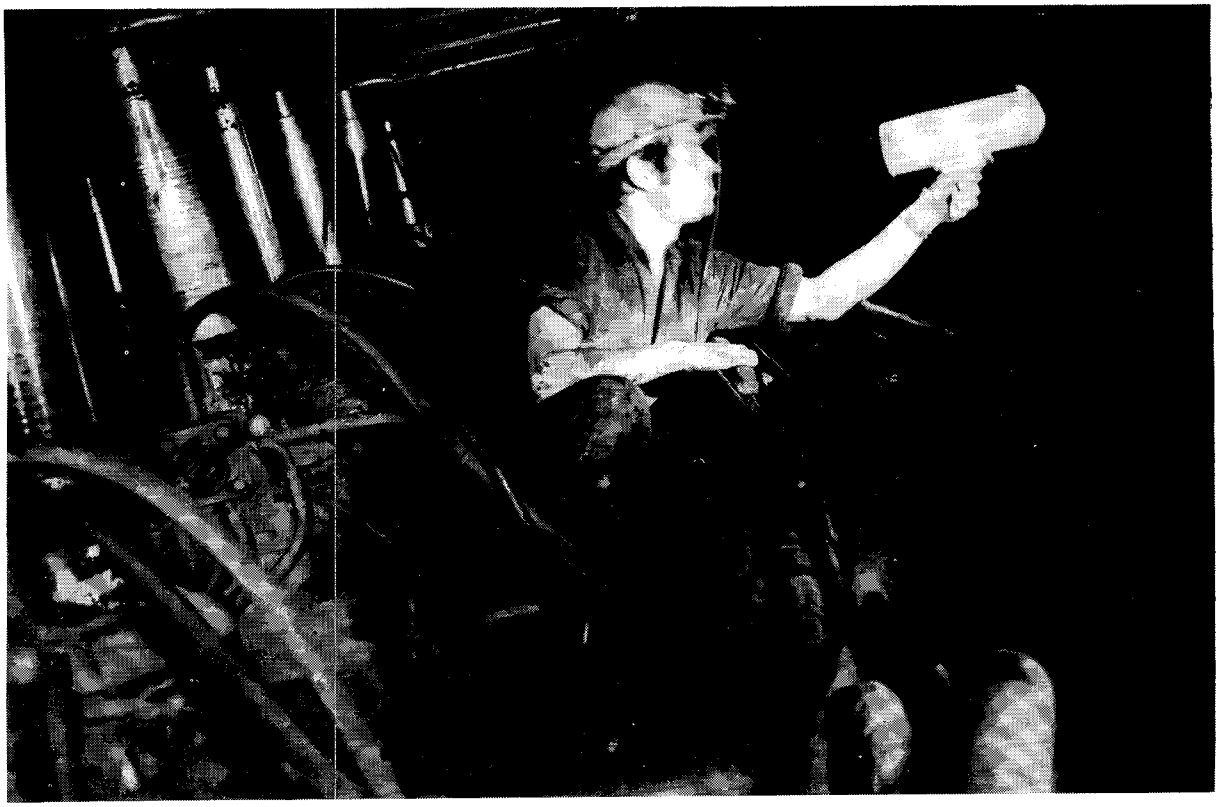


Рис. I.

хода воздуха через забой выработки, мощности пласта и скорости перемещения комбайна. По этим данным по формуле

$$X_{уд} = \frac{(C_1 - C_0)\, Q_в}{100 P},$$

где C_0 - фоновая концентрация метана;

C_1 - концентрация метана за зоной выемки;

P - фактическая производительность комбайна

определяется удельное метановыделение из отбитого угля.

Установлено, что величина удельного метановыделения в геологических нарушениях в 2-5 раз больше, чем за пределами этой зоны, а в местах непосредственного проявления внезапных выбросов в I0-I5 раз выше, чем на невыбросоопасных участках. В подобной аномальной газовой обстановке формируется выбросоопасная ситуация в забое, которая своевременно (за 6-20 м) может быть предсказана по устойчивому возрастанию газовыделения при разрушении угля по мере приближения линии забоя к выбросоопасной зоне.

Отработка угольного пласта в выбросоопасных зонах, выявленных прогнозом, должна осуществляться с применением противовыбросных мероприятий. Но в зонах геологических нарушений часто возникают случаи, при которых принятые параметры ме-

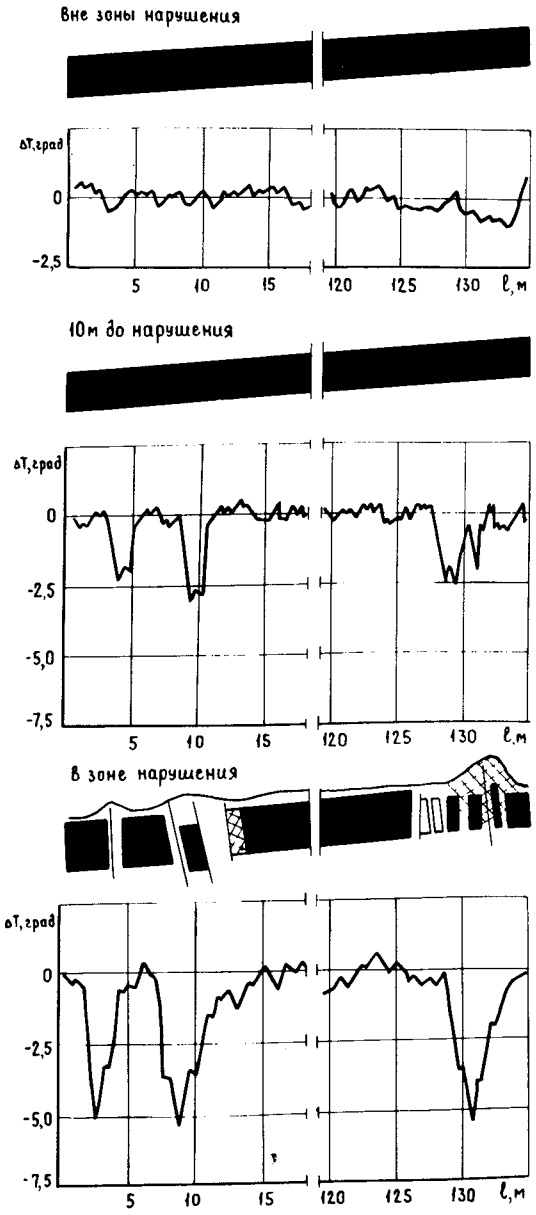


Рис. 2.

роприятий оказываются невыполнимыми. В связи с этим возникла необходимость контролировать эффективность их выполнения путем объективного инструментального измерения показателей, легко поддающихся автоматизации.

Существующие региональные и локальные противовыбросные мероприятия направлены на снижение напряжений в горном массиве и его дегазацию. Поэтому выполнение противовыбросных мероприятий должно сопровождаться повышением концентрации метана в выработке во время и некоторое время после его выполнения. Это положение было использовано для разработки способа контроля эффективности противовыбросных мероприя-

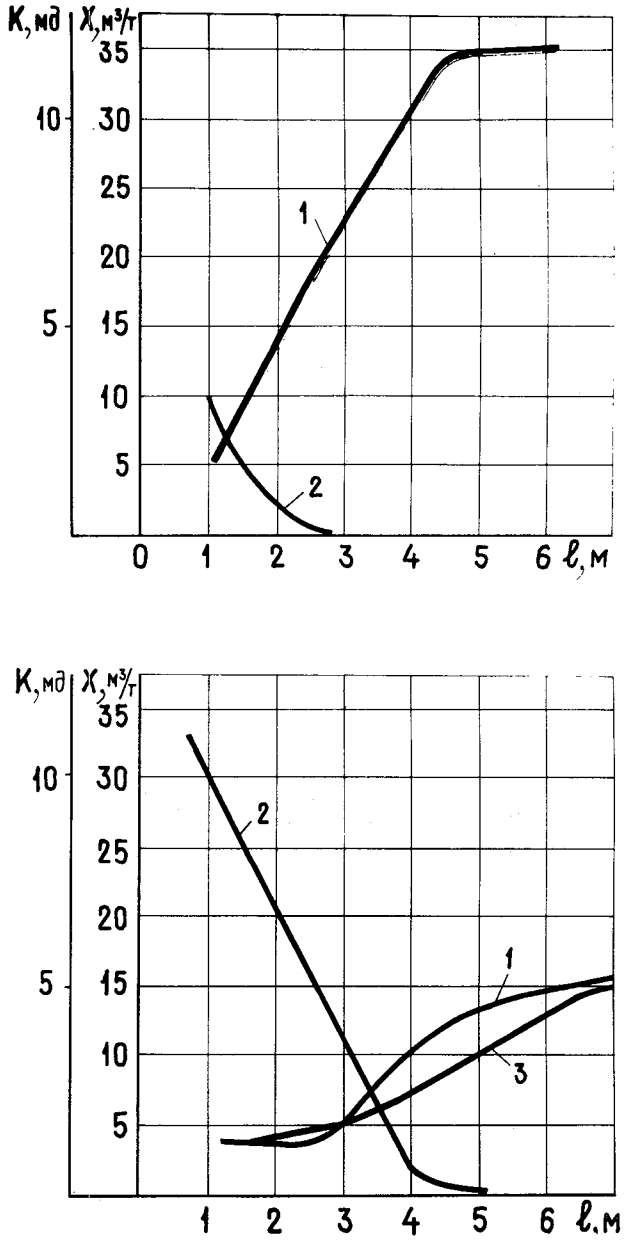


Рис. 3.

тий с использованием аппаратуры контроля метана и ЭВМ. Способ основан на регистрации во время выполнения противовыбросного мероприятия концентрации метана и расхода воздуха через выработку, по которым рассчитывается величина удельного метановыделения. По разности газоносности пласта на глубину влияния способа и удельного газовыделения определяется величина оставшейся газоносности пласта (χ^0), которая сравнивается с критическим значением ($\chi_к$) для каждой конкретной выработки. При $\chi^0 < \chi_к$ выдается заключение об эффективно выполненном способе, при $\chi^0 \geq \chi_к$ -

мероприятие неэффективно. Принципиальная схема автоматизированного метода контроля эффективности противовыбросных мероприятий приведена на рис.4.

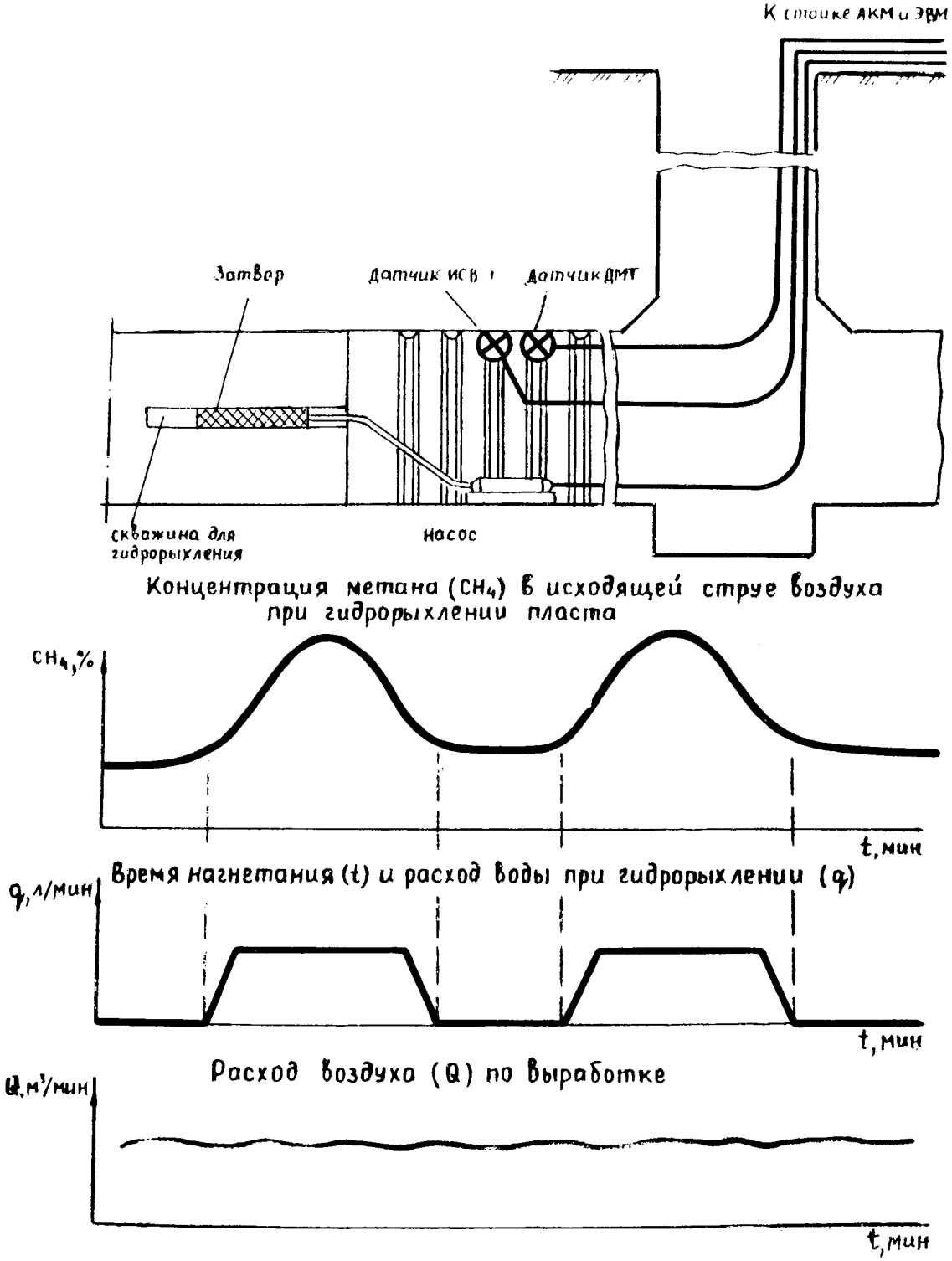


Рис. 4.

Применение автоматизированных методов прогноза и контроля эффективности выполнения противовыбросных мероприятий позволяет исключить субъективный фактор при выполнении работ, повысить надежность определения выбросоопасных ситуаций, снизить трудоемкость прогноза и контроля и затраты времени на их выполнение.

I. Измерение радиационной температуры забоя для оценки структурной нарушенности пласта по длине очистной выработки.

2. Изменение радиационной температуры забоя по длине очистной выработки при приближении к геологическим нарушениям и в самом нарушении:

ΔT - отклонение радиационной температуры забоя угольного пласта от температуры окружающей среды в месте ее измерения, К;

ℓ - расстояние по длине очистного забоя, м.

3. Изменение газоносности и газопроницаемости угольного массива в выбросоопасной и неопасной зонах:

I - изменение газоносности в выбросоопасной (а) и неопасной (б) зонах геологического нарушения;

2 - изменение газопроницаемости в выбросоопасной (а) и неопасной (б) зонах геологического нарушения;

3 - изменение газоносности за зоной влияния геологического нарушения.

4. Схема автоматизированного метода контроля эффективности выполнения противовыбросных мероприятий.

AUTOMATIC SYSTEMS FOR FORECAST AND MONITORING OF THE EFFICIENCY OF MEASURES TO PREVENT COAL AND GAS OUTBURSTS

O.A. Kolesov and L.A. Vainshtein

The improved seismoacoustic method of forecasting zones liable to coal and gas outbursts, developed at the MakNII Institute, on the basis of the seismoacoustic activity of the coal seam has been introduced at more than 150 coal faces and development headings. Seismoacoustic methods are reliable techniques for evaluating processes of breaking in the solid strata; signals related to both conventional rock pressure manifestations and outburst risks are recorded for direct evaluation outburst hazards. In outburst-prone zones, machine haulage has been decelerated or even stopped when the activity of acoustic emission is critical. Specific methane emission during mechanized coal winning in geologically faulted zones is 2 to 5 times greater than beyond the boundaries of such a zone; in areas of outburst, direct manifestation is 10 to 15 times greater than in non-hazardous sections. In such abnormal gas conditions, an outburst-hazardous situation is being formed at the coal face and can be predicted in advance (6 to 20 meters) from a steady increase of gas emission as the coal face approaches the geological fault. Over 80% of outbursts occur in structurally disturbed coal seams. A method has been developed for detecting such zones based on the radiation temperature of the face surface; it allows a low-amplitude geological fault to be detected at a distance of 7 to 10 meters, its influence zone to be identified, and efficient prevention measures to be selected.

SYSTEMES AUTOMATIQUES DE CALCUL ET DE SURVEILLANCE, ET EFFICACITE DES MESURES DE PREVENTION DES DEGAGEMENTS SOUDAINS DE CHARBON ET DE GAZ

O.A. Kolesov et L.A. Vainshtein

La méthode sismoacoustique améliorée de détection des zones sujettes aux dégagements soudains de gaz et de charbon, développée à l'Institut MakNII sur la base de l'activité sismoacoustique des couches de charbon, a été introduite dans plus de 150 fronts de tailles et avancées de charbon. Les méthodes sismoacoustiques sont des techniques fiables permettant de déterminer les processus de fractures dans les strates solides; on enregistre des signaux se rapportant à la fois aux manifestations habituelles des pressions rocheuses et aux risques de dégagements soudains; ces signaux permettent d'évaluer directement les risques de dégagements brusques. Dans les zones qui y sont sujettes, on freine ou on arrête même les transports par machines si les émissions acoustiques deviennent critiques. Au cours de l'extraction mécanisée du charbon, les émissions spécifiques de méthane sont de 2 à 5 fois plus importantes dans les zones de failles géologiques qu'au-delà des limites de ces zones; dans les régions de dégagements, les manifestations directes sont de 10 à 15 fois plus importantes que dans les sections non sujettes aux risques. Lorsqu'on se trouve en présence de ce type d'émission anormale de gaz, un risque de dégagement se forme au front de taille du charbon; on peut le détecter à l'avant (6 à 20 mètres) par l'augmentation régulière de l'émission gazeuse à mesure que le front de taille du charbon se rapproche de la faille géologique. Plus de 80% des dégagements soudains se produisent dans des couches de charbon dont la structure est perturbée. On a développé une méthode permettant de détecter ces zones grâce au rayonnement thermique de la surface du front de taille. En suivant cette méthode, on peut détecter une faille géologique de faible amplitude à une distance de 7 à 10 mètres, calculer sa zone d'influence et adopter des mesures de prévention efficaces.

AUTOMATISCHE SYSTEME ZUR VORHERSAGE UND ÜBERWACHUNG DER EFFEKTIVITÄT VON MAßNAHMEN ZUR VERHÜTUNG VON KOHLE- UND GASAUSBRÜCHEN

O. A. Kolesov, L. A. Vainshtein

Die verbesserte seismo-akustische Methode zur Vorhersage von Kohle- und Gasausbruch-gefährdeten Zonen, die vom MakNII-Institut auf Grund der seismo-akustischen Aktivität des Kohleflözes entwickelt wurde, ist in mehr als 150 Streben und Streckenvortrieben eingeführt worden. Seismo-akustische Methoden sind sehr verläßliche Techniken zur Bewertung des Aufbruchprozesses des festen Gebirges. Signale, welche durch normalen Gebirgsdruck sowie durch einen bevorstehenden Ausbruch hervorgerufen werden, werden für eine unmittelbare Bewertung der Gefahr aufgezeichnet. In ausbruchsgefährdeten Zonen wird der Vortrieb verlangsamt oder sogar unterbrochen, wenn die seismo-akustischen Emissionen kritische Werte erreichen. Die spezifische Ausgasung während des mechanisierten Abbaus in geologisch zerklüfteten Gebieten ist zwei bis fünf Mal größer als in ungestörten Gebieten. Unter diesen anormalen Umständen entwickelt sich eine Ausbruchgefahr im Streb, welche 6 bis 20 Meter im Voraus durch einen stetigen Anstieg der Ausgasung vorhergesagt werden kann, während sich der Streb der Störungszone nähert. Über 80% der Ausbrüche treten in strukturell gestörten Flözen auf. Eine auf der Temperataurausstrahlung des Kohlenstoßes basierende Methode zur Aufspürung solcher Zonen wurde entwickelt. Sie ermöglicht die Aufspürung geringfügiger Störungen in einer Entfernung von zwei bis zehn Metern, die Identifizierung ihrer Einflußgebiete und das Ergreifen effektiver Verhütungsmaßnahmen.

防止煤与气体突出措施功效的自动的预测和侦察系统

Q。A。柯来索夫 及 L。A。凡许探

在MakNII研究所根据煤层地震音响作用判定的预测煤与气突出区域的新方法已被用在一百五十个以上的煤层和开采面。地震音响法是测定固体层破裂很为可靠的方法；有关普通岩石压力现象和突出危险的信号都记录下来以作测定分析之用。在易有突出危险地带，当音响放射值达到邻界值时，机器运输必须减速，甚至停止。在机械采煤时地质断层区的甲烷比发射度较这区以外的地区高至2－5倍；在突出区域里，读数较无危害的区域里高10－15倍。在这种反常的气体情形下，煤层表面产生突出危害的情况，可以从煤层表面接近地质断层时发射率渐增而提前测定（6至20公尺）。百分之八十的突出发生在结构扰动的煤层中。我们发展了一种用煤层表面辐射温度来测定这种区域的方法；它能在7至10公尺内测出低幅度的地质断层和它的影响范围并选定适当有效的预防措施。

POSSIBILITY OF EMPLOYING OF RADON EMISSION AND MIGRATION TO FORECAST OF SUDDEN OUTBURST

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ABSTRACT

Presented paper is a continuation of previous work published on the Conference in Beijing, 1987. During two years observations of changes in radon concentration have been extended to different workings, including sandstone. The radon emission from long drainage boreholes was measured, and the results have been used for the estimation of degasification of boreholes impact. The most essential result is the real possibility to use this phenomenon, this is the changes in radon emission, to forecast outbursts in coal seams and as a means of warning in sandstones.

INTRODUCTION

In the paper presented on 22-nd conference (Lebecka et al. 1987) it was stated that "three outburst occurences which were experienced during the test period were always accompanied by a considerable decrease of radon concentration prior to outbursts and an intense increase after the occurrence". During two years a further work confirmed this regularity.

The general method of observations and measurements was described in the previous paper, mentioned above, Some small modifications to the detector of radon were introduced, increasing its sensitivity. The gas samples were taken from 3-6 m leng, test boreholes, drilled in coal at faces of driven roadways. Generally radon concentrations varied from below 20 Bq/m3 to 2500 Bq/m3.

THE CHANGES IN RADON EMISSION IN DRIVAGES

Three drivages were systematically observed. One was the transportation dip 6 W in 672 seam from - 250 m level to - 350 m level, the second was in 671 seam on - 250 level, both in the Thorez Colliery. The data of outburst occured are given in Table 1, and the results of observations of radon concentrations changes are given in Figure 1. In this case, during the last days of monitoring the radon concentrations dropped below the detectable limit. Four days later after about 3,2 m of face advance an outburst occured in which 70 tonnes of coal and 15000 m3 of CO_2 were released.

In both headings not only radon concentration but also gas pressure and desorption intensity were measured as well. The measured values significantly exceeded the polish critical indices values. There were no correlations found between the gas pressure, desorption intensity and radon concentration for all samples taken. This problem will be discussed more widely later.
The third drivage submitted to detail observation was the C-Drivage, in sandstone on - 110 level in Nowa Ruda Colliery. This gallery is driven in the direction of the coal seam which has been closed in 1939. In 1988 the drivage was worked from 1962, to 1982,1, exclusively in sandstone

Table 1

Outburst in Thorez Colliery during the observation period

Outburst number	Datum	Face	Amount released of coal	
			tonnes	CO_2 (m^3)
150	85.03.28	Transportation dip 6 W in 672 seam	16000	15000
153	85.05.28	"	10000	12500
155	85.07.17	"	16000	18890
159	87.11.11	Drivage in 571 seam - 250 m level	7000	15000

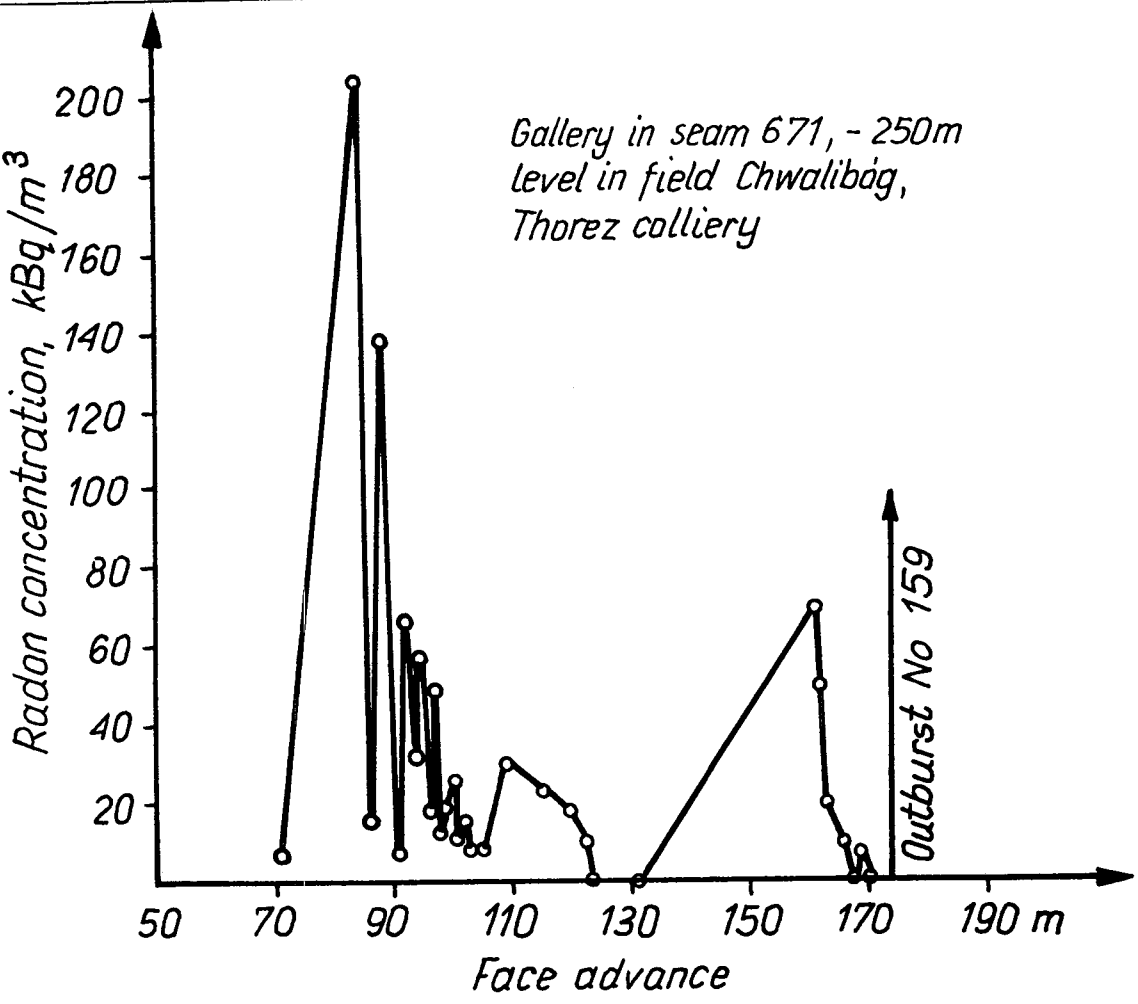


Fig. 1. Radon concentration as a function of face advance

Table 2

Physical parameters of sandstones	prone to outburst	not prone to outburst
Porosity (%)	2,7 - 5,4	8 - 12
Filtration coefficient (md)	0,1	0,1 - 654,2
Gas capacity (ml/g)	0,05 - 0,51	0,05 - 0,85
Compression strenght (MN/m^2)	80 - 160	40 - 80
Tensite strenght (MN/m^2)	6 - 10	3 - 5
Crushing index	5 - 10	15

rock, without crossing the coal seams. Only two outburst occured, both sandstone and CO_2 one with 4000 ton of outbursted mass. The prevention against most frequent outburst was assured by drilling long destressing boreholes and relaxing shotfiring. The outburst in sandstone are rather rare and sandstones prone to outburst have very different physical properties than those which are not prone to outburst. This differences are given in Table 2.

Athough the gas content in sandstones considerably (10-20 times) lower than the gas content in coal, the outburst are often more dynamical than in coal seams.

The observations of changes in radon concentration has been conducted for 6 month, from April 27 to October 7. No coal seams were passed. The radon concentrations were very high, from 470 Bq/m3 to 2150 Bq/m3. Apart from two outburst mentioned above, a very stong one was observed just before the beginning of the observations. The results of measurements are presented on Fig.2
Prior to both outburst the decrease of radon concentrations is observed, but just before the outburst one may observe the increase. It is seen, that in this case the changes in radon concentrations have a quite different character than those in coal seams. It results from different properties of rocks - in sandstones the gas is free, not adsorbed like in coal. The outburst zones are surroundet by zones of high stresses. Approaching the outburst zone one passess high stressed rocks, with tighten pores. There the radon liberation is very small. After that, one passess into the zone with higher porosity, and gas content. There radon is easily liberated and its concentration increases. In zones with high porosity and casy liberation of radon the outburst is possible.

OBSERVATIONS OF CHANGES IN RADON CONCENTRATION IN GASES FROM LONG DISTRESSING BOREHOLES

These investigations were done as an addition to explanation of migration of gas in rocks prone to outbursts. Measurements were made of outflowing gases from long distressing boreholes in the shale strata in Nowa Ruda Colliery. Exploitation of this shale was necessary to distress the overlying strata highly saturated with carbon dioxide.

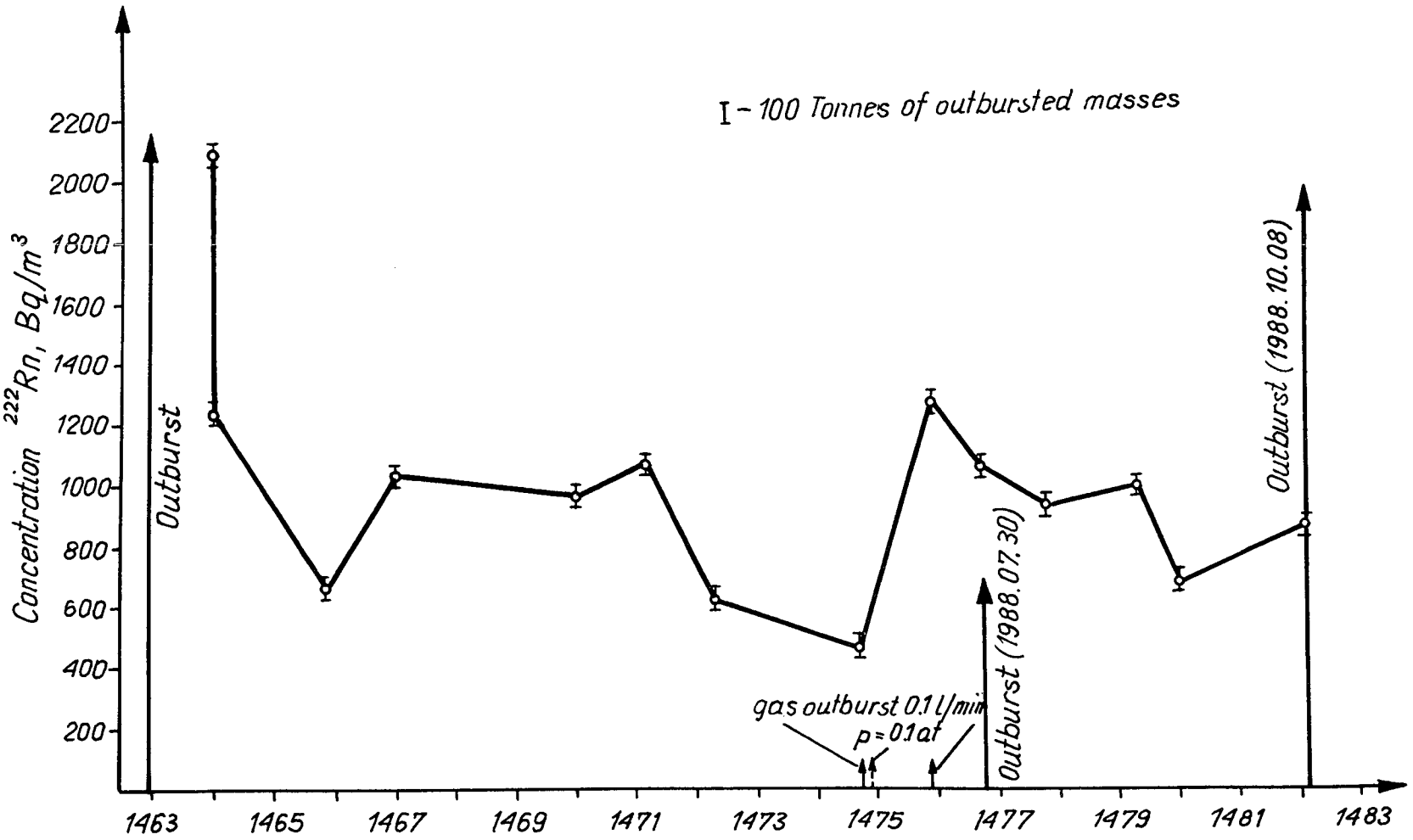


Fig. 2. Concentration changes ^{222}Rn in gases from testing boreholes drilled at the face of the testing gallery C in the direction of Ludmita shaft of „Nowa Ruda" colliery

During exploitation of shale the carbon dioxide transferred from sorbed to free state gas, and migrated down in large quantities into the deepest galleries due to gravitional forces. In these conditions it was impossible to maintain a low level of CO_2 content in air which is required by regulations. Boreholes 54 to 77 m long were obliquely drilled in the direction of the overlying beds. Gas flow and CO_2 concentration were systematicaly monitored, Also radon concentration in exhaust gases was measured from time to time.

In three of the five boreholes the radon concentration increased in time. In the remaining two boreholes radon concentration was also constant. The increase in radon concentration is explained by casier emission of radon with the decreasing rock pressure. The example of changes in radon concentration in gases from long distressing boreholes is given on the fig.3.

On the basis of these results a minimum radius of degasification borehole impact was estimated using two methods. One was based on the radon emission rate and radon emanation rate (E_{Rn}) from a given type of coals and the other was based on the gas content in the coals.

For estimation of minimum borehole impact based on radon emission rate the following simplifying assumptions were made:

- the radon emanation rate is constant in the neighbourhood of the borehole,
- the radon flows only according to pressure gradient i.e. in the direction of borehole,
- the radioactive decay of radon was not taken into account,
- the radon emanation rate is independent of pressure in the range of pressure changes in the area from which the gas was collected.

On this basis the radius of minimum impact of borehole was estimated. Assuming a radon emission rate of Q_{Rn} = 17 Bq/s, a borehole lenght l = 54,35 m with a radius of r = 0,075 m, radon emanation rate of $E_{Rn} = 5 \cdot 10^{-7}$ Bq/kg.s and average rock density of $q = 2.10^3$ kg/m^3, the minimum radius

$$R = 10,5 \text{ m}$$

was calculated

The deriving of formula is given in Appendix 1a.

To estimate the minimum borehole impact based on gas content the followign assumptions were made:

- the rocks in neighborhood of borehole are uniformly saturated with gas,
- during the degasification the saturation of rocks by gas is a function of time and grows linearly with the distance from the borehole, Based on these assumptions the calculated, minimum borehole impact would be

$$R = 10.8 \text{ m}$$

The deriving of formula is given in Appendix 1b.
For estimation of time of gas migration to the borehole the following assumptions are made:

- gas is not sorbed on the path of migration,
- the gas flows in the direction of the borehole only,
- the medium is uniformly porous.

This estimation can give information for the model of gas migration trough the rock body, since radon is radioactive and rather short living and therefore decay while passing through the rocks.

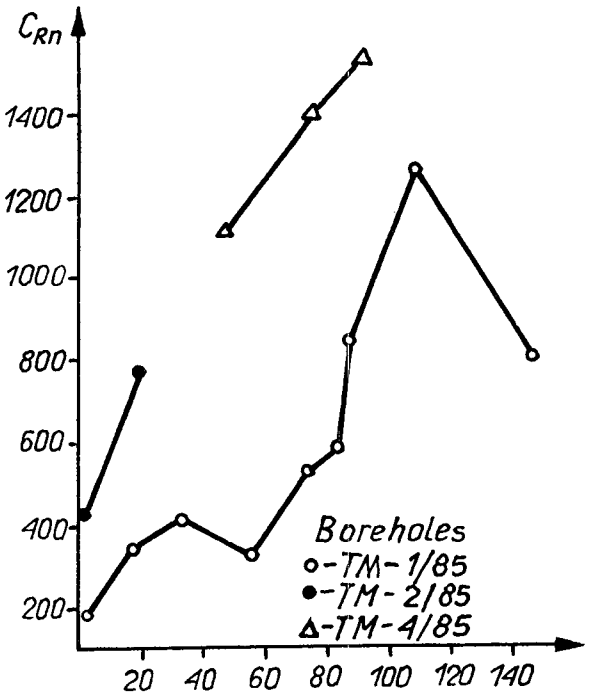


Fig. 3. Radon concentration changes in gases of long degassing boreholes.
C_{Rn} - Radon concentration
t - Time

The deriving of formula is given in Appendix 1c in which the table of results is given as well.

The data shows that from a distance of 10 m the gas should flow for 11 days, which is three times longer than half-lefe of ^{222}Rn isotope. The quantity of radon flowing from borehole is rather high, so one may conclude that the migration of gas is not likely to be in an uniformly porous media, but there exist priviledged paths (fissures and faults) by which gas must migrate quickly.

THE POSSIBILITY OF OUTBURST FORECASTING

In the table 3 is given the summing up of results of measurements of changes in radon concentration in seams prone to outburst.

It is seen that radon concentration depends sharply on the kind of tock. The highest concentrations are observed in sandstone, the lowest in shales, the concentrations in coal are medium. In eight cases of outbursts that occured in coal and shale with coal interlaying. Radon concentration was never higher than $20^{+}10$ Bq/m3 prior to outburst.
It is also true that the cases of so low concentrations occured without outbursts, but in seams prone to outburst the radon concentration changes very much. The symptom of dropping of radon concentration is characteristic just the same as raising of desorption and pressure. The fact is that there is no correlation between obligatory indexes, desorption and pressure. The correlation coefficients are very low no more than 0,2.

In pure shales and sandstones the situation is different. In shales the radon concentrations are generally low, and changes in radon concentration are generally difficult to observe.For this reason the radon method of forecasting does not work for shales.

In sandstones the course of radon concentrations is quite different than in coal and shales, The concentrations are generally high, and do not drop to values near zero. But in two cases of outburst the drop prior to outburst is observed. The number of observations is still very small, nevertheless considerable drop and after that a raise of radon concentration in sandstone have to be considered like a serious symptom of the approaching of coming outburst.

Conclusions

In spite of small number of observations of changes in radon concentration before and aften outburst it is evident that clear regularities different for different rocks exist. In coal seams, the very low concentrations are observed prior to outbursts. In sandstones the drop

Table 3

The results of measurement of changes in radon concentrations in different places with outburst hazard

	workings					
	1	2	3	4	5	6
Type of rock	coal	coal	shale with coal interlayer	coal	coal	sandstone
Rn concentration Bq/m3						
maximum	690	205	143	193	175	2105
minimum	11	2	2	13	101	474
average	168	30.5	16.4	108.1	130.3	781
Number of outbursts	3	1	4	0	6	2
Number of cases C C_{kr}	5	6	26			
Number of measurements	36	35				
Outbursts parameter						
Mass released (tons)	160,100, 160	70	90,120, 40,100			1400, 4000
Gas volume (10^3 m^3)	15/12.5/ 18.8	15.0	7.6/13.0/ 3.44/7.02			12, 40.0
Rn concentration prior to out-	20+10 14+10 11+10	2+6	6+6 14+6 12+6 0+5			474-1290 691-873
Obligatory hazard indexes						
Desorption mm H_2O			462, 360 418			
pressure			2.0 0.3 1.0 0.5			0.1 0.0
C_{kr}	20, 19 11	5	15			two times drop
Adevance	1, 5, 6	4	1, 3, 1, 1,	-	-	11, 11

Workings designation

1 - Thorez Colliery - transportation dep.
2 - Thorez Colliery drivage in seam 671
3 - Nowa Ruda - drivage in shale
4 - Nowa Ruda - 101 longwall face
5 - Nowa Ruda - rise gallery
6 - Nowa Ruda - C drivage

before and a raise of concentration after is observed. For this reason the average level of radon concentration for a given seam must be established. The big changes give evidence about changes of strength in strata - accumulation or disstressing.

It would be very useful to test the radon method in different conditions not only in Poland where the number of seams prone to outburst is very small.

The results of radon observations may be used now at least like auxiliery information in forecasting of outburst, particularly in coal seams where sharp drop of radon concentration may be considered as a warning. Also in sandstones the phenomenon of sharp drop and increase of radon concentration is a signal of the coming outburst.

The relations between results of radon concentrations measurements and seismic phenomena must be found.

APPENDIX

1a) Calculation of minimum boreholes impact based on radon emission rate

Assumptions are given before
The flow of radon from borehole is

$$Q_{kn} = E_{Rn} \times M = E_{Rn} \; l \; (R^2 - r^2) \quad (1)$$

where

E_n - radon emanation rate Bq/kg.s

M - mass of rocks which emanate observed radon amount (kg)

l - lenght of borehole (m)

ϱ - rock density kg/m3

r - borehole diameter (m)
R - borehole impact

Thus

$$R = \sqrt{\frac{Q}{E_{Rn} \; \pi \; l \; \varrho} + r^2} \quad (2)$$

1b) Calculation of minimum borehole impact based on gas content

Assumptions are given in the main text
The degree of saturation of rock by gas is a function of time and grows linearly with the distance from borehole

$$G(x,t) = a(t).x \quad (3)$$

Out of borehole impact the degree of saturation is

$$G(R,T) = G_o \quad (4)$$

The volume of gas flow from borehole

$$V = \frac{1}{2} \; \pi \varrho \; l \; (R^2 - r^2) \; . \; G_o \quad (5)$$

This volume is also:

$$V = \int_0^T Q(t) \; . \; sdt \quad (6)$$

Q(t) - the gas flow m3/s

s - the concentration of gas (CO_2 or CH_4) in gas from borehole

T - time of life of borehole

From formula 5

$$R = \sqrt{r^2 + \frac{2 \; V}{G_o \; \pi \; \varrho \; l}} \quad (7)$$

V - must be calculated from form.(6)

1c)Estimation of migration time time to the borehole

Assmptions are given in the main text.
The total flow of gas by laterel surface of borehole (concentric cylinders with radius r and R) is:

$$Q_r = Q_R = Q_x \qquad (8)$$

$$Q_x = 2\pi v \cdot l \qquad (9)$$

where v - the gas velocity in distance x from borehole.
From formula (9) we have

$$V = \frac{Q}{2\pi 2x}$$

The time of ags flow $dt = 2\frac{1}{Q}\,dr$

and

$$\int_0^t dt = \int_r^R \frac{2\pi}{Q}\,dr$$

$$t = \frac{\pi}{Q}(R^2 -,r^2)$$

Example:

For Q = 1 m^3/min for R = 10 m the time of gas flow is 284.7 days.

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UTILISATION POSSIBLE DE L'EMISSION DE RADON ET DU PHENOMENE DE MIGRATION POUR PREVOIR LES DEGAGEMENTS SOUDAINS

Jolanta Lebecka, Kazimierz Lebecki, Zbigniew Kobiela

Ce rapport constitue la seconde partie des travaux d'observation et de mesure de l'émission de radon dans les couches sujettes aux dégagements, travaux qui ont été présentés lors de la 22ème conférence à Beijing.

Il décrit les essais effectués sur l'émission de radon de trous de mine de drainage forés dans les couches sujettes aux dégagements. L'impact du rayon minimum de dégazage d'un trou de forage a été calculé sur la base de la mesure de la concentration de radon. Ces calculs ont utilisé deux méthodes : l'une est basée sur l'émission de radon et l'autre sur la teneur en gaz. Les résultats obtenus sont semblables: 10,5 et 11,4 m.

On a évalué le temps minimum de migration des gaz de la portée d'impact du trou de forage. Les modifications de la concentration du radon avant et après le dégagement ont également été analysées. Nous décrivons un modèle simple de dégagement.

MÖGLICHKEIT, RADONAUSSTRAHLUNGEN UND -AUSBREITUNGEN ZUR VORHERSAGE VON PLÖTZLICHEN GASAUSBRÜCHEN ZU BENUTZEN.

Jolanta Lebecka, Kazimierz Lebecki, Zbigniew Kobiela

Der Vortrag stellt den zweiten Teil von Untersuchungen und Messungen von Radon- Ausstrahlungen in Flözen dar, die zu plötzlichen Gasausbrüchen neigen, welche bereits auf der 22igsten Konferenz in Peking vorgestellt wurden.

Die Untersuchung beschreibt Tests mit der Ausstrahlung von Radon aus Drainagelöchern, die in Flöze mit Neigung zu plötzlichen Gasausbrüchen gebohrt wurden. Der Mindestumkreis der Entgasung um das Bohrloch wurde basierend auf der Radonkonzentration errechnet. Die Berechnungen wurden nach zwei Methoden durchgeführt: basierend auf der Radon-Ausstrahlung und basierend auf dem Gasgehalt. Die erzielten Ergebnisse sind sich sehr ähnlich: 10,5 und 11,4 m.

Die Mindestzeit der Gasausbreitung von der Bohrlocheinflußzone wurde bewertet. Die Veränderung in der Radonkonzentration wurde ebenso vor und nach Gasausbrüchen analysiert, und ein einfaches Vorhersagemodell wurde entworfen.

ВОЗМОЖНОСТЬ ИСПОЛЬЗОВАНИЯ ЭМИССИИ И МИГРАЦИИ РАДОНА ДЛЯ ПРЕДСКАЗАНИЯ ВНЕЗАПНЫХ ВЫБРОСОВ ГАЗА

Иоланта Лебецка, Казимир Лебецки, Збигнев Кобиела

Доклад описывает вторую часть проекта по наблюдению и измерению выделения радона в пластах, в которых могут произойти внезапные выбросы этого газа. Первая часть была доложена на 22-ой конференции в Пекине. Описываются тесты, определяющие выделение радона из дренажных скважин в пластах, склонных к внезапным выбросам этого газа. Минимальный радиус воздействия дегазационной скважины был рассчитан на основе измерения концентрации радона. При расчетах использовались два метода: 1) метод, основанный на выделении радона и 2) метод, основанный на содержании газа. Были получены близкие результаты: 10,5м и 11,4м. Было определено минимальное время миграции газа из зоны воздействия скважины. Были также проанализированы изменения в концентрации радона до и после внезапного выброса и дана простая модель для предсказания этого явления.

用氡气的放射和渗移性来预测突然突出的可能性

周阐塔．勒贝卡，卡席米尔兹．勒贝奇，史华格钮．柯别拉

本文为在北京第二十二届煤矿安全学术会上发表的有突出倾向煤层氡气发射的观察与测定的论文的第二部分．这里叙述有突出倾向煤层排泄孔内的氡气发射测量试验．最小除气孔半径根据氡气浓度的测定结果来计算．用了根据氡气发射量和气体量的两种方法来计算，结果很相似：10．5 与 11．4m．

文章里求出了气体从孔眼冲击区渗移出来的最短时间，并分析了突出前后氡气浓度的变化．本文还给出了一个简单的突出预测模型．

ОЦЕНКА ОПАСНОСТИ ОТ ВНЕЗАПНЫХ ВЫБРОСОВ УГЛЯ И ГАЗА ПРИ ВЛИЯНИИ ГОРНОГО ДАВЛЕНИЯ

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РЕЗЮМЕ

Проведны исследования для оценки опасности от внезапных выбросов угля и газа при наличии влияния опорного горного давления, путем измерения: показателя скорости десорбции V_1,газового давления $P_г$, сближения вмещающих пород и других показателей.На основе полученных результатов установлена возможность отмены мероприятий для разгрузки и дегазации пластов,а также разработаны требования и рекомендации при работе на выбросоопасных пластах. Практическое применение результатов привело к существенному повышению эффективности борьбы с внезапными выбросами угля и газа, а также к снижению себестоимости добытого угля.

1. ВВЕДЕНИЕ

Одним из трех главных факторов, обуславливающих возникновения внезапных выбросов угля и газа, как известно, является горное давление, соответственно возникающие под его влиянием напряжения в угольных пластах.Часто однако, этот фактор учитывают весьма относительно или в виде общих требований и рекомендаций относительно порядка итехнологии ведения горных работ.Опыт работы в Балканском бассейне в Болгарии дает основания считать, что оценка выбросоопасности с конкретным учетом влияния горного давления позволяет не только целенаправлено устранять предпосылки к возникновению внезапных выбросов, но часто и весьма выгодно использовать благоприятное воздействие в отношение выбросоопасности. При этом,речь идет главным образом о повышенном опорном горном давлении,которое в общем плане считается повышающим выбросоопасность. В настоящей статье рассмотрим коротко наиболее интересные результаты исследования и практического опыта в Балканском бассейне в этом плане.

2. ГОРНО-ГЕОЛОГИЧЕСКИЕ И ДРУГИЕ ОСОБЕННОСТИ В БАССЕЙНЕ

2.1.Горно-геологические условия Угленосная толща бассейна,сеноманско-мелового возрастта,включает 8 угольных пластов,не все с промышленном значением и местами чередующиеся в связи с весьма сложной тектоникой.Часто встречаются разрывы угленосной свиты с амплитудой от несколько метров или километров до десятков километров. Наблюдаются тектонические структурные формы разного типа:нормальные и перевернутые складки и надвиги с внутренними усложнениями,много продольных и поперечных дизьюнктивов с большим или меньшим протяжением и амплитудой. Ввиду интенсивных тектонических сдвижений,характерным в бассейне является весьма частое изменение распространения,залегания и мощности угольных пластов, даже на небольших расстояниях по простиранию и падению.Мощность угольных пластов меняется в пределах от 0,1-0,6 до 20-30m, но для промышленных чаще всего от 0,6-1 до 1,5-2,5m .Падение

пластов в основном крутое, чаще всего 70-90°, но встречаются и участки с пологим и наклонным залеганием от 5-15 до 25 - 45°. Угольные пласты характеризуются высокой степени тектонической нарушенностью / III - V тип, соответственно среднее расстояние между микротрещинами от 0,02 до 0,1-0,15mm /, а также сложной структурой, нередко с прослоями породы и силицитизированного угля. Уголь имеет весьма небольшую крепость: коэффициент крепости по методу толчения f от 0,12-0,16 до 0,25-0,37; крепость на одноосное сжатие при неправильной форме образцов - от 0,02 до 0,1-0,35МРа, а для наиболее крепкого угля при стандартных образцах достигает до 0,7-2,5 МРа. В бассейне добывают уголь марки Ж, ПС, Т и К, с выходом летучих веществ от 9-17 до 25-38%. Вмещающие породы чаще всего представлены аргиллитами и глинистыми мергелями с крепостью от 20-35 до 50-70 МРа, но встречаются и песчаники с крепостью от 60-70 до 100-140МРа. Мощность междупластий меняется в границах от 6-15 до 20-45m. Следует отметить, что отдельные угольные пласты в свите, а также одноименные по стратиграффическому порядку, в разных частях бассейна и даже на соседних шахтах и участках отличаются по своим характеристикам, иногда весьма существено.

2.2. Особенности проявлений внезапных выбросов

Внезапные выбросы угля и газа в бассейне регистрируются с 1933г. До сих пор отмечено более 250 газодинамических явлений с выделением метана, в том чесле: собственно внезапные выбросы угля и газа - 49%; выдавливания и высыпания угля с повышенным газовыделением - 30-35% и с попутным газовыделением - 15-20%. Обратим внимание, что выдавливания и высыпания угля с повышенным газовыделением мы рассматриваем в качестве переходных к выбросам и пласты, где наблюдаются, относятся к угрожаемым, т.е. склонным к внезапным выбросам угля и газа. Газодинамические явления в целом и внезапные выбросы в частности в бассейне регистрируются главным образом при проведении подготовительно-нарезных выработок, с наибольшей интенсивности при вскрытии пластов квершлагами - до 350t При ведении добычных работ отмечены чаще всего выдавливания и обрушения угля с повышенным и попутным газовыделением, но и несколько собственно внезапных выбросов угля и газа. Минимальная глубина проявления первого выброса на различных угольных пластах меняется в весьма широких пределах - от 130 до 390m, в зависимости от крепости, газоносности и других свойств пластов. Наблюдается четкая тенденция повышения интенсивности внезапных выбросов и выбросо - опасности в целом с увеличением глубины горных работ от поверхности.

2.3. Горно-технические условия

В бассейне применяется прямой порядок отработки шахтных полей. Высота эксплоатационных горизонтов-этажей составляет 50m. Подготовка осуществляется групповым полевым штреком, квершлагами через 100-120m по простиранию пластов, пластовыми штреками и скатами-углеспусками. Эта схема со значительным удельным весом подготовительно-нарезных выработок обуславливается весьма большой тектонической нару - шенностью и невыдержанностью угольных пластов и сложностью условий. В связи с этим, механизировать горные работы в бассейне не удается, хотя эксперименты в этом плане продолжаются и теперь. Проходка подготовительно-нарезных выработок и очистные работы проводятся отбойными молотками и буро-взрывным способом. При очистной выемки применяют три системы разработки: с подэтажным обруше - нием - до 65-70% от добычи по бассейну; "лава-этаж" с подвиганием забоя по простиранию пласта - 25-30%; горизонтальные слои с сухой закладкой по вос-

станию - 5%. Горные работы в настоящее время ведутся на глубине от поверхности от 130 до 500m , но чаще всего от 250 до 350m .В связи с горным рельефом,на данном эксплоатационном горизонте глубина от поверхости может меняться на 100 - 150m .

3. РЕЗУЛЬТАТЫ ИССЛЕДОВАНИЙ И ПРАКТИЧЕСКОГО ОПЫТА

Рассмотрим коротко наиболее интересные результаты в отношении защитных пластов, разных систем очистной выемки и других выработок, отметив сначала некоторые общие принципы при исследованиях и ведении горных работ в целом.

3.1.Основные методические принципы при исследованиях

В качестве основного методического принципа было принято исследование проявлений горного давления совместно с оценкой напряженного газодинамического состояния предзабойного угольного массива. Последняя проводилась на основе измерения ряда индиректных характеристик в этом отношении, использующиеся также для оценки опасности от внезапных выбросов угля и газа, как то:показатель скорости десорбции V_1, газовое давление в угольных пластах $P_г$, количество угольной мелочи при бурении скважин V_0 и др. Оценка проявлений горного давления проводилась на базе измерений сближений боковых пород главным образом, в выработанном пространстве и в выработках впереди забоя. Для этой цели закладывали через 3-8m по простиранию или падению пару металлических или деревянных реперов на глубину 0,5-1,0m и периодически измеряли сближений. При этом использовались динамометрические теллескопические стойки,при необходимости с индикаторами, с точностью отсчетов соответственно 0,5-1,0mm и 0,01mm .Контрольные исследования проводили аналогичным образом относительно выдавливания угля из предзабойного массива при помощи глубинных реперов,а также относительно нагрузки на угольный пласт с использованием гидравлических динамометров.Проводились также измерения метановыделения и отмечались особенности при работе,и прежде всего при выемке угля.

3.2. Некоторые общие принципы при ведении горных работ

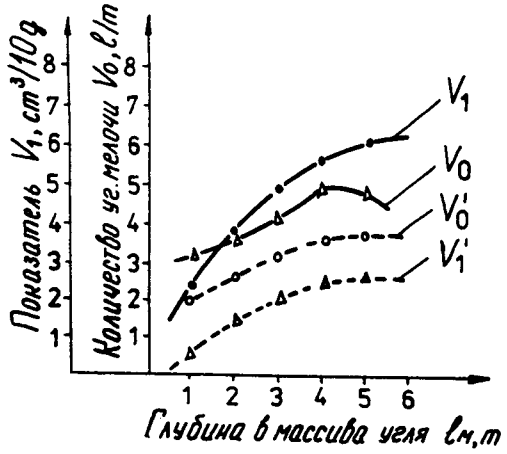
Коротко отметим некоторые общие принципы и требования при ведении горных работ в бассейне, связанные с учетом горного давления на выбросоопасность. В качестве основного принципа принято и в нормативных документов по безопасности регламентируется выполнение прогноза и ведение горных работ в соответствии с требованиями по выбросоопасности при глубине больше минимальной критической глубины выбросоопасности в бассейне, которая является 130m при падении пластов до 45°, и 150m - более 45°. Она определена на основе статистических данных о регистрированных в бассейне внезапных выбросах угля и газа, а также других газодинамических явлений упомянутого переходного типа. В отношении выбросоопасности,эта глубина рассматривается как обуславливающей нагрузку и соответствующих напряжений в угольных пластах, при превышении которых возможно разрушение последних в форме внезапных выбросов угля и газа при специфических условиях в бассейне.При особенностях последних,воспринятый принцип имеет важное практическое значение,ввиду исключительно сложной обстановки и особенно то обстоятельство, что горными работами не пройдена вероятная выбросоопасная глубина на многих пластах,пока что неопасных,но с рядом характеристик соответствующих выбросоопасным. На этих пластах проводится локальный прогноз выбросоопасности,с целью своевременно их отнести к угрожаемым при необхо-

димости.Он основывается на восьми признаках, из которых два характеризуют горное давление: глубина от поверхности и крепость вмещающих пород. При практическом применении локального прогноза и критической глубины в течении более 15 лет не отмечено неожиданных выбросов, что потверждает их надежность, а также надежность принятых принципов в целом. В отношении локального прогноза, на наш взгляд это обуславливается в значительной мере, ввиду конкретного учета горного давления (Нановска, 1986) Другой принцип, использующийся обычно на всех бассейнах с выбросоопасностью, является порядок ведения горных работ, предусматривающий использования во всех случаях, где это возможно, разработку склонных к выбросам пластов /опасные и угрожаемые/ с опережающей выемкой защитных пластов. Некоторые результаты в этом отношении мы рассмотрим несколько дальше.

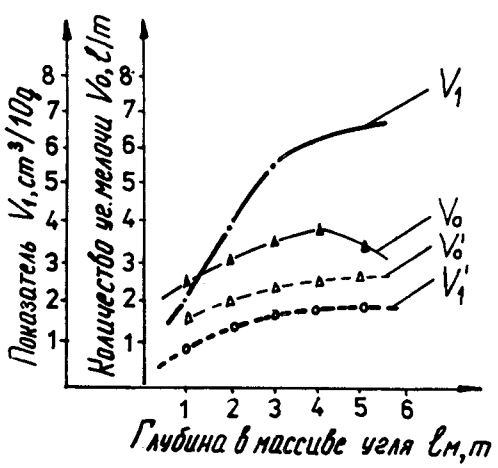
При применяемом в бассейне прямом порядке отработки шахтных полей и сплошных систем очистной выемки, важным является принцип проведения подготовительно-нарезных выработок, а именно: чтобы пластовые штреки проходились с опережением линии очистных работ не менее 50m ,а нарезные выработки - не менее 20m .Это требование и параметры обоснованы результатами обширных исследований о распространении опорного горного давления под влиянием очистных и других выработок при различных горно-геологических и горно-технических условиях и характеристиках. Результаты этих исследований используются широко при разработке технологических параметров, требований и мероприятий, способствующих снижению выбросоопасности при ведении очистной выемки и проходке подготовительно-нарезных выработок, а также для решения ряда практических других вопросов по борьбе с внезапными выбросами, как будет показано далее. При соблюдении упомянутых требований для подготвительно-нарезных выработок существенным образом снизилось число внезапных выбросов в зонах повышенного горного давления под влиянием очистных и других выработок. В последних 15 лет отмечено всего 3-5 случаев, в том числе два выброса, вызванных сотрясательным взрыванием, а в предъидущих годах более 60% выбросов происходили в таких зонах.

3.3. Защитные пласты

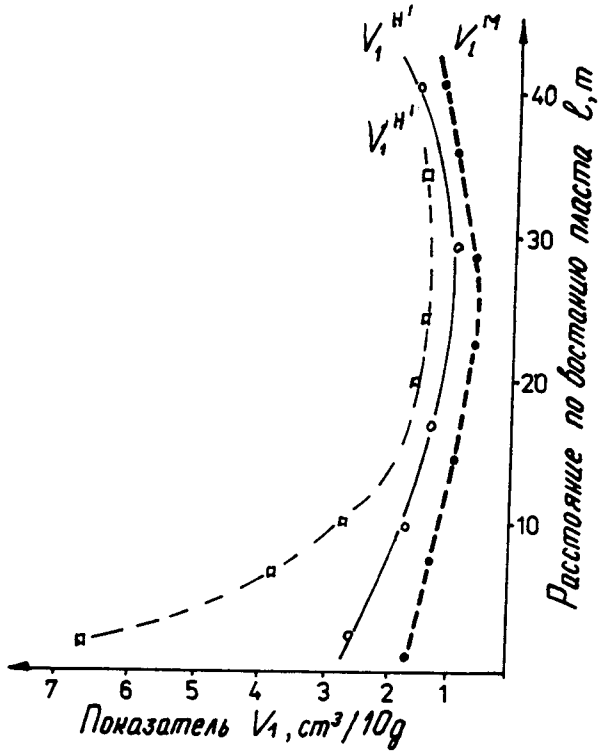
Несмотря на казалось бы благоприятные условия, разработка склонных к выбросам пластов с опережающей выемкой защитных удается только для около 10 - 30% случаев в бассейне. Это обусловлено непостоянным распространением, залеганием, мощностью и структурой потенциально защитных пластов, а также горнотехническими особенностями. Анализом практического опыта в предидущих годах и экспериментированием, была установлена возможность использовать в первом приближении для определения границ защищенной зоны по востанию и падению, а также незащищенной зоны под влиянием целиков в выработонном пространстве, соответственно применяемых в СССР схемы и углы сдвижения-защиты (Инструкция .. 1977) При этом, зоны повышенного горного давления под влиянием целиков и краевых частей выработанных пространств определяли на основе результатов проведенных в бассейне исследования о распространении опорного горного давления. И именно в эти зоны попадают чаще всего откаточные штреки и сбойки над ними, являющиеся одними из наиболее опасными по выбросам. В связи с этим, при разнообразных условиях были приведены исследования для оценки выбросоопасности, путем измерения показателя V_1, количества мелочи при бурении скважин V_0 и других. В качестве примера на Рис.1 а,б показано изменение этих показателей в глубине предзабойного массива при проходке подготовительно-наре-


Показатель V_1, см³/10г
Количество уг.мелочи V_0, л/т
V_1
V_0
V_0'
V_1'
Глубина в массиве угля $\ell_м$, т

а)


Показатель V_1, см³/10г
Количество уг.мелочи V_0, л/т
V_1
V_0
V_0'
V_1'
Глубина в массиве угля $\ell_м$, т

б)


$V_1^{н'}$
$V_1^{м}$
$V_1^{н'}$
Расстояние по бостанию пласта ℓ, т
Показатель V_1, см³/10г

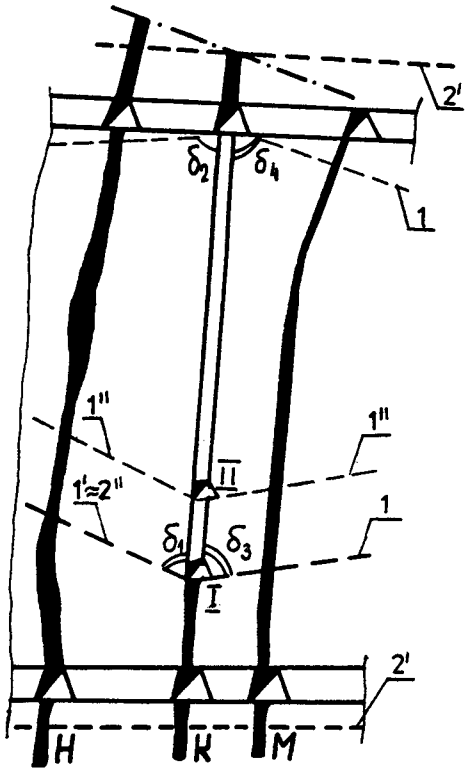

2'
δ_2
δ_4
1
1''
1''
II
1'≈2''
1
δ_1
δ_3
I
2'
Н
К
М

Рис. 1

зных выработок в незащищенных/V_1, V_0/ и защищенных зонах /V_1', V_0'/ соответственно на двух выбросоопасных пластах. Наблюдаются значимые отличия при защите и без ней, особенно для показателя V_1. Установлена и регламентирована возможность на основе измерения последнего уточнять границы защиты, включая опасного влияния целиков. На Рис.1в показан пример из практики в этом отношении. Опережающая выемка защитного пласта К обуславливает разгрузку и дегазацию расположенных в 6 и 10 m в почве и кровле пластов М и Н и соответственно снижение показателя V_1 под граничным значением выбросоопасности $V_1=3\,cm^3/10g$, в том числе и в зоне повышенного горного давления по падению и востанию от границ защиты 1-1, определенных по соответствующей схеме и углам сдвижения /Рис.1в -V_1^M и V_1^H/. Устанавливается возможность расширения защищенной зоны по данным измерения - прогноза-/2-2 -Рис.1в/. Выклинивание защитного пласта и необходимость оставления породного целека на около 15 по востанию от откаточного штрека вызывает измининие в границах защиты, определенных по схеме и прогнозом /1',2' -Рис.1 в /. Практическое применение более 10 лет оценки выбросоопасности измерением показателя V_1 в зонах повышенного горного давления показало высокую надежность- не было выбросов в неопасных по данным прогноза занах, где снимали мероприятия по предупреждению внезапных выбросов угля и газа.

3.4. Система с подъэтажным обрушением

Эта система нашла большое распространение в бассейне, ввиду ряда преимуществ при сложных условиях, и прежде всего гибкости. Из недостатков отметим большой удельный вес подготовительно-нарезных выработок, что весьма неблагоприятно в отношении склонных к выбросам пластов, т.к. подъэтажи приходится проходить в зоне повышенного горного давления под влиянием выемочных полей по востанию и простиранию. В нача - льном варианте длина выемочных блоков составляла 30 m по про - стиранию пластов, спредварительной нарезкой подъэтажей через 6-10 m по падению. В целях совершенствования системы провелись эксперименты при длине блока 100-120 m и подъэтажей соответственно 50-60 m и расстояние между ними 5-6 m, в том числе и на склонных к выбросам пластах. При этом выполнен комплекс исследований для оценки выбросоопасности при проходке подъэтажей в зоне повышенного горного давления. Результаты показали, что при этом скорость сближений боковых вмещающих пород в 2-3 раза больше по сравнению с отка - точным штреком, при близким другим условиях. Это можно рассматривать в смысле увеличения в 2-3 раза деффформации угольного пласта под влиянием повышенного горного давления. При систематических исследованиях при проходке штрека установлены значения показателя V_1 в пределах от 0,8 до 2,2 $cm^3/10g$, что в 2-3 раза м еньше соответствующих при проходке откаточного штрека вне заны влияния опорного давления очистных работ. Измерение стоимости показателя V_1 в подъэтажных штреках меньше граничного опасного значения, что означает разгрузка и дегозация пласта в безопасной степени. Это потвердилось и небольшими значениями измеренного газовызеления из герметизированных шпуров - от 0,1 до 0,3 l/min, без особых изменений до глубины 5-6 m. При проходке подъэтажей, в том числе и при выемке угля, измерено небольшое газовыделение и абсолютную газообильность от 0,1 до 0,3 m^3/min. Рассмотренные выше результаты получены на основе исследований при проходке более 10 подъэтажей в выемочном поле дленой 100 m и высотой 50 m. Следует отметить, что в вверхней части блока имеется частичная защита от подработки соседним пластом. При данных исследованиях не был

особо отмечен переход границы и работа в зоне повышенного горного давления под влиянием краевой части защитного пласта. Обратим внимание,что район исследования характеризуется значительной тектонической нарушенностью,значительным изменением мощности пласта:от 0,1-1,0m на уровне откаточного штрека до 4-10m в середине поля по востанию и снова уменьшение до выклинивание ниже вентиляционного горизонта. Отмеченный внезапный выброс при проходке ската-углеспуска в середидине блока,где проводились исследования, а также наблюдавнные предупредительные признаки при проходке штрека, указывают на выбросоопасную зонну в районе исследования.

Таким образом,результаты исследования показали,что при проходке подъэтажных штреков в зоне повышенного горного давления под влиянием верхнего обрушенного поля, при расстоянии до него не более 5-6m ,не устанавливается повышенной опасности от выбросов,а наоборот - наблюдается благоприятное влияние горного давления,обуславливающее разгрузку и дегазацию угольного пласта в такой степени, что по данным прогноза возможно отменить мероприятия по предупреждению внезапных выбросов. На основе результатов исслидования разработаны требования и рекомендации по безопасному ведению горных работ на склонных к выбросам пластах, при системе с подъэтажным обрушением при длине блока 100-120m и соответственно подъэтажей 50 - 60m .Они относятся к: порядку нарезки и обрушения подъэтажей, проветриванию, оценке выбросоопасности и возможности снятия мероприятий по предупреждению выбросов и т.д. Не будем их рассматривать,т.к. они специфичны для бассейна.Отметим только,с их использованием на основе результатов контрольных исследований,расширилось применение рассмотренного усовершенствованного варианта системы с подъэтажным обрушением на всех шахтах бассейна,в том числе и на склонных к выбросам пластах.Не отмечены выбросы при ведении работ согласно требований,хотя в большинстве случаев отменяли мероприятий по данным прогноза. Однако, при увеличении высоты обрушаемого целика от 5-6 до 10m и больше, выработка попадает в зону неблагоприятного воздействия повышенного горного давления,нет разгрузки и дегазации пласта и весьма часто наблюдаются предупредительные признаки возникновения внезапных выбросов угля и газа,а в двух случаях были вызваны такие сотрясательным взрыванием. На Рис.2 в качестве примера показана выемка с подъэтажным обрушением по весьма опасному пласту.По результатам измерений показателя V_1 в целях оценки выбросоопасности, построены граффики изменений значений по падению пласта в трех местах.При высоте обрушаемого целика до 5-6m ,угольный пласт разгружен и дегазирован в безопасной степени,показатель V_1 ниже граничного значения выбросоопасности и работают без мероприятий.С увеличением высоты до 6-7m ,стоимости показателя V_1 начинают возрастать и при 10-12m превышают граничного значения,что обуславливает необходимость возобновить мероприятий по борьбе с выбросами - Е6, Е6, Е5, Е2 - рис.2.На разрезе б-б -Е5 наблюдается разгрузочно-дегазирующего влияния каверны от выброса,которое наблюдается на расстоянии от ее контура до 7-10m . Этим примером хорошо иллюстрируется надеждность оценки опасности от внезапных выбросов в зонах повышенного давления,имея ввиду высокую естественную выбросоопасность в районе блока: в предидущих годах регистрировано 8 внезапных выбросов,при удельной частоте 0,6-0,7 случаев на 1000 m^2 отработанной площади или более 1000 на 1mil t добытого угля от пласта, если пересчитат. Результаты проведенных исследо-

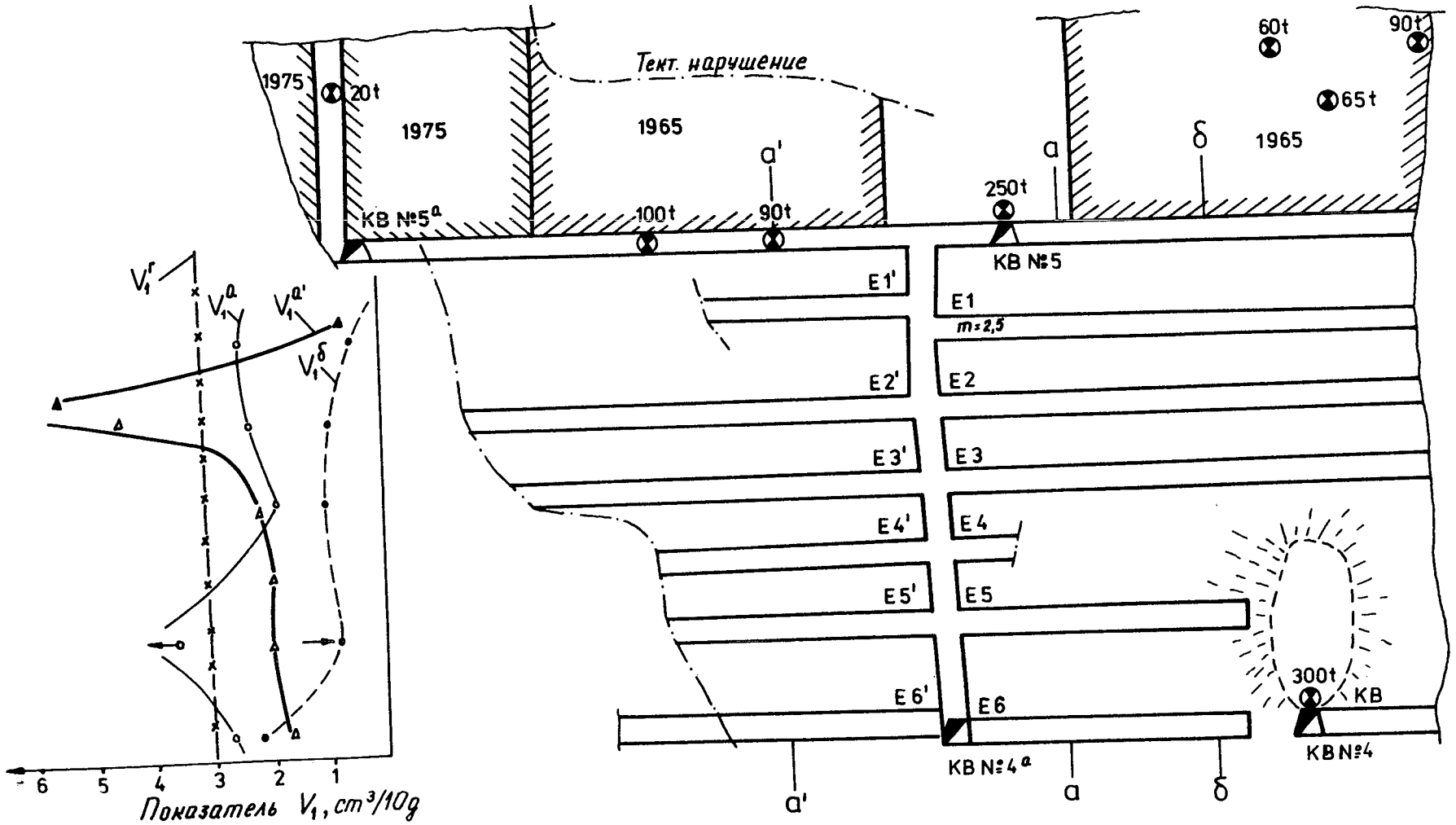

Тект. нарушение
1975
20t
1975
1965
1965
60t
90t
65t
250t
100t
90t
КВ №5а
КВ №5
E1'
E1
m=2,5
E2'
E2
E3'
E3
E4'
E4
E5'
E5
E6'
E6
300t
КВ
КВ №4
КВ №4а
а'
а
б
$V_1^г$
$V_1^а$
$V_1^{а'}$
$V_1^б$
6 5 4 3 2 1
Показатель V_1, см³/10g

Рис.2

ваний весьма полезны для практики, ввиду создания на их основе предпосылок для безопасного и эффективного применения сестемы с подъэтажным обрушением на склонных к выбросам пластах. Возможность снятия мероприятий по предупреждению внезапных выбросов и других режимных требований, позволяет снизить расходы на проведение подъэтажей на 15-30%, а также увеличить добычу угля в участке на около 5-7,5 %.

3.5. Система "лава-этаж"

Система разработки известна и кратко отметим некоторые особенности. Выемка угля проводится одним уступом, отбойными молотками или буро-взрывными работами. Крепление и управление кровлей: деревянные стойки и костры, плавное опускание, обрушение, иногда сухая закладка полосами параллельно линии забоя.

На основе результатов исследования проявлений горного давления в выработанном пространстве и впереди забоя лавы, а также напряженного газодинамического состояния предзабойного массива, разработаны рекомендации, относительно: числа и глубины рабочего - добычного и магазинного уступов, крепления и управления кровлей, с допустимым расстояниями отставания крепи, закладки и т.д. При соблюдении требований-рекомендаций, способствующих снижению выбросоопасности, допускается вести выемку при системе "лава-этаж" на склонных к выбросам пластах без применения мероприятий для разгрузки и дегазации пластов. Это допускается на основе результатов представительных исследований при разных условиях, которыми установлено, что в условиях бассейна в большинстве случаев предзабойный угольный массив, перед лавой является разгруженным и дегазированным в безопасной степени относительно возникновения выбросов на глубину не менее 4-5m . Это получается в связи небольшой скорости подвигания лавы, небольшой крепости угля и т.д. На Рис.3а показан пример в этом отношении. Повышение скорости сближений вмещайших пород соответствует повышению деффорации пласта в сторону забоя, т.е. снижению нагрузки и пластового газового давления. Отметим, что результаты исследования показали, что по характеру и величине выдавливание угля на глубину 1m перед забоем выработки соответствует почти полностью скорости сближения боковых пород на 1-3m от забоя. Практическим опытом и исследованиями установлено, что иногда наблюдаются случаи недостаточной разгрузки и дегазации предзабойного массива угля. Например, когда из-за недостаточно своевременного обрушения кровли она зависает на значительном расстоянии в выработанном пространстве. На Рис.3 показан такой пример для угрожаемого по выбросам пласта, когда при работе в лаве отмечены предупредительные признаки и произошло высыпание угля с повышенным газовыделением. Замерами показателя V_1 и количество угольной мелочи V_0 установлено, что угольный пласт на 2m перед забоем неразгружен /$V_1^н$, $V_0^н$ -Рис.3б/. Разгрузку и дегазацию пласта выполнили скважинами Ø120mm /$V_1^р$, $V_0^р$/. Повышение горного давления наблюдается и в связи с отставанием, крепи или закладки/Рис.3в V_1' /на расстоянии больше рекомендованных значений. С углублением горных работ проводятся контрольные исследования для проверки возможности работы в лавах без мероприятий для разгрузки и дегазации пластов. Разработаны также методы контроля выбросоопасности на основе измерения показателя V_1 и количество буровой мелочи V_0. Многолетняя практика в бассейне потвердила надежность разработанных мероприятий при их соблюдений не отмечено выбросов.

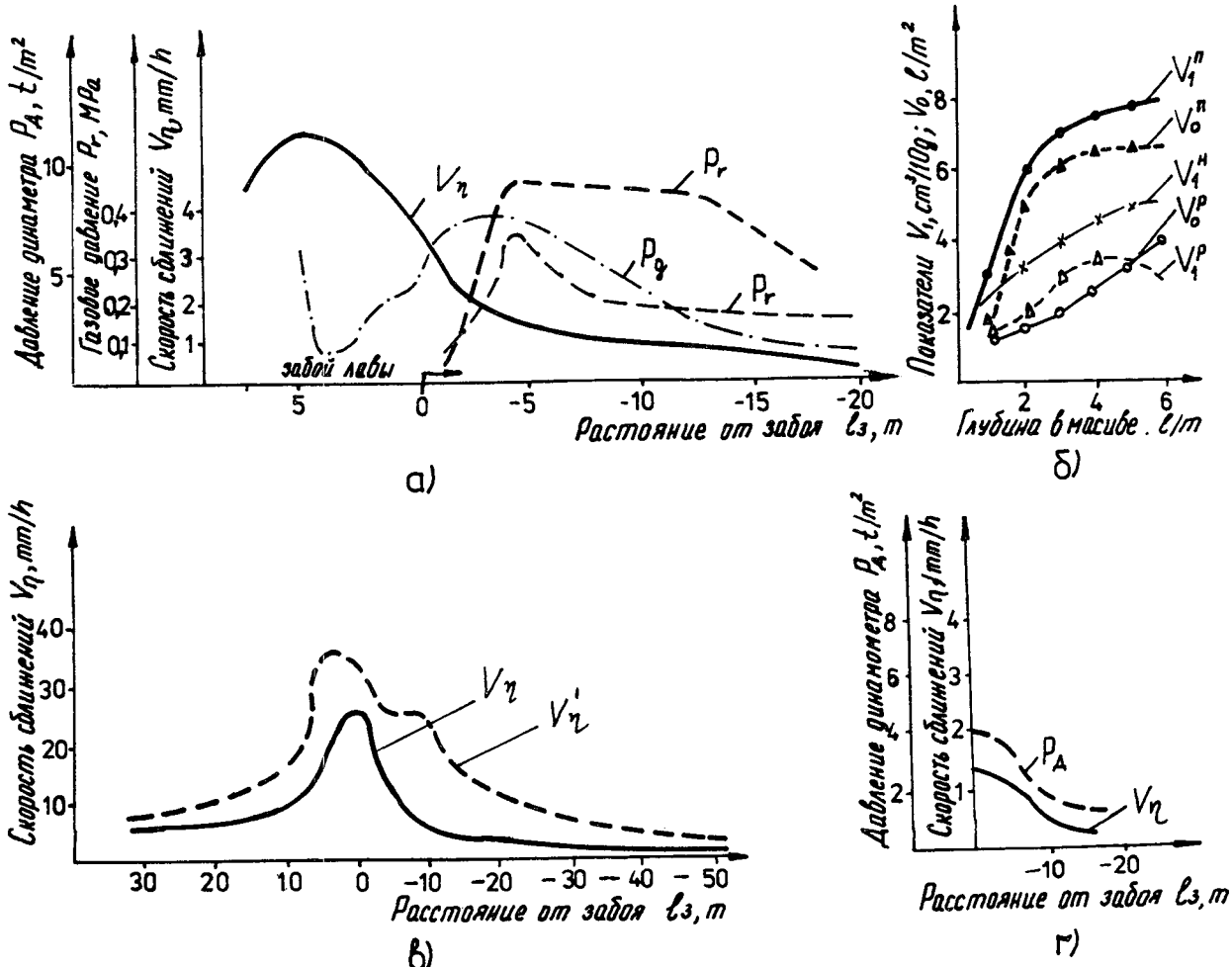

Давление динамометра P_A, t/m²
Газовое давление P_r, MPa
Скорость сближений V_η, mm/h
V_η
P_r
P_g
забой лавы
Растояние от забоя $\ell_з$, m
а)
Показатели V_1, cm³/10g; V_o, l/m²
$V_1^п$
$V_o^п$
$V_1^н$
$V_o^р$
$V_1^р$
Глубина в массиве . l/m
б)
Скорость сближений V_η, mm/h
V_η
V'_η
Расстояние от забоя $\ell_з$, m
в)
Давление динамометра P_A, t/m²
Скорость сближений V_η, mm/h
P_A
V_η
Расстояние от забоя $\ell_з$, m
г)

Рис.3

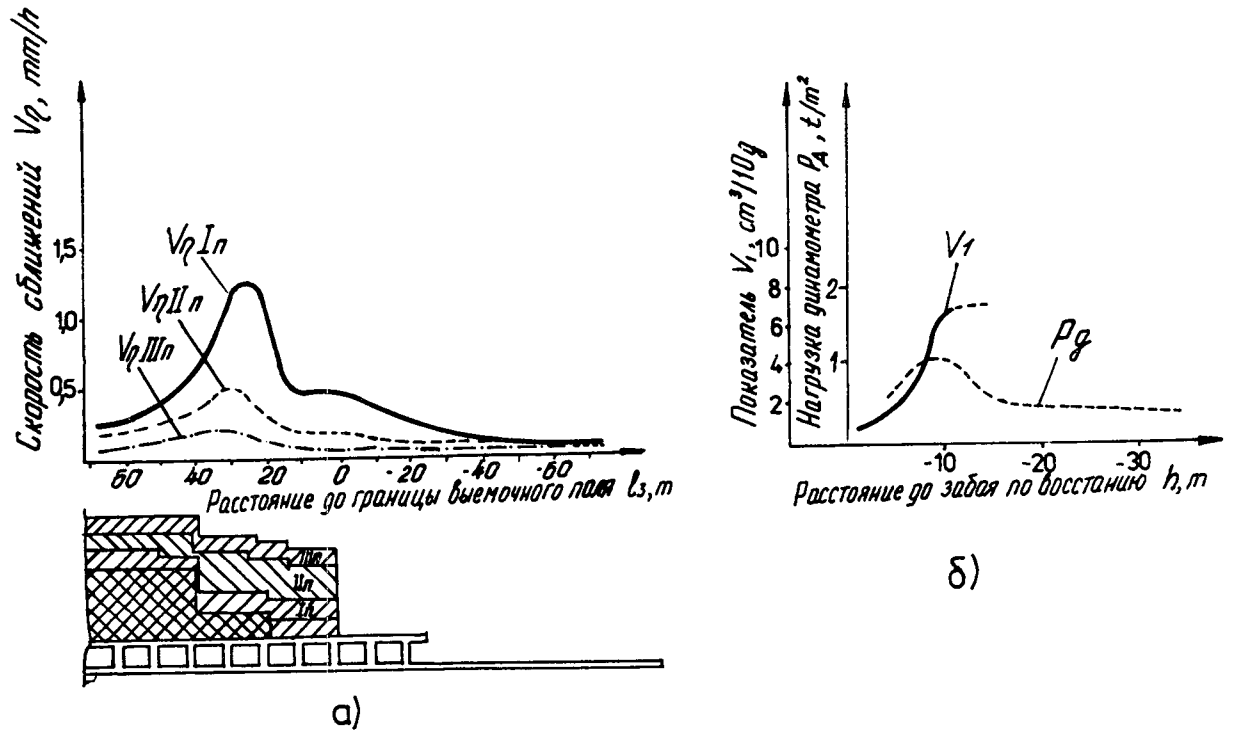

Скорость сближений V_η, mm/h
V_η I п
V_η II п
V_η III п
Расстояние до границы выемочного поля $\ell_з$, m
а)
Показатель V_1, cm³/10g
Нагрузка динамометра P_A, t/m²
V_1
P_g
Расстояние до забоя по восстанию h, m
б)

Рис.4

3.6. Система с горизонтальными слоями по востанию

Эта система применяется редки, ввиду небольшой производительности.Выемка угля проводится отбойными молотками или буровзрывными работами, полосами по $2m$,с последующей закладкой выработанного пространства породой из включений в пласте или почвы и кровли. Как и при системе "лава -этаж", устанавливается разгрузка и дегазация угольного пласта перед забоем и по востанию,даже в большей степени,т.к.собственный вес способствует этому. На Рис.4 а,б показано соответственно распространение опорного горного давления при выемке отдельных полос и изменение показателя V_1 и нагрузка на пласт по востанию. Для этой системы также разработаны мероприятия,параметры,требования и рекомендации,способствующие снижению выбросоопасности на склонных к выбросам пластах. При этом допускается работать мероприятий по предупреждению выбросов, а такие применяют при необходимасти по ваным прогноза.

4. ЗАКЛЮЧЕНИЕ

Проведенные исследования о проявлениях горного давления при очистных и подготовительных выработок совместно с характеристиками о напряженном газодинамическом состоянии предзабойного угольного массива позволили получить весьма интересные и полезные результаты. На основе этих результатов разработаны и соблюдаются ряд мероприятий,рекомендации,параметров и требований при ведении очистной выемки и других горных работ. При этом создаются предпосылки для использования благоприятно влияния повышенного опорного давления для разгрузки и дегазации предзабойного угольного массива и допускать отмены мероприятий по предупреждению внезапных выбросов угля и газа. При таком подходе работы не были отмечены до сих пор выбосы, а то же время улучшаются технико-экономически е показатели работы.Таким образом исследования и практический опыт показали целесеобразность и большую полезность оценки опасности от выбросов в зонах влияния повышенного горного давления.

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ESTIMATING STRESS-INDUCED GAS AND ROCK OUTBURST HAZARDS

S. Nanovska, H. Betchev, and I. Tcanev

Research to estimate gas and rock outburst hazards under rock stress is based on measurement of the gas desorption rate index V_1, gas pressure, adjacent rock convergence and others. Results indicate the possibility of promoting stress relief and gas outflow through preventative measures. Rules and instructions for mining in hazardous strata are developed. Practical implementation of the results leads to effective gas and rock outburst prevention and to reduction of mining costs.

EVALUATION DES RISQUES DE DEGAGEMENTS DE GAZ ET DE ROCHES DUS AUX CONTRAINTES

S. Nanovska, Tcv. Voutov, Chr. Betchev, Iv. Tcanev

Les recherches sur l'évaluation des risques de dégagements de gaz et de roches dûs aux contraintes des roches sont basées sur les mesures de l'indice V_1 du taux de désorption des gaz, sur la pression des gaz, la convergence des roches voisines, et autres. Les résultats présentés examinent la possibilité d'exclure le soulagement des contraintes et le dégagement des gaz à la suite des mesures de prévention. Des règles sont développées, ainsi que des instructions d'extraction dans les strates présentant des risques. L'application des résultats permet une prévention effective des dégagements de gaz et de roches, ainsi qu'une réduction des coûts d'extraction.

GEFAHRENABSCHÄTZUNG VON SPANNUNGSBEEINFLUßTEN GAS- UND GESTEINSAUSBRÜCHEN.

S. Nanovska, Tcv. Voutov, Chr. Btechev, Iv. Tcanev

Untersuchungen zur Gefahrenabschätzung von spannungs-beeinflußten Gas- und Gesteinsausbrüchen beruhen auf der Messung der Ausgangsrate V_1, des Gasdrucks, der Konvergenz des umliegenden Gebirges und anderer Parameter. Die vorgestellten Ergebnisse beschreiben die Möglichkeit, Entspannung und Gasausfluß von den Verhütungsmaßnahmen zu eliminieren. Vorgehensweisen und Vorschriften zum Abbau in gefährdeten Gebieten werden entwickelt. Praktische Anwendungen der Ergebnisse führen zu einer effektiven Verhütung von Gas- und Gesteinsausbrüchen und damit zur Senkung der Gewinnungskosten.

对由应力引起的气体和岩石突出危险性的估计

S．拿那夫士卡，Tcv．夫托夫

Chr．贝柴夫，Iv．察涅夫

估计由于岩石应力影响所引起的气体和岩石突出危险的研究是根据下列几条：气体解吸率指数V1的测定，气体压力，相邻岩石的汇聚等等．本文指出应力消除和气体外流的可能性，以作预防的方法．文中列出有危险性的煤层中开采时应遵照的条例及方法．这些结果的现场应用，能有效地防止气体和岩石的突出，并可降低开采成本．

MICROSEISMIC MONITORING IN COAL

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ABSTRACT

The U.S. Bureau of Mines has actively conducted microseismic research in coal mines for many years. The purpose of this research is to provide the mining industry with reliable techniques to warn of imminent coal mine bumps and outbursts so that miners may evacuate dangerous areas prior to ground failure. In this paper, the Bureau's research program in microseismic monitoring is discussed, including recent achievements of microseismic techniques applied to recognizing impending catastrophic ground failure in coal mines and the development of an automatic digital microseismic monitoring system. Event source location accuracy in coal is examined providing recommendations for future research to improve source locations.

INTRODUCTION

A coal mine bump is a sudden and sometimes catastrophic ground failure within the mine structure. This type of rapid failure is capable of causing multiple injuries or fatalities in addition to damaging a working section in a mine. An outburst is a sudden release of energy accompanied by a large volume of gas, usually methane, and coal dust. In addition to the potential for worker injury from flying rock, outbursts can leave an explosive mixture of methane, air, and coal dust. For example, on April 15, 1981, 15 miners died as a result of an explosion after an outburst in Dutch Creek #1 Mine near Redstone, Colorado (Elam et al. 1981).

Literally, a microseism is a "small earthquake." The word is derived from the Greek words "micros," meaning small, and "seismos," meaning earthquake. As detected in a mine, a microseism is sometimes an audible, sometimes subaudible, noise resulting from vibrations introduced into the mine structure by rock fracturing or slipping. There is no strict magnitude definition of a microseism. On the lower end of the scale, a practical limit is the sensitivity of the actual detection system installed; on the upper end, most all coal mine related bumps can be included because they are considered small by earthquake standards. However, some large mining induced earthquakes have been reported with Richter magnitudes greater than 5 (Mendecki et al. 1988).

Microseisms in coal mines are detected by securely attaching an array of velocity gages to the rock surrounding the area of interest, usually near the face. Since these transducers are "listening" for rock vibrations, they are generically called "geophones." When a rock noise, or microseism, is detected by several geophones, the origin of the noise can be calculated from the relative arrival times of the signal at the different geophones.

Vibrations travel through the mine at the speed of sound in rock, which usually varies along the path of the sound wave. As an example, if a geophone is 30 m farther from a

microseismic source than the closest geophone, the signal will arrive at the more distant geophone 10 ms later than at the closest geophone, assuming a 3000 m/s sound velocity. These time differences are used to calculate source locations that best fit the data.

Microseismic research was initiated in the late 1930's by Leonard Obert of the U.S. Bureau of Mines. It was an outgrowth of exploratory investigations to devise a method for determining the pressures on rock columns and arches in mines. One approach suggested to solve this problem was based on the possibility that as pressures develop on rocks there may be microscopic readjustment that would produce sound vibrations capable of detection if sufficiently amplified (Obert 1939). The initial field testing of microseismics, initially called "subaudible noises," was reported by Obert in 1941. Over the last 5 decades, instrumentation, recording capability, and understanding of microseismic techniques have advanced significantly.

Since sources of microseismic activity are caused by rock fracturing or slippage along planes of weakness within the rock mass, locating and plotting sources of microseismic activity can point to areas of a mine adjusting to changing stresses induced by mining. Microseismic techniques have been used to delineate potential areas of ground failure. If a dangerous area can be identified days in advance of a failure, then there may be sufficient time to modify mining plans to prevent the failure. If a failure cannot be prevented, then microseismics may be used to provide warning so that miners may move to safety before the failure.

The Denver Research Center of the U.S. Bureau of Mines had a microseismic monitoring project at the Dutch Creek Mine, near Redstone, Colorado (e.g. Leighton 1984). The mining method employed at the Dutch Creek Mine is advancing longwall, extracting a 3 m thick coal seam, dipping at 13° under nearly 900 m of overburden. A second coal mine is 120 m above the workings. Microseismic activity has been monitored at the Sunnyside Mine, Sunnyside, Utah (a retreating longwall under 400-750 m of overburden). Through a contract with the Colorado School of Mines, the Bureau monitored a mountain bump-prone area of the Olga Mine, Caretta, West Virginia (a retreating room-and-pillar mine, Descour and Miller 1988).

This paper outlines current microseismic systems available for coal mines, presents an example of monitored activity prior to a failure, and indicates areas of research necessary to fulfill the promises of microseismics.

INSTRUMENTATION

To successfully predict bumps, the ability to monitor and locate microseismic activity must be at least 3-4 orders of magnitude lower than that which needs to be predicted. Thus, to predict a catastrophic failure of Richter magnitude 2, microseismic activity whose magnitudes are as low as -2 must be successfully monitored and located. There are two reasons for looking at small events. First, Bureau experience has shown that major failures are usually preceded by an increase in the number and intensity of background microseismic events. Second, extrapolating a series of large events to predict a future large event is not very meaningful to the mining industry. Prediction must be based on observations of smaller events. The Bureau has, therefore, concentrated its microseismic research efforts on monitoring individual areas of mining with a local geophone network. A local network is designed to monitor a relatively small section of a mine, a section 100-200 m in extent. In a mine with several working sections, complete coverage requires several

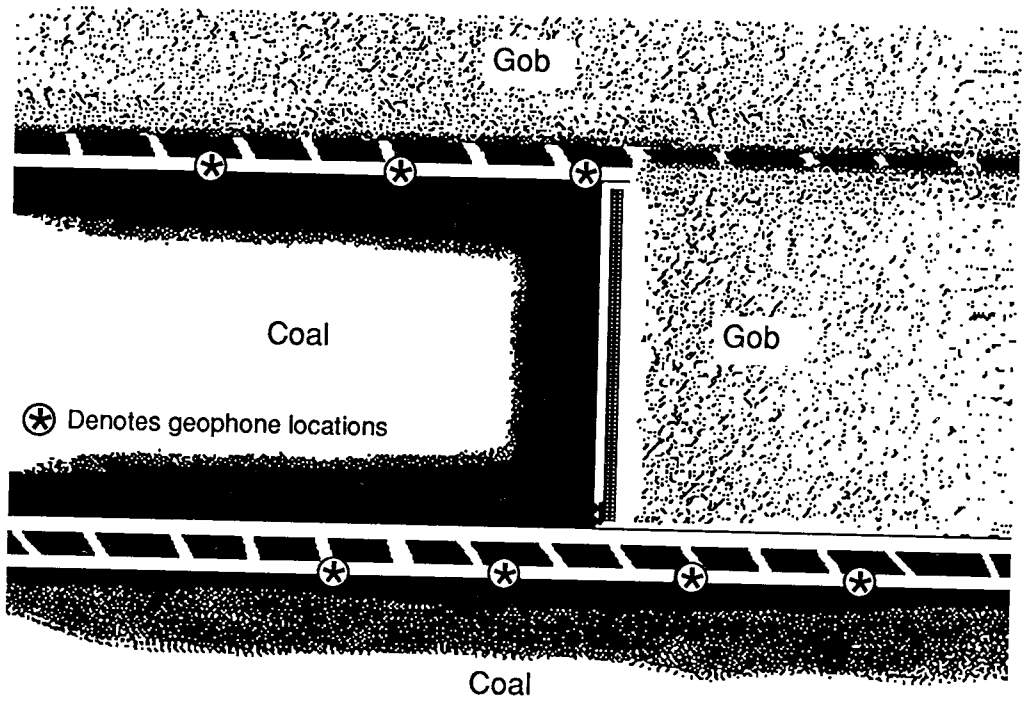


Figure 1. Plan view of a typical geophone layout for a longwall coal mine. Geophone spacing is about half the width of the longwall panel.

local systems which can be tied together such that data can be shared to make a mine-wide system.

A typical array at a longwall coal mine is two dimensional, consisting of at least seven geophones, Figure 1. A two dimensional array means that all geophones are located on a single plane, the plane of the coal seam. Through experience, it has been found that geophone spacing along the gate roads should be approximately half the width of the longwall face. More geophones closer together in the gate roads would slightly improve location accuracy, but at a considerable increase in cable cost. Actual geophone locations are subject to roof conditions and proximity to noisy equipment, such as pumps and transformers.

A three dimensional array has geophones placed significantly off the plane of the seam. The advantage of a three dimensional array is that accurate source locations can be determined in three dimensions. A two dimensional array allows good resolution of source locations on the plane but does not allow accurate resolution off the plane. For example, an array of geophones placed around a horizontal coal seam does not allow for distinguishing whether the event took place in the floor, coal, or roof. In order to determine this information, additional geophones have to be mounted in the roof and floor.

MICROSEISMIC MONITORING

The basis of microseismic monitoring is the detection of signals and the location of their sources. Plots of event locations are superimposed on a mine map for given periods of time, (e.g. Kneisley 1989). Events in a given area are counted to produce rate plots. For example, the number of microseismic events per day may be plotted against the days of the month. An anomalous increase in the daily event rate may presage a bump. Sometimes contours are sketched to show areas of event concentration (Blake et al. 1974 and Leighton et al. 1974). However, mining progress must be considered because there is

usually more microseismic activity during coal extraction than during non-mining cycles. Data presentation and interpretation, an area of microseismics requiring more research, are the keys to predicting bumps.

Microseismic activity in coal mines has indicated something anomalous happening prior to several major bumps (Kneisley 1989 and Descour and Miller 1986). For example, at the Olga Mine, Caretta, W.V., an anomalous increase in microseismic activity was observed by the on-site observer prior to a massive series of bumps that occurred on Monday, June 3, 1985, destroying approximately 100 coal blocks. This increase in activity occurred even though mining was limited to pulling fenders which did not normally induce much microseismic activity. The buildup in microseismic activity began during late May. Unfortunately, after the tape ran out on Saturday, June 1, 1985, no more data were available until Monday after the tape was changed (Descour and Miller 1986).

The Bureau's microseismic monitoring system for coal mines has been based on a commercial analog tape recording unit that records up to 14 channels of data on a 7200 ft. (~2200 m) reel of magnetic tape, at 15/16 in/s (~24 mm/s), giving a total available recording time of 25+ hours. After data have been recorded on analog tape, the tape is played back on a second tape deck while new data are being recorded on the first. Under the best of circumstances, when the on-site investigator can immediately begin data analysis, it is still yesterday's data being analyzed. The 25 hour limitation of the tape deck has also proven to be a short-coming of the analog system. Without changing tapes every day, data are lost.

An automatic microseismic monitoring system has been available for hard rock mining for several years (Coughlin 1982). The Bureau has attempted to transfer the knowledge gained from automatic rock burst monitoring directly to coal with marginal success. Sources of strong microseismic events whose signals had sharp first breaks have been accurately located, automatically. The rest of the locatable events were not accurately located. The principle of operation of a rock burst monitor (RBM) is the determination of arrival times by detecting the times when the microseismic signal arriving at each geophone crosses a preset threshold. These relative arrival times are then sent to a computer for calculating sources. The signal wave forms are not recorded and there is no possibility of reconstructing signals to check arrival times. The only cross check of data comes from the attempt to calculate source locations that best fit the measured arrival times. If a reasonable source location cannot be calculated from the arrival time data, nothing further can be done. After the source location that best fits the measured arrival times has been found, relative arrival times at each geophone of a microseismic signal emanating from this source can be calculated. The reasonableness of a source location is determined by comparing arrival times calculated for the "best fit" source to the measured arrival times. If the comparison is good, the solution and data are assumed good. Otherwise, the raw data are considered suspect. Out of the comparison of measured vs. calculated arrival times, an estimate of the uncertainty of the solution is calculated and displayed.

Automatic systems in coal mines have failed because signal frequencies observed in coal mines are much lower and signal amplitude relative to noise is much lower than in hard rock, Figure 2. The major differences in signals observed, hardrock vs. coal, are due to several conditions: 1) hard rock is a better transmitting medium than coal and sedimentary materials typically found above and below coal beds; 2) since a coal seam is one of many layers, the resonant frequencies and absorptive characteristics of the

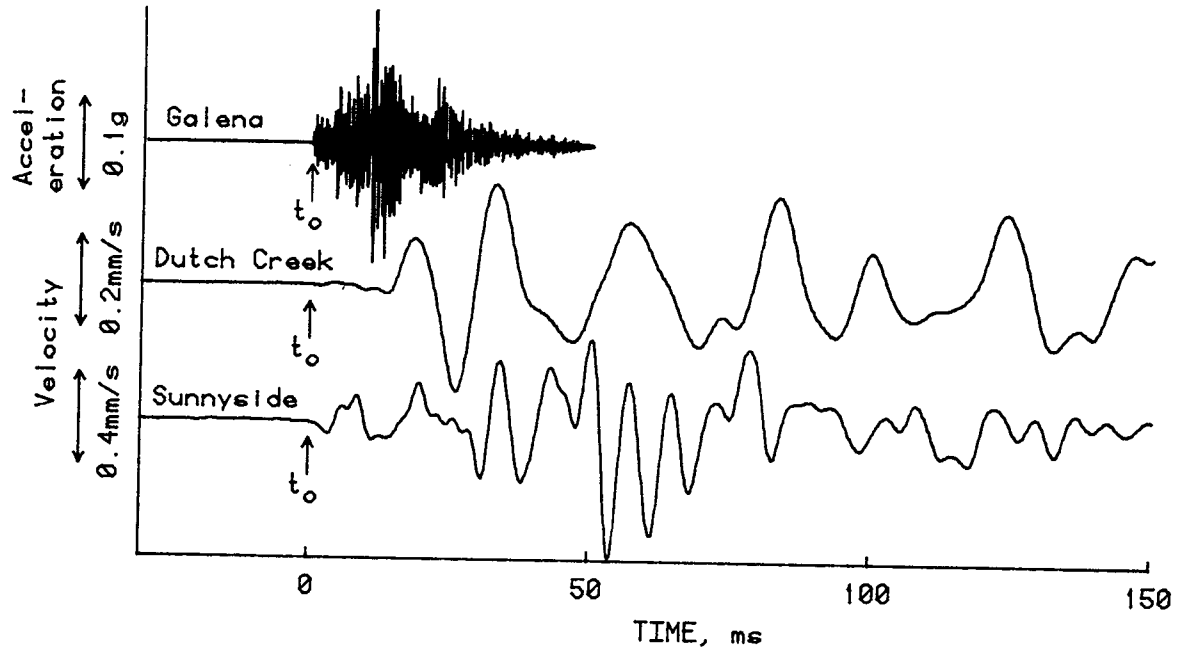


Figure 2. Comparison of typical microseismic signals from a hard rock mine (Galena Mine) and two coal mines. Microseismic signals from hard rock mines have much higher frequency content and are much shorter in duration than signals from coal mines.

layers probably dominate observed signal frequencies and amplitudes; 3) mine entries and working areas are generally much larger in coal mines than in hard rock mines resulting in lower resonant frequencies; 4) in coal mines, geophones are farther apart and generally farther from signal sources allowing greater attenuation of higher frequencies; and 5) a typical microseismic event detected in a coal mine is larger than a typical event detected in a hard rock mine.

The order of magnitude differences in signal frequency and duration between coal and hard rock (Figure 2) have contributed to the success and failure of automatic monitoring systems in each type of mine. Because signal frequencies and amplitudes are low in coal mines, automatic first break picking has not been sufficiently accurate. When using a simple threshold picking routine, the first breaks of the closest geophones may be picked accurately while the first break picks on more distant geophones may miss the first cycle and catch the 2nd or 3rd cycle or even miss the P-wave altogether and pick a point on the shear wave.

The location error resulting from missing the first or second cycle is worse in coal than in hard rock because the frequencies are much lower. When the first cycle of a 2 kHz signal is missed in hard rock because its amplitude is too low, the resulting error in time (0.5 ms) caused by picking the second peak corresponds to a distance error of less than 3 m (sound velocity ~5600 m/s). When the first cycle of a 50 Hz signal in coal is missed, the time error of 20 ms corresponds to a distance error of 60 m (sound velocity ~3000 m/s) resulting in an unacceptably poor location. Final location errors are usually less than those just cited because of the error averaging characteristics of location routines. However, when the first cycle of a microseismic signal is detected from some geophones and the second cycle from other geophones, the calculated source location can be

meaningless in coal, but still usable in hard rock. When an automatic system is able to detect the first break of corresponding cycles in the signals received from each geophone in a coal mine, it does very well in calculating accurate source locations.

The approach to minimizing the problem of first break picking errors has been to judiciously adjust signal amplification and detection threshold so that the threshold is barely above the noise level of the ambient signal. Any signal above the noise level is then detected and picked. This approach has been effective in hard rock mines, but not in coal. Figures 3 and 4, showing two microseismic signals recorded about two hours apart at Sunnyside Mine, illustrate the problem. Figure 3A shows a signal recorded while the shear was not operating; Figure 3B keeps the same time scale but is expanded vertically by a factor of twelve to show details of the signal near its first breaks. In this example, the relative differences between hand picked arrival times and threshold picked arrival times ranges up to 9 ms (geophone 1). The difference in calculated source locations was 14 m. Using hand picked arrival times, the source was calculated to have been in the coal ahead of the face; using threshold picks, the source was calculated to have been in the gob. With errors of this magnitude, conclusions drawn from microseismic data might be suspect.

Based on Figure 3, one might conclude that location accuracy might be improved by tightening the threshold. The dilemma is illustrated in Figure 4 which shows a microseismic signal recorded while the longwall shear was operating. Any reduced threshold would be continuously surpassed by the noise, as is the case with geophone 6. Thus microseismic source location would be impossible while the shear was operating. With the threshold set as indicated in Figure 4, the difference between locations calculated using hand picked arrival times vs. threshold picked arrival times (ignoring geophone 6) is about 10 m, but the calculated uncertainty in location is double that estimated when the shear was not operating. In comparing Figures 3 and 4, it is apparent that there is no correct threshold adjustment level for coal. Because of this, simple automatic detection devices have had only limited success in automatically locating sources of microseismic activity in coal. The automatic system used at the Olga Mine did prove useful in indicating the amount of microseismic activity and the times of event occurrences; it was only location accuracy that could not be trusted.

AUTOMATIC MICROSEISMIC MONITORING SYSTEM (AMMS)

Analog and digital recording of microseismic data have advantages and disadvantages. The advantage of an analog tape recording system is that all data are recorded on the tape and can be played back at any time. No signals are missed as long as the tape is running. Disadvantages of a tape system include: the tape must be changed before it runs out (data gathered at coal mines has had gaps over weekends and vacations because man power was not available to change tapes), data storage on magnetic tape is expensive (over $100 per reel which lasts one day), retrieving data is expensive (the linagraph print paper used with the visicorder usually costs over $50 per day), time consuming (one person would have trouble retrieving and analyzing more than 25 events every day), data analysis takes place a day late, at best, and finally, analog tape noise (hiss) is added to the signal, which lowers the signal to noise ratio making accurate first break picking more difficult.

The current rock burst monitor (RBM) used at the Galena Mine has several advantages over a tape system. Data reduction is fast (less than a second between signal occurrence and completion of source location

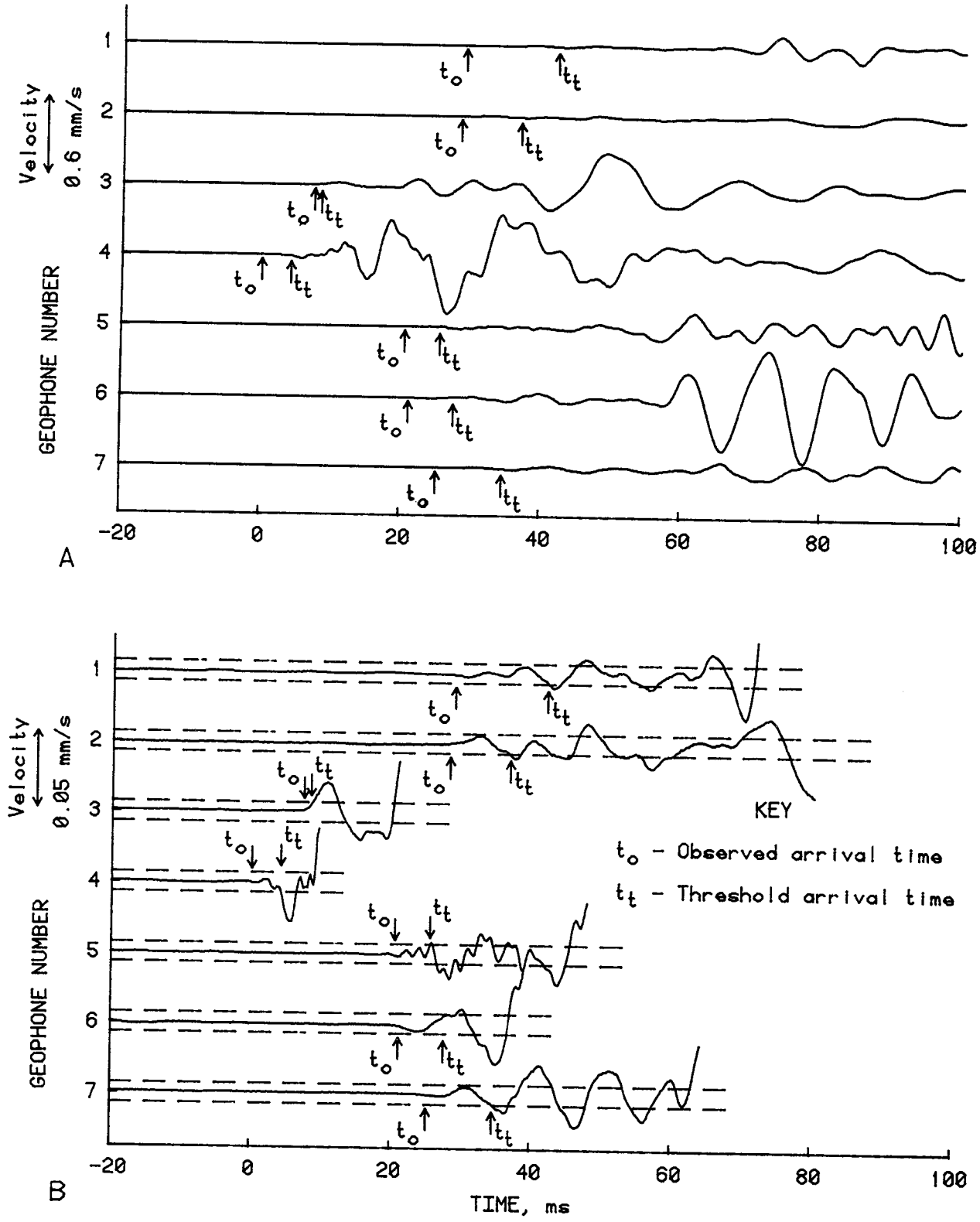


Figure 3. Vibrations from a background microseismic event at Sunnyside Mine while the shear was not operating. Dashed lines indicate possible rock burst monitor thresholds.

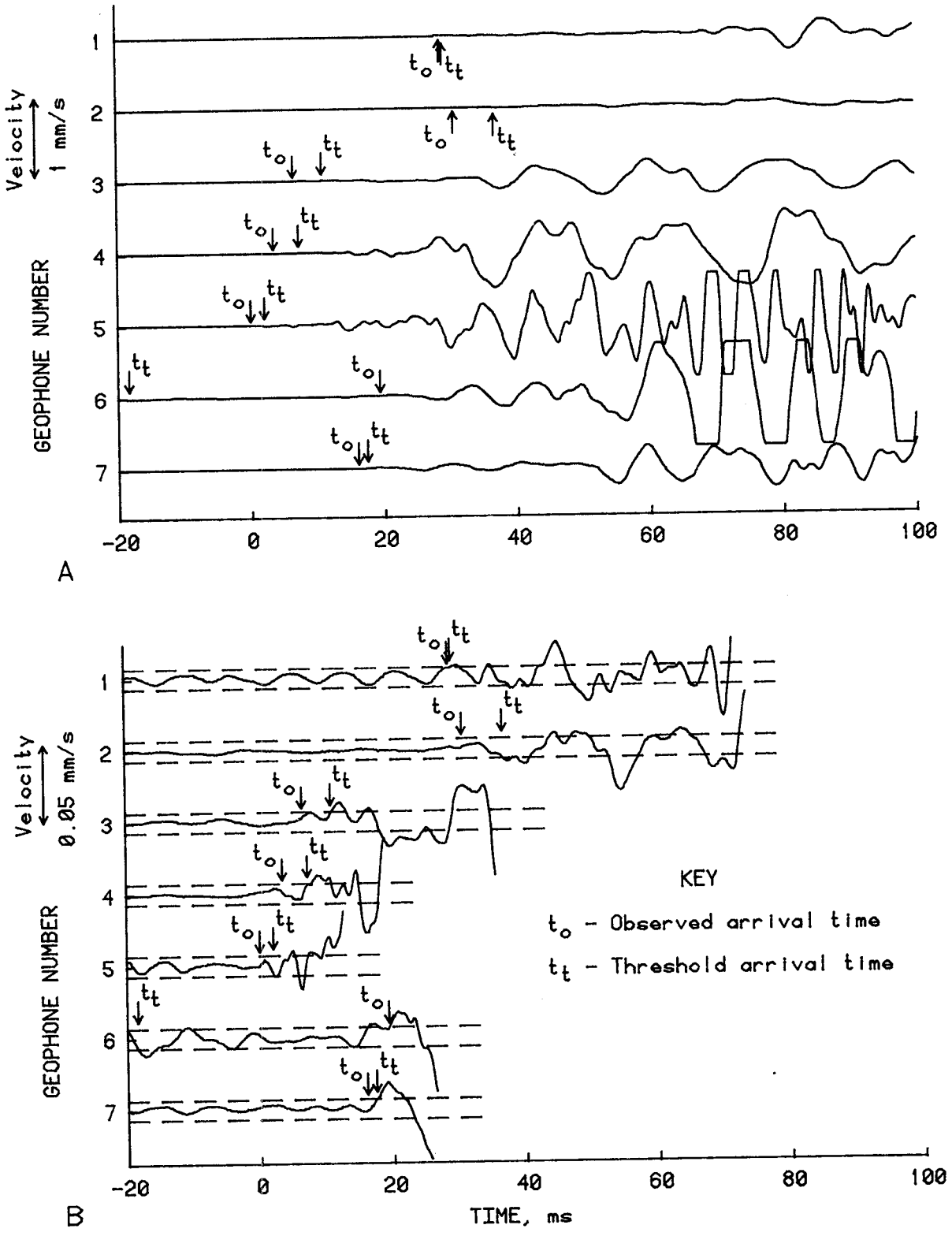


Figure 4. Microseism recorded at Sunnyside Mine two hours later than the signal shown in Figure 3. The longwall shear was operating which caused increased background noise clearly visible in the 20X vertically expanded figure, part B.

calculation) and automatic. Data storage is cheap; an average week of data can be stored on 1 or 2 floppy disks. The system can run unattended and is fairly simple to use. With current hard disk storage capacity, the system can run over a long weekend without losing data. The primary disadvantage of the RBM is that only arrival times and an approximate energy are retained, and this information cannot be double checked. All other signal characteristics are lost.

A full digital system will record the complete microseismic signal from each geophone for automatic and immediate analysis, retaining the advantages of both the analog and RBM systems. By eliminating periods of no data and judicious elimination of meaningless data, storage of digitized signals can be kept reasonable.

Since standard RBM systems have had only limited success in coal mines, the analog tape system, with all its disadvantages, has been the standard for microseismic monitoring in coal mines. The necessity of having an automatic and continuous microseismic monitoring system in coal has been apparent for some time. Development of an AMMS for coal is currently a high priority. There are two goals of an automatic system. First, the source location and signal characteristics of a microseism should be automatically and accurately calculated and displayed within seconds of an event's occurrence. Second, the system must be able to run for extended periods of time, unattended. In order to accomplish these goals, the Bureau has been investigating full digital data acquisition systems, in which the computer receives, from each geophone, a complete voltage vs. time history. The computer can then look at each time history to determine relative arrival times of a signal at each geophone. If the algorithm for determining relative arrival times is good enough, accurate source locations will follow. Developing such an algorithm will require extensive field trials on real microseismic signals.

To further the objective of automating microseismic monitoring in coal, two commercial computer data acquisition systems were purchased for testing and evaluation. The MASSCOMP System 5450 (now Concurrent Computer Corporation - reference to a specific trade name is made for identification only and does not imply endorsement by the Bureau of Mines) was chosen for further development because it has more computer power and is about half the cost of the other. It is capable of simultaneously digitizing signals from 16 geophones at sampling rates of 20,000 samples/s, 12 bit analog to digital conversion resolution. The data can be automatically recorded, stored, analyzed, and displayed in a multitasking mode of operation. This unit was field tested during the summer of 1988 at the Galena Mine and will be placed in operation at a coal mine during 1989. This type of system is expected to become the standard for coal mine microseismic monitoring, relegating the analog system to the roll of back-up to be used mainly during anticipated high activity periods, such as destress volley firing.

LOCATION OF RECORDING ROOM

There are three general locations for a central recording room: in the mine, on the surface at the mine site, and remote from the mine. The Bureau has engaged in microseismic research using underground enclosures and surface trailers for the central recording room, the work station of the on-site investigator.

In a coal mine, the recording equipment must be in fresh air; the geophone array must be permissible. In any mine, the recording equipment must be far enough away from active mining to minimize the chance of equipment damage from ground motion caused by a large bump.

The advantages of an underground recording room are routed in its

close proximity to the area of mining being monitored. First, cable runs are shorter and less expensive to install. Second, it is quicker to find and repair a geophone or cable problem in a short cable run than a long cable run. Third, the time involved in getting to a problem, repairing it, and getting back to the recording room can be excessive when mine entrance and exit are dictated by man trip timing as compared to the on-site investigator already being underground for the shift. Fourth, a better rapport with the miners is established by being underground. Fifth, it is easier for the observer to keep track of the mining sequence and progress when underground every day. Sixth, the data are readily available to the underground boss.

There are two advantages of a central recording room on the surface. Access to equipment is not hindered by temporary mine closures and some people consider it safer, for both equipment and personnel, outside a mine than inside.

The only reasons for having the central recording area remote from the mine site are when several mines are being monitored at one location and/or no qualified engineer or scientist is willing to work at the mine site. The more remote the observer is from the microseismic activity, the less complete his knowledge will be of what is really going on in the mine and the less likely the project will succeed. Modern telecommunications are marvelous for transmitting data, but no replacement for on-site observation and contact with the mining situation. In research, the post analysis of data can be done anywhere, anytime, but data analysis loses its immediacy as the observer gets farther away from the mine.

LOCATION ACCURACY

The location accuracy of events from within a geophone array, after obtaining arrival times from real data obtained from geophones placed in a mine, is about 10% of the geophone spacing. In a longwall mine with a 200 m face and geophone spacing of about 100 m, the expected uncertainty in microseismic source locations is about 10 m, for strong events with easily discernable arrival times. Location accuracy for weak events and those from outside an array is worse. Even with accurate arrival times, there are many potential errors that creep into source location calculations. Many articles have been written on the subject of location algorithms for earthquakes, microearthquakes, and microseismic events. Prugger and Gendzwill (1988) give a bibliography of many of the papers written on the subject. A series of assumptions is usually made to solve for source locations in a reasonable time with known information. Usually, it is assumed that the acoustic wave travels from a point source to the geophone in a straight line, at an average velocity. For lack of better information, the velocity is usually assumed to be the same from the source to each geophone. Sometimes the velocity is given and sometimes it is calculated by the algorithm to give a best fit.

Solutions can be refined using results from a velocity survey. In practice, this is seldom done after the initial survey because the extra effort is time consuming and the new information usually does not improve accuracy enough to make the effort worthwhile. The assumption of straight line travel of the signal from source to geophone is very optimistic. The fastest travel path from source to geophone is most likely not a straight line, especially in a coal mine where the fastest path will usually be through a sandstone layer above or below the coal seam.

Various schemes to calculate location errors based upon errors in the assumed velocity and measured arrival times have been used to optimize geophone arrays. These exercises provide valuable insight for laying out geophone arrays (within the constraint of mine layout) and

evaluating calculated source locations from real data (Redfern and Munson 1982). Location accuracy will improve to better than 10% of geophone spacing as detection of first arrivals improves, as the layout of geophone arrays improves, and as location algorithms and velocity models improve.

DISCUSSION

Control of unstable ground in mining is a world wide problem. As mining goes deeper to obtain natural resources, problems with ground control will probably get worse. Microseismic monitoring needs to be a partner with other rock mechanics techniques to aid mining difficult-to-retrieve resources.

Techniques of microseismic monitoring have advanced significantly over the past few decades. Microseismic monitoring works well as long as the observer is aware of both the strengths and pitfalls of current microseismic monitoring techniques. The strength of microseismic monitoring is its ability to locate sources of microseismic activity. Data interpretation has advanced to allow the prediction of major events under some circumstances. Accurate predictions of rock bursts have been made in hard rock mines, but prediction of catastrophic bursts is not totally reliable, at this time. Predictions could have been made in coal mines had the data been available for automatic and immediate analysis. The accuracy and reliability of predictions will improve as the understanding of rock fracture improves and the ability to obtain a complete and accurate history of microseismic activity in a mine improves.

As computer technology has advanced, the price of digital data acquisition has decreased. As research aspects of microseismic monitoring are completed and requirements for a production system are determined, the price of microseismic monitoring systems will continue to fall. Bureau research suggests that, after further development, a complete microseismic monitoring system based on a desk top computer can be put together for $10,000 plus cable cost. Signal transmission cable or fiber optic cable running from the geophones to an underground instrumentation room may add another $10,000.

The ultimate goal of microseismic monitoring is not to predict catastrophic ground failures, but to aid in preventing them from happening. The Bureau's goal is to anticipate catastrophic failures in time to modify mining procedures to prevent failure. Until catastrophic failures can be reliably anticipated long enough in advance and until methods for controlling ground failure are developed and understood, the immediate goal is to predict all imminent catastrophic failures in time to allow miners to leave the mine safely.

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SURVEILLANCE MICROSISMIQUE DU CHARBON

Glen A. Rowell

Depuis de nombreuses années, le Centre de Recherches de Denver (DRC), Bureau des Mines Américain, mène d'actives recherches microsismiques dans les mines de charbon. Le but de ces recherches est de fournir à l'industrie minière des techniques fiables d'alerte de secousses et de dégagements imminents dans les mines de charbon, de manière à permettre aux mineurs d'évacuer les zones dangereuses avant l'effondrement du sol. Notre rapport présente la situation et les progrès du programme de recherches du DRC; il fournit de récents résultats des techniques microsismiques permettant de détecter l'imminence d'effondrements catastrophiques du sol dans les mines de charbon; il décrit le développement d'un système automatique de surveillance microsismique numérique de 16 pistes, d'une résolution de 12 bits, à l'intérieur d'un appareil disponible dans le commerce. Il étudie l'exactitude de la détection de la source de l'évènement et fournit des recommandations de futures recherches sur l'amélioration de la surveillance microsismique.

MICROSEISMISCHE ÜBERWACHUNG VON KOHLE.

Glen A. Rowell

Das Denver Research Center (DRC), U. S. Bureau of Mines, unternimmt schon seit Jahren microseismische Untersuchungen in Kohlegruben. Zweck dieser Forschung ist Entwicklung einer zuverlässigen Methode zur Warnung gegen drohende Gebirgsschläge und Gasausbrüche, damit das Gefahrengebiet rechtzeitig evakuiert werden dann. Der Vortrag befaßt sich mit dem Stand und Fortschritt des DRC Forschungsprogramms, einschließlich der neuesten Erkenntnisse in der microseismischen Technik zur frühzeitigen Erkennung von bevorstehenden katastrophalen Gebirgsschlägen in Gruben. Es wird außerdem die Entwicklung eines automatischen digitalen microseismischen Aufzeichnungssystems mit 16 Kanälen und 12 bit Wiedergabeschärfe vorgestellt, das in Gestalt eines im Handel erhältlichen Gerätes vorliegt. Die Genauigkeit, mit der Auslöser des Gebirgsschlages lokalisiert werden können, wird untersucht. Weitere Forschungsvorschläge zur Verbesserung der microseismischen Überwachung in der Zukunft werden vorgelegt.

МИКРОСЕЙСМИЧЕСКИЕ НАБЛЮДЕНИЯ НА УГОЛЬНЫХ МЕСТОРОЖДЕНИЯХ

Глен А. Роузлл

Денверский исследовательский центр Горного бюро США уже много лет осуществляет интенсивные микросейсмические исследования на угольных месторождениях. Целью этих исследований является обеспечение горной промышленности надежными методами предупреждения угрожающих угольных отслоений и выбросов, чтобы шахтеры смогли очистить опасные отсеки до обвала. В докладе обсуждается состояние исследовательской программы на сегодняшний день и достигнутый прогресс, включая новейшие микросейсмические методы, с помощью которых было своевременно распознано приближение катастрофических обвалов в угольных шахтах. Большое значение придается разработке 16-ти канальной, с разрешающей способностью в 12 бит, автоматической цифровой системы слежения, которая может быть использована с уже продающейся аппаратурой. Рассматривается точность локализации источника выброса угля. Предлагаются рекомендации для будущих исследований по улучшению микросейсмических наблюдений.

煤矿中微地震监测法

格兰 A． 罗威尔

美国矿务局丹佛研究中心（ＤＲＣ）几年来一直积极地从事煤矿微地震探测法的研究．此项研究工作的宗旨是给煤炭工业界提供一个可靠的技术来报警即将发生的煤矿石突出和塌方事件，从而矿工们能在地面崩塌之前撤离危险地段．本文将阐述丹佛研究中心（ＤＲＣ）的研究现况和进展，其中包括新近取得的预告煤矿内即将发生的灾难性地塌的微地震探测技术；本文还提到了一个１６频道，１２位解析度的微地震自动数字监测仪的研制，此仪器组装在一个单元里并可从市面上购得．震源在煤矿里位置的准确度也在这里提到．本文并提供了微地震监测今后的研究和改进方向．

DATA MANAGEMENT AND SCANNING SYSTEM FOR GAS OUTBURSTS

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University for Heavy Industry, Hungary

ABSTRACT

The Mecsek coal deposit is one of the most endangered mining area by gas outbursts. During 90 years nearly 600 rock and gas outbursts took place, the greatest of them exceeds the gas output of 200.000 m^3 and the rock thrown out of 1500 tons. To reveal the complexity of causes of rock and gas outbursts and the circumstances of outbursts a computer aided data management and scanning system was created.

Above the data of rock and gas outbursts the prevention processes were recorded for evaluation of effectivity of them. The further development of the data management contains the description of the rock environment, which makes a significant extension for exploration of causes.

At present the scanning system gives possibility to perform a statistical examinations and regression analyses for finding functional relationships of one and more variables.

The system was developed on IBM-PC/AT in program languages FORTRAN and C.

From among the coal presences in Hungary only coal deposits of the coal basin at the Mecsek-mountain are dangerous for coal and gas eruption. Since the 1st unexpected coal and gas eruption in 1894 nearly 600 such events took place. By increasing the extraction depth also intensity of these eruptions have been increased; the greatest of them reached or exceeded the methane yield of 200 000 cu. m and that of 1000 to 1500 erupted solid material. The majority of this material is coal, but some percentage (3 to 8 %) is spoil.

Great many years of research, collection of experiences and experiments leaded to the use of succeseful protective methods that may be shown by the decrease of numbers of unexpected rock and gas eruptions in a year.

We form the analysing system registrating the data of gas eruptions and related activities for the sake of a wider finding the relation between the different facts and effects.

Construction of the designed program system may be seen on Fig.1. and consists of 3 main

units:

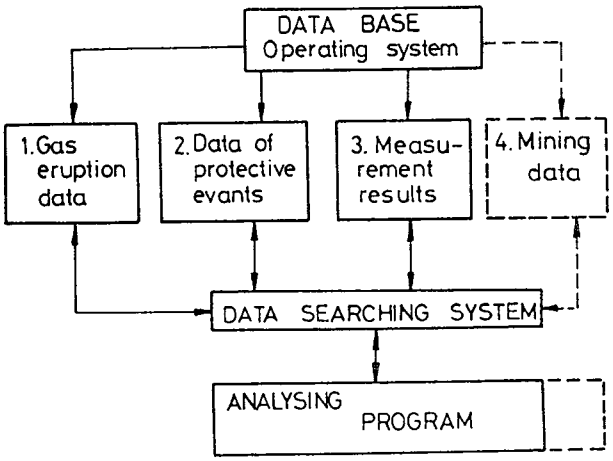


Fig.1. Structure of the designed program system

1. Data base handling system
2. data searching system
3. system of analysing programs

Data base contains beside the occurred gas eruptions the most important caharcteristics of the coal deposited unity (measurement data), and the data of the protection events. Filing of mining data may be the result of further developments at the area of areas endangered by gas eruption.

No. of all the gas eruptions may exceed 80.

Among them the most important are: time of eruption with exactness in min.; type of eruption, its exact location within the mine, typical data of the coal deposit, name of base and roof stone, type of mining area, its direction of moving related to the deposit, geometrical location, length, cut; data of securing, way of breaking, geological irregularities, protection methods, gas draining, gas pressure, discharged methane, cast-out solid material, rate of filling-up slope length, flying coal dust, and humidity, ash content, related to the outcast coal; volatile material, porosity, grain distribution, sorbtion, desorbtion, indications for the eruption; data for the accidents, rescue, escape.

So, data filing does not contain only the basic data for the eruption, but it also describes the circumstances and consequences of it. Some data is derived from the analysis in laboratory of the coal samples taken after the gas eruption. Registration of these data required a separate solution since the data base is of sequential construction, where the data set of every gas eruption means one separate record at a steady length. At the same time, number of data from samplings depends on the quantity of the outcast solid material. Because of the above, data from the samplings (e.g. ash contant, porosity, etc.) get into an outer, changed-structure data set.

A quick finding of eruption data is ensured by a separate indexing set.

In that set many written value will be registered (name of Co., name of drift, denomination of minerals, geological trouble, etc.) at which the unified denomination and mode of writing is important, for in the opposite, searching methods mentioned later may become unsure. For the data input we ordered a data set to be enlarged willingly, of which, in case of the presence of all the data, actual denomination may be selected (e.g. name of Co., denomination of mineral, etc.).

Since the 1st gas eruptions occurred more than 80 years, it is obvious that essentially fewer data of them we have,

than could be stored. Complete data sets can be registered mostly the eruptions of the last 10-15 years. In the data set data lacks are differred from the zero values.

Formation of data to be filed referring for the protective actions is on the way, now. Our goal is by registering these data that the effect of the protective methods may be evaluable for overcoming gas eruptions on the base of the detailed evaluation of these methods.

Since in the mining at the area near the Mecsek occurs mostly during shafting, first of all we like to register protective methods at these workplaces. During it every workplace will be registered - said to be dangerous for the gas eruption - from the start of the drift driving till its end. Place, time, technical parameters of the protective events (preliminary drilling, material take--out) are stored in every case. Handling of the great amount of data is facilitated by connecting data referring to the same drift in records or by chaining them.

With the Mecsek Mining Co. documentation of the protective methods is been doing for several years, so getting back in time, at the start, a great deal of data set may be registered.

Taking into account the mining data is considered to be realizable if at the Co. a mining CAD system has been adapted, data set of which may be reached also by the gas eruption registrating system.

To the data base a data searching system is connected that can do selection of any type in the whole data set. Researching acc. to the simple key ensures the quick finding of the data of gas eruptions. When searching for parameters any number of parameter may be given as searching point, so the selection of the data set may be done acc. to the most different points of view.

When searching acc. to limits, the upper one or intervals may be given for every parameters (it may theoretically be used for the written parameters, too, since at the lining we follow the order of the English alphabet). For giving the points for the searching of data acc. to parameters, we use the same screen picture as for data input that facilitates greatly the handling. Data evaluating programs are connected to the complete system that the continuous enlarging possibility is given. Since events, processes relating to the gas eruptions are of statistical character because of the many "ad hoc" effects, firstly we prepared a methematical-statistical program set for the analysis.

Over the basic statistical data (mean, dissipation) hystogram and empiric distribution may be prepared by calculating the inclination and oblateness.

The second great unit is a program set containing regression calculation of two-variables where the determination of 11 different functions my be possible. Here are polynoms from the linear to fifth degree. Different hyperbolas, exponential functions $y = ax^b$, $y = ax^b+c$, and an exponential function $y = ae^{bx}+c$. The determination of the regression constants is possible only on

the iteration way at most functions, since the normal equation system is not linear.

Analysing system is supplemented by a graphic display, that makes a possibility in case of the regression functions to check the position of the point; then you may select a function type in the required number and display the Figure on the screen.

When doing regression calculations correctness of the estimation is characterised by the estimated distribution of the estimation itself, tightness of the equation relation by a correlation index.

For a multivariable regression calculation we prepared a program that makes possible the selection of the function type belonging to the variables, from linear, quadratic and hyperbola. So variable functions may by composed up to 5 independent variables.

For the multivariable regression analysis you may have two more power functions. $y = ax_1^b . x_2^c ... x_2^g$, or supplemented by a constant. A possible enlarging of the data analysing program happens acc. to needs arriving during the analysis.

During this latter only trial running have been made since the "filling-up" of the data set is still under way.

Development of the programs occurs on an IBM.PC-AT/XT machine, partly in the FORTRAN and on C-language.

We suggest that by the aid of the computer aided data filing and processing system efficiency of protection against the unexpected coal- and gas eruptions may be increased, for which we are in ever greater need because of mining in the greater and greater depths.

SUMMARY

Authors sum up in that paper the issues relating to the design of ventillation systems. After the erection of the designing system the let us know about the developments that they made during the last 10 years in the field of design.

Air quantity determination methods, computer simulation, system calculation and the determination of the regulations are the principal elements that would back the designing system from the computer point of view. Computer developments are made on machines at PC-level for they might be able to introduce in the mining industry.

SYSTEME DE GESTION DES DONNEES ET DE SCANNING POUR LES DEGAGEMENTS SOUDAINS

J. Janositz, Z. Buocz, F. Kovacs et L. Meszaros

Le gisement de charbon de Mecsek est l'une des régions minières les plus sujettes aux dégagements soudains. En 90 ans, près de 600 dégagements brusques de roches et de gaz s'y sont produits, dont les plus importants dépassaient 200.000 m^3 de gaz et 1.500 tonnes de roches. Pour étudier la complexité des mécanismes de dégagements de roches et de gaz, ainsi que les circonstances entourant ces dégagements, l'on a créé un système informatisé de gestion des données et de scanning.

L'efficacité des processus de prévention a été évaluée sur la base des données des dégagements soudains de roches et de gaz. La gestion des données donne en outre une description de l'environnement rocheux.

A l'heure actuelle, le système de scanning permet de procéder à des études statistiques et à des analyses par régression pour déterminer les relations fonctionnelles d'une ou plusieurs variables.

Ce système a été développé sur IBM-PC/At en langages FORTRAN et C.

DATENMANAGEMENT UND -ABFRAGESYSTEME FÜR AUSBRÜCHE

J. Janositz, Z. Buocz, F. Dovacs, L. Meszaros

Die Mecsek Kohlelagerstätte ist eine der am stärksten ausbruchgefährdeten Lagerstätten. In 90 Jahren kam es zu annähernd 600 Stein- und Gasausbrüchen, wovon die größten mehr als 200,000 m^3 Gas und 1,500 Tonnen Stein lösten. Um die Komplexität der Stein- und Gasausbruchsmechanismen und die Umstände des Ausbruches zu studieren, wurde ein computerunterstütztes Datenmanagement- und Abfragesystem erstellt.

Ausgehend von den Daten für Stein- und Gasausbrüche, wurden Verhütungsmaßnahmen auf ihre Wirksamkeit hin untersucht. Eine Weiterentwicklung des Datenmanagement-Systems wird eine Beschreibung des Gebirgsumfeldes beinhalten.

Zur Zeit ermöglicht das Abfragesystem statistische Untersuchungen und Regressionsanalysen zur Bestimmung funktionaler Zusammenhänge einer oder mehrerer Variablen.

Das System wurde auf einem IBM-PC/AT in FORTRAN und C entwickelt.

ОБРАБОТКА ДАННЫХ И СИСТЕМА СКАНИРОВАНИЯ НА ВЫБРОСЫ

Я. Яносиц, З. Буоч, Ф. Ковач и Л. Месзарос

Мешекское угольное месторождение является одним из наиболее выбросопасных. Около шестисот выбросов породы и газа произошло за последние 90 лет, в наихудшем случае - 200.000 кубических метров газа и 1500 тонн породы. Для комплексного изучения механизма выбросов породы и газа, и обстоятельств, при которых произошли выбросы, была создана компьютерная система обработки и сканирования данных. Эффективность предупредительных мер оценивалась на основе имеющихся данных по выбросам породы и газа. Дальнейшее усовершенствование системы включает описание геологических условий. В настоящее время система сканирования дает возможность производить статистическую обработку и регрессионный анализ с целью определения функциональной зависимости одной или нескольких переменных. Система была разработана на Ай-Би-Эм-(PC/AT) в языках ФОРТРАН и Си.

突出的数据处理和扫描系统

J。詹诺锡兹， Z。布屋兹，
F。柯凡克士 及 L。美萨罗士

美散克煤矿是矿场中最易发生突出现象的．九十年来近六百起岩石与气的突出曾经发生过，最大的突出牵涉到超过２００００立方公尺的气体和１５００吨的岩石．我们发展了计算机辅助的数据处理和扫描系统来研究岩石与气体突出的作用．防止的措施曾根据岩石与气体突出的数据来衡量它的有效性．数据处理以后的工作将牵涉到岩石周围环境的叙述．目前扫描系统能用统计分析和回归分析来决定变数之间的关系．这系统是用ＦＯＲＴＲＡＮ和Ｃ语言在ＩＢＭ－ＰＣ／ＡＴ上使用．

BUMPS IN INDIAN COAL MINES - OVERVIEW OF RECENT R&D EFFORTS

A. K. Ghose
Indian School of Mines, India

ABSTRACT

The paper presents a critical analysis of experiences of coal mine bumps in India and traces chronologically the methods of contending with bumps and the impacts on the hazard dimensions. The results from a major R&D programme initiated by the coal industry during 1977-1986 in assessing and controlling the severity of the problem are discussed in the paper with special reference to the following:

(i) Analysis of the decadal experience (1977-1986) with coal mine bumps vis-a-vis geomechanical and mining parameters;

(ii) Assessment of bump proneness using laboratory tests and drilling yield tests at different coal mine sites, both in board and pillar and longwall workings;

(iii) Problems of caving under massive coalmeasure roof rocks and results from seismic monitoring at Chinakuri 1 & 2 Pits Colliery; and

(iv) Overall assessment of future problems with extension in depth and new mining strategies, including proposed trials with hydraulic fracturing, for controlling the bump problem.

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Manuscript was not received in time to be included in Proceedings.

SECOUSSES DANS LES MINES DE CHARBON DE L'INDE VUE GENERALE DES EFFORTS RECENTS DE R&D

A.K. Ghose

Ce traité présente une analyse critique des secousses dans les mines de charbon en Inde; il donne l'historique des méthodes de lutte contre ces secousses et leur impact sur l'envergure des risques. Il examine les résultats d'un important programme de R&D lancé par l'industrie charbonnière entre 1977-1986 pour évaluer et contrôler la gravité du problème; les éléments suivants y figurent en particulier:

(i) Analyse de l'expérience acquise durant cette dernière décennie (1977-1986) sur les secousses dans les mines de charbon en fonction de paramètres géomécaniques et miniers;

(ii) Evaluation de la prédisposition aux secousses au moyen d'essais en laboratoire et d'essais de forage effectués dans diverses mines de charbon - chantiers de bords, de piliers et de longues tailles;

(iii) Problèmes de sous-cavage sous des roches massives de toits de charbon et résultats des surveillances sismiques des Puits 1 & 2 des Houillères de Chinakuri;

(iv) Evaluation générale des futurs problèmes posés par des profondeurs plus importantes et de nouvelles stratégies minières, y compris des essais proposés de fractures hydrauliques pour contrôler le problème des secousses.

GEBIRGSSCHLÄGE IN INDISCHEN KOHLEGRUBEN - ÜBERBLICK ÜBER DIE JÜNGSTEN FORSCHUNGS- UND ENTWICKLUNGSVORHABEN.

A. K. Ghose

Der Vortrag gibt eine kritische Analyse der Erfahrungen, die auf indischen Gruben mit Gebirgsschlägen gesammelt wurden, und zeigt chronologisch die Gegenmaßnahmen und ihre Auswirkungen auf. Die Ergebnisse eines wichtigen Forschungsprogrammes, welches von 1977 bis 1986 von der Kohleindustrie eingeleitet wurde, um den Ernst des Problems abzuschätzen und es zu kontrollieren, werden in Bezug auf die folgenden Punkte diskutiert:

(i) Analyse der Erfahrungen zwischen 1977 und 1986 mit Gebirgsschlägen gegenüber geomechanischen und bergbaulichen Parametern;

(ii) Bewertung der Gebirgsschlagneigung durch Labortests und Bohrlochtests in verschiedenen Gruben mit Kammer - Pfeilerbau und Strebbau;

(iii) Probleme mit dem Bruchbau unter massivem Deckgebirge und Ergebnisse der seismischen Überwachung auf den Steinkohle-Gruben Chinkuri 1 und 2; sowie

(iv) Allgemeine Abschätzung der Probleme in der Zukunft mit wachsender Tiefe und neuen Abbaustrategien, z.B.: Vorschlag von Versuchen, das Gebirge hydraulisch zu lockern, um Gebirgsschläge zu kontrollieren.

ПРОБЛЕМА ОТСЛОЕНИЯ УГЛЯ В ИНДИЙСКИХ УГОЛЬНЫХ ШАХТАХ: ОБЗОР НАУЧНЫХ ИССЛЕДОВАНИЙ ПОСЛЕДНИХ ЛЕТ

А.К.Гоус

Доклад представляет собой критический анализ опыта работы с отслаивающимися пластами угля на месторождениях Индии, прослеживает хронологически развитие методов борьбы с отслоением угля и рассматривает влияние катастрофических случаев. Обсуждаются результаты основных научных и прикладных исследований, проведенных угольной индустрией с 1977 по 1986 г.г. с целью определения серьезности этой проблемы и ее предотвращения. Особое внимание уделяется следующим вопросам:

1) Анализ десятилетнего опыта работы (1977-1986) с отслоением угля в сопоставлении с геомеханическими и горными параметрами;

2) Оценка склонности к отслаиванию, основанная на использовании лабораторных результатов и данных, полученных при бурении различных видов угольных выработок: штреков, целиков и длинных забоев;

3) Проблемы просадки верхней пачки угольных пластов и результаты контроля, полученные сейсмическим методом, на месторождении Чинакури, 1-й и 2-ой угольные карьеры;

4) Общая оценка проблем отслоения угля, которые могут возникнуть при углублении карьеров и внедрении новой горнодобывающей технологии, включая попытки применить метод гидравлического разрыва пластов.

印度煤矿井中矿石的突出 -- 最新研究和发展的概况

A. K. 戈斯

本文对印度煤矿的矿石突出作了重要分析，并且追溯了过去每一时期对突出的处理方法及其对灾害的影响。本文将讨论一个 1977-1986 年间由煤炭工业界发起的针对控制该问题的严重性的研究项目所得的一些结果。有如下几个主要讨论题目：

(i) 1977-1986 十年来煤矿突出与地力学及采矿参数之间关系的分析；

(ii) 用试验室试验和不同的残柱式及长壁式矿场中的采矿量试验评定突出的易发性；

(iii) 巨型煤层顶板岩石的塌落问题和钦那库里第 1 号与第 2 号煤矿的地震探测结果；及

(iv) 对今后开发深度的加大和新采矿技术的总体评价，其中包括建议试用水力破裂来控制岩石突出。

COAL MINE BUMP RESEARCH: PROVIDING TOOLS FOR RATIONAL LONGWALL DESIGN

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ABSTRACT

Coal mine bumps, the violent failures of over stressed coal, present a safety hazard to miners in underground coal mines. To help alleviate this hazard, the Bureau of Mines is pursuing an extensive research program with the ultimate goal of developing a mining layout design procedure that minimizes bump hazard. The foundation of this research program is the delineation of the detailed reactions and physical properties of the bump-prone coal strata through in-mine evaluation of various mining layouts. One room-and-pillar retreat section and two longwall gate road configurations have been intensively instrumented in deep Southern Appalachian coal mines with a history of coal bumps. Stress and deformation measurements from these instrumented areas have provided valuable insight into the strata response associated with pillar bumps. Currently the gob loading distribution and the strata caving characteristics at the longwall mine are under investigation through instrumentation installed under the panels and in surface boreholes. Data from these field investigations will be used in the development of a non-linear, displacement-discontinuity model that will use energy changes caused by mining to predict a bump hazard. Using this model, multiple trial mining layouts and sequences can be quickly and easily simulated in order to minimize the relative bump hazard.

INTRODUCTION

Full extraction retreat coal mining concentrates stress in the coal adjacent to expanding gob areas. When mining is conducted at great depth and between unyielding roof and floor strata, over stress of the coal may result in violent pillar failures. These "bumps" vary from minor vibrations without significant strata movement, to notable earth tremors with thousands of tons of coal ejected into the mine workings.

The Southern Appalachian Basin of the United States has had a long history of coal bumps and coal bump research. One of the earliest detailed reports about coal bumps in this area was compiled by Rice (1935). This report identified numerous sites of coal pillar bumps in Eastern Kentucky and Southwestern Virginia, and noted that in one 4 month period, eight miners were killed and a number injured by bumps. In another report, Holland and Thomas (1954) examined 177 instances of pillar bumps, most of which were in the Southern Appalachian Basin, and found that the primary cause of these bumps was "unfavorable" mining practices in abutment areas. Talman and Schroder (1958) noted the importance of very stiff overlying strata to the occurrence of bumps, thereby emphasizing the influence of local geologic conditions. Comprehensive laboratory and field data on the physical properties of two bump-prone coals from Southern Appalachia (the Pocahontas No. 3 and No. 4) were reported by Wang et al. in 1976. No characteristics of these coal beds indicated a particular preference for them to bump.

The Bureau of Mines re-initiated coal mine bump research in the Southern Appalachian Basin due to a rash of bump-related fatalities and injuries in 1984 and 1985 (Campoli, et al., 1987). A preliminary investigation of the geologic, mining, and engineering parameters at five sites in West Virginia and Virginia, where recent miner fatalities or injuries were associated with coal bumps, revealed that stiff, competent associated strata and the high stresses generated by retreat mining were common to all five sites. These findings supported the earlier conclusion of Holland and Thomas (1954) that the probability of bump occurrence is increased by certain mining practices that concentrate stresses during retreat mining. This preliminary report recommended that mine designs should avoid the development of pillar line points or long roof spans which propagate bumps. Also, retreating longwalls with carefully designed gate roads were recommended over room-and-pillar retreat in deep, bump prone mines (Campoli, et al., 1987).

The Bureau's bump research program initiated the development of design criteria for controlling bumps with

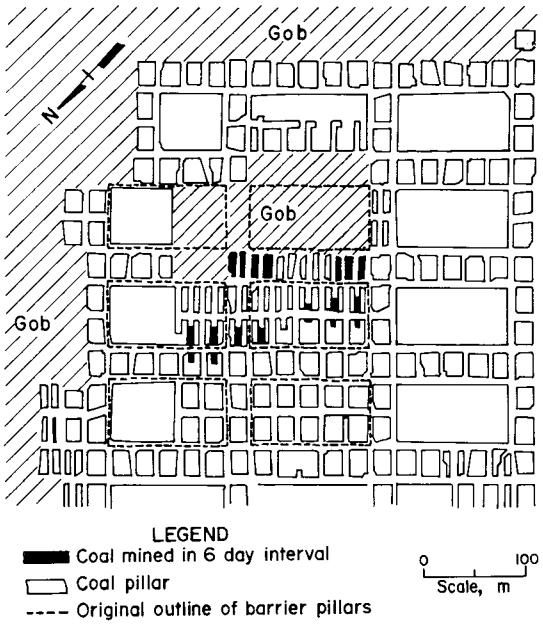


Figure 1.

an in-mine evaluation of a novel extraction sequence used to control bumps during room-and-pillar retreat coal mining at the Olga Mine, McDowell Co., West Virginia. This novel retreat mining system, which conducted mining over three pillar rows outby the gob, distributed abutment loads up to six pillar rows outby the newly formed gob (figure 1). Microseismic monitoring was also conducted in the Olga Mine study area, the results of this effort were reported by Condon and Munson (1987).

Roof-to-floor convergence monitoring at this site proved to be a very valuable tool in evaluating the novel mining method, and localized destressing techniques. Consistent indications of pillar loading and stress redistribution resulted; because both the roof and floor rocks apparently acted as large thick plates during the entire mining period, crushing the friable Pocahontas No. 3 Coalbed.

Using roof-to-floor convergence and coalbed pressure measurements many facets of bump-prone strata were investigated at this site. Maximum strain energy storage in the chain pillars occurred just prior to the first of four chain pillar split cuts. Before these split cuts, a 4.6 m wide yielded perimeter confined the core of the 16.8 by 21.3 m pillars, permitting the soft coalbed to support tremendous loads. Splitting the chain pillars into two 5.3 by 21.3 m wings removed the confining pressure. From convergence and stress measurements it was apparent that structural failure of the pillar occurred after the third pillar split cut. Thus, by strategically softening pillars in the abutment zone, the novel mining method successfully redistributed excessive strain energy away from the pillar line.

Shot fire and auger drilling destressing techniques were also evaluated at this mine. Both

techniques induced instantaneous roof-to-floor convergence, indicating a softening of the subject chain pillars. This resulted in a reduction of their energy storage capacity. Both roof-to-floor convergence and cuttings volume measurement indicated that the drilling of 10.2 cm auger boreholes, was effective for stress reduction on highly stressed pillars. Thus, both localized destressing techniques enhanced the effectiveness of the mining method in redistributing strain energy (Iannacchione, et al. 1987 and Campoli, et al. 1989).

The remainder of this paper outlines the current status of the Bureau's research program to develop a design criterion that can be used to minimize the bump hazard of mining layouts. The foundation of this program is the delineation of the detailed reactions and physical properties of the bump-prone coal strata through in-mine evaluations. The results of the Olga Mine study are enhanced by an evaluation of two different, bump control, longwall gate road configurations in a deep bump-prone coal mine, in the Southern Appalachian Basin. In order to obtain an understanding of the far-field loading mechanism in the unique bump-prone strata of Southern Appalachia and to build upon the stress and deformation data, a field project specifically designed to characterize the longwall gob behavior at the study mine has been initiated by the Bureau of Mines. The accumulated knowledge base resulting from these research efforts will be utilized in the development of the mine design procedure. The design methodology will be implemented through a numerical model that can be used as a tool for designing coal mines to minimize bump hazards.

THE LONGWALL GATE ROAD DESIGNS

A section of the mine plan showing the two instrumented gate roads is presented in figure 2. In this area of the mine, the longwall panels are roughly 180 m wide and 1,700 m long. The first detailed study area, located in the A-development gate road, is centered approximately 1,430 m from the start-up entry of panel A, under 610 m of overburden. Panel A is the sixth panel in a series of longwall panels extracted from north to south. The second detailed study area, located in the B-development gate road, is centered approximately 1,400 m from the start-up entry of panel B, under 640 m of overburden. Thus, as shown in figure 2, the two study areas are located adjacent to each other on opposite sides of panel B.

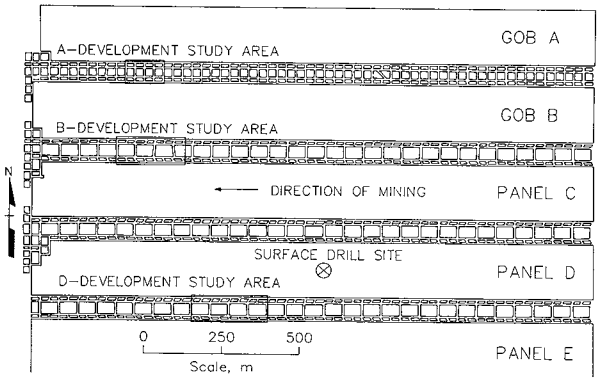


Figure 2.

A detailed map of the A-development study area is presented in figure 3. The A-development gate road employs a yield-abutment-yield configuration with the yield pillars on 15.2 m centers, the abutment pillars on 30.5 m centers, and all crosscuts on 30.5 m centers. This gate pillar configuration has replaced an older design which consisted of three square pillars with both entries and crosscuts on 30.5 m centers. In this previous design, the 24.4 m square pillars frequently experienced heavy bumps directly adjacent to the tail drive causing coal to be thrown into the face area (Campoli et al, 1987). However, in the newer A-development design, the 24.4 by 9.1 m pillars

essentially yield on the headgate pass of the face, thereby eliminating their potential to bump. The yielded pillars effectively shield workers

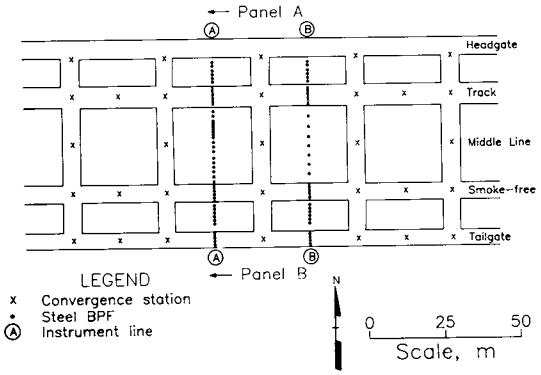


Figure 3.

from coal thrown in the event the 24.4 by 24.4 m abutment pillar bumps during the subsequent tailgate pass.

This technique of shielding the workers from the potential abutment pillar bumps is satisfactory if the abutment pillar bumps occur behind the longwall face in the gob or adjacent to the tail drive. However, as the mine expanded under deep cover, the abutment pillars began bumping in advance of the face. In A-development, bumps of this nature completely closed the 6.1 m wide crosscuts between the abutment pillars as shown in figure 4. Also, when the tailgate abutment pillars started failing



Figure 4.

outby the tail-drive, bumps began to occur directly on the tailgate side of the face.

In an attempt to control the tailgate bumps and to standardize their gateroad design, the mine switched to an alternate yield-abutment-yield configuration in B-development. This new design consists of yield pillars on 12.2 m centers and abutment pillars on 42.7 m centers (figure 5). Between the yield pillars, the crosscuts were driven at 60 degree angles on 30.5 m centers, whereas between the abutment pillars the crosscuts were driven at 90 degree angles on 61.0 m m centers. This layout results in 6.1 by 24.4 m by yield pillars on either side of a 36.6 by 54.9 m abutment pillar. The larger abutment pillars in this new design are intended to carry the applied abutment loads, and thus prevent the ground stresses from overriding the longwall face. This juxtaposition of these two strategically different gateroad designs in highly stressed strata provides a unique opportunity for obtaining a better understanding of the strata movements and coal behavior associated with the bump phenomena.

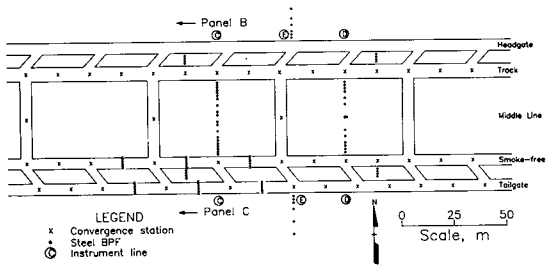


Figure 5.

PHYSICAL PROPERTIES OF THE BUMP PRONE COAL STRATA

The study areas are located in the Pocahontas No. 3 Coalbed under overburden ranging from 550 by 670 m in thickness. The immediate roof in these areas consists of

approximately 9.1 m of widely jointed siltstone overlain by a massive quartz arenite sandstone (figure 6). Similarly, the first 12.2 m of the mine floor consists of a combination of competent siltstone and sandstone. Underground observations reported by Iannacchione (1988) indicate a persistent absence of prominent roof and floor fractures or joints and that the main roof, dominated by the thick quartz arenite sandstone, is exceedingly difficult to break. These unique geologic conditions apparently cause greater pillar loads in the study area than would be predicted by conventional, empirical abutment load calculations, such as those proposed by Mark (1987).

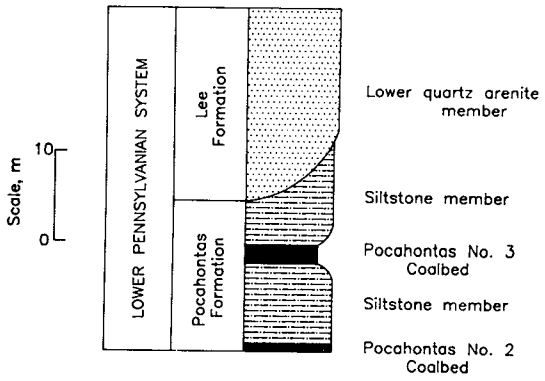
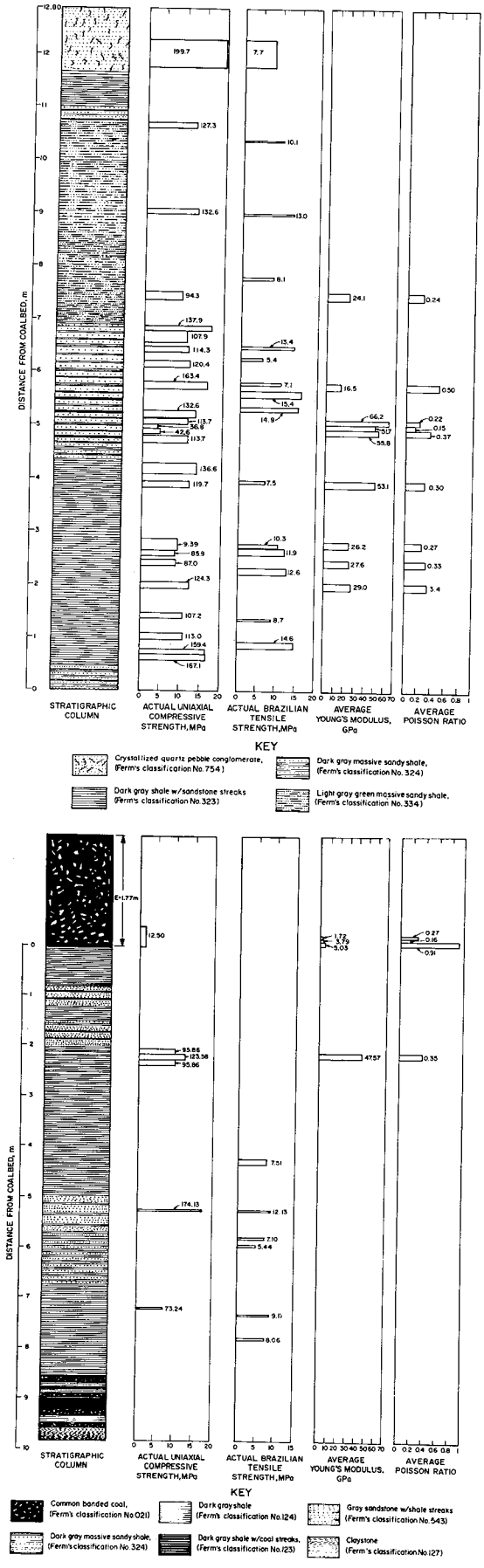


Figure 6.

NX-size coreholes were drilled 12.8 m into the immediate roof and 9.8 m into the immediate floor within the gate road evaluation study area. Detailed lithologic logs were produced from visual examination of the recovered core. Also, unconfined compressive and Brazilian tensile tests were performed on selected core samples. The Young's modulus and Poisson's ratio of the rock were provided from these laboratory tests. The lithologic logs and the physical properties of the immediate roof and floor are presented in figures 7 and 8, respectively. From these figures, it is apparent that the



Figures 7 and 8.

strength and stiffness of the strata surrounding the Pocahontas No. 3 Coalbed in the study area are uncommonly high for coal measure rocks. Similarly, high strength and stiffness properties were reported by Iannacchione (1987) for the strata surrounding the Pocahontas No. 4 Coalbed in bump prone areas. In a special effort to determine the confined elastic moduli and coal strength for modeling purposes, a suite of laboratory tests were conducted on specially prepared samples of the Pocahontas No. 3 Coalbed. Initially, to establish a datum, three unconfined compressive tests were conducted on 5.1 cm cubic samples taken from the study area. The nominal unconfined compressive strength in these cubic tests was found to be approximately 12.4 MPa. Three triaxial compression tests were performed on 10.2 cm cubic samples of the coalbed. In these triaxial tests, horizontal confining pressures of 13.8, 20.7, and 27.6 MPa resulted in vertical failure stresses of 53.1, 69.6, 84.8 MPa, respectively. This increase in vertical coal cube strength with increasing confinement pressure (figure 9) helps explains how a soft and friable coal pillar can sustain the tremendous pressures that precede a bump occurrence.

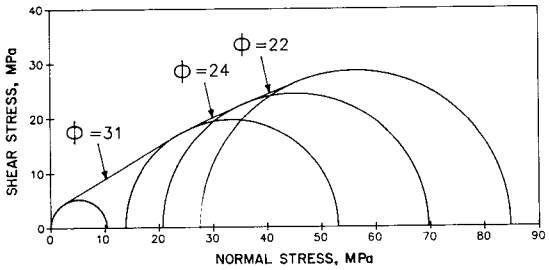


Figure 9.

FIELD INSTRUMENTATION

The complete instrumentation arrays installed in the two study areas consist of convergence stations for measuring entry closure, stainless steel borehole platened flatjacks (BPF) for indicating changes in pillar stress, coal extensometers for measuring pillar dilation, overcoring sites for determining initial in situ horizontal stresses in the immediate roof, differential roof-sag indicators for monitoring bedding separations in the roof, and differential floor-heave indicators for monitoring relative floor heave. Since the intent of this paper is to provide a general outline of the Bureau's coal mine bump research program, only a small portion of the BPF and convergence information will be presented herein. Heasley and Barron (1988) and Iannacchione (1988) have reported the results from the A-and B-development instrument arrays in more detail.

The BPFs, used in this study to indicate pillar stress changes, consist of a stainless-steel hydraulic flatjack and two aluminum platens designed to fit the flatjack to a 5.1 cm diameter holes. Unless otherwise indicated, the BPFs described in this paper were oriented to indicate the pillar stress changes in the vertical direction. It should also be noted that the BPFs do not directly measure the pillar stress changes; but rather, indicate the hydraulic pressure change of the flatjack which is related to the true pillar stress change by some "stress intensity factor." All of the BPF pressure information presented in this paper were "corrected" to pillar stress changes by a stress intensity factor developed through the method outlined in Heasley (1989).

The convergence stations, used in this report to measure entry closure, consist of permanent pins installed in the roof and floor, and a portable, telescoping rod which can measure the distance between the permanent pins to within several thousandths of an inch. It should be noted that it is not possible to directly distinguish between roof-sag,

floor-heave, or pillar strain with this instrument; therefore, only the relative closures between the roof and floor are reported in this paper.

The exact locations of the convergence stations and coal cells installed in A-development and B-development are shown in figures 3 and 5, respectively. In general, the convergence stations were installed in all of the nearby intersections and at the mid-points of the four instrumented abutment pillars in order to provide an overall view of the entry closure in the study areas. However, at certain locations, coinciding with the lines of BPFs, five convergence stations were installed in a line across the entry in order to obtain a profile of the relative roof-to-floor closure. In contrast to the widespread installation of the convergence stations, the main goal behind the BPF installation was to obtain pressure profiles across the width of the gateroads. Therefore, all of the BPFs were installed in two transverse lines across the gateroad with each line encompassing two yield pillars and an abutment pillar. In all, 133 convergence stations and 83 BPFs were installed and monitored in the two instrument arrays. However, the remainder of this paper will focus only on the information from the A and C instrument lines (figures 3 and 5), since the additional information from the remaining instrument lines approximately duplicate the information which is presented for the A and C instrument lines.

In general, the 36.6 m wide abutment pillars have been considerably more stable than the 24.4 m wide abutment pillars. However, the face of panel C has yet to mine past the instrument array in B-development. Thus, a direct comparison of the two designs is possible only for the headgate pass. The stress levels in the abutment pillars of the A instrument lines from both gate roads (at a point when the first face is 38.1 m past each instrument line) are shown in figure 10. These two curves vividly demonstrate the reduced pillar stress levels that result from the increase in abutment pillar size and help explain the increased stability of the 36.6 m wide abutment pillar as compared to the 24.4 m wide abutment pillar.

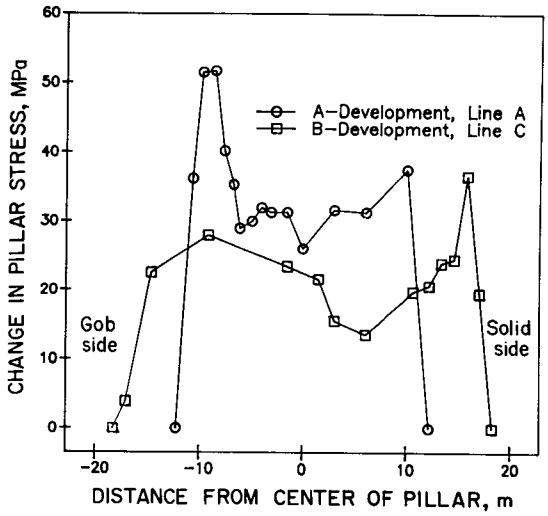


Figure 10.

The convergence data from the A and C instrument lines, when the headgate is 38.1 m past the instrument lines, are shown in figure 11. This convergence data coupled with in-mine observations indicate that the 24.4 m wide abutment pillar in the A-Development design is undergoing more strain than the 36.6 m pillars and permitted more pronounced roof tilt toward the gob. This greater deformation over the entire 24.4 m width of the abutment pillar, indicates that the abutment pillar is failing before the immediate roof or floor. In contrast, the 36.6 m wide abutment pillar in the B-Development design is sufficiently stable to induce fracturing and heaving of the floor as noted by in-mine observation. The floor failure at the gob side pillar edge and in the adjoining entry is demonstrated by the over

5.1 cm convergence displayed in figure 11. This situation is also verified by the relatively low stress changes on the gob side edge of the 36.6 m abutment pillar displayed in figure 10.

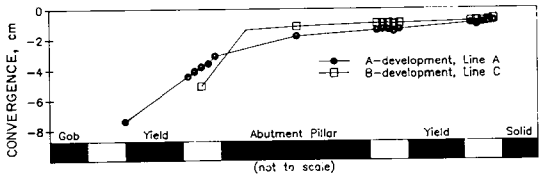


Figure 11.

Both the stress and the complimentary convergence information from the headgate pass indicate that the 36.6 by 54.9 m abutment pillars will be significantly stronger than the previously employed 24.4 m square pillars. It is anticipated that the additional gate road pillar load bearing capacity will shield the tail of the longwall face from excess abutment loads and prevent the face bumps that occurred during the mining of Panel B with the 24.4 m abutment pillars (figure 2). However, a final determination of the effectiveness of the B-development design must be withheld until the mining of panel C is completed.

CHARACTERIZATION OF LONGWALL GOB BEHAVIOR

Earlier, it has been noted that the pillar loads in the study area are greater than would be predicted by conventional abutment load calculations. An understanding of this unusual strata behavior and its impact on the distribution of the overburden loads is critical to the development of a design procedure for minimizing bumps. A field project specifically designed to characterize the strata behavior at the study mine is in progress. This project consists of measurement of the gob loading distribution and delineation of the strata caving characteristics as induced by longwall mining in bump-prone strata.

The gob load, and also the gate road floor load, are being measured with an array of steel BPFs. This instrumentation array is located in the D-development gate road approximately 1220 m from the start of panel D (figure 2). The BPFs are being placed in boreholes drilled in the siltstone floor below the Pocahontas No. 3 Coalbed from a sump located in the center of the D-development gate road (figure 12). A total of 27 BPFs will be installed in boreholes under the panels D and E, and under an abutment pillar. BPFs are also being installed in the coal directly above each BPF installed under the abutment pillar. Also, the Bureau of Mines' cylindrical pressure cell (CPC) is being employed in both the coalbed and the floor to determine the in situ modulus of deformation of these stratigraphic units. A comparison of the response of the BPF in the floor with the response of the paired BPF in the coalbed, combined with the in situ modulus of elasticity of the host rock provided by the CPC, will allow for accurate reduction of the BPF pressure readings.

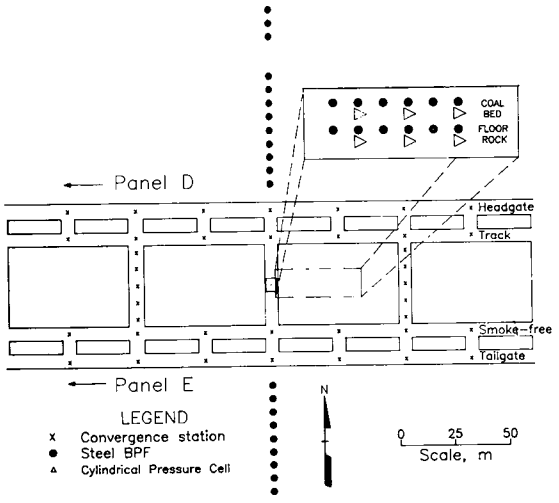


Figure 12.

Since most of the strata reactions occur after mining of the tailgate panel it is imperative to be able to measure these reactions at this point in time. Therefore, a remote, permissible data acquisition system is being utilized at the site so that data can be collected (without human exposure) after the site has been encompassed by gob on both sides.

The measurement of the strata caving characteristics is oriented towards determining the far-field effect of longwall mining on the strata above the mined panel. Specifically, the height of the immediate cave zone and the fractures and slips in the strata above the mined panel are being determined. The structural effects of gob formation will be studied pre- and post-mining, from one location in the center of panel D (figure 2).

A pre-mining NX-size corehole, from the surface to 91.4 m below the coalbed, is being drilled. The drill site was selected based on the panel geometry, overburden thickness, and surface accessibility (figure 2). The detailed lithologic description and the precise fracture locations are being established by on-site core logging. Selected samples of the continuous core are being collected for laboratory estimations of rock properties. A geophysical logging crew will directionally survey the borehole and perform two additional logging runs. This geophysical logging will aid in determining fracture locations, and vertical and horizontal strata deformation, prior to mining.

A second corehole will be drilled in close vicinity to the first corehole after the extraction of panel D. The physical and geophysical logging process will then be repeated in this second corehole. A comparison of the fracture locations and strata deformations between the two coreholes will delineate the structural effects of gob formation. This information combined with the results of the gob loading measurements will provide an understanding of the far-field loading mechanism, which is critical to the development of a realistic design procedure.

NUMERICAL MODELING

The ultimate goal of these combined research projects is to provide a design procedure that can be used as a tool for minimizing the bump hazard of coal mines. This model should be capable of accurately simulating the physical response of the coal throughout relatively large areas of the mine. The formulation of the model needs to be fairly efficient in order to allow for quick and easy simulation of multiple trial mining layouts and sequences. Also, it is useful for the model to simulate the non-linear and post-failure behavior of the coal; and for predicting the bump potential, the model needs to be capable of tracking the energy changes associated with the mining process. The concept of using the mining-induced energy changes to predict bump potential has been used fairly successfully in hard rock mining (Brady and Brown, 1985). However, to the best of our knowledge, mining-induced energy changes have never been applied in a coal mining environment for predicting bumps. With the in-depth field data that has been obtained from the instrumentation sites, a perfect opportunity exists for analyzing the utility of energy calculations as they are applied to a burst-prone coal mine.

From all of the possible numerical formulations (finite element, finite difference, discrete element, etc.), the displacement discontinuity method (DDM), a variation of the boundary element formulation, was chosen because it shows the most promise for fulfilling the requirements stated above. The displacement discontinuity method essentially analyses the mining process as

stresses and displacements induced at the surface of a narrow slit in an infinite (or semi-infinite) elastic medium. This specific numerical formulation provides several advantages for analyzing bumps over other mathematical methods. Since the stresses and strains in the out-of-seam materials are not specifically calculated (unless requested), the number and complexity of the equations that are needed for determining the response of the seam are minimal; and therefore, the model can efficiently simulate relatively large areas of the mine and multiple trial mining layouts and sequences. Also, because of the numerical formulation of the DDM, calculating the energy changes associated with mining is fairly trivial, and since there are no arbitrary boundary surfaces in the DDM as there are in other numerical methods, the calculated energy changes are truly appropriate for an infinite (or semi-infinite) body.

The specific displacement discontinuity program chosen for performing the modeling work in this project was the program MULSIM. This program is in the public domain; and therefore, it is easily modified to suit our particular purpose and can be disseminated to interested parties. At the present time, the ability to calculate the energy changes associated with mining and the ability to accept non-linear properties for the seam material are being added to the program. Once these additions have been completed, a model of the field sites will be developed and calibrated to the specific geologic conditions in the mine by adjusting the input parameters to provide model results that match with the measured stress and displacements values from the field instrumentation. It will be possible to verify the utility of using energy calculations for predicting bumps by applying the calibrated model to several previously recorded bump occurrences. If the model proves successful in using energy calculations to predict coal bumps, it will be an extremely useful tool for simulating multiple trial mining layouts and sequences for minimizing bump hazards.

SUMMARY

The Bureau of Mines re-initiated coal mine bump research in the Southern Appalachian Basin due to a rash of bump-related fatalities and injuries in 1984 and 1985. A preliminary study identified the inherent benefits of retreat longwall mining over room-and-pillar retreat methods for bump control. However, violent pillar bumps have occurred in at least two longwall mines in the Southern Appalachian Basin in the past few years. Operating on the premise that longwall layout design changes can further advance the bump control aspects of retreat longwall mining, two different bump control longwall gate road configurations are under evaluation in a deep bump-prone coal mine in this area. Stress and deformation measurements from these instrumented areas have demonstrated that mining layout modifications can enhance bump control. However, to obtain a better understanding of the unusual strata behavior in the bump-prone strata of Southern Appalachia and to build upon the stress and deformation data collected on the gate road pillars, a field project specifically designed to characterize the longwall gob behavior at the study mine is underway. The ultimate goal of these combined research projects is to facilitate the development of a design procedure that can be used for minimizing the bump hazard of mines. This procedure will be implemented with a numerical model in order to allow for quick and easy simulations of multiple trial mining layouts and sequences.

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RECHERCHES SUR LES SECOUSSES DANS LES MINES DE CHARBON : OUTILS PERMETTANT DES CONCEPTIONS RATIONNELLES DE LONGUES TAILLES

A.A. Campoli, K.A. Heasley

Les secousses se produisant dans les mines de charbon, c'est-à-dire la violente rupture du charbon dûe à une contrainte trop forte, présentent un risque pour la sécurité des mineurs et de l'équipement minier des mines de charbon souterraines.

Les secousses se produisent lorsqu'on effectue des excavations entre des strates compétentes voisines. Le Bureau des Mines poursuit un programme de recherches pour développer un modèle permettant d'optimiser des conceptions minières minimisant les risques de secousses. Ce programme est fondé sur l'établissement des caractéristiques des propriétés physiques des strates de charbon sujettes aux secousses; ceci se fait par l'évaluation de divers agencements miniers à l'intérieur même des mines. A l'heure actuelle, on évalue deux configurations de voies de longues tailles, conçues pour contrôler les secousses, dans une mine de charbon profonde sujette aux secousses. Les mesures des contraintes des couches de charbon et des déformations des strates qui y sont liées démontrent que les contraintes des couches de charbon diminuent à mesure que la surface de résistance des piliers des voies augmentent. La réaction d'ensemble des strates aux ruptures produites par les excavations minières se caractérise également par l'évaluation de la charge du remblai et de la hauteur du sous-cavage. Toutes ces informations seront cruciales pour le développement et la vérification d'un modèle numérique exact.

GEBIRGSSCHLAGFORSCHUNG IN KOHLEGRUBEN: HILFSMITTEL FÜR DIE RATIONELLE ABBAUPLANUNG.

A. A. Campoli, K. A. Heasley

Gebirgsschläge, gewaltsame überspannungs-bedingte Ausbrüche der Kohle, stellen eine Gefahr für die Grubenbelegschaft sowie für die Ausrüstung in Untertagegruben dar. Gebirgsschläge treten auf, wenn zwischen benachbarten Schichten Abbau betrieben wird. Das Bureau of Mines verfolgt ein Forschungs-Programm mit dem Ziel, ein Modell für eine optimale Abbauplanung mit minimierter Gebirgsschlaggefahr zu entwickeln. Ziel dieser Untersuchungen ist die detaillierte Beschreibung der physikalischen Eigenschaften der zu Gebirgsschlägen neigenden Kohleschichten durch untertage vorgenommene Analysen verschiedener Abbauauslegungen. Zwei Abbaustreckenkonfigurationen, die zur Kontrolle von Gebirgsschlägen entwickelt wurden, werden zur Zeit in einer Grube, die eine gebirgsschlagreiche Vergangenheit hat, beobachtet. Messungen der Spannung in der Lagerstätte und der Verformung der Nebengesteinsschichten zeigen, daß die Spannung in der Kohle reduziert wird, wenn die Tragfläche der Abbaustreckenpfeiler erhöht wird. Die allgemeine Reaktion der Gebirgs-Schichten auf Gebirgsschläge, hervorgerufen durch Abbau, wird ebenso durch die Bewertung der Bruchzonenlast und Bruchhöhe charakterisiert. Die verschiedenen Untersuchungen sind zur Entwicklung und Bestätigung eines genauen numerischen Modells unabdinglich.

ИССЛЕДОВАНИЕ ОТСЛОЕНИЙ НА УГОЛЬНЫХ ШАХТАХ: РАЦИОНАЛЬНОЕ ПРОЕКТИРОВАНИЕ ДЛИННОГО ЗАБОЯ

А.А. Камполи и К.А. Хизли

Отслоения в угольных шахтах, эти сильнейшие угольные обвалы, обусловленные сверх-стрессом, представляют опасность для шахтеров и горного оборудования в подземных угольных шахтах. Отслоения случаются когда добыча производится между соседними компетентными пластами. Горное бюро США проводит исследовательскую программу с целью разработки модели усовершенствования конструкции шахт, которая доведет до минимума опасность отслоений. В основе этой программы лежит детальное описание физических свойств выбросоопасных угольных пластов посредством оценки на местах различных планов шахт. Две конфигурации нарезного штрека длинного забоя, которые были спроектированы с целью предотвращения отслоений, в настоящее время оцениваются в глубокой угольной шахте, известной своими прошлыми отслоениями. Измерения стресса в угольных пластах и связанной с ним деформации пластов показывают, что уровень стресса в угольных пластах уменьшается с увеличением опорной площади целиков нарезного штрека. В докладе также характеризуется, посредством оценки выработанного материала и высоты завала, общая реакция пластов к обвалам, вызванным добычей угля. Вся эта информация очень важна для разработки и проверки точной количественной модели.

煤矿突出的研究：长壁巷道合理设计的工具

A．A．堪坡里 及 K．A． 赫士里

在地下煤矿矿井，煤矿里的突出是由煤矿石在过力状态下的一种剧烈的破坏作用，这对矿工和井下设备的安全都是极大的威胁．突出发生于两个相邻强劲地层界面的开采过程．矿务局正在开展一项研究方案，旨于建立一个能优化矿井设计和降低矿石突出危害模式．此研究方案是基于一些不同设计矿井内有突出倾向的煤层物理性质的现场调查资料的．我们正在一个有矿石突出历史的深矿井内试用两个为防止突出而设计的长壁坑道设计图纸．从煤床应力和连带形变的测定值来看，在采空区运输巷道增加支承面能使煤床应力减少．地层对采矿引起的破坏的总体反应用废石载荷和淘空高度来衡量．最后，所有这些资料对于建立和验证一个准确的数值模型都是十分关键的．

Session IX

EXPLOSIVES

THE TESTING AND TRIALS IN COAL MINES OF EMULSION EXPLOSIVES IN THE UK

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ABSTRACT

The examination, testing, and pit trials of emulsion explosives in the UK are discussed.

These explosives, which are sensitised by microballoons, have different gas ignition characteristics from nitroglycerine sensitised types. This has caused the UK testing authority to impose a temporary increase in the severity of some of its test requirements.

Ease of initiation to detonation is less than with traditional explosives, and the stability of emulsions may still be a problem.

Other characteristics of emulsions which have been examined have not revealed any serious problems.

Pit trials have shown their performance to be similar to nitroglycerine sensitised explosives in depth of pull. Fragmentation is marginally better. Their most marked advantage is in the reduced make of fume.

Shotfirers found that the relatively soft cartridges made hole charging more difficult, but detonator insertion easier.

Following extensive pit trials, one type of P1 emulsion has been approved for general use as a Permitted explosive.

INTRODUCTION

Most safety explosives currently in use in British mines are sensitised by mixtures of nitroglycerine and ethylene glycol dinitrate (subsequently referred to as NG), and are tested for their safety features, e.g. incendivity in methane/air mixtures, in steel apparatus that was devised during the course of their development. Appendix 1 outlines the main tests for P1 explosives.

NG explosives range from gels containing about 30% of NG, the remainder being crystalline inorganic oxidisers and fuel molecules, to powders containing about 10% of NG.

Emulsion explosives differ from NG types both in composition and consistency. They are composed of micron sized particles of very concentrated oxidising salts surrounded by a thin layer of fuel in the form of wax and oil held in place by powerful emulsifying agents. The oxidisers and fuels require a sensitiser to initiate them to reaction, and this is achieved in the case of emulsion explosives by means of the "hot spot" phenomenon which occurs in

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the vicinity of density discontinuities such as micro-balloons. A number of mechanisms have been proposed for "hot spot" formation in solid or granular explosives. These include adiabatic compression of gas bubbles and inter-granular friction (Taylor 1952), viscous and elastic flow effects, and shock interaction with density discontinuities (Mader 1976). This last category gives rise to effects which include jetting, void collapse, shock collision, and shock reflection. It seems likely that all these processes can make a contribution to causing the required localised temperature increases, and hence rapid initiation of reaction in emulsion explosives.

The microballoons normally used as sensitisers in emulsion explosives are made of either plastic or glass, and occupy about 30% by volume of the composition. The consistency of the emulsion can be varied, and the density can be changed by altering the concentration of microballoons.

The velocities of detonation of emulsion explosives are in the range four to five thousand metres per second which is considerably higher than the two to two and a half thousand metres per second of currently approved NG group P1 explosives. It therefore seemed appropriate to be cautious when testing this new type of explosive in tests developed for NG explosives, because of their relatively high velocity of detonation, and hence faster energy release rates. The rapid rate of release of energy in emulsions is due to its small droplet size which provides a very large initial contact area for reaction between fuel and oxidiser of about 3000 m^2/kg.

FACTORS AFFECTING SAFETY IN GAS

In general increasing the weight of an explosive charge detonated in a steel test apparatus increases the probability of causing an ignition of gas. However, studies of detonating mining safety explosives of the NG type, (Taylor 1949) showed that the propensity to ignite methane/air mixtures when charges of fixed weight were fired from cannons in the test galleries was generally reduced by reducing power output and increasing the proportion of flame inhibitor, such as common salt, in the composition. Increasing velocity of detonation did not appear to have a marked effect on the incendivity of an explosive in cannon tests, but did increase incendivity when explosives were detonated freely suspended in gas.

HISTORY OF TESTING EMULSIONS IN THE UK

The first experimental emulsion explosives to be tested in the UK were manufactured overseas. These explosives contained no recognisable flame retardant, such as salt, and caused some very violent ignitions during testing. Some cartridges failed to detonate with a number eight strength detonator, having a base charge of 4.5×10^{-4} kg of PETN, and as a consequence failed the test. These early compositions tended to fail in the 0.8 kg stemmed shot test, which is carried out as described in Appendix 1. Experimental compositions containing a flame retardant were then submitted for test and the violence of the ignitions was reduced, as might be expected, because of better flame inhibition and lower power.

Subsequently further compositions, containing varying amounts of flame inhibitor were examined; of these, several failed due to failure to detonate, but others passed the full official test.

A UK manufacturer has had several emulsion explosives tested, and one composition passed the full official group P1 test in July 1987. Trials of this explosive, which are taking place at selected collieries are

described later in the paper. This explosive is sensitised with glass microballoons, contains a flame inhibitor, and caused no ignitions during official testing. It was found that charges of the explosive that were substantially heavier than the normal test weight of 0.142 kg were required to cause ignitions in the blown-out-shot test, which is carried out as described in Appendix 1.

CONCENTRATION OF METHANE IN THE TEST GAS

Experience has shown that 9% methane/air mixtures are the ones most easily ignited by all types of NG explosives when detonated in the testing galleries. This point has recently been checked in connection with the development of emulsion explosives. Four different emulsion explosives have been fired in the test galleries, at fixed charge weights, in methane/air mixtures containing 8%, 9%, and 10% methane, and it has been found in all cases that the nine percent mixtures are the most easily ignited. Since the explosives differed in composition the fixed charge weight for each explosive was set at a level that would give approximately 50% ignitions in 9% methane/air mixtures.

DIAMETER EFFECTS

A limited number of tests comparing cartridges of 32mm and 37mm diameter indicate that the larger diameter cartridges tend to be slightly more incendive at a given charge weight.

ADJUSTMENT OF TEST CRITERIA

The violent ignitions which occurred during the testing of some of the early overseas compositions, and their relatively high velocities of detonation caused the UK Testing Authority to reconsider the test requirements for emulsions submitted for test as group P1 explosives. It was agreed that until more experience and knowledge becomes available the blown-out-shot test, in which up to thirteen ignitions are allowed in twenty six shots, would be set such that no more than six ignitions would be allowed; this change of test condition means there is a significant increase in severity at the 95% confidence level. The stemmed shot test requirement was also changed temporarily so that no ignitions in ten shots are required rather than the normal no ignitions in five shots; this requirement was changed because there has been a tendency for explosives of the non-NG type to cause ignitions in the stemmed shot test. These extra requirements should help to prevent the introduction into safety lamp mines of explosives that might pass existing tests but subsequently prove dangerous in use; experience may show that they can be relaxed in due course.

COAL DUST SHOTS

Tests in coal dust as described in Appendix 1 have been carried out on emulsion explosives. There have been no ignitions with any of the explosives tested.

Some tests have also been carried out in which the explosive has been detonated in the test cannon into a pre-formed coal dust cloud such that the coal/air mixture was approximately oxygen balanced, and again no ignitions were observed, whereas ignitions have been observed with corresponding NG compositions.

INITIATION OF DETONATION

All NG mining safety explosives have been subjected to gas ignition tests using No. 6 strength detonators containing a base charge of 2.5×10^{-4} kg of PETN, which until recently was the only type approved for use in gassy mines. However, detonators containing a base charge of 4.5×10^{-4} kg of PETN are now also approved, and are gradually being introduced into mines; these will provide more powerful and

reliable initiation for emulsion explosives.

AGEING OF EMULSIONS

Samples of overseas manufactured emulsion explosives that have successfully passed the group P1 test have been kept in our magazine and their detonator sensitivity examined at regular intervals using a standard range of detonators of steadily increasing size of base charge. It has been found that the detonator sensitivity of some emulsions has deteriorated within three months of manufacture to a point where they could not be detonated, even with a detonator having a base charge of 8×10^{-4} kg of PETN. In some cases the loss of sensitivity to detonator initiation was associated with a phenomenon termed "egg-shelling" which arises as a result of the instability of the emulsion.

The UK testing Authority has not yet had sufficient experience of emulsions to be clear about the effects of ageing on detonator sensitivity, but No. 6 strength detonators reliably initiated a freshly made group P1 explosive that passed the official test. Samples of this explosive were stored for subsequent testing to examine the effects of ageing on sensitivity. After six and twelve months determinations of velocity of detonation and strength gave almost identical results to those obtained during testing for approval, but cap sensitivity had decreased slightly.

AIR GAPS AND STRENGTHS OF EMULSIONS

Measurement of air gaps, and strengths in the ballistic pendulum, indicate that results for group P1 emulsions are similar to those for existing NG group P1 explosives.

CANNON BORE EROSION

Because of higher velocities of detonation it seemed likely that emulsion explosives might cause more severe erosion of the bores of test cannons than corresponding NG types, and this could affect test results, particularly if erosion is relatively rapid. Obviously it is difficult to obtain precise estimates of the extent of bore erosion because of the pitting that occurs especially in the region where the charges are placed. However, estimates of the extent of erosion are being made with a three legged micrometer which is used to measure the apparent diameters of cannon bores at 50mm intervals along their lengths. So far it appears that wear is not increased significantly by firing emulsion explosives containing flame inhibitors.

The effect of bore erosion has been examined in the past and a number of different conclusions reached. (Grimshaw & Payman 1931) concluded that the bore volume could be increased by as much as 50% before test results were affected, however, in a later review, (Coward 1962) the French and German testing authorities were quoted as saying that a volume increase of about 20% is allowable before results are affected, and this seems to be in agreement with recent experience of testing explosives in the UK.

Firm rules about changing cannons because of wear cannot be arrived at in the UK because we test a number of different groups of explosives of different strengths, and in different configurations in the cannons. With NG types most wear is caused by group P1 explosives and least by the weak group P4 explosives. In general, cannons are useable for about five hundred shots.

EARLY TRIALS UNDERGROUND

Trials of the first emulsion explosive (a non-permitted variant) were carried out in a salt mine in September 1985. During a visit to the site in October 1985 it was apparent that the 37mm x 400mm

cartridges in plastic film sealed by steel end clips were detonator sensitive and gave good coupling through 5 to 7 cartridges in 5m long horizontal holes. Similar trials with narrow diameter cartridges were carried out in two gypsum mines, a tin mine and a rock tunnel at a quarry. The first permitted explosive was tried in a small naked light coal mine and a barytes mine.

PIT TRIALS OF APPROVED EXPLOSIVE

Comparative trials are not easy to organise because of the nature of cross measures drifting. It is difficult to ensure the same number of shotholes or even identical strata. British Coal Corporation, and HSE agreed on trials at several mines in varying conditions in different parts of the country. The trials were to take place in a period of one year, and include 250 tonnes of explosive. The explosive was considered to be as effective as a well established NG-sensitised type, but there were difficulties in charging the shotholes and cartridge diameter was subsequently reduced from 37mm to 35mm.

Environmental monitoring carried out by British Coal Corporation at two collieries showed that the peak concentrations of carbon monoxide produced by the emulsion explosive were on average 40% of the peak concentrations of the NG-sensitised type. Also the volumes of nitrogen dioxide and oxides of nitrogen produced were on average 2.5% and 20% respectively of the amounts of those gases produced when using an NG-sensitised explosive.

British No. 6 strength detonators were used in the trials, and although no misfires were reported, the trials were continued with stronger detonators, recently approved for use in safety lamp mines and designated No. 8 strength.

The trials were subject to field support by the explosives manufacturer, the regular submission by the mine manager of appraisal reports to HM Inspectorate and a 'teach-in' at the mine concerned. Environmental monitoring, including fume sampling, was also undertaken during shotfiring. These trial arrangements worked very well and were enthusiastically carried out at the mines concerned. During the monitoring exercises support was provided by British Coal Corporation Scientific Departments and colliery ventilation staff. It is significant that the first two field trials were in Areas or mines where accidents had occurred to shotfirers who had been overcome by nitrous fumes following shot-firing.

EXTENDED TRIALS UNDERGROUND

The trials were slow to be taken up although sufficient experience was obtained to widen them. The decision was made to extend them for a second year to any mine which wished to participate, with the proviso that the trial conditions were maintained. During 1988 some 32 coal mines held trials varying in amounts from 100 kg at a licensed mine to 27,000 kg at a British Coal mine. The greatest use has been at a Potash mine where there is a flammable gas outburst hazard. Additionally 13,000 kg were used in a salt mine and 75 kg used as primer charges at two quarries. Comments by shotfirers generally confirmed that the explosive is effective and produces very little after fume. The explosive compared well with the NG-sensitised type, and yielded depths of pull up to 2.4m in stone drifts. Fragmentation was either satisfactory or excellent, with a reduced need to break large stones manually in the rock pile.

COMMENTS ON THE FIELD TRIALS

Handling and loading the emulsion explosive into the shotholes revealed that slightly different techniques were required to those

acquired over many years with other permitted explosives. At first this caused problems. When first introduced the 0.35 kg cartridges of explosive were physically limp to handle and almost twice the length of the 0.25 kg cartridges previously used. Some existing bulk transport box or compartment sizes and also miners personal carrying cases were not suitable. The cartridges are secured at each end by a steel clip.

The sausage-shaped end of the cartridge does not present a hard face to any debris which may have been left in a shothole and a wedge action may result during charging, particularly in shotholes inclined downwards. The shotfirer may burst the cartridge if he persists in ramming and one reason for some sockets left in the face after firing may result from the explosive not being inserted to the back of the shothole. This has led to requests from shotfirers for stronger cartridge film and there may be some validity in this request but greater care in cleaning out the shothole helps. One benefit of the softer explosive composition is that the shotfirer only has to pierce the cartridge film prior to inserting the detonator easily into the primer cartridge. The fact that the explosive is only now manufactured in the 0.35 kg size at present is considered a disadvantage by shot-firers where applications may require less than a complete cartridge. For most applications this is not a problem, although the manufacturer has the matter under review for other groups of safety explosives. One reason for not opting for the conventional 0.2 kg cartridge is that in most applications the full charge weight of 1.4 kg is obtained in four cartridges. The explosive is not as sensitive in coupling through the gaps between cartridges, and the greater the number of gaps the greater the chance of misfire. However, cartridge separation has not proved to be a problem.

PROBLEMS

Misfires have been reported at 4 mines. One of these was a procedural error by the shotfirer and the other three were complete failures of initiation of the explosive. Tests later proved that the explosive was insensitive to No. 8 strength detonators and the manufacturer withdrew the material. Although samples have detonated successfully at 18 months age the explosives involved in the misfires appeared to have aged prematurely and had a crystalline eggshell appearance. This matter is coupled to comments by one of HM Inspectors inspecting the trials. The Inspector determined difficulties in charging the shot-holes were due to misalignment or deviation within the holes caused by a hand held drilling machine and by leaving the holes in the face drilled for a long period of time before charging. The softness of the explosive was also apparent because the explosives at the bottom of the box were deformed by flattening and those boxes stored upright resulted in cartridges having bulges at the bottom. These matters have largely been corrected by the manufacturer by process improvements which have resulted in a more rigid cartridge. There have been fewer complaints since and no further problems with initiation.

FINAL APPROVAL

A review of the trials has resulted in the emulsion explosive being Approved for general use as a Permitted Explosive on 7 January 1989.

THE FUTURE

It seems likely that over the next few years attempts will be made to replace all NG explosives with emulsions. An approved group P1 emulsion explosive has been formulated and trials so far have proved satisfactory.

The next step would appear to be the

development of a group P3 explosive possibly by reducing the amount of ammonium nitrate and increasing the sodium chloride content.

The development of a group P5 explosive may be more difficult because of the ease with which the present formulations deflagrate and it could be some time before a solution to this problem is reached. To produce a group P4/P5 emulsion may require the explosive to have a much lower velocity of detonation than the explosives tested so far but emulsion explosives having velocities of detonation below two thousand metres per second have been produced using foamed polystyrene spheres of large particle size as sensitisers and density control agents.

CONCLUSIONS

The examination, testing, and pit trials of emulsion explosives in the UK have been discussed. These explosives, which are sensitised by microballoons have different gas ignition characteristics from NG types. This coupled with some severe ignitions that occurred during testing of early compositions, caused the testing authority to increase temporarily the severity of its test requirements, until experience of the use in mines of these explosives shows this to be unnecessary.

It has also been apparent that ability to be detonated is considerably less than with corresponding traditional explosives, and there are still stability problems.

Other aspects of testing which have been examined have not revealed any problems that have not or cannot be solved.

Pit trials have shown their performance to be similar to nitroglycerine sensitised explosives in depth of pull. Fragmentation is marginally better. Their most marked advantage is in the reduced make of fume.

Clearly progress is being made and it seems likely that in time these newer types of explosives will replace existing NG types completely, leading to a worthwhile increase in safety in their transport, storage, and use.

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APPENDIX 1

DESCRIPTION OF THE TESTS FOR INCENDIVITY IN METHANE/AIR AND COAL DUST/AIR MIXTURES

The official tests for permitted explosives are described in the HSE Testing Memorandum No. 2 (revised 1988). The main tests for group P1 explosives are summarised here.

GROUP P1 EXPLOSIVES

There are three test conditions for

these explosives.

BLOWN-OUT-SHOT TEST

A 0.142 kg charge of the explosive is placed at the back of the cannon of bore 55mm and length 1200mm. The detonator is placed at the back end of the charge (inverse initiation) which is fired into a gallery containing nine percent methane/air. In order to pass, an explosive must give not more than thirteen ignitions in twenty six shots. This simulates the hazard of a "blown-out" shot.

STEMMED SHOT TEST

A 0.8 kg charge, which is the maximum weight of this group of explosives that may be used when firing in or near a coal face, is fired from the cannon, the charge being stemmed with a dry fireclay plug 25mm in length. The detonator is placed at the end of the charge nearest to the mouth of the cannon (direct initiation). There must be no ignitions in five shots.

COAL DUST TEST

The above test is repeated with the explosive firing into a gallery containing a deposit of coal dust. Again there must be no ignitions in five shots.

The stemmed shot and coal dust tests represent practical conditions except that little or no stemming is used.

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ESSAIS ET EPREUVES D'EXPLOSIFS EMULSIONS DANS LES MINES DE CHARBON DE GRANDE-BRETAGNE

N.L. Heathcote, M. Kennedy, F. Thompson

Le présent rapport décrit les examens, les essais et les épreuves des explosifs d'émulsion effectués dans les mines de Grande-Bretagne. Les caractéristiques d'inflammation au gaz de ces explosifs - sensibilisés par microballons - sont légèrement différentes de celles des explosifs sensibilisés par nitroglycérine. C'est pourquoi les organismes d'essais anglais ont été obligés d'augmenter temporairement les critères de rigueur de certains de ces essais. Les détonations se déclenchent moins facilement qu'avec les explosifs traditionnels et la stabilité des émulsions peut encore présenter un problème. Les autres caractéristiques des émulsions examinées n'ont pas révélé de sérieux problèmes.

Les résultats des essais de mines effectués ont démontré que ces performances sont semblables à celles des explosifs sensibilisés par la nitroglycérine en profondeur de traction. La fragmentation est marginalement meilleure. La quantité moins élevée de fumée créée constitue leur avantage le plus marquant. Les tireurs ont trouvé que la relative souplesse des cartouches rendait plus difficile le chargement des trous, mais que l'insertion du détonateur était facilitée. A l'avenir, les émulsions pourront remplacer totalement dans les mines de Grande-Bretagne tous les explosifs existants sensibilisés chimiquement.

VERSUCHE UND EXPERIMENTE MIT EMULSIONSPRENGSTOFFEN IN BRITISCHEN KOHLEGRUBEN.

N. L. Heathcote, M. Kennedy, F. Thompson

Der Vortrag berichtet über Untersuchung, Tests und Untertageversuche mit Emulsionssprengstoffen in England. Diese durch Microballons aktivierten Sprengstoffe unterscheiden sich in ihren Gasentzündungseigenschaften geringfügig von Sprengstoffen, die mit Nitroglycerin aktiviert werden. Aus diesem Grund hat die britische Zulassungsbehörde eine zeitweise strengere Überwachung der Testbedingungen erlassen. Die Detonation läßt sich mit diesem Sprengmittel weniger leicht einleiten als mit traditionellen Sprengmitteln, und die Stabilität der Emulsion kann auch dann noch problematisch sein. Andere Eigenschaften der Emulsion, die beobachtet wurden, zeigten keine größeren Schwierigkeiten.

Untertageversuche zeigen, daß die Leistung im Hinblick auf Zugtiefe ähnlich der der mit Nitroglycerin aktivierten Sprengstoffe ist. Die Zertrümmerungs-Wirkung ist marginal besser. Der bemerkenswerteste Vorteil ist die verringerte Rauchgasentwicklung. Sprengmeister bewerten das Besetzen der Bohrlöcher wegen der relativ weichen Patronen als schwieriger, das Einbringen des Zünders allerdings als leichter. Es ist zu erwarten, daß die Emulsionen in der Zukunft alle bisherigen chemisch aktivierten Sprengstoffe in britischen Gruben ersetzen werden.

ИСПЫТАНИЯ ЭМУЛЬСИОННЫХ ВЗРЫВЧАТЫХ ВЕЩЕСТВ В УГОЛЬНЫХ ШАХТАХ ВЕЛИКОБРИТАНИИ

Н.Л. Хифкоут, М. Кеннеди и Ф. Томрсон

Доклад посвящен проверке, а также лабораторным и полевым испытаниям в карьерах эмульсионных взрывчатых веществ в Великобритании. Эти взрывчатые вещества, чувствительность которых была повышена микробаллонами, имеют газозажигательные свойства, слегка отличные от нитроглицериновых типов. Вследствие этого, временно были повышены требования к полевым испытаниям. Детонация достигается в этих веществах не так легко, как в традиционных взрывчатых веществах, и эмульсионная стабильность попрежнему остается проблемой. С другими изученными в работе свойствами эмульсий никаких серьезных проблем обнаружено не было. Полевые испытания показали, что эмульсионные взрывчатые вещества работают на глубине так же хорошо, как нитроглицериновые. Фрагментация при их использовании слегка улучшается. Их наиболее заметное преимущество проявляется в меньшем количестве дыма. Подрывники заметили, что относительно мягкие гильзы затрудняли минирование, но облегчали помещение детонаторов в шпуры.Возможно, что с течением времени эмульсии заменят все используемые в английских шахтах взрывчатки с химически повышенной чувствительностью.

大不列颠联合王国在矿井内乳胶炸药的检定和试验

N．L．茹士可特，M．肯乃迪，F．汤姆孙雷

本文将讨论大不列颠联合王国乳胶炸药的分析，检定和坑道试验．这种炸药用微球引爆，与硝化甘油引爆的炸药有稍微不同的燃烧性能．这使英国检验当局对其某些试验要求附加了一些更严格的暂行条例．这种炸药不如普通炸药容易引爆，而且乳胶的稳定性尚是个问题．在检定过的乳胶的其他性能里还未发现什么严重的问题．

坑道内试验结果指出，这种炸药的爆破深度与硝化甘油引爆的炸药大致相同，但破坏力会稍大些．这种炸药最大的优越性是硝烟少．爆破工人觉得这种炸药比较软，不容易装入炮眼，但比较容易安插雷管．在大不列颠联合王国的矿井中，这种乳胶炸药迟早会取代所有现用化学引爆炸药．

LIGHTNING AND SAFETY IN SHALLOW COAL MINES

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ABSTRACT

The problem of lightning causing dangerous disturbances underground in shallow coal mines in certain areas of South Africa is discussed. A brief background to the problem is given. The criterion of a shallow coal mine is explained and exceptions outlined.
The mechanisms responsible for the energy transfer from a lightning strike into the mine are described in some detail. A detailed explanation of the new theory of how this energy then couples into the detonator circuits is given.
Finally, a yardstick by which a particular mine is judged in terms of seriousness of the problem is given. Several new recommendations are made to reduce the hazard due to lightning in these mines.

1 INTRODUCTION

The problem of lightning influencing safety in certain shallow South African coal mines was identified several years ago (Geldenhuys et al, 1985). A number of reported underground incidents have been related to the passage of thunderstorms. These incidents include electric shocks, sparking from underground equipment, premature and unexpected detonation of explosives and methane gas ignitions. This is a serious problem, with a risk of serious injury or possibly death, to personnel in the vicinity of such an incident. The grave concern of the industry regarding this safety hazard culminated in the formation of a sub-committee of the Explosions Hazards Advisory Committee of the South African Coal Mining Research Controlling Council in 1978. This committee's task has been to carry out an indepth study of the problem and to formulate appropriate safety measures.

Thus far work has been carried out in two directions; a purely theoretical study of the problem has been undertaken, and secondly, an experimental program measuring and identifying the transients produced underground by surface lightning is underway (Geldenhuys, 1988). This paper will concentrate on the former. As a starting point, some background to the problem and the relevant physical properties of lightning and how they are modelled will be briefly discussed. Thus far, several mechanisms have been identified as possibly being responsible for the transfer of energy from the lightning strike into the mine. The paper will describe each in some detail, giving the key points necessary to solve the problem. Several case studies will be tabulated and will serve as a guide line when drawing up a "risk factor" scenario. The paper will conclude by suggesting further steps which need to be taken down the road toward solving the lightning hazard in shallow coal mines problem. The work described has been carried out under the auspices of the Lightning and Stray Current Sub-committee of the Explosions Hazards Advisory Committee.

2 LIGHTNING INCIDENTS

After an extensive analysis of reported lightning related incidents, Zeh (1987a) confirmed that most incidents still occur in the Eastern Transvaal, as reported by Geldenhuys et al (1985). Here the regional ground flash density is high, approximately 7 to 9 ground flashes per square kilometer per year, and the mines are

Date	Location	Nature
08/04/80	Douglas Colliery, Van Dyks Drift	5 hole exploded. 1 person killed. Electric shock felt 50m away. (Possibly illegal use of the methanometer?).
09/09/80	Landau III	Detonation of Shotholes in 4 charged faces. Depth of 48m. Shocks recorded. Sparks observed. Within 30m of underground conductive system.
04/02/82	New Largo	Shocks recorded. Flash observed between borehold and power cable. Depth 34m.
18/03/82	Springbok	Electric shocks. Depth 25m.
06/11/82	Ermelo	Methane explosion, sinking incline shaft with one fatality.
30/12/83	Douglas Colliery	5 shotholes fired. Flashes observed. Near borehole. Depth 20m.
20/2/84	Douglas Colliery, Van Dykes Drift	Shothole explosion. Near borehole. Steel ladder used. Wet conditions underground. Shocks reported. Sparks observed.
23/10/84	Landau - SACE	1 out of 10 charged holes fired. No 5 delay detonator not connected. Dyke close by. 18h30.
30/09/85	Arnot Colliery	1 of 6 vertical pop-holes detonated. Type 1 high energy IED. Stat-safe delays.

Table 1 Several Lightning Related Incidents. After Zeh (1987a).

generally shallow. Incidents naturally occur in the summer months when the incidence of thunderstorm activity is highest. Table 1 lists a number of recent incidents, giving all relevant detail. Lightning related incidents are generally restricted to a depth of 100m. If one examines the typical geological fingerprint of this area, a soil-sandstone-shale-coal-sandstone-shale-coal pattern is observed (Zeh, 1987b). The incidents occur in the coal layers which have a considerably higer resistivity than the ground layers; 15 to 1000 $k\Omega\ m$ in relation to 0.3 to 1.0 $k\Omega\ m$. Since the depth of penetration of lightning current, which has high frequency components, is proportional to resistivity, the thickness of the ground layer above the seam is a restricting factor. Where there are dykes or geological faults of high resistivity, this criteria no longer holds and a different scenario is found.

Zeh (1987b) noted several important features of the problem. The general working conditions and the stage of preparation of the face appear to be important. Only a few detonators in a face fire, due to the local distribution of lightning currents and/or to the mode of current coupling into the detonator circuit. An incident can be very extensive, covering distances up to 800m between sites of disturbances. A feature common to several case studies is the proximity of a borehole, dyke or geological fault. The location of conductive structures underground is also important. The significance of these factors will become clear during the following discussion. Their relevance in determining a "risk factor" will also be shown.

3 COUPLING MECHANISMS

Several important features of a typical shallow, South African coal mine as well as possible lightning disturbance mechanisms are illustrated in Figure 1. Three basic mechanisms, shown in the diagram, are thought to be responsible for energy transfer into the mine. These are direct strike, direct conduction and electromagnetic induction. For all three the energy transfer may result ultimately in a voltage surge propagating through the mine via any conductive system present, for example the conveyor system, water pipes, housing of electrical equipment, armouring of power cables, etc.

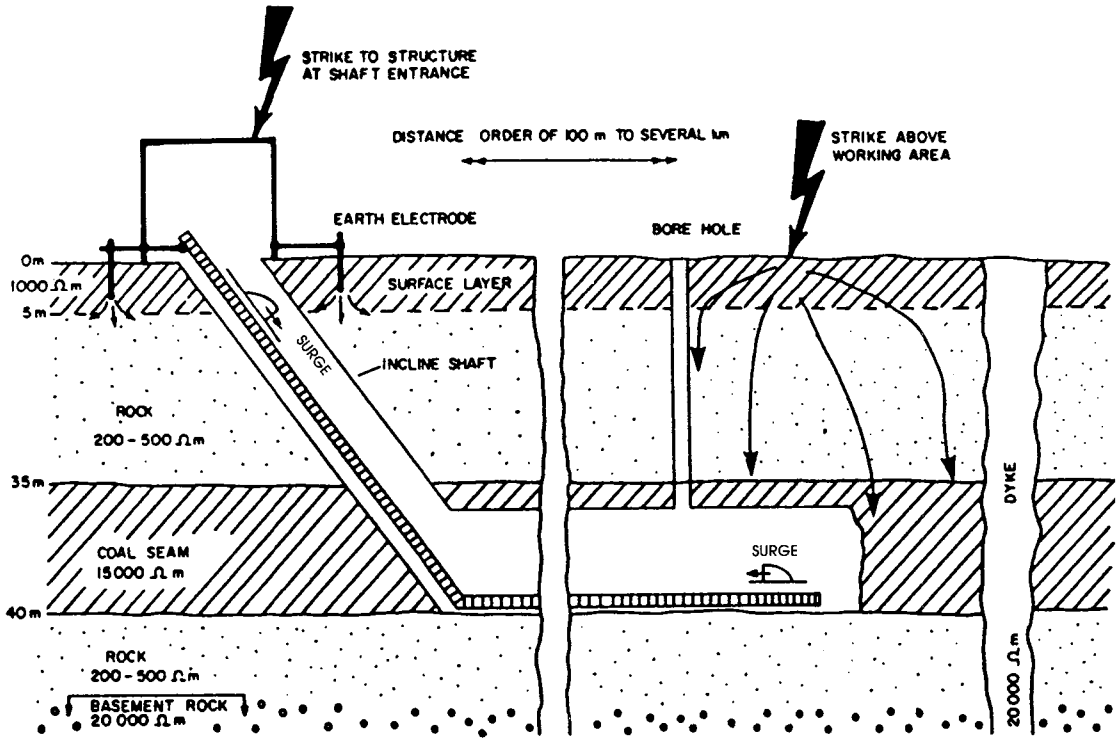


Figure 1. Simple representation of the colliery scenario. After Geldenhuys et al (1985)

Conventional transmission line theory governs the behaviour of this surge, with the surge being attenuated as it propagates through the mine, due to losses in the transmission path, and reflected at any change in path impedance i.e. any parallel path junction or termination. The attenuation is determined by, among other factors, the line impedance at that frequency and the line configuration. The reflection co-efficients are given by

$$k = \frac{Z_t - Z_0}{Z_t + Z_0} \qquad \text{(Kraus, 1985)} \qquad (1)$$

where Z_0 is the characteristic impedance of the line and Z_t is the "load" impedance.

The attenuation factor is modified by the quality of contact between the conductive system and the rock strata as this introduces further parallel paths. This contact, however, is usually poor (Geldenhuys et al, 1985) and the wave can still propagate to the furthest extremities of most mines.

The end of the conductive structure usually constitutes an open circuit and the surge would typically be reflected and doubled (Z_t in equation (1) is infinity). This leads to an increased concentration of potentials. Due to the fact that most conductive structures underground have a branching nature this is a complex transmission line problem best solved making extensive use of computer facilities. The ways in which the travelling wave is initiated vary for the different mechanisms as discussed later. Coupled to this is the fact that the lightning parameters used differ for the different mechanisms as explained below.

3.1 Lightning parameters

The magnitude and shape of the lightning current pulse are important when considering all three mechanisms. Lightning current magnitude has a probabilistic distribution as shown in Figure 2 (Anderson and Eriksson, 1980) which should be taken into account if meaningful results are to be obtained. As a starting point, however, the highest expected current of approximately 140 kA may be used to determine the "worst case" solution.

The shape of the lightning pulse, shown in Figure 3 (Anderson and Eriksson, 1980) and its corresponding frequency spectrum are also relevant. For the "direct strike" and "conduction" mechanisms these are important in the form presented here as they determine the "shape" of induced voltage and conduction current directly.

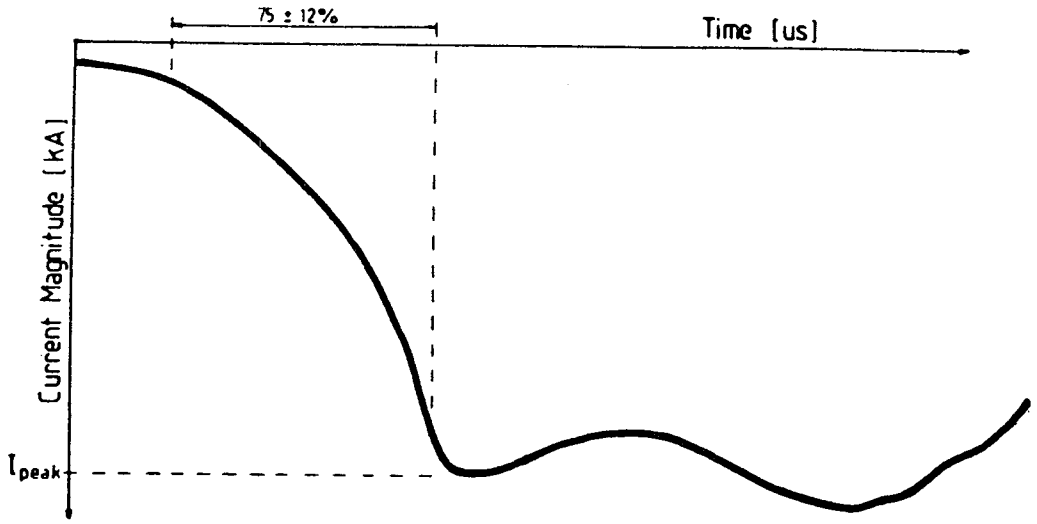


Figure 3. Lightning Current Pulse. After Anderson and Eriksson (1980).

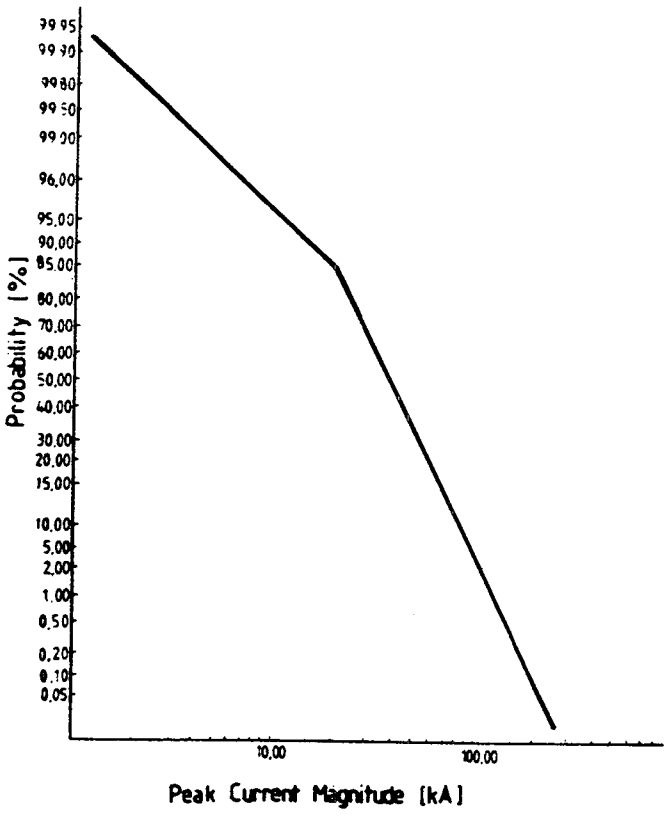


Figure 2. Distribution of lightning current magnitudes. After Anderson and Eriksson (1980).

The "electromagnetic" mechanism is, however, more complex as the electromagnetic field associated with such a lightning current must be determined. This is achieved by finding the transient solutions to Maxwell's equations.

The lightning strike is represented as a current element and if the sperical co-ordinate system shown in Figure 4 is used then the transient fields at $P(z,r)$ are given by the equations in Table 2 (Waters, 1983). An order of magnitude estimation is obtained when these equations are solved.

Once the source, lightning, has been accurately modelled the energy transferred into the mine via each mechanism can be determined as outlined below.

3.2 Direct strike

A direct strike to a structure at any shaft entrance or to the shaft itself constitutes the most simple transfer mechanism. Here the voltage surge discussed previously is the only energy transfer mechanism involved. The earth electrodes around the shaft and the conductive structures going into the mine form parallel paths. The lightning current will divide between these depending on their relative impedance magnitudes. The electrode impedance is usually a much lower magnitude, of the order of a few ohms, which is further reduced under transient conditions, and can easily be calculated. The impedance of the conductive structure is very much more complex and generally not well

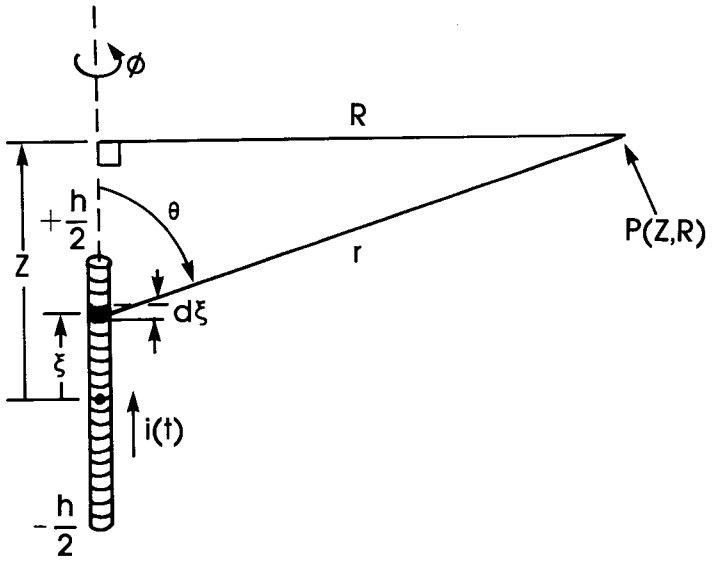


Figure 4. Co-ordinate system for calculating **E** and **H** for a Finite current. After Waters (1983)

$$\frac{dH_\Theta}{dt}(z,R) = \{\frac{di(t)/dt}{4\pi R}[\frac{u}{(R^2+u^2)^{1/2}}] + \frac{d^2i(t)/dt^2}{4\pi c}[\tan^{-1}(\frac{u}{r})]\}_{u=z-\frac{h}{2}}^{u=z+\frac{h}{2}}$$

$$E_R = -\frac{R}{4\pi}[\frac{\mu di(t)/dt}{(R^2+u^2)^{1/2}} + \frac{\eta i(t)}{(R^2+u^2)} + \frac{q(t)}{\varepsilon(R^2+u^2)^{3/2}}]_{u=z-\frac{h}{2}}^{u=z+\frac{h}{2}}$$

$$E_z = \frac{1}{4\pi}[\mu\frac{di(t)}{dt}(\frac{u}{(R^2+u^2)^{1/2}}) + \eta\frac{i(t)}{2} \{\frac{1}{R}\tan^{-1}(\frac{u}{R}) - \frac{u}{R^2+u^2}\} - \frac{3q(t)u}{\varepsilon(R^2+u^2)^{1/2}}]_{u=z-\frac{h}{2}}^{u=z+\frac{h}{2}}$$

with

r^2	=	$R^2+(z-\varsigma)^2$
$\sin\Theta$	=	R/r
u	=	$z-\varsigma$
$\cos\Theta$	=	$(z-\varsigma)/r$
$H\phi(t)$	=	magnetic field Am^{-1}
$E_R(t)$	=	radial electrical field Vm^{-1}
$E_z(t)$	=	transverse electric field Vm^{-1}
$q(t)$	=	$\int_o^t(t)\,dt\,C$ ie. charge and includes any initial charge
η	=	characteristic impedance of the medium Ω
h	=	length of current element m
μ	=	magnetic permeability Hm^{-1}
ε	=	electric permittivity Fm^{-1}
$i(t)$	=	current element A
c	=	velocity of light ms^{-1}

Table 2 Equations for the Transient Fields at $P(z,r)$.

known. Although for accurate results the "percentage current" should be calculated considering current division, as a starting point one can assume all the lightning current goes to earth via the electrode system. This causes the potential of the structure to rise in relation to a remote earth which results in the propagation of the voltage surge. The magnitude of the surge is calculated simply from the lightning current magnitude and electrode system impedance under transient conditions. The surge shape follows that of the lightning current.

3.3 Conduction

A more complex situation arises when the lightning strike position is away from the shaft. Here conduction and/or electromagnetic induction are the relevant energy transfer mechanisms.

For the case of conduction the conductive path through the interviewing rock and soil media is augmented underground by any conductive systems. The current magnitude and distribution is determined in the usual way considering ohms law in the frequency spectrum i.e. the initial current magnitude and the impedance of all conductive paths need to be considered. A voltage surge results from the steep potential gradients formed when this current distribution is distorted. Local variations in the terrain, such as dykes, boreholes and geological faults, the mine excavations and the underground conductive systems can all locally distort the current and initiate the surge. Due to the large number of conductive paths available this problem cannot be solved without extensive computer manipulation. A cased borehole provides an excellent conductive path form the surface into the mine.

3.4 Electromagnetism

The electromagnetic fields associated with the discharge of the lightning channel, and calculated as shown previously, penetrate through the overlying ground into the mine tunnels. The situation is complicated by the fact that the mine constitutes a stratified medium. To calculate the magnitude of the electromagnetic field in the mine tunnel or coal seam, Maxwell's equations are solved for each horizontal layer. Proper account of reflections and boundary conditions i.e. changes in permeability and permittivity, must be taken. Also the fact that the source itself is in a different medium must be considered.

As a first approximation the system can be treated as having perfectly homogeneous layers with smooth boundaries. The incident wave is, however, not a plane wave and the equations Wait (1962) developed have to be significantly modified. Once one is confident that these modified equations are correct, the boundaries can then be made irregular and dykes and faults of different permittivities and permeabilities can be introduced and their effect on the electromagnetic field distributions studied. This becomes an extremely complex problem, solvable only through intensive computation and careful thought.

The voltage surge induced into any conductive system situated in the electromagnetic field has magnitude given by (Kraus, 1985)

$$\mathbf{V} = \int \mathbf{E}.dl$$

where $\mathbf{V}$ = voltage induced V
$\mathbf{E}$ = Electrical field component Vm^{-1}
d = path of integration m

As well as this surge then propagating through the mine the electromagnetic field itself can propagate along the tunnel. A wave - guide effect may also come into play as the coal seam (or tunnel if the ratio of wavelenth to tunnel dimensions is high) of high resistivity, is surrounded by shale of low resistivity. The theory of waveguides with conducting material inserted (Kraus, 1985) is modified to suit a mine situation. A similar effect is often used during underground communications in coal mines (Austin and Lambert,1985). Here guided modes are important with natural stratification and incidental guiding structures playing significant roles. The frequencies are different to those associated with lightning but this avenue does warrant further research. Another area which also needs further consideration is that of a capacitive effect being present with the coal seam or tunnel acting as the dielectric.

3.5 Methane Explosions

The problem of lightning initiated methane explosions is an extremely important one. Methane explosions are believed to be initiated

when electric sparks with sufficient energy occur in the proximity to pockets of methane air mixtures with the appropriate proportions (Lightning and Stray Currents sub committee, 1986). The same energy transfer mechanisms as described above are believed to be significant. This area requires further investigation.

3.6 Coupling into Detonator and Exploder circuits

Zeh (1987b) postulated that except in the special case of conduction through the soil and rock above or behind the face, any currents must be electromagnetically induced into the circuits for all the energy transfer mechanisms concerned. The conductive systems which act as transmission paths for the travelling voltage surge now behave as "radiating antennas", with the eletromagnetic wave being radiated from the termination point. The magnitude of this wave depends on very many factors including the strength of the source i.e. voltage pulse magnitude. Hence the fact that the pulse is reflected and doubled at the termination is very significant. The amount of energy available for coupling into the exploder and detonator circuits also depends on many different factors, including the distance between the "radiating antenna" and the coal face. Thus the positioning of conductive structures relative to the face is extremely important. The radiation, induction and quasi-static field components could all be relevant depending on the distance between source and receiver. As stated earlier an electromagnetic field alone could transmit energy from lightning strike to the coal face.

For all cases of energy transfer the detonator and exploder circuits act as "receiving antennas" (Zeh, 1987b). The amount of energy extracted from the incident electromagnetic field depend on several factors including "receiver antenna" geometry, length and orientatation relative to the incident field vector. Hence the stage of preparation of the face, detonator placement and face area are all important. The energy coupled into the circuit is a maximum when the receiving antenna is made of single wires of a frequency dependent length but is greatly reduced by mutual coupling if the wires are twisted together. This "transmitter receiver antenna" postulate does explain several key factors noted in the case studies.

A natural progression after understanding the above principles is to determine which mines are at risk and to make recommendations to reduce this hazard to a minimum.

4 RISK FACTORS

The results of this research will lead to a better understanding of the hazard produced by lightning. Steps can then be taken to illiminate this problem. These steps may present a financial burden to a mine so it is important to identify where they need to be applied and how strictly they must be followed. Each mine is a unique situation and it is proposed that a "risk factor" criteria be developed. This will assist the mine manager in deciding (Geldenhuys, 1988):

- whether or not to install a lightning warning system
- when to stop handling explosives
- what risk there is of methane explosions
- whether or not to use safety detonators.

From our knowledge of the situation to date, both from a theoretical viewpoint and from analysing the case studies as depicted in Table 1, several conclusions can be drawn in this regard. These will briefly be discussed.

The first consideration is a climatic one, i.e. lightning intensity and rainfall figures are relevant. The higher the lightning ground flash density the greater the risk but the greater the rainfall figure the less the risk. Strata resistivity is inversely related to the rainfall and higher resistivities allow greater depths of penetration. There is an increase in resistivity during the dry season which could indicate that the beginning of the lightning season should be the most dangerous. Thus the risk factor is seasonal.

The geology and topography of the region is also important. The probability of a direct strike to a shaft which is surrounded by kilometers of flat terrain is higher than if there were other high points in the area. The number of dykes and geological faults is significant with risk factor increasing as this number increases. The geological strata are also relevant as is the soil resistivity (as well as the change with rainfall). A mine which has high resistivity rock and

soil is at a greater risk than one with lower resistivity over burden.

The mine excavations themselves are important. Generally, except where geological considerations dictate otherwise, a shallow mine would have a greater risk factor than a deeper one. The number and nature i.e. cased, uncased or filled, of boreholes influences the risk with many cased boreholes being undesirable.

As regards the shaft as stated previously its height relative to the surroundings is important. The risk will be greatest when several "tall" shaft structures have been used. Obviously if fewer such structures exist the risk will be lower. The quality of earthing at the shaft is very significant with a poor earth posing the greatest risk. The type of shaft structure should also be considered when determining the risk. A good conducting structure will present the greatest risk.

Once underground the general mining conditions become important. The risk would be highest where conductive structures are placed close to a working face. A damp mine would have a higher risk factor then a dry one as water could act as a conductive path. Obviously, also a mine with high methane emissions will need to take greater care. The nature of explosives could also be relevant and this is being investigated at present.

All the above factors related only to physical parameters of the mine and not to the energy transfer mechanisms. At present the main emphasis of the research is centred on clarifying the relative "risk factors" of all the energy transfer mechanisms. Both this risk factor and the risk factor determined by the actual mine could change as the mine develops and conditions change. Thus the risk factor must be easy enough to calculate so that a re-evaluation of the mine overall risk factor can be made on a regular basis.

The surge detector measurement program (Geldenhuys et al, 1987) could be used to verify the risk factor determined for an individual mine.

5 RECOMMENDATIONS

An extensive set of recommendations have already been issued from the Government Mining Engineer's Office under the guidance of the Lightning and Stray Currents sub-committee (Code of Practice, 1981). As this research program progresses it is envisaged that these will be updated if necessary. The issue here is twofold; understanding the mechanisms may lead to more extensive recommendations and defining a risk factor indicates the degree to which these recommendations need to be enforced.

From the study to date the recommendation of twisting the detonator wires together has been reinforced as this reduces electromagnetic energy transfer by increasing mutual coupling. Other recommendations which still need to be put through a feasibility test include:

- changing the termination points of all conductive system from an open circuit to a short circuit configuration, thus collapsing the pulse rather than doubling it.
- improving the contact between the conductive systems and the strata reducing the travelling pulse magnitude.
- working with the conductive systems placed at the maximum convenient distance from the face reducing the electromagnetic field strength at the face.
- never using metal ladders against the face, rather using wooden ones.
- providing a lightning conductor and earth mat near each shaft but separate from it thus reducing the probability of the direct strike mechanism.
- removing as much unused conductive material as possible, including old roof bolts and supporting wire which could themselves act as radiating antennas.
- improving the procedure of warning workers underground to stop charging up operations, i.e. ensuring the effectiveness of any early warning lightning detectors is not reduced if the telephone system is being used.

All the recommendations listed in the Code of Practice need to be implemented on all shallow coal mines in the Witbank area of South Africa until a reliable risk factor can discriminate between them.

6 CONCLUSION

The problem of lightning causing disturbances underground in shallow coal mines in the Eastern Transvaal, South Africa, is a very real one. Case studies (Zeh, 1987a) and experimental data (Geldenhuys, 1987) support the postulate that lightning currents do penetrate to sufficient depths to be of concern.

As this study continues and the mechanisms become well understood and quantified, hazardous mines and situations can be more definitely identified. Thus safety precautions may be enforced to improve the safety of the coal mine without unduly disrupting normal production, thus minimising the financial loss to the mine without compromising its high safety standard.

In the meanwhile the Code of Practice as it stands issued in 1981 has done much to remove the risk to personnel during a storm, provided it is followed correctly.

The problem of sparking causing methane explosions still exists so this work needs to be continued. Much work is still needed in this area but much progress has been made towards understanding and solving this problem.

7 ACKNOWLEDGEMENT

The author would like to thank all members of the Lightning and Stray Current sub-committee of the Explosion Hazards Advisory Committee for their support, both financial and otherwise, AECI South Africa for supplying the computer equipment used for many of the computations required and the Department of Electrical Engineering, University of Witwatersrand, South Africa for their continued advice and support.

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FOUDRE ET SECURITE DANS LES MINES DE CHARBON PEU PROFONDES

K.A. Zeh

Ce rapport constitue la suite de celui présenté à la Conférence de Sydney en 1985.

Il expose brièvement le problème de la foudre qui provoque des risques à l'intérieur des mines de charbon peu profondes. Les dangers proviennent de détonations prématurées, de chocs et d'explosions de méthane. Plusieurs incidents causés par la foudre ont fait l'objet de relevés, à partir desquels on a trouvé des facteurs communs. Notre rapport examine également les critères d'une mine de charbon peu profonde et indique les exceptions existantes.

Il contient principalement le détail des mécanismes responsables du transfert énergétique du gisement à l'intérieur de la mine. Il comporte les équations qui s'y appliquent ainsi que les résultats obtenus pour une mine de charbon "typique". Il fournit également une explication détaillée du couplage énergétique dans le circuit "chargé".

Enfin, il donne un standard permettant de juger de la gravité du problème dans une mine spécifique et fait plusieurs recommandations concernant les détonations prématurées et les explosions de méthane; ces recommandations permettent d'atténuer le risque provoqué par la foudre dans ces mines.

BLITZSCHLAG UND SICHERHEIT IN OBERFLÄCHENNAHEN GRUBEN.

K. A. Zeh

Dieser Vortrag ist eine Fortsetzung des an der Konferenz in Sydney im Jahre 1985 gehaltenen Referates. Eine kurze Einleitung befaßt sich mit den Hintergründen zu den Problemen und Gefahren, die sich durch Blitzschlag in oberflächennahen Gruben bilden. Die Gefahren entstehen durch verfrühte Detonationen, Erschütterungen und Methangasexplosionen. Verschiedene durch Blitzschlag hervorgerufene Vorfälle werden aufgelistet und einige gemeinsame Faktoren aufgezeigt. Das Kriterium für eine oberflächennahe Grube wird erklärt und die Ausnahmen davon werden erläutert.

Der Hauptteil des Vortrages befaßt sich mit einer detaillierten Beschreibung des Mechanismus, welcher für den Energietransport des Blitzes in die Grube verantwortlich ist. Eingeschlossen werden entsprechende, für eine "typische Kohlegrube" erhaltene Gleichungen und Ergebnisse. Eine detaillierte Erklärung für die Energieverkopplung in den "geladenen" Stromkreis wird gegeben.

Der Vortrag schließt mit Richtlinien, nach denen die Vorfallsgefahr für eine bestimmte Grube bewertet werden kann. Bestimmte Empfehlungen bezüglich der Frühzündung und Methangasexplosion werden erläutert, um die Gefahren durch Blitzschlag in solchen Gruben zu verringern.

МОЛНИИ И ПРОБЛЕМА БЕЗОПАСНОСТИ В НЕГЛУБОКИХ УГОЛЬНЫХ ШАХТАХ

К.А. Зе

Настоящий доклад является продолжением доклада, представленного на конференции в Сиднее в 1985 году. Кратко описывается история проблемы молний, являющихся причиной опасности работы в неглубоких подземных шахтах. Опасность возникает от преждевременных детонаций, шоков и взрывов метана. Приводится несколько случаев, связанных с ударами молний, и описываются некоторые общие факторы. Обьясняется, что имеется в виду под неглубокими угольными шахтами, и приводятся исключения.

В основной части доклада детально обьясняются механизмы, способствующие переходу энергии от удара молнии в шахту. Включены важные уравнения и примеры, выведенные для "типичной" угольной шахты. Дана детальная интерпретация энергии, соединенной в "заряженную" цепь. И, наконец, дан стандарт, по которому определяется серьезность проблемы в отдельно взятой шахте. Предложено несколько новых рекомендаций, основанных на преждевременных детонациях и взрывах метана и направленных на то, чтобы уменьшить опасность от удара молний в неглубоких шахтах.

浅煤矿矿井中的闪电与安全问题

K. A. 隋

本文为 1985 年悉尼会议上发表的文章的续篇.

本文简要讨论了浅层地下煤矿矿井中由闪电造成的危险情况. 危险是致于炸药的过早引爆, 震动和甲烷爆炸. 文章用表格型式列出了几个与闪电有关的事故, 并指出了其中的一些共同因素. 文中也说明了浅层煤矿的定义准则及例外情形.

文章的主体详细说明了闪电能量传递到矿井内原理, 同时还给出了为一个 "典型" 的矿井推导出的方程式和结果. 文章里详细介绍了闪电能如何感应到带电的线路里.

最后, 本文提出一种尺码来衡量矿井里此类事故的严重程度, 并例举了其在一矿井中的应用. 为了降低闪电对此类矿井造成的危害, 对炸药的过早引爆和甲烷爆炸问题也提出了一些新的建议.

AN INVESTIGATION INTO THE THERMAL BEHAVIOUR OF EMULSION EXPLOSIVE AND DETONATING FUSE

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ABSTRACT

A number of coal mines in India are on fire. A number of these fires are several years old and a majority of them are confined to near the outcrops. Preventive measures adopted to contain and control the fires were only partially successful at exorbitant cost. Removal of coal on fire with due precautions is now recognized as one of the attractive alternatives. The technique of removal involves blasting in hot overburden strata. In order to select safe and suitable commercial explosives for blasting, investigations by the authors are in progress to study the thermal behaviour at elevated temperatures up to 200° C. The paper details some of the interesting features observed so far.

A system to determine the thermal stability of full cartridge of explosive has been set up for the purpose. Physical explosive parameters at elevated temperatures were determined in the laboratory under field simulated conditions and also at the blasting sites. Variety of explosives including NG based, slurries, ANFO, emulsions, detonators and detonating fuses were investigated. NG based commercial explosives and detonators were unsafe at temperatures above 80° C and therefore, they were not permitted in hot holes. The detonating fuses were not safe at temperatures exceeding 120° C due to low thermal resistance of PVC sheathings. Slurries tended to loose their gel structure at temperatures above 80° C, however, they retained explosive properties up to 120° C and therefore, were used in hot holes. Emulsion explosives were the safest and could withstand temperatures up to 170° C. In general, use of the explosives at temperatures exceeding 120° C is not recommended. For blasting at temperatures exceeding 80° C special blasting methods were experimented by the authors for use of explosives.

INTRODUCTION

A number of surface coal mines are on fire and many underground workings are also sealed due to spontaneous heating and in some cases blazing fires. A substantial amount of money has been spent on preventive measures in the past and is also being spent at present to contain and control the mine fires. In some mines the preventive measures yielded good dividends. However, in many other cases they were not very effective because control at one place led to eruption at the other adjoining place. A few years back successful attempts were made to excavate the coal on fire thereby permanently removing the source from that particular site. Extraction of coal on fire involved removal of over burden rock by blasting which was heated due to the fire below.

Rock blasting in hot holes particularly under confined conditions, using commercial explosives and blasting agents is not considered safe because of the possibilities of premature detonation of explosives. Premature detonation took place in a coal mine while charging a NG based explosive in an apparently hot blast hole

killing six miners at the spot. (Gupta and Nabiullah 1985) investigated the thermal behaviour of NG based explosives and concluded that they were not safe at temperatures exceeding 80° C. The use of all NG based solid explosives in hot holes is now prohibited by regulation. They also studied the thermal stability of slurry explosives (1988). The study indicated that the slurries lost their gel structure at temperatures in excess of 80° C but did not detonate up to 150° C. The thermal behaviour of detonating fuses and detonators was also studied by (Nabiullah and Gupta 1987). It was observed that detonating fuses burned at 130° C under unconfinement but partially detonated under confinement. All electric detonators and relays exploded between 100-130° C at heating rates between 0.4-1.5° C per minute. (Sato et al. 1977) observed using a differential thermal analyser (DTA) that the decomposition of dynamite starts at 130° C and of slurries at 150° C. (Omura and Sato 1983) developed water gel explosive which was stable up to a temperature of 140° C for 24 hours. (Miron et al. 1976, 1979) observed a self sustained exothermal reaction in an aluminized ANFO mixed with some pyrite up to 131±5° C. Some of the prominent authors who investigated thermal behaviour of explosives include (Nanko et al. 1977), (Tanaka et al. 1977), (Ronald 1984), (Baner et al. 1984) and (Matsugama et al. 1977). (Schulz 1987) used a fibre optic probe to determine the velocity of detonation of explosives down the holes.

Investigations by the authors revealed the difference in results with small quantities of explosives tested on a DTA/DSC apparatus as compared with the results on full cartridges. Therefore, the thermal stability investigations were conducted either at the blasting sites or under simulated bore hole conditions in the laboratory using full explosive cartridges. The P and S wave velocities of hot overburden rocks were determined using seismographs to estimate the blastability index. It was observed that due to heat the micro and macro cracks appeared in the rock. A charge factor of 0.3-0.4 kg/m^3 was sufficient for good fragmentation of overburden rocks in fire areas as compared to 0.4-0.6 kg/m^3 in normal conditions.

EXPERIMENTAL PROCEDURE AND RESULTS

The following tests were done to investigate the thermal behaviour of explosives:-

(a) Drop and impact test on hot surface

(b) Detonation test on heating in confined and unconfined state

(c) Heat precariousness and kinetic study

(d) Blasting method in hot holes

(e) Influence of heat on dynamic elastic properties of rock mass.

DROP AND IMPACT TEST

The test procedure involved dropping of an explosive cartridge on a hot sand bed through a cement pipe from a height using a Cr-Al thermocouple. The experiment was repeated several times with gradual increase in temperature and time until the explosive burned or detonated. The experimental set-up is shown in Figure 1; the results are detailed in Table 1.

The impact sensitivity of explosives was determined using the conventional fall hammer apparatus. Steel hammers weighing 5 and 10 kgs were dropped from different heights on a 200 gm explosive. The height of fall at which detonation/ explosion occurred was recorded. The experimental set-up is shown in Figure 2 and the results are tabulated in Table 2.

Table 1: Drop test on hot surface

Type of explosive	Temperature of sand bed °C			
	100	150	200	250
Emulsion	No fumes or ignition up to 2.5 hours.	No ignition up to 2 hours.	Ignition after 1/2 hour.	Burned after 15 minutes.
Slurry, Organic nitrate sen.	No ignition up to 2 hours. There was vapour after 1 hour.	Nitrous fumes after 1/2 hour but slowly-burned after 1 hour.	Ignited after 15 minutes and burned out but, no explosion.	Ignited within 10 minutes and burned but no explosion.
ANFO	No ignition or fumes up to 3 hours.	No ignition up to 2 hours.	Nitrous fumes after 15 minutes but slowly burned	Ignition within 15 minutes and burned.

Table 2: Impact tests

Type of explosive	Weight of hammer (kg)	Height of fall (m)	Observation
Emulsion	10.0 5.0	3.0 3.0	No explosion No explosion
Slurry	10.0 5.0	3.0 3.0	No explosion -do-
ANFO	10.0 5.0	1.0 1.5	Detonation Detonation

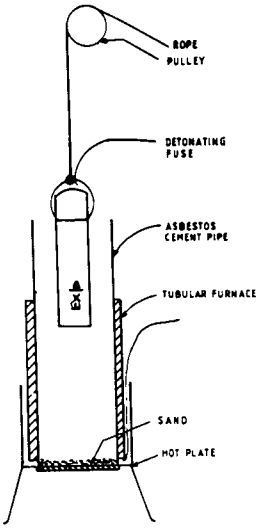


Figure 1. Arrangement for impact sensitivity test on hot surface

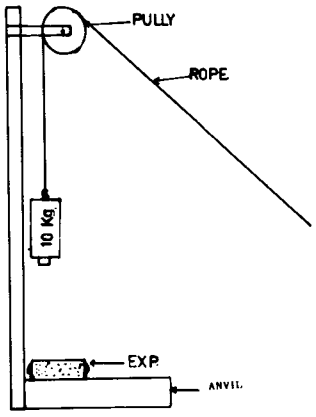


Figure 2. Arrangement for impact sensitivity test

DETONATION TEST ON HEATING

The method of testing and the experimental set-up have been described in the work of (Nabiullah and Gupta 1985). Explosive cartridges were heated in an electric tubular furnace at different temperatures for varying periods. The temperature of furnace and of the explosive cartridge at the periphery and at the centre was recorded using Cr-Al thermocouples. Tests were carried out under confined and unconfined states stimulating a hot blast hole in the rock. The furnace diameter was same as that of blast holes at the site. The cartridges were heated at constant temperature up to 4 hours. The decomposition, if any, of the explosive was recorded after cooling the cartridges. The detonating property of explosives at elevated temperatures was recorded by firing the cartridges using a detonating fuse of 10 gm/m. The velocity of detonation (VOD) was recorded by a digital electronic timer while firing the cartridges using ionization probe method. The experimental set-up is shown in Figure 3 and 4. Results are given in Table 3.

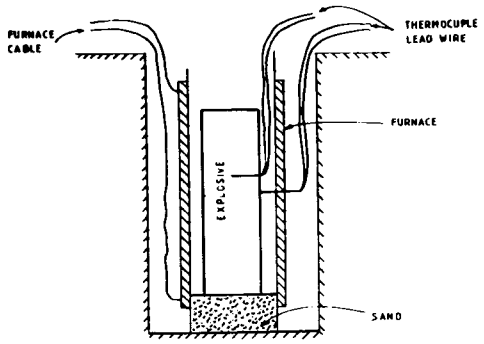


Figure 3. Arrangement for heat test in unconfined condition

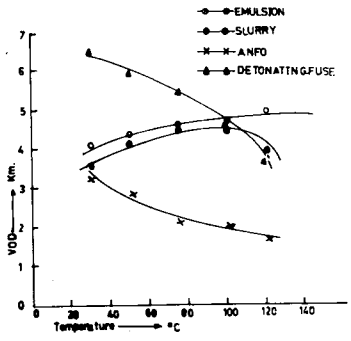


Figure 4. Influence of temperature on VOD

The ignition temperature of explosives was recorded by heating cartridges in an electric tubular furnace at a rate of 1-1.5° C/min. The explosive and furnace temperatures were recorded using Cr-Al thermocouples. The self sustained ignition temperature of explosives was recorded.

Table 3: Ignition temperature and VOD

Type of Explosive	Ignition Temperature (°C)		VOD (M/sec) Unconfined state (°C)				
	Unconfined	Confined	30	50	75	100	120
Emulsion	200±5	180±5 (Explode)	4100	4400	4620	4852	4925
Slurry	166±10	160±5	3600	4150	4530	4500	3700
ANFO	260±5	-	3341	2850	2200	2160	1700
Detonating fuse	135±5	135±5	6600	6060	5534	4753	3800*

* Misfire was observed in four out of five tests.

Table 4: Arrhenious reaction kinetic from DTA curve

Type of Explosive	B °C/min	T Kelvin	E K Cal/Mole	Z Min^{-1}	Correlation Coefficient
Emulsion	2 5 10 15 20	478 483 487 598 503	38.05	$1.9X10^{14}$	-0.9414
Slurry	2 5 10 15 20	420 430 435 450 454	23.12	$1.08X10^{13}$	-0.9661
ANFO	5 10 15 20 30	553 569 573 579 593	28.63	$4.99X10^{10}$	-0.9870

THERMAL BEHAVIOUR OF DETONATING FUSE

Detonating fuses of 2 m lengths were heated in an electric tubular furnace of 22 mm internal diameter and 1 m length. The two ends of the tubular furnace were sealed using clay as stemming material. The furnace was heated at different temperatures each time for one hour. The heated detonating fuse was fired with No. 6 detonator and the VOD was recorded with a digital electronic timer using 'off' and 'on' probe method. The arrangement is shown in Figure 5 and the results are given in Table 3. Misfires were observed at temperatures greater than 100° C.

The density of explosives at different temperatures was also recorded to determine the change in density with temperature.

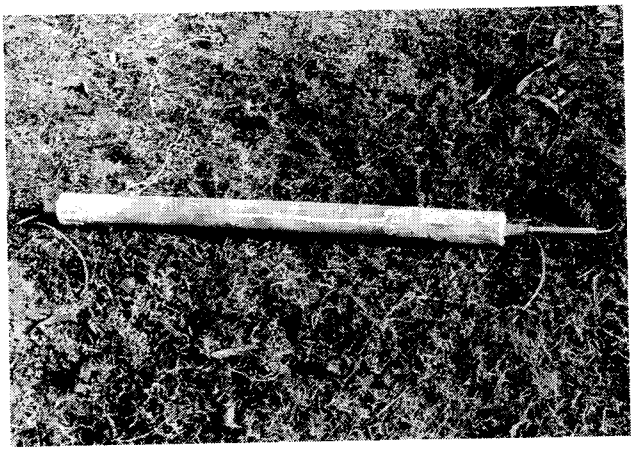


Figure 5. Arrangement for heat sensitivity test of detonating fuse

HEAT PRECARIOUSNESS AND KINETIC STUDY

A DTA apparatus was used to test various explosives including NG based solid explosives, slurries, emulsions and ANFO. Nitrogen was used as a purge gas in the DTA and the weight of each sample was 20-26 mg. The ignition temperatures for kinetic study were noted from the strip chart records of DTA for rate of heating between 2-20° C. The reference sample was α-Al_2O_3. The results are summarized in Table 5. Empirical relationships between 1/T (where, T is the ignition temperature in kelvin) and the rate of heating B (°c/min) were established using least square method of regression analysis. The results are plotted in Figure 6. The activation energy E and the pre-exponent factor Z were computed using (Ozwa 1970) equation. Results are given in Table 4.

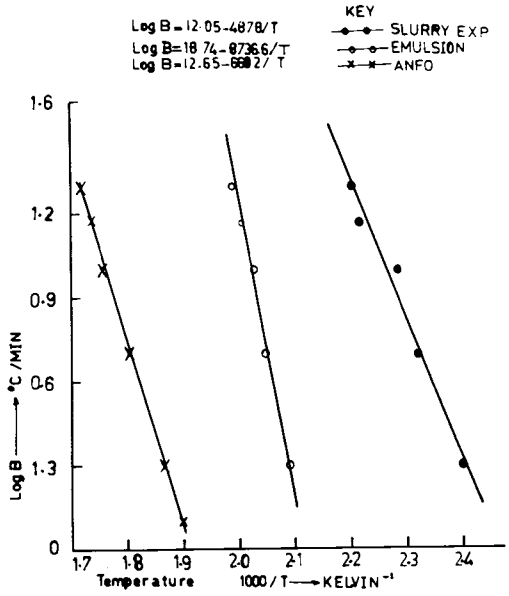


Figure 6. Arrhenious plot between log (B) and 1/T of explosive (B is rate of heating $°C^{-1}$ and T is ignition temperature in Kelvin)

BLASTING METHOD IN HOT HOLES

Use of NG based solid explosives in hot holes are not permitted by mines regulation since 1984. The slurry and emulsion explosives are permitted in holes where the temperature is below 80° C. However, where the hole temperature cannot be reduced below 80° or where there is a probability of increase in temperature because of stemming, special blasting techniques are recommended. No blasting is done in holes having a temperature of more than 120° C.

Table 5: Blasting parameters in hot overburden rock

Blast No.	1	2	3	4	5	6	7	8	9	10
Rock Type	Sandstone (Hard)	Hard S.St.	Hard S.St.	Med. hard S.St.	Hard sand S.St.	Med. hard S.St.	Coal	Hard sand Stone	Coal	Coal
Hole Temperature °C	90±10	70±10	70±10	70-100	90-108	115-120	55-70	40-72	60-72	53-80
No. of holes	7	6	8	9	5	3	9	6	6	9
Diameter(mm)	250	250	250	150	150	150	100	250	100	100
Depth(m)	10	8.3	15.1	11.0	12.0	12.5	3.75	16	3.75	3.75
Type of Explosive	Emulsion	Emulsion	Emulsion	Slurry	Slurry	Slurry	Slurry	Slurry	Slurry	Slurry
Total Explosive (kg)	1137.5	650.0	1800.0	468.7	317.5	250.0	75.0	1500	50	75.0
Charge factor Kg/M^3	0.4	0.4	0.4	0.4	0.5	0.4	0.2	0.5	0.2	0.2
Initiation	D.F.	D.F.	D.F.	D.F	D.F.	D.F.	D.F.	D.F.	D.F.	D.F.
Detonation	Good	Good	Good	Good	Good	Good	Good	Good	Good	Good
Fragmentation	Fine	Fine	Fine	Fine	Good	Good	Good	Good	Good	Good

Table 6. Static and dynamic elastic properties of rock

Name of mine	Rock sample					In-situ condition			
	Compressive strength	Tensile strength	Young's Modulus	Rigidity	Bulk Modulus	Young's	Rigidity	Bulk	Poission Ratio
	kg/cm^2	kg/cm^2	10^{11}dyne/cm^2	10^{11}dyne/cm^2	10^{11}dyne/cm^2	10^{11}dyne/cm^2	10^{11}dyne/cm^2	10^{11}dyne/cm^2	10^{11}dyne/cm^2
Sandstone Block II (BCCL)	421.0	49.0	1.696	1.388	0.364	0.027	0.013	0.012	0.088
Shaly hard sandstone (Sudamdih)	786.0	164.0	1.665	0.872	0.710	0.299	0.127	0.160	0.179
Fine grained Sandstone (Sudamdih)	494.0	103.0	2.073	1.108	0.878	0.021	0.095	0.089	0.098

The probability of misfires of fuses increased with increase in temperature particularly above 100° C.

The first step for blasting in hot overburden rocks is the measurement of temperatures in holes. Observations indicated that the temperature varied between 50-300° C excepting in a few holes where the temperature recorded was up to 600° C. The identified hot holes are flushed with water at least 12 hours prior to blasting. The temperature of flushed holes is recorded at regular intervals. Holes which do not retain the water are again flushed with bentonite-sodium silicate mixture until the water is retained in such holes. Gaur gum (up to 5 percent mixed in water was also flushed in cracked holes to seal the fractures and was effective. Only those holes where the temperature is below 80° C are charged with explosives. In a few holes pre-calibrated thermistors were embedded with the explosive cartridges to monitor the change in drill hole temperature with elapsed time. Stemming in holes is done after charging all the holes to be blasted. Wet or water saturated coarse and angular rock chips/drill cuttings are recommended for stemming. The number of holes in a round of blast are selected such that the charging operation is completed within 2 hours. Generally not more than 12 holes are blasted at a time.

In holes where the temperature is above 80° C but below 120° C, and where it is difficult to lower the temperature, extra precautions are taken. In such cases an asbestos-cement pipe is lowered into the drill holes. A mixture of bentonite-sodium mixed in water is flushed around the pipe to fill the gap between the pipe and the hole. Explosive cartridges are then lowered inside pipe and empty space is filled with bento silicate-water mixture up to the hole mouth. Holes are stemmed as described above. Not more than 4 holes are charged so that the charging operation is completed within an hour. Precalibrated thermistors are embedded in explosive cartridges to monitor the rise in temperature inside the charged holes.

Investigation indicated that in all such cases the temperature of the charged explosives varied between 40-70° C.

ELASTIC PROPERTIES OF ROCK MASS

The static and dynamic elastic properties of hot overburden rocks were determined to assess the blastability index. P and S waves were measured in the field using Signal Enhancement Seismograph. Seismic waves were generated by detonating 100 gms of explosives at 7-10 m distance from the geophones. The velocity between the arrays of geophones was computed from the strip chart records by dividing the distance between geophones with the elapsed time between geophones with the elapsed time between their respective break points. The dynamic elastic properties were computed using relationships suggested by (Broadbent 1974). The results are briefed in Table 6. Some static properties of rocks were computed in the laboratory using samples collected from the blasting sites. The seismogram record is shown in Figure 7.

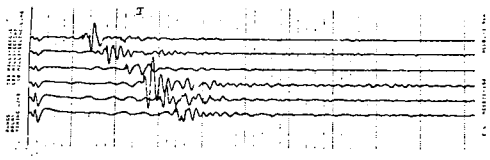


Figure 7. Seismogram record (in-situ condition)

DISCUSSION

The investigation into the thermal behaviour of the most commonly used emulsion, slurry and ANFO explosives indicated that they were safe up to 100° C for a period of 3 hours. No fumes were detected and no premature detonation or deflagration was observed when they were heated in

confinement at 100° C for 3 hours. Detonation was observed when emulsions and slurries were heated in confinement at 170±15° C. Slurries lost some water and their gel structures were broken when heated above 80° C. However, slurries retained detonating properties up to 120° C, and were sensitive to No. 6 blasting cap. The ignition temperatures of emulsions slurries, ANFO and detonating fuse was 200±5° C, 166±10° C, 260±5° C and 135±5° C, respectively. It may be seen from Table 3 that the ignition temperature in confinement was less than in unconfined state.

The influence of temperature on VOD is shown in Figure 4. There is an increase in the VOD emulsion explosive with temperature up to 130° C. In case of slurries, VOD increases up to 100° C, therefore, it drops down. There is a significant decrease in VOD of ANFO and detonating fuse with increase in temperature. Misfires were recorded when the detonating fuse was heated above 100° C. In one of the experiments, four out of five shots misfired. Underwater blasting tests indicated that a small reduction in VOD leads to significant reduction in the available energy of explosive and vice versa (McKenZie 1987). The implication is that emulsion explosives would perform better at elevated temperatures as compared to others. It may be seen from Table 4 that the activation energy of emulsion explosives is the highest 38.05 K cal/mole as compared to 23.12 and 28.63 K cal/mole for slurries and ANFO, respectively. However, the frequency factor of ANFO was least as compared to slurry and emulsion.

Emulsion and slurry explosives burned when dropped on a hot surface heated at 200° C. However, in confined condition they detonated. The impact sensitivity of emulsion and slurry explosives was poor. No detonation was recorded when hammer of 10 kg was dropped from a height of 3 m. However, ANFO was detonated when a hammer of 10 kg was dropped from a height of one metre.

It is concluded that the use of emulsion explosives in hot holes is more efficient and safer as compared to other commercial explosives. The explosive consumption in hot holes was half that of normal holes. The optimum charge factor for medium to hard sand stone was 0.4 kg/m^3.

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RECHERCHES SUR LE COMPORTEMENT THERMIQUE DES EXPLOSIFS EMULSIONS ET DES CORDEAUX DETONANTS

Md. Nabiullah, R.N. Gupta, B. Singh

Un certain nombre de mines de charbon de l'Inde sont en feu; la majorité d'entre elles sont situées près des affleurements. Des mesures de contrôle ont permis d'obtenir un succès partiel, mais à un coût exorbitant. On réalise aujourd'hui que l'enlèvement du charbon enflammé, en prenant les précautions appropriées, représente une solution alternative attrayante. Il faut pour cela effectuer des tirages dans des strates brûlants. Pour déterminer quels sont les explosifs commerciaux sûrs, adaptés au tirage, on étudie actuellement leur comportement thermique par des températures élevées (jusqu'à 200°C). On a calculé leurs paramètres physiques en laboratoire, avec simulations des terrains, ainsi que sur les sites des tirages. On a étudié les explosifs à base de nitroglycérine, les boues explosives, les ANFO, les émulsions, les détonateurs et les cordeaux détonants. Les explosifs et les détonateurs commerciaux à base de nitroglycérine se sont révélés dangereux par des températures supérieures à 80°C; leur utilisation est donc interdite dans les orifices brûlants. Les cordeaux détonants se sont révélés dangereux par des températures supérieures à 120°C, en raison de la faible résistance thermique de leurs revêtements de chlorure de polyvinyle. Les boues explosives avaient tendance à perdre leur structure colloïde au-delà de 80°C mais, comme elles conservaient leur propriétés explosives jusqu'à 120°C, on les a utilisées dans les orifices brûlants. Les explosifs émulsions se sont révélés les plus sûrs; ils supportaient en outre des températures allant jusqu'à 170°C. D'une manière générale, l'utilisation d'explosifs par des températures supérieures à 120°C est déconseillée. Les auteurs ont fait des essais de méthodes spéciales de tirages à des températures supérieures à 80°C.

EINE UNTERSUCHUNG DES THERMISCHEN VERHALTENS VON EMULSIONSSPRENGSTOFF UND ZÜNDER

Md. Nabiullah, R. N. Gupta, B. Singh

In Indien steht eine Reihe von Gruben in Brand, die meisten davon nahe den zutage tretenden Schichten. Bekämpfungsmaßnahmen waren teilweise erfolgreich und äußerst kostspielig. Die vorsichtige Entfernung von brennender Kohle wird als attraktive Alternative angesehen. Dieses Verfahren macht Sprengarbeiten in heißen Deckschichten notwendig. Um einen sicheren und brauchbaren kommerziellen Sprengstoff auszuwählen, wurde das thermische Verhalten bei erhöhten Temperaturen (bis zu 200°C) studiert. Die physikalischen Explosivparameter bei erhöhten Temperaturen wurden im Labor unter simulierten Bedingungen und am Sprengort selbst bestimmt. Auf N-G basierende Sprengstoffe, explosive Schlämme, ANFO, Emulsionen, Zünder und Zündschnüre wurden untersucht. Die auf N-G basierenden kommerziellen Sprengstoffe und Zünder waren bei über 80°C nicht mehr sicher und können für Sprengungen in heißen Löchern nicht zugelassen werden. Die Zündschnüre waren bei über 120°C unsicher, was auf die geringe thermische Widerstandsfähigkeit der PVC-Ummantelung zurückzuführen ist. Die Schlämme neigten dazu, ihre Gelstruktur bei über 80°C zu verlieren, sie behielten aber ihre Explosiveigenschaften bis zu 120°C und wurden in heißen Bohrlöchern verwandt. Die Emulsionssprengstoffe waren die sichersten und überstanden Temperaturen bis zu 170°C. Im allgemeinen wird die Verwendung von Sprengstoffen bei Temperaturen von über 120°C nicht empfohlen. Bei Temperaturen von über 80°C wurde mit speziellen Sprengmethoden experimentiert.

ИЗУЧЕНИЕ ПОВЕДЕНИЯ ЭМУЛЬСИОННЫХ ВЗРЫВЧАТЫХ ВЕЩЕСТВ И ВЗРЫВАТЕЛЕЙ В ЗАВИСИМОСТИТ ОТ ИЗМЕНЕНИЯ ТЕМПЕРАТУРЫ

Мд. Набиулла, Р.Н. Гупта и В. Сингх

На нескольких угольных шахтах Индии происходят пожары, главным образом в местах выхода угля на поверхность. Контрольные меры имеют определенный успех, однако стоимость их очень высока. Извлечение горящего угля, проводимое с определенными предосторожностями, признается в настоящее время в качестве привлекательной альтернативы. В горячих поверхностных слоях угля эта работа производится с помощью взрывов. Проведено изучение поведения взрывчатых веществ в условиях повышенных температур (до 200°С) с целью отбора наиболее безопасных взрывчатых материалов, пригодных для промышленного использования. Физические параметры взрывчатых веществ в условиях повышенных температур изучались на лабораторном полигоне и в реальных условиях. Исследовались взрывчатые суспензии на базе нитроглицерина, ANFO, эмульсии, взрыватели и запалы. Промышленные взрывчатые вещества на базе нитроглицерина и взрыватели небезопасны при температурах, превышающих 80°С, и поэтому их использование не разрешено в горячих шпурах. Запалы не были безопасны при температуре выше 120°С из-за низкой теплоустойчивости полиэтиленовой оболочки. Суспензии теряли свою желатиновую структуру при температуре выше 80°С, однако, они сохраняли свои взрывчатые способности до 120°С и были использованы в горячих шпурах. Эмульсионные взрывчатки являются наиболее безопасными, выдерживая температуру до 170°С. В целом применение взрывчатых веществ при температурах выше 120°С не рекомендуется. Авторы провели экспериментальные взрывы особым методом при температуре выше 80°С.

乳胶炸药和引爆引信热力状态的研究

Md。奈布西拉， R。N。葛布塔， B。辛格

在印度很多煤矿着火，大多数发生在露头处．控制的措施有部分的效益，但花费极大．另外较好的办法是小心地除去着火的煤．这办法是在炙热过载的岩层处爆炸．要选择安全和适当的商用炸药，必先研究它在高温（到２００摄氏度）的热力性能．高温的炸药物理参数是在模拟现场情形的实验室及在爆炸现场所得出．ＮＧ基，塑胶炸药，ＡＮＦＯ，乳胶，引爆剂，导爆索都要经过详细研究．ＮＧ基的商用炸药及引爆剂在８０摄氏度以上便不安全，故不能在热矿井内使用．导爆索因为ＰＶＣ外壳的热阻较低在　１２０摄氏度以上便不安全．塑胶炸药在８０摄氏度以上会失去胶质组织，但能在１２０摄氏度保持它的爆炸性能，故可用在热矿井里．乳胶炸药最为安全，可以维持到１７０摄氏度．通常在１２０摄氏度以上的温度不宜使用炸药．作者曾在８０摄氏度试验过一些特殊爆炸方法．

ПОВЫШЕНИЕ БЕЗОПАСНОСТИ ВЗРЫВНЫХ РАБОТ В УГОЛЬНЫХ ШАХТАХ, ОПАСНЫХ ПО ГАЗУ И ПЫЛИ

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РЕЗЮМЕ

В связи с совершенствуванием технологии добычи угля создано новое взрывчатое вещество "Выгленит-2", предназначенное для открытого взрывания в шахтах, опасных по газу и пыли . Из антистатичного полимерного фолийного материала разрабатаны изделия для осуществления забойки взрывных шпуров, для создание инертных водянных заслонов , для водоизоляции зарядов взрывчатого вещества. При практическом применении установлено существенное повышение надежности, эффективности и безопастности взрывных работ, а также улучшение санитарно-гигиенических условий и технико-экономических показателей горных работ.

ВВЕДЕНИЕ

Разработываемые угольные месторождения в НРБолгарии характеризируются весьма сложными горно-геологическими условиями. Поэтому большой удельный вес подготовительных выработок и очистных работ выполняются буровзрывными работами, ввиду невозможности или неэффективности применения механизации. В связи с этим, совершенствувание технологии и средств взрывания является очень важной проблеммой на нашых угольных шахатах, как с точки зрения повышения безопастности, так и в отношении повышения эффективности и производительности сответствующих технологических операции . Наряду с традиционными проблеммами безопастности буро-взрывных работ при сложных горно-геологических условиях в угольных шахтах у нас отмечаются и некоторые специфические проблеммы. Например, в немеханизированных лавах с индивидуальной крепью, часто наблюдается зависание кровли в отдельных участках и возникает необходимость ее принудительного обрушения. При этом шахтерам приходилось работать в непосредственной близости к местам с частично нарушенной кровлей и крепи в выработном пространстве. Попытки избежать присутствия людей в этих случаях путем использования разных механизмов и сооружении привели только к частичным улучшениям. По данным практики при этом технологическом процессе регистрируются немало несчастных случаев, иногда тяжелых.Статистический анализ последних пяти лет показал, что около 20 % общего числа аварий в угольных шахтах, на которе выежали горноспасители, связаны с обрушениями кровли, нависающими породамми и г.д..Таким образом возникла необходимость разработки более безопастных технлогий в этом отношении, с применением открытых и коротких зарядов взрывчатого вещества, безопастного в условиях шахт, опасных по газу и пыли. Такое взрывчатое вещество необходимо также в ряде других случаев практики; например разрушение крупных кусков породы (негабаритов), откупорка углеспусков и бункеров, при подрывке породы по периметру выработок в целях увеличения сечения при перекреплении и т.д..В практической деятелности на угольных шахтах,

опасных по газу и пыли, были установлены некоторые недостатки применяемых водонаполняемых ампул, мешков и др. средств из полимерных материалов. Возникли предпосылки и были проведены исследовательско-экспериментальные работы для их совершенствования. В настоящей статье коротко отметим некоторые результаты проведенных работ по совершенствованию технологии взрывных работ на шахтах, опасных по газу и пыли в этих двух планах.

ВЗРЫВЧАТОЕ ВЕЩЕСТВО ДЛЯ ОТКРЫТЫХ ЗАРЯДОВ В ШАХТАХ ОПАСНЫХ ПО ГАЗУ И ПЫЛИ

Для решения ряда практических проблемм, как было упомянуто, возникла необходимость применения открытых зарядов или зарядов в весьма коротких шпурах, с использованием взрывчатого вещества, удовлетворяющего требованиям безопастной работы в условях угольных шахт, опасных по газу и пыли. В других странах такие взрывчатые вещества производится и успешно применяют, например "Угленит-5" в СССР, GS-16 в Франции и т.д. У нас было принято решение разработать и производить отечественное взрывчатое вещество на основе местного сырья и по собственной технологии. Это осуществлено коллективом сотрудников Испытательного центра для взрывчатых материалов при "Минпроект"-ДНИ. Уже началось серийное производство этого нового взрывчатого вещества, названного "Выгленит-2". Эго прямое предназначение, как уже говорилось, для проведения открытых взрывных работ в условиях угольных шахт, опастных по газу и пыли. Оно имеет следующие храктеристики: работоспособность по Трауцэлю-60 cm^3 ; бризантность по Гесу-5 mm ; критический диаметр-9mm; передача детонации -4 cm; скорость детонации-1600 m/s; чувствительность к удару на копре-50 %; безопастность по газу и угольной пыли при граничном заряде-600 g . Взрывчатое вещество "Выгленит-2" разработано на основе аммиачной селитры, нитроэфиров, поваренной соли и других веществ. Оно патронируется в бумажных гильзах диаметром 29 и 36mm и весом 75, 100 и 150 g . Взрывчатое вещество "Выгленит-2" принято для работы в ряде угольных басейнах: Перницком, Бобовдолском, Балканском и других. Разработаны технологии-паспорта для безопастной и безаварийной работы при разных технологических процессах: разрушении деревянной крепи в выработанном пространстве, дроблении негабаритных кусков, обрушении нависших кусков породы, подрывание пород для увеличения сечения выработок и т. д.. Отметим коротко некоторые характерные моменты при разных случаях взрывания на основе требований специально разработанной Инструкции для безопасной работы с взрывчатым веществом "Выгленит-2". При разрушении деревянной крепи заряды (патроны ВВ) закрепляются на ней или вставляются в нее без забойки, но при этом количество одного заряда не должно превышать 150 g . Крупные (негабаритные) куски пород разрушаются открытыми зарядами количеством до 600 g выполнением забойки сверху (покрытие патронов ВВ) толщиной не менее 20 mm . При подрывке пород с применением шпуров, масса зарядов не должна превышать 700 при длине забойки не менее 50 g. При взрывании открытых зарядов в камерах, в бункерах для отстранения закупорок и т.д., количество взрывчатого вещество одного заряда не должно превышать 600g. Взрывчатое вещество "Выгленит-2" нашло весьма хороший прием на шахтах, т.к.работы с ним проводятся быстро и удобно, при существенном снижении трудоемкости и повышении безопасности работ. При практическом применении установлено существенное снижение травматизма при принудительном

обрушении кровли и других упомянутых нами операциях и соответственно значительное повышение безопастности труда. После введения новой технологии принудительного обрушения кровли с использованием взрывчатого вещества "Выгленит-2" не отмечены несчастные случаи. Кроме того, продолжительность выполнения соответствующих процессов снизилась около 4 раз. С используванием нового взрывчатого вещества обрушение кровли лавы средней длинны осуществляется за около 2 часов, вместо в течении одной смены при применяемом ранее способе. При этом, шахтерам не приходится работать при небезопастных условиях в выработанном пространстве. Увеличивается на 25-30 % производительность работ при необходимости подрывки пород для уширения и перекрепления подготовительно-нарезных выработок, в том числе и на сопряжениях лав. Значительно облегчается выпуск угля, в связи с возможностью быстрого и безопастного устранения закупорок. Таким образом, разработка нового взрывчатого вещества и соответствующих технологий его применения создало возможность решить ряд проблемм, производственной практики, и соответственно повысить существено безопастность и эффективность некоторых технологических процессов на шахтах опасных по газу и пыли.

СОВЕРШЕНСТВОВАНИЕ НЕКОТОРЫХ СРЕДСТВ ДЛЯ ПОВЫШЕНИЯ БЕЗОПАСНОСТИ ПРИ ВЗРЫВНЫХ РАБОТАХ

В связи с изменениями предприятий-производителей, создались предпосылки для совершенствования полимерных материалов, технологии и конструкции некоторых изготовляемых из них средств для повышеная безопасности при взрывных работах, как : водонаполняемые ампулы и мешки, рукава из полимерной пленки для изоляции взрывчатого вещества в обводненных забоях и т.д.. Работы в этом направлении выполнены специалистами Испытательного центра для взрывчатых материалов при "Минпроект"-ДНИ совместно со специалистами соответствующих заводов-производителей полимерных материалов и упомянутых средств. Коротко отметим результаты этих работ по разным видам средств-изделий. Для водонаполняемых ампул было необходимо совершенствование материала, конструкции и технологии их изготовления. Разработаны состав и технология, а также организовано производство полиэтиленовой пленки (фолио) с низкой плотностью, предназначенной для применения на шахтах опасных по газу и пыли. Оно соответствует определенным требованиям относительно сопротивления на разрыв (10 MN/m^2) устойчивости при высоких и низких температурах,

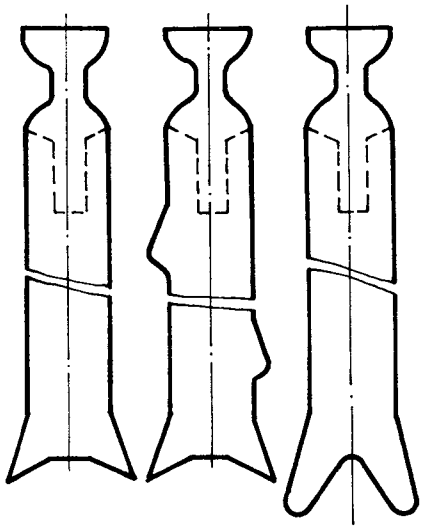


Рис.1 Разновидности водонаполняемых ампул

антистатичных свойств (удельное поверхностное электрическое сопротивление не более 1.10 Ω) и т.д.. Разработаны и производятся водонаполняемые ампулы диаметром от 28 до 50 mm и длиной от 150 до 350 mm с применением высокочастотной сварки. Имеется несколько разновидностей в отношении конструкции . Наиболее интерес-

ными являются конструкции, обеспечивающие самозаклинивание ампул при заряжении снизу вверх вертикальных или наклоненых шпуров. (Рис. 1) Эти ампулы имеют ряд преимуществ и весьма удобны при проходке восстающих выработок, при системе с подэтажным обрушением и т.д.
При испытаниях и практическом применении установлено удобное заполнение ампул водой и эффективная закупорка после этого. В отношении материала и конструкции водонаполняемых ампул защищено 2 авторских свидетельства. На этапе опытной и пробной серии водонаполняемых ампул проводились исследования в отношении выделения токсических газов при взрывании зарядов взрывчатого вещества с использованием ампул в забойке. Не было установлено увеличение выделениия токсических газов выше допустимых норм. Промышленные испытания и практическое применение водонаполняемых ампул проводились в сответствие с разработанными рекомендациями-инструкциями.
Как обыкновенно принято, водонаполненые ампулы размещают в шпурах за зарядом. А после размещени ампул необходимо выполнить забойку из глины, длиной 120-150 *mm*. При небольших зарядах взрывчатого вещества до 800 *g* используют одну водонаполненую ампулу длиной не менее 270 *mm*. В длинных шпурах при количестве взрывчатого вещества более 2 *kg* следует применять не менее 3 водонаполненных ампул. В зависимости от конкретных условий рекомендуется располагать водонаполненную ампулу на дне (в конце) шпура. Примерная конструкция зарядов показана на (Рис.2). Заполнение ампул водой проводится из рудничного водопровода на забоях или на специально устроенных для этой цели пунктах. В последнем случае ампулы в забой доставляют в ящиках. При промышленных испытаниях и практическом применении усовершенствованных водонаполненных ампул в основных угольных бассейнах установлено следующее: - значительно повышается коэффициент использования шпуров до 0.9-095 и более ; -подвигание забоя за цикл увеличивается на

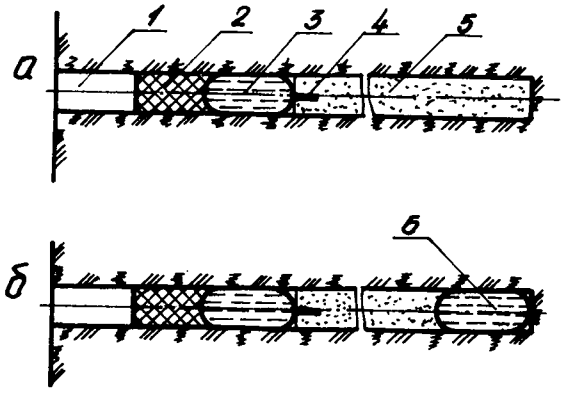


Рис.2. Заряженные взрывные шпуры 1/Взрывный шпур; 2/Глинянная забойка 3/ Водонаполняемая ампула 4/ Электро детонатор ЕДМАН 5/Взрывчатое вещество 6/Водонаполняемая ампула

8-15 % ; - продолжительность заряжания шпуров снижается на 24-40 %;-пылевыделение в забое выработки 5 *min*. после взрывания снижается на 50 % по сравнению с случаями без применения водонаполненных ампул, т.е. они обезпечивают эффективность обеспыливания при взрывных работах порядка 50% ; содержание азотных окислов в рудничной атмосфере после взрывания снижается на 10-12 %; время снижения вредных примесей в рудничном воздухе забоя после взрывания до допустимых норм уменьшается 2-2,5 раза по сравнению с глинисто-песчаной забойкой без ампул и достигает 7-10 *min*. .Таким образом применение усовершенствованных водонаполняемых ампул способствует не только повышению безопасности взрывных работ на опасных по газу и пыли шахтах , а также и улучшению санитарно-гигиенических условий труда

и эффективности работы. Рукава (шланги) из полиэтиленовой пленки низкой плотности, склееные или завязаные на одном конце, применяются для водоизоляции взрывчатого вещества при заряжании шпуров в обводненных забоях. Разработана технология и налажено серийное производство таких рукавов (шлангов) диаметром 36-40 *мм* с толщиной пленки 0,8 *мм* из материала соответствующего требованиям антистатичности и т.д. для шахт, опасных по газу и пыли. Имея ввиду, что широко применяемые у нас предохранительные взрывчатые вещества не являются водоустойчивыми,таким образом была успешно решена одна немаловажная проблемма безопастности и эффективности взрывных работ в обводненых забоях. Изготовляемые рукава (шланги) нашли широкое применение и отличный прием на шахтах. Практический опыт показал, что их использование способствует предупреждению отка+зов и опасности связанные с их ликвидацией. При этом эффективность работ в обводненных забоях повысилась более чем на 35 %. Для водонаполненных мешков разработана конструкция, технология и налажено серийное производство из антистатической полиэтиленовой пленки (фолио), соответствующей требованиям использования в шахтах, опасных по газу и пыли. Мешки изготавливают из пленки толщиной 0,1-0,12 *мм* , с вместимостью 20 *дм³* воды. Они устроены так, что их можно повесить к потолку выработок или положить на пол. Вода разбрызгивается взрыванием заряда в количестве 80-100 *г* водоустойчивого или неводоустойчивого взрывчатого вещества, расположенного в середине мешка в специально устроенном "зарядном рукаве" (Рис. 3). Водонаполняемые мешки используются, как известно, для создания водяных заслонов при взрывных работах в подготовительных и других выработках, с целью повышения безопастности на шахтах, опасных по газу и пыли. Согласно рекомендациям по применению водонаполняемых мешков, они используются самостоятельно или вместе с другими водяными заслонами.

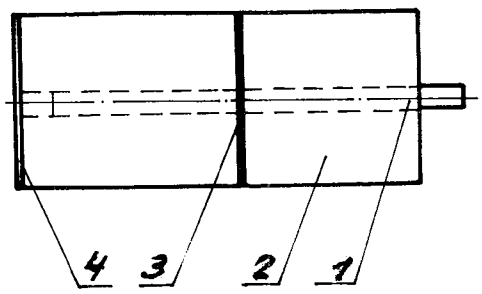


Рис.3.Конструкция водонаполняемых мешков 1/Зарядный рукав(шланг) 40 . 2/Водонаполняемый мешок 3/Маркировка уровня воды 4/Двойной шов

После заряжения забоя, мешки подвешиваются к потолку или располагаются на полу выработки, причем их инициирующий (разбрызгивающий) заряд включается в забойную взрывную цепь, с расчетом воспламенения его капсулы-детонатора на 25 *мс* раньше, чем заряды забоя. Мешки располагают таким образом, чтобы расстояние до груди забоя и стенок выработки не превышало 1,8 *м* . Количество водонаполняемых мешков определяется исходя из этого условия и из расчета чтобы расход воды на 1 *м²* сечения выработки был не менее 3 *дм³* . Защитное действие воднянного заслона из водонаполняемых мешков в отношении воспламенения метановоздушной взрывоопасной смеси (8,6-9,5% метана по объему) испытывалось в экспериментальной штольне Института "Минпроект"-ДНИ. Установлена высокая надеждность в этом отношении. Кроме повышения безопасности взрывных работ в отношении вероятного воспламенения возможных скоплении метана, применение водонаполняемых мешков способствует значительному

улучшению санитарно-гигиенических условий и уменьшению времени проветривания забоев выработок после взрывания.

ЗАКЛЮЧЕНИЕ

Результаты проведенных работ способствуют существеному повышению безопастности и эффективности взрывных работ на шахтах, опасных по газу и пыли. Созданно, производится и применяется на шахтах отечественное взрывчатое вещество для открытого взрывания на опасных по газу и пыли шахтах с неплохими показателями относительно надеждности и эффективности. Примемение разработанных технологий с его использованием обеспечили существенное снижение травматизма и повышение безопастности, а также производительности работ при принудительном обрушении кровли в немеханизированных лавах, при ремонте и расширении сечения выработок, при разрушении крупных (негабаритных) или нависших кусков породы и т.д. Существенно усовершенствованы средства повышения безопасности при взрывных работах из полимерных материалов: водонаполняемые патроны и мешки, рукава (шланги) для водоизоляции взрывчатого вещества в обводненных забоях и т.д. Применение этих средств на практике способствует значителному повышению безопастности и эффективности взрывных работ, улучшению санитарно-гигиенических условий в выработках после взрывания и т.д..

INCREASING THE SAFETY OF BLASTING IN COAL MINES IN THE PRESENCE OF COAL DUST AND GAS

A. Panajotov, D. Dekov, V. Simeonov, and K. Cholakov

In work to improve coal production, a new explosive called "Uglenit - 2" has been developed for open blasting in gassy and dusty coal mines.

Products have been developed from antistatic polymer foil for blasting holes, for use with passive water barriers and for water-proofing of explosive charges.

The results have been a large increase in the reliability of blasting operations, their effectiveness and safety, as well as the improvement in health and the technical and economic characteristics of mining.

AMELIORATION DE LA SECURITE DES OPERATIONS DE TIRAGE DANS LES MINES DE CHARBON SUJETTES AUX RISQUES DUS AUX POUSSIERES DE CHARBON ET AUX GAZ

A. Panajotov, D. Dekov, V. Simeonov, K. Cholakov

Un nouvel explosif appelé "Uglenit-2" a été fabriqué dans le cadre de l'amélioration de la technologie de la production charbonnière. Cet explosif est destiné aux tirages ouverts dans les mines sujettes aux risques causés par les poussières de charbon et par les gaz.

Certains produits, fabriqués avec des matériaux de feuilles polymère antistatiques, sont destinés aux abattages de mines par trous, aux protections d'eau inactive et à l'isolation des charges explosives par l'eau.

Ils ont permis d'améliorer la fiabilité, l'efficacité et la sécurité des opération de tirages, ainsi que les conditions de salubrité et les facteurs techniques et économiques des exploitations minières.

ERHÖHUNG DER SICHERHEIT BEI SPRENGARBEITEN IN GAS- UND STAUBHALTIGEN GRUBEN.

A. Panajotov, D. Dekov, V. Simeonov, K. Cholakov

Im Zusammenhang mit Fortschritten in der Kohlegewinnungs-Technik wurde ein neuer Sprengstoff mit dem Namen 'Uglenit - 2', der zum offenen Sprengen in gas- und staubhaltigen Wettern gedacht ist, entwickelt.

Einige Produkte sind aus antistatischer Polymerfolie hergestellt, z. B. : Wasserbesatzschläuche und wäßrige Isolationsmittel für Explosivstoffe.

Praktische Anwendungen bestätigen, daß diese Produkte die Sicherheit der Sprengarbeiten, den Wirksamskeitsgrad und die Sicherheit, sowie die gesundheitlichen, technischen und ökonomischen Eigenschaften der Gruben verbessert haben.

有煤尘及气体侵害的煤矿爆破作业安全性提高问题

A．班奈白托夫，D．戴可夫，V．锡米诺夫，及 K．却拉可夫

为了改进产煤量，一种新型炸药 "乌格兰镍特 - 2" 研制成功了，这种炸药主要适用于有煤尘及气体侵害的煤矿敞式爆破．

我们研制了一些由抗静电高分子箔制成的用于炸洞，惰性水档水用具，和炸药防水绝缘上的产品．

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FIREDAMP AND COAL DUST SAFETY OF PERMITTED EXPLOSIVES - OXYGEN INDEX AS TESTING CRITERION

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Experiments in the 50ies in the US Bureau of Mines Laboratory have established that the ignitability of methane-air mixtures by small explosive charges can be increased gradually by adding oxygen - i. e. without changing the chemical reactive components. Recent investigations at BVS have shown that by this method the safety of permitted explosives in firedamp as well as in coal dust clouds can be determined, under official testing conditions and in other blasting arrangements, by the oxygen concentration necessary for a 50 % ignition probability. Test results obtained with German permitted explosives of different safety classes are reported. In these experiments a closed test gallery is used which prevents the discharge of shock waves. The control technique of the test parameters, necessary to obtain reproducible results, is described. Additional investigations in a 2-m^3-chamber under increased total pressure (simulating air pressure increase in greater depths) with regard to the influence of ingitability are also reported.

GENERAL REMARKS ON TESTS IN FIREDAMP WITH PERMITTED EXPLOSIVES

For nearly 100 years permitted explosives have generally been classified according to the results of defined blasting tests in chambers, filled with firedamp. The test arrangements were designed to simulate extremely dangerous conditions, encountered in practical operations. Here, it is known from experience that the bigger the charge that does not ignite firedamp in the tests the higher the safety of the explosives in firedamp. With the increasing safety of explosives in firedamp it became necessary to increase the charge for testing or, if this was not possible any more, to apply more severe test arrangements. So, in the course of the years different test arrangements and testing philosophies were developed in the individuel countries. Some do the tests with charge weights as small as possible for the determination of the charge with a 50 % ignition probability, other countries carry out the tests with charge weights used in practical application to determine the maximum charge producing no ignition under such conditions. Several safety classes of explosives have been developed in the individual countries that are applied in different operational conditions, i. e. different danger situations.

As a rule, the tests are carried out in a near stoichiometric methane-air-mixture. The use of other flammable gas-mixtures is prescribed only in exemptional cases.

DEVELOPMENT OF OXYGEN ADDITION TEST METHODS

Already in the 30ies Beyling and Schulze-Rhonhof used balloons, filled with flammable methane-oxygen-mixture, in front of the boreholes, for the determination of deflagrating explosive particles. In the 50ies Lewis and von Elbe developed a method for the observation of firedamp ignition with very small charges in a miniature borehole-mortar and gallery. They varied the ignitability of a

methane-air-mixture by different quantities of added oxygen. They found that for the same explosive the transition from a non-ignitible to an ignitible gas mixture is in the region of 2 - 3 % b.v. oxygen and they defined the 50 %-oxygen value as a mean between the ones for 5 ignitions and 5 non-ignitions of the respective oxygen concentrations. They also established that the 50 %-oxygen value is typical for a specific explosive mixture and that explosives could be classified with regard to their safety in firedamp according to this value. But certain explosives, known as highly incendive showed extremely high oxygen values with this method which obviously means great safety. This misinterpretation led to the conclusion not to take the oxygen method as basis for an official approval.

Because of noise reduction a closed test gallery for explosives, similar to the one in Bruceton, was built at BVS in the 70ies. Here it showed that far more violent explosions without being a nuisance to the environment could be initiated than in the open test gallery. On the occasion of a special test of permitted propellant cartridges, first tests with oxygen addition were also carried out. These tests established that the findings of the US Bureau of Mines proved valid even for other ignition sources.

So test series on the ignition probability of permitted detonators and single cartridges of permitted explosive were carried out, the results of which were reported on the International Conference of Safety in Mines Research Institutes in Sheffield (Franke and Zimmermann, 1983). These tests clearly showed by the different "oxygen indices" the difference in safety of equal charges of different classes W I to W III of German permitted explosives (see Figure 1) but also the safety increasing effect of one explosive cartridge compared to the permitted detonator itself as the less safe component of the charge. Figure 2 shows that the quantity of a charge is as well influencial as the test arrangement. It was noted that the W III explosive used in the shown angle-shot mortar-set-up, which must never - according to the test regulations - ignite firedamp in any shot, has obviously the required safety, but without a clear margin.

TEST ON FIREDAMP SAFETY UNDER INCREASED PRESSURE

At the time mentioned the winning depth in Ruhr area coal mines was planned to be increased up to 1600 m. Because of the bariometric pressure increase in great depth it was assumed that the partial pressure of the oxygen in air would reach values there, which had caused ignitions in the test gallery with a W III explosive, even though at lower total pressure. One of the findings of a therefore started fundamental research project on minimum ignition energy of electrical sparks in relation to oxygen partial pressure and to total pressure was a significant decrease in minimum ignition energy with oxygen addition to methane-air-mixtures at normal pressure.
This decrease was confirmed if not fully, but clearly with increased oxygen partial pressure by increasing total pressure. Therefore the aim was to investigate whether the above phenomena also applied for permitted explosives as potential ignition source. In contrast to electrical sparks, permitted explosives are regarded, because of the energy released, as rather strong ignition sources.

The BVS-test galleries are designed to resist high pressure shocks; but they are not, in the precise sense of the word, gas pressure tight. Therefore, static overpressures of some tenth bar, required for such investigations, cannot be produced. So the tests were carried out in a high-grade steel clad gas-tight 2-m^3-blasting chamber, which is equipped with an electric heater, gas feeders, and measuring instruments for gas concentration and pressure (see Figure 3). Tests under these very well reproducible conditions were carried out with permitted W III-explosives. The total pressures were varied between 1020 and 1900 mbar; partial pressures for methane between 75 and 280 mbar and for oxygen between 210 and 840 mbar. This corresponds to concentrations by volume for CH_4 between 7.4 and 18.6 % b.v. and for O_2 between 20 and 44 % b.v. The charges were 300 g W III- explosive in usual

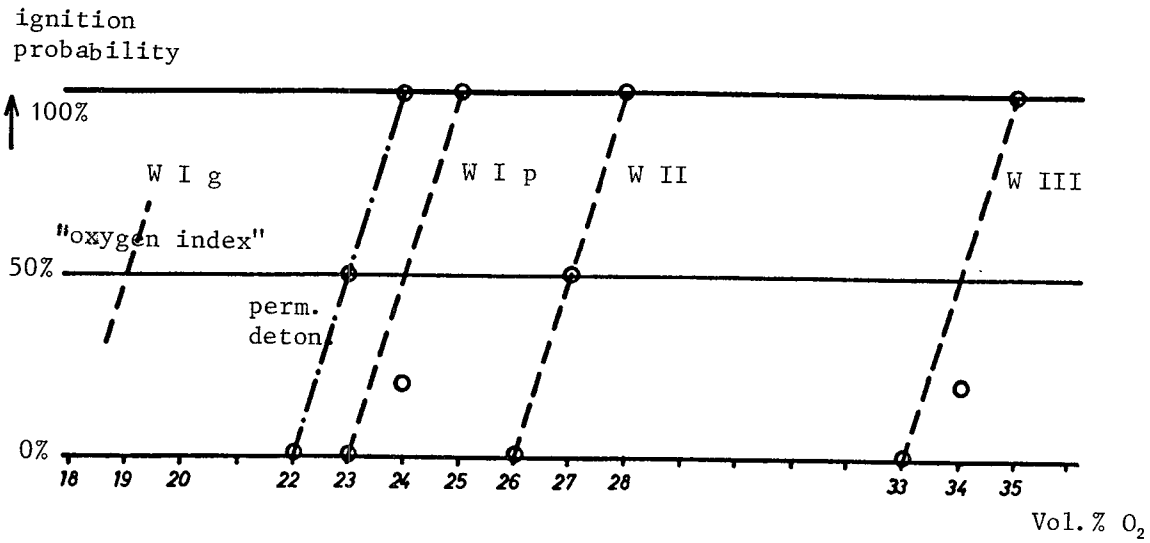


Figure 1: Oxygen index of 1 cartr. perm. explosive (freely suspended in the gallery)

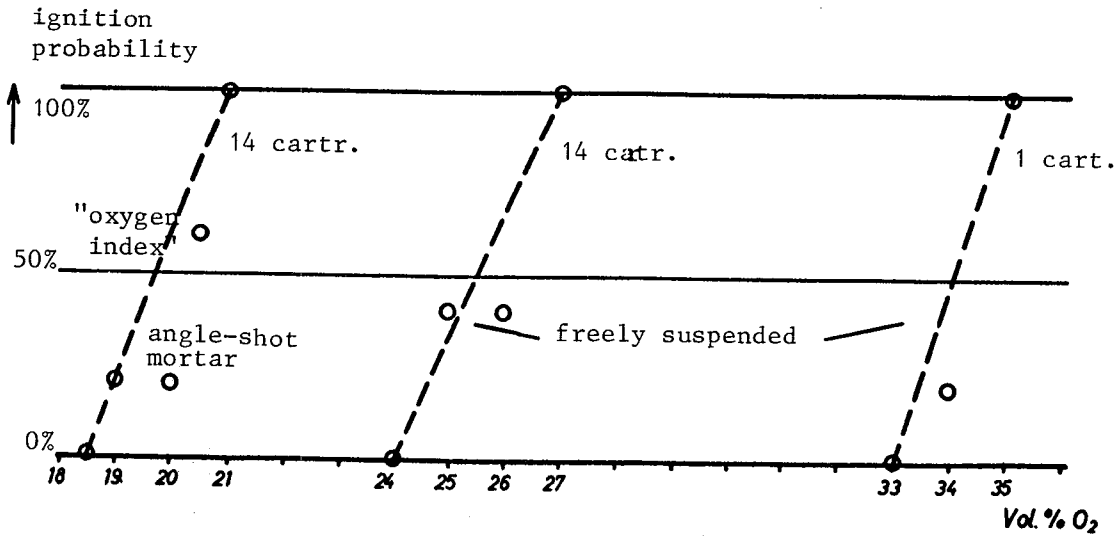


Figure 2: Oxygen index of different charges W III-explosive

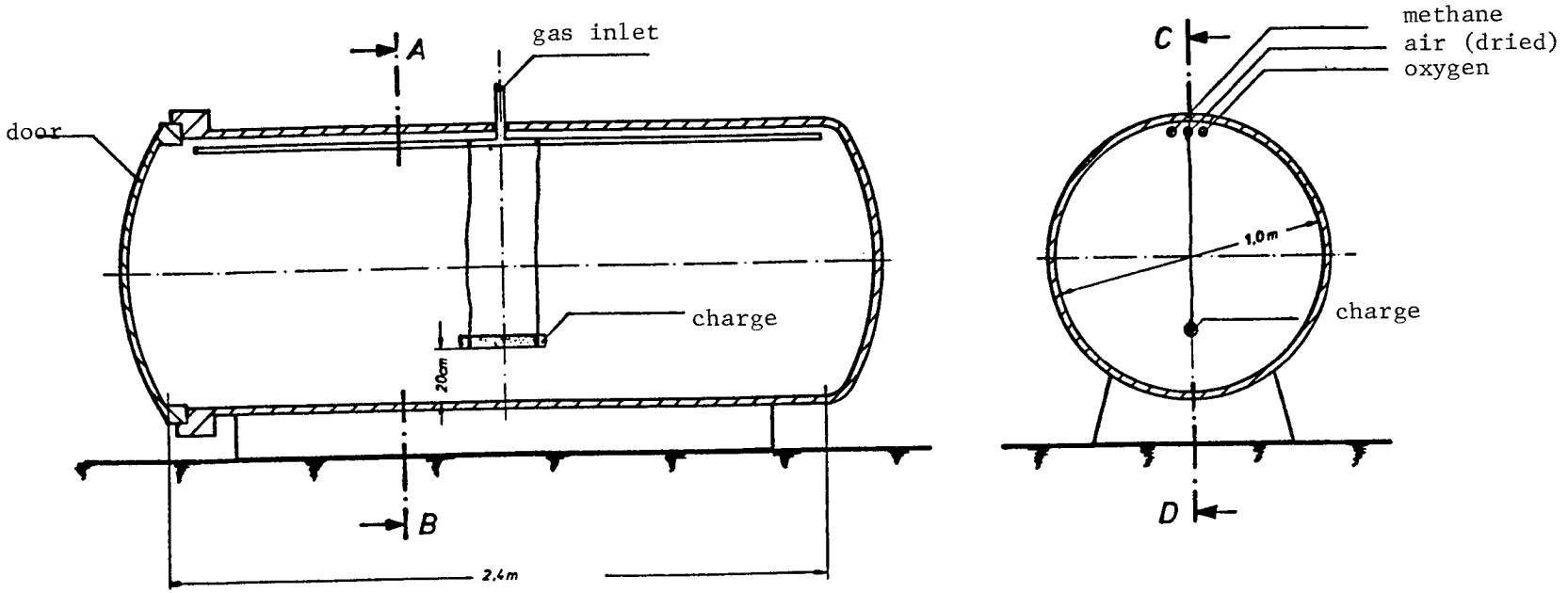


Figure 3: Pressure-tight-2-m^3-blasting chamber

cartridge diameter, detonated either freely suspended with 20 cm wall distance or in a small borehole mortar open at both ends.

These tests showed that for obtaining 50 % explosion probability, the content by volume of the gases in the mixtures was more important rather than the partial pressures. In the case of suspended charges all oxygen characteristic values for 50 % explosion probability are plotted as a function of the methane content, independent of the total pressure, shown in Figure 4. If the explosive was fired in a borehole mortar, a slight tendency to increased oxygen index was noted as the total pressure increased (Figure 5).

The test results established another interesting result: if the oxygen addition, necessary to get a 50 % explosion probability, is regarded as a measure for the ignition sensitivity of gas mixture, obviously the sensitivity for under-stoichiometric methane-air-mixtures decreases very strongly, whereas the sensitivity of over-stoichiometric mixtures in fact decreases too, but not so much. It is also noted that the minimum of the curve corresponding to maximal ignition probability is, for the explosive used, above the stoichiometric mixture.

In practical operations with short delay detonators and careful gas measurement before blasting, flammable gas mixtures will be produced as a lean mixture and only in very few cases will get stoichiometric or over-stoichiometric in the very short time of 500 ms at most. On the other hand the test regulations do not prescribe the very reproducible test conditions in the over-stoichiometric region.

THE CLOSED TEST GALLERY AND ITS MEASURING AND CONTROL TECHNIQUE

If a test gallery is not tight the methane content can decrease rather quickly in the time between gas fed and initiation of the test charge, which clearly influences sensitivity. Therefore it is necessary to control exactly these parameters and keep them constant. For this purpose a computerized measuring and control system has been developed for the closed test gallery. Figure 6 shows a schematic diagramm of this gallery and its operation system.

Instead of the commonly applied system process control, a modular control system was used, which is as simple to program but which has some powerful characteristics, important for the specific control applications, which normally are typical only for larger process computers.

The following input parameters are registered

- temperature in the test gallery (several test points)
- methane concentration
- oxygen concentration
- bariometric pressure
- valve and ventilator operation position

The calculated values for the

- gas concentration

are preset via a keyboard.

The respective operations and conditions of the test gallery are displayed on a colour monitor.

The installation can be operated

a) manually: the operation procedure can be controlled via a keyboard similar to the operating agents of a switchboard; to make operation easier several partial operations are linked by a porogram (the command "gas circulation on" comprises for instance the synchronized operation of several valves and electrical switches for ventilator).

b) semi-automatically: on the basis of the preset calculated values, the test gases are roughly dosed. Fine dosing is carried out manually via keyboard. When the exact values are reached, the exploder is unlocked so that the test charge can be detonated. the fumes exhaust, the opening of the

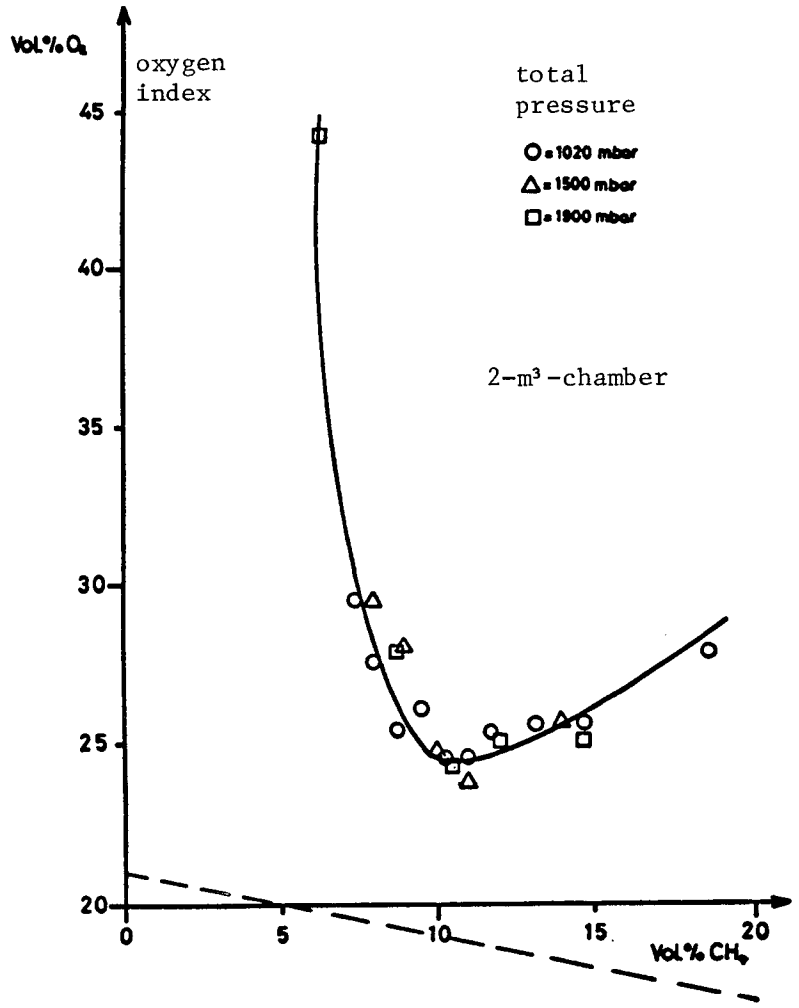


Figure 4: 300g W III freely suspended (wall distance 20 cm)

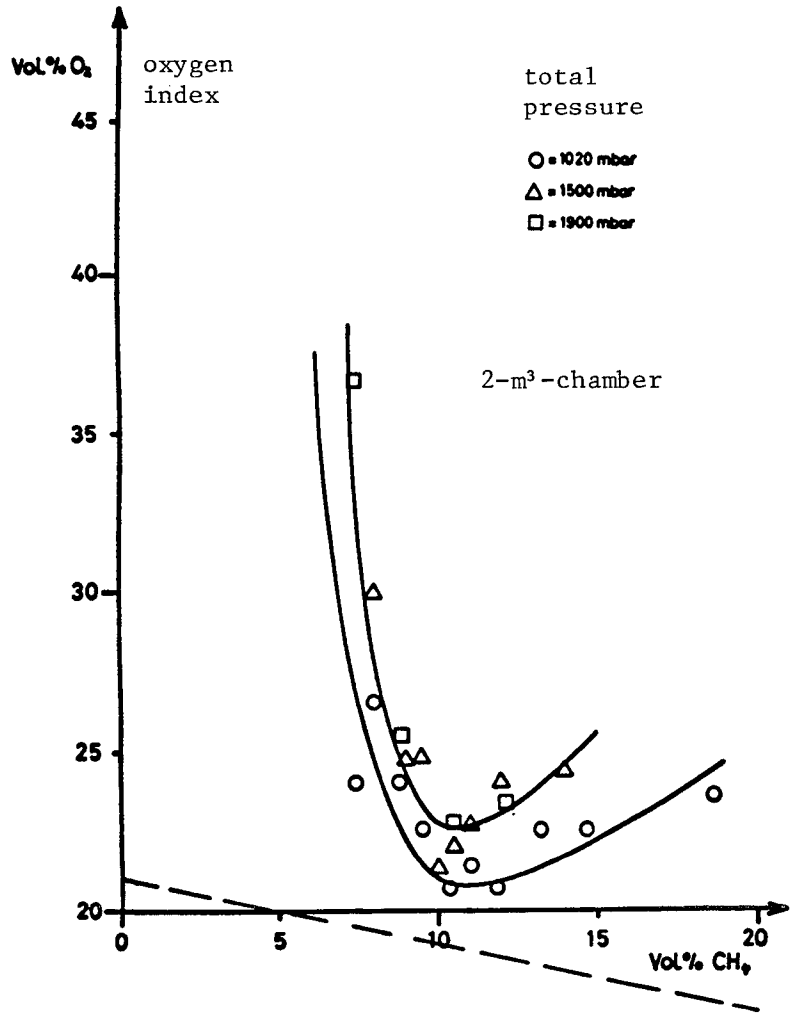


Figure 5: 300g W III in borehole mortar (Ø 50 mm)

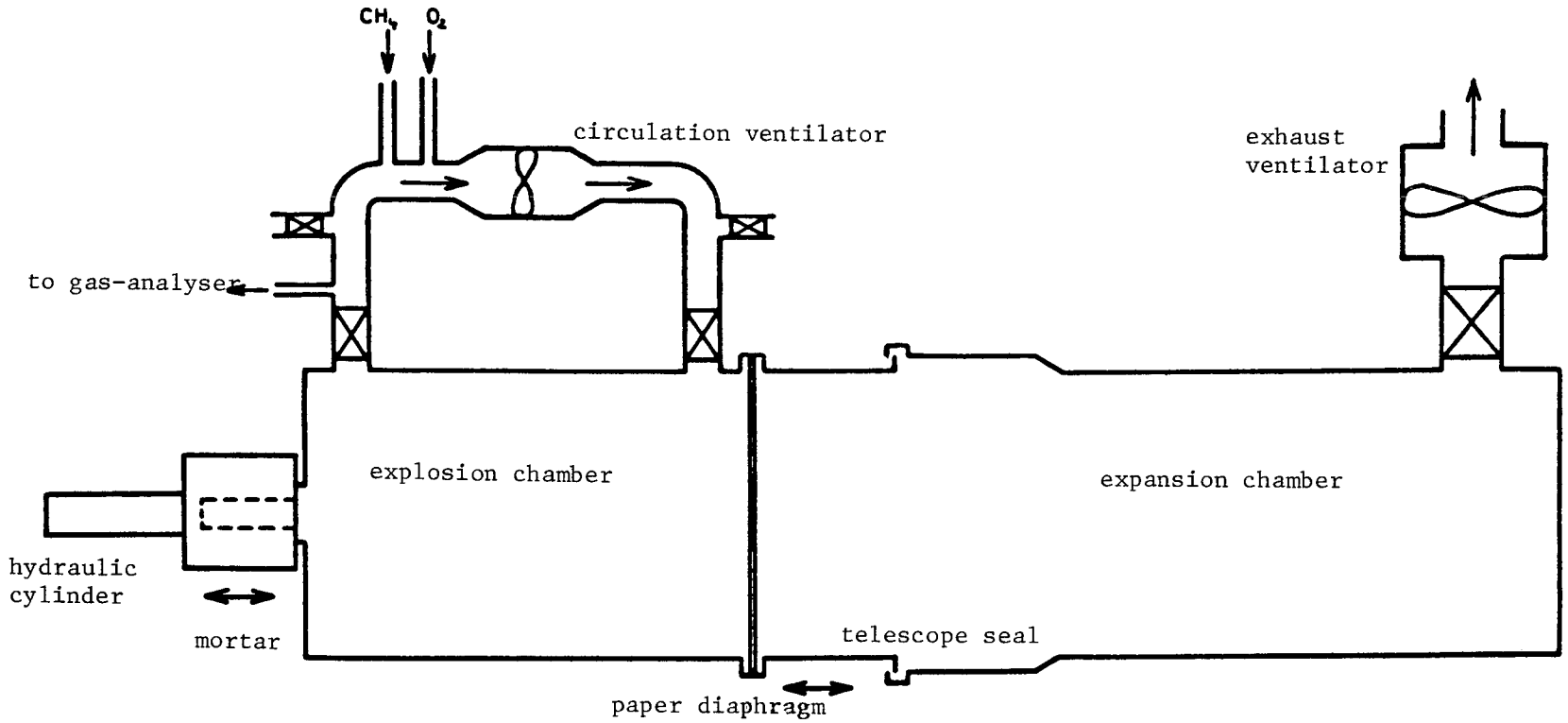


Figure 6: Schematic diagram of the closed gallery

test gallery, and, in the case of high temperatures, sprayed water for cooling, and other operations necessary before placing a new test charge in the gallery are carried out automatically with regard to the measured parameters.

c) fully automatically (under development): on the basis of the experiences made with the mode of operation b) a mathematical model is being developed, which automatically allows exact dosing of the test gas concentration and then unlocks the ignition installation.

The system is programmed with easily understandable clear text like control comands, which are generated in the programming language FORTH. the special interactive character of a FORTH-system allows to apply the three above described modes of operation simultaneously and alternatively, without the need to load the control software anew.

RESULTS OF THE OXYGEN TEST METHOD WITH CLASS III PERMITTED EXPLOSIVES

According to the results in the 2-m^3-chamber the highest ignition probability was found in slightly over-stoichiometric methane-air-mixtures. In order to adjust the testing regulations it was necessary to reproduce these results in the test gallery under normal pressure with the usual test charges. A W III-explosive with a charge of 10 cartridges at 25 cm wall distance was used, which had proved as the severest test arrangement in former tests. Here the results, plotted in Figure 7, show that the maximum ignition sensitivity is between 10 and 10.5 % b.v. methane concentration. The sensitivity curves are basically equivalent to those obtained in the 2-m^3-chamber.

In further tests the influence of possible impurities in the test methane was investigated. These are the combustible gases ethylene and hydrogen. The content of ethylene in the test gas is about 2000 to 3000 vpm, sometimes higher. Normally there is no hydrogen content, sometimes, however, it can be present up to 1000 vpm. The point of sensitivity of a mixture just not leading to an ignition has been set for a W III-permitted explosive in a 10.5 % b.v. methane-air-mixture by oxygen addition. It was then investigated with which addition of ethylene, or hydrogen respectively to the given oxygen concentration, ignitions were surely obtained. The test results, shown in Figure 8, establish that this was the case with 0.8 % b.v. ethylene and 2 - 3 % b.v. hydrogen in the mixture. This means that the content in the methane would have to be ten times these values. So it can be concluded that the normal impurities in the test gas do not have any significant influence on the ignition probability. It also was noted that the addition of combustible gases does not lead to an as clear transition from non-ignition to ignition as the relatively small percentual changes in oxygen concentration by volume.

GENERAL REMARKS ON COAL DUST TESTS WITH PERMITTED EXPLOSIVES

A similar test situation as described for highly safe permitted explosives, exists for the most types of permitted explosives with regard to their safety in coal dust. Although plausibility and experience lead to suppose that explosives, safe in firedamp, meet far higher demands with regard to safety in coal dust; this is not at all a compulsory conclusion, because other mechanisms become effective in firedamp explosions than in coal dust explosions. This was confirmed in the 60ies in special angle-shot-tests with the formerly not deflagration-proof W III permitted explosives. In the tests on safety in coal dust there are, in addition to the above mentioned different geometries in test arrangements, two other possible variants:

at first, the tested charge had to raise the coal dust by its own fumes;

at second, the coal dust is raised generally 0.5 to 1.0 seconds prior to the ignition of the test charge.

In both types of test arrangements the addition of oxygen to the

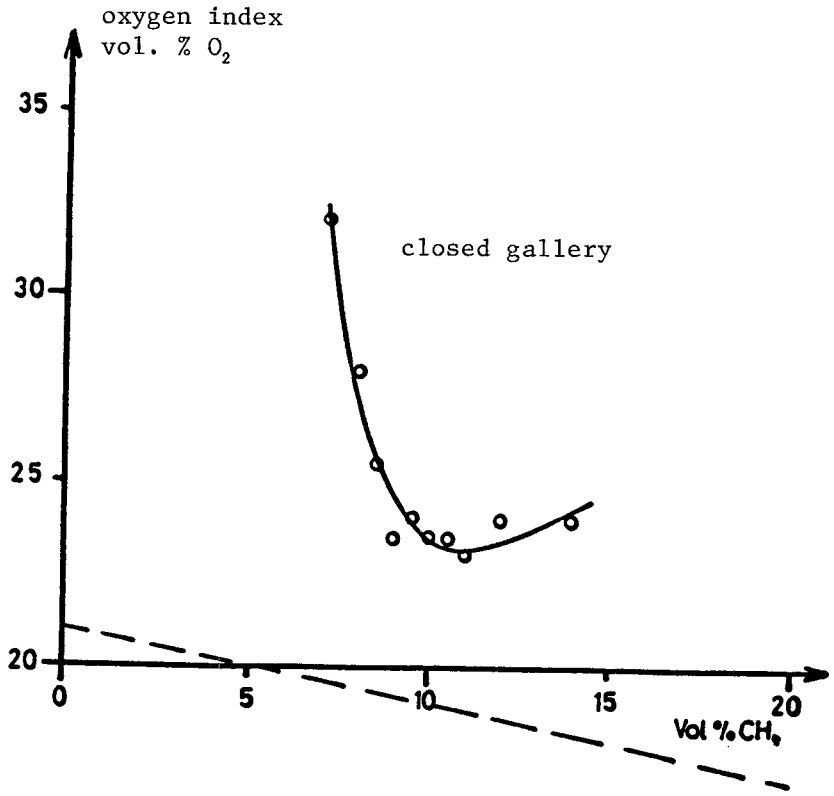


Figure 7: 10 cartridges of W III-explosive in angle-shot-mortar

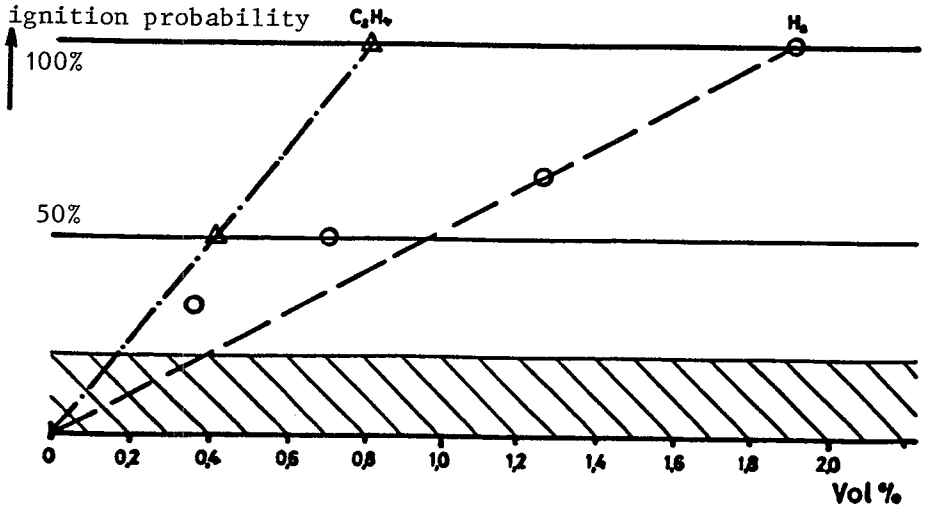


Figure 8: Influence of impurities in methane

atmosphere in the test gallery yields destinctive features.

In the tests, generally bituminous coal dust is used with high content of volatible matter, low content of ash and water, being crushed and milled under constant conditions to a grain size, characterized by median values between 10 - 30 µm.

PRELIMINARY RESULTS OBTAINED WITH THE OXYGEN-TEST-METHOD OF THE SAFETY IN COAL DUST OF W I-PERMITTED EXPLOSIVES

In the fundamental investigations to the safety of powderform W I-permitted explosives that have replaced the gelatinous W I-explosives at the beginning of the 80ies, it was not only realized that the powderform explosives in freely suspended arrangement were considerably safer in firedamp than the gelatinous explosives but also that in the tests in coal dust even under the test conditions for W II- and W III-explosives no ignitions were obtained (Zimmermann, 1981).

To gain more information on the safety of W I-explosives in different types of coal dust, test series were carried out similar to the official test arrangement. In the prescribed test, a charge of 60 cm length is fired in a mortar with a 60 cm borehole into a coal dust cloud, pre-raised a short time earlier by an auxiliary charge of 50 g of the same explosive. The test charge is initiated from the bottom of the borehole. To obtain a 50 % ignition probability, the oxygen content in the test gallery atmosphere was increased in steps of 2 % b. v. under the above conditions.

In the test series bituminous coal and dry lignite dusts were used in different quantities and raise times. The results in Figure 9 show no significant differences of ignition sensitivity, if the raise time remains within 0.5 to 1.5 seconds and the dust quantity between 1.5 and 3 kg for bituminous coal and 3 and 5 kg for lignites. Therefore, a good reproducibility of the test results can be anticipated.

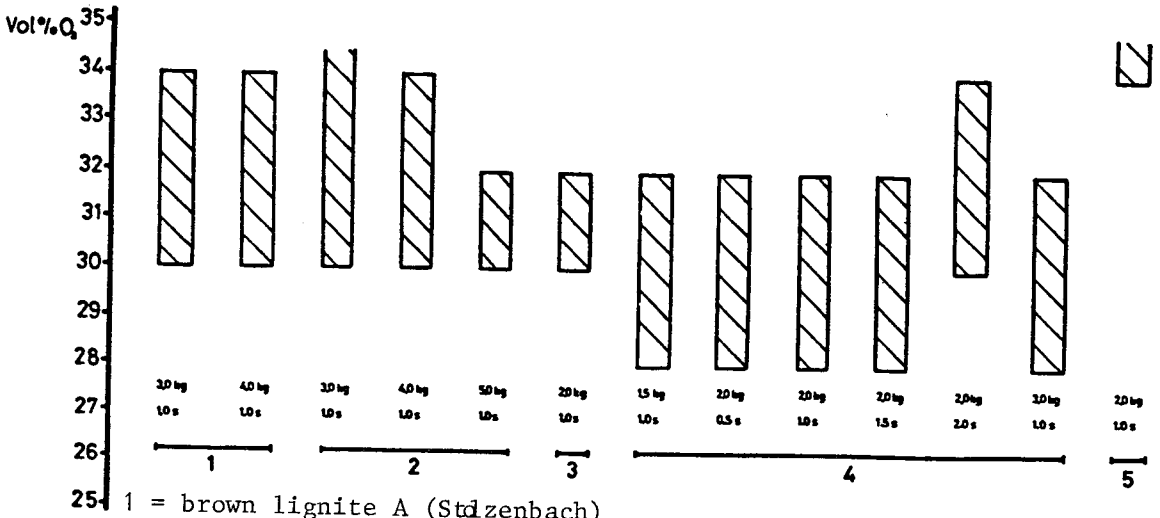
All coal dust tests must be carried out in the open gallery, because it is not possible to remove the dust completely from the expansion part of the closed gallery.

PRELIMINARY RESULTS FOR COMPARING THE COAL DUST SAFETY OF DIFFERENT PERMITTED EXPLOSIVES

The above reported test series has established the optimum parameters for raise time and coal dust quantity. So another test series was carried out to get a basic comparison on coal dust safety of all classes of permitted explosives. The prescribed test for W II- and W III-explosives was used. The charges, 2 m long, consisting of 30 mm-cartridges of W I-, W II- or W III-explosives, were fired in a mortar with a 2 m-borehole, 40 mm in diameter. Additional tests were done with similar charges of W III-explosives in the angle-shot-mortar, as well as with freely suspended instantaneous detonators of the permitted type with copper-shell and of the non-permitted type with aluminium-shell. The test results in Figure 10 show significant differences in the values obtained with permitted explosives as well as with detonators. Remarkable is the strongly decreased oxygen-index of the W III-explosive in angle-shot-mortar, in spite of the fact that this charge evaluates only a small part of its total energy. It should be emphasized that in normal atmosphere such differences are not detectable, as by no means ignitions occur, except for aluminium-detonators that ignite coal dust with a low probability.

CONCLUSIONS

With the presented test results it can be shown that by adding oxygen in tests on safety in firedamp and coal dust the evidence of the tests can be considerably enlarged from a yes/no decision to a semi-quantitive measuring method, which allows the comparison of different explosives having very different ignition risks with equal test methods and the comparison of different test methods with the same explosive. Also, the assessment of the safety distance of specific test arrangements to



1 = brown lignite A (Stdzenbach)
2 = brown lignite B (Rheinbraun)
3 = gas flame coal (Ensdorf)
4 = fat coal (Dortmunder Normalstaub)
5 = anthracite (Ibbenbüren)

Figure 9

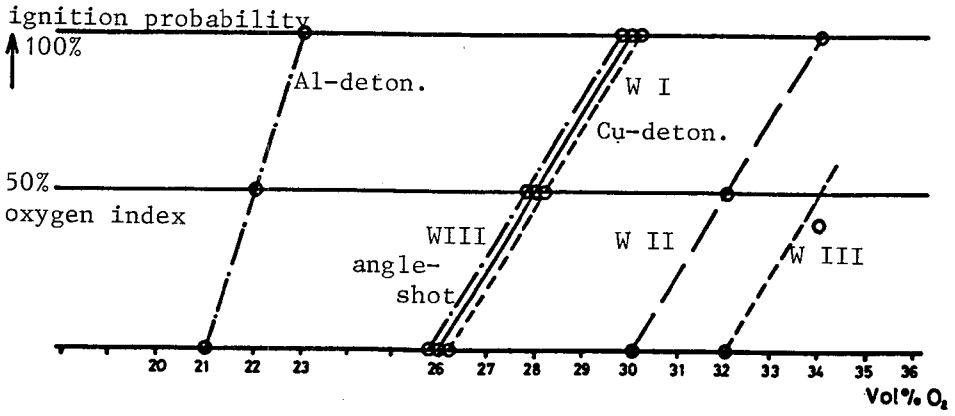


Figure 10: 2 m charges of different permitted explosives in borehole mortar, angle-shot mortar, and freely suspended detonators in bituminous coal dust

operational conditions, and the optimisation of the test parameters is possible.

This method is planned to be increasingly applied in the Federal Republic of Germany for the assessment of new developments of explosives and for the comparison with types of explosives being in use since several years. Moreover the method should be applied in underground experimental mines in order to get estimated values of the safety margin of explosive charges used in conditions being hazardous in firedamp or coal dust.

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SICHERHEIT VON WETTERSPRENGSTOFFEN IN SCHLAGWETTERN UND KOHLENSTAUB- SAUERSTOFFKENNWERT ALS PRÜFKRITERIUM

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Durch Experimente im Labormaßstab wurde in den 50er Jahren im US Bureau of Mines gezeigt, daß die Zündempfindlichkeit von Methan-Luft-Gemischen gegen Sprengstoffe durch Sauerstoffzusatz kontrolliert gesteigert werden kann, d. h. ohne Änderung der chemischen Reaktionspartner. Neuere Untersuchungen der BVS haben gezeigt, daß diese Methode geeignet ist, die Sicherheit der Wettersprengstoffe sowohl gegen Schlagwetter als auch gegen Kohlenstaub nicht nur unter den offiziellen Prüfbedingungen, sondern auch in anderen Sprenganordnungen durch eine Kennzahl, nämlich der zur Erzielung einer Entflammwahrscheinlichkeit von 50 % erforderlichen Sauerstoff-Konzentration, zu beschreiben. Über die mit deutschen Wettersprengstoffen verschiedener Sicherheitsklassen erhaltenen Ergebnisse wird berichtet.

Zur Durchführung derartiger Experimente wird eine geschlossene Prüfstrecke benutzt, deren Konstruktion das Austreten von Druckwellen verhindert. Es wird die Regeltechnik derjenigen Versuchsparameter erläutert, die zur Erzielung reproduzierbarer Untersuchungsergebnisse wesentlich sind. Zusätzlich wird über Untersuchungen bei erhöhtem Gesamtdruck in einer 2 m^3-Kammer (zur Simulation der Luftdruckzunahme in großen Teufen) hinsichtlich ihres Einflusses auf die Entflammhäufigkeit berichtet.

ALLGEMEINES ÜBER SCHLAGWETTER-TESTS AN WETTERSPRENGSTOFFEN

Die Einteilung der Wettersprengstoffe erfolgt seit nunmehr bald 100 Jahren nach dem Ergebnis bestimmter Prüfungen, d. h. durch Sprengversuche in schlagwettergefüllten Kammern. Dabei sollen die verwendeten Prüfanordnungen in reproduzierbarer Weise die als besonders gefährlich erkannten Bedingungen in der Praxis simulieren. Als plausible und allgemein anerkannte Erfahrungsregel gilt dabei, daß die Schlagwettersicherheit eines Sprengstoffes um so höher einzuschätzen ist, je größer die Ladung werden kann, die noch keine Schlagwetter-Explosionen verursacht. In dem Maße, wie man lernte, die Schlagwettersicherheit der Sprengstoffe zu steigern, wurde es erforderlich, auch die geprüften Ladungsmengen zu steigern oder, wenn dies nicht mehr möglich erschien, eine schärfere Prüfanordnung zu wählen.

So entwickelten sich im Laufe der Zeit in den verschiedenen Ländern unterschiedliche Prüfanordnungen und Prüfphilosophien. Während man in einigen Ländern versucht, mit möglichst kleinen Lademengen auszukommen, und bei den Prüfungen die Lademenge mit einer 50%igen Entflammhäufigkeit ermittelt, werden in anderen Ländern Prüfungen bevorzugt, die sich an den in der Praxis verwendeten Lademengen orientieren und bei denen unter diesen Bedingungen überhaupt keine Entflammungen auftreten dürfen. Außerdem haben sich in einzelnen Ländern mehrere Sicherheitsklassen der Sprengstoffe herausgebildet, die in jeweils unterschiedlichen Betriebs- und damit Gefahrensituationen verwendet werden müssen.

In aller Regel finden die Prüfungen in einem nahezu stöchiometrischen Methan-Luft-Gemisch statt. Die Anwendung anderer, zündempfindlicherer Brenngase ist nur in Ausnahmefällen vorgeschrieben.

ENTWICKLUNG VON TESTMETHODEN MIT SAUERSTOFF-ZUSATZ

Bereits in den 30iger Jahren benutzten Beyling und Schulze-Rhonhof zum Nachweis deflagrierender Sprengstoffteilchen vor den Bohrlöchern Ballons, welche mit sehr zündempfindlichen Methan-Sauerstoff-Mischungen gefüllt waren.

In den 50iger Jahren entwickelten Lewis und von Elbe im Bureau of Mines eine Methode zur Beobachtung von Schlagwetterentflammungen mit sehr kleinen Ladungen in Miniaturmörsern. Die Zündempfindlichkeit des Schlagwettergemisches regelten sie durch einen Zusatz von Sauerstoff. Sie stellten bereits fest, daß bei ein- und demselben Sprengstoff der Übergang von einer nichtzündbaren zu einer zündbaren Gasmischung in einem Bereich von 2 - 3 Vol.% Sauerstoff erfolgt und definierten einen 50%-Sauerstoffwert als Mittelwert zwischen dem Sauerstoffgehalt bei fünf Zündungen bzw. fünf Nichtzündungen. Sie stellten weiterhin fest, daß dieser 50%-Sauerstoffwert typisch war für eine bestimmte Sprengstoffmischung und daß sich mit diesem Wert die Sprengstoffe nach dem Grad ihrer Schlagwettersicherheit einordnen ließen. Allerdings zeigten gewisse Sprengstoffe, die aus der Praxis als sehr zündgefährlich bekannt waren, bei dieser Methode extrem hohe Sauerstoffwerte und damit scheinbar eine hohe Sicherheit. Diese augenscheinlichen Fehlbeurteilungen ließen es ratsam erscheinen, die Ergebnisse der Sauerstoffmethode nicht als Grundlage für eine offizielle Zulassung heranzuziehen.

In Deutschland war in den 70iger Jahren bei der BVS zunächst aus Gründen der Lärmverminderung eine geschlossene Sprengstoffprüfstrecke ähnlich der in Bruceton vorhandenen gebaut worden. Es zeigte sich, daß hierin auch wesentlich heftigere Explosionen ohne Belästigung der Umgebung ausführbar waren als in der bisher vorhandenen offenen Prüfstrecke. Anläßlich einer Sonderprüfung von schlagwettersicheren Treibladungskartuschen wurden hierin auch erste Versuche mit Sauerstoffzusatz durchgeführt. Diese zeigten, daß sich die im Bureau of Mines gewonnenen Erkenntnisse auch bei Verwendung anderer Zündquellen bestätigten.

Darauf wurden einzelne Versuchsreihen auch über die Zündwahrscheinlichkeit von schlagwettersicheren Sprengzündern und von Einzelpatronen verschiedener Wettersprengstoffe durchgeführt, über die bereits auf der Grubensicherheitskonferenz in Sheffield (Franke und Zimmermann, 1983) berichtet wurde. Hierbei zeigte sich an den verschiedenen Sauerstoff-Kennwerten deutlich die unterschiedliche Sicherheit gleichgroßer Ladungen aus verschiedenen Wettersprengstoffklassen W I bis W III und gleichzeitig die die Sicherheit verbessernde Wirkung der Sprengstoffpatronen gegenüber dem Sprengzünder als dem am wenigsten sicheren Bestandteil der Ladung (Bild 1). Der Einfluß der Ladungsmenge sowie der Prüfanordnung zeigt (Bild 2). Man erkannte weiter, daß der verwendete Wettersprengstoff der Klasse III in der dargestellten Kantenmörseranordnung, bei der er lt. Prüfvorschrift in keinem Fall Schlagwetter entflammen darf, die geforderte Sicherheit offenbar zwar noch erreichte, aber keine deutliche Reserve mehr besaß.

TEST DER SCHLAGWETTERSICHERHEIT UNTER ERHÖHTEM ANFANGSDRUCK

Zu dieser Zeit war in Westdeutschland begonnen worden, den Abbau der Steinkohle bis zu Teufen von 1600 m zu planen. Aufgrund der barometrischen Druckzunahme in großen Teufen war zu vermuten, daß der Partialdruck des Luftsauerstoffs dort bereits Werte annehmen würde, bei welchen in der Prüfstrecke - allerdings bei einem niedrigeren Gesamtdruck - bereits Entflammungen mit dem Wettersprengstoff der Klasse III beobachtet worden waren. Ein daher zwischenzeitlich aufgenommenes Untersuchungsvorhaben über die Mindestzündenergie von elektrischen Funken in Abhängigkeit vom Sauerstoffpartialdruck und Gesamtdruck hatte ergeben, daß die bei Sauerstoffzusatz zu Schlagwettergemischen unter Normaldruck beobachtete Verringerung der Mindestzündenergie bei Erhöhung des Sauerstoffpartialdruckes durch Gesamtdruckerhöhung zwar nicht in vollem Umfang, aber dennoch deutlich nachzuweisen war. Es sollte daher untersucht werden, ob dieser Effekt auch für die Zündquelle Wettersprengstoffe zuträfe. Im Gegensatz zu elektrischen Funken sind Wettersprengstoffe bezüglich der umgesetzen Energie eher als starke Zündquellen anzusehen.

Die bei der BVS vorhandenen Prüfstrecken sind zwar darauf ausgelegt, hohe Druckstöße zu ertragen; sie sind jedoch nicht im eigentlichen Sinne gasdruckdicht ausgeführt. Daher lassen sich statistische Überdrücke von einigen Zehntel bar, wie sie für derartige Untersuchungen erforderlich wären, in ihnen nicht einstellen. Es wurde daher auf eine vorhandene, edelstahl-plat-

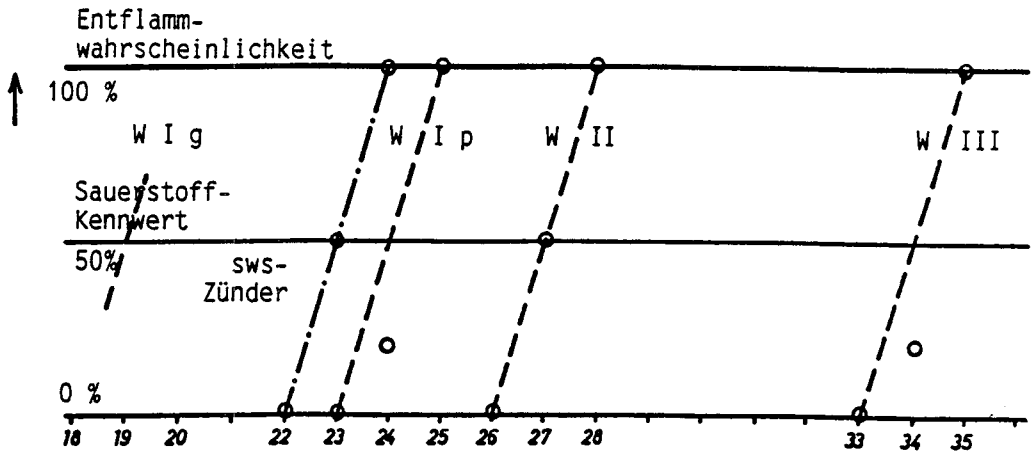


Bild 1: Sauerstoff-Kennwert von 1 Patr. Wettersprengstoff (freihängend in der Prüfstrecke)

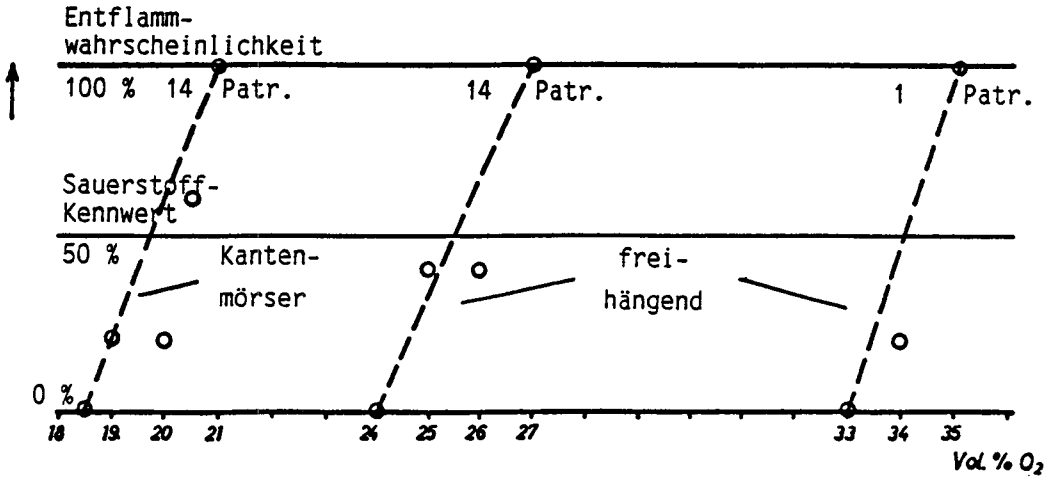


Bild 2 Sauerstoff-Kennwert verschiedener Ladungen von W III - Sprengstoff

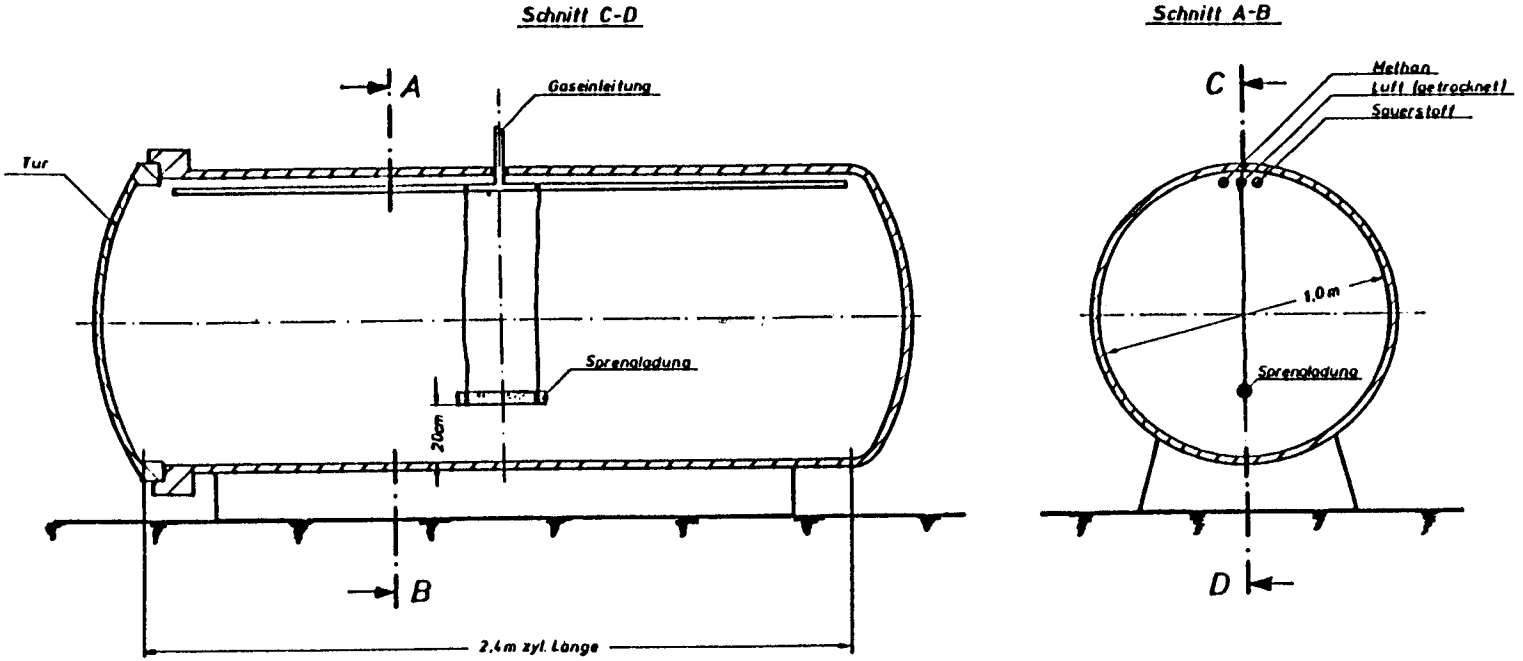


Bild 3 Druckdichte 2m³-Kammer

tierte gasdichte Sprengkammer von 2 m^3 Volumen zurückgegriffen, die mit elektrischer Heizung und Gaszuführungen sowie Gaskonzentrations- und Druckmeßgeräten ausgerüstet wurde, so daß hierin unter sehr gut reproduzierbaren Bedingungen Versuche an Modell-Ladungen von W III-Sprengstoff durchgeführt werden konnten (Bild 3). Die Gesamtdrücke wurden zwischen 1020 und 1900 mbar variiert; die Partialdrücke für Methan zwischen 75 und 280 mbar und für Sauerstoff zwischen 210 und 840 mbar. Dies entspricht Volumenkonzentrationen zwischen 7,4 und 18,6 % für CH_4 und 20 bis 44 % für O_2. Als Ladungen wurden jeweils 300 g W III-Sprengstoff entweder freihängend mit 20 cm Wandabstand oder in einem beidseitig offenen Bohrlochmörser gezündet.

Hierbei zeigte es sich, daß es, um eine Explosionshäufigkeit von 50 % zu erreichen, offenbar nicht auf die Partialdrücke, sondern auf die Volumenanteile der Gase im Gemisch ankam. Wie Bild 4 zeigt, lassen sich für die freihängende Ladung alle Sauerstoffkennwerte für die 50%ige Entflammhäufigkeit als Funktion des Methangehaltes, unabhängig vom Gesamtdruck, auftragen. Bei Zündung des Sprengstoffes in einem Bohrlochmörser wurde mit steigendem Gesamtdruck sogar eine leicht zunehmende Tendenz der Sauerstoffkennwerte beobachtet (Bild 5).

Die Versuche zeigten aber noch ein weiteres interessantes Ergebnis: Wenn man den zur Erreichung einer 50%igen Explosionswahrscheinlichkeit erforderlichen Sauerstoffzusatz als Maß für die Zündempfindlichkeit des Gasgemisches ansieht, so ist offenbar eine sehr starke Abnahme der Empfindlichkeit bei unterstöchiometrischen Methan/Luft-Gemischen gegeben, während bei stark überstöchiometrischen Gemischen die Empfindlichkeit zwar auch wieder erkennbar abnimmt, aber doch bei weitem nicht so stark. Man kann auch feststellen, daß das Minimum der Kurve, entsprechend dem Maximum der Zündwahrscheinlichkeit, zumindest bei dem hier verwendeten Sprengstoff etwas oberhalb des stöchiometrischen Gemisches liegt.

In der Praxis wird bei der Anwendung von Kurzzeitverzögerungs-Zündern und sorgfältiger Gasmessung vor dem Sprengen die Entstehung explosionsgefährlicher Gasgemische von der brenngasarmen Seite erfolgen und wahrscheinlich nur in seltenen Fällen in dieser kurzen Zeit von max. 500 ms bis zum stöchiometrischen Gemisch oder darüber führen. Andererseits ist nach den bisherigen Prüfvorschriften die offenbar gut reproduzierbare Prüfung im überstöchiometrischen Bereich nicht vorgesehen.

DIE GESCHLOSSENE PRÜFSTRECKE UND IHRE MESS- UND REGELTECHNISCHE AUSRÜSTUNG

Durch Undichtigkeiten in den Prüfstrecken kann es im Zeitraum zwischen Abstellen der Gaszufuhr und Zünden der Prüfladung zu einem recht schnellen Abfall des Methangehaltes kommen, der die Empfindlichkeit bereits deutlich beeinträchtigt. Es ist daher erforderlich, diese Parameter genau zu kontrollieren und konstant zu halten. Dazu wurde ein rechnergesteuertes Regel- und Meßsystem für die geschlossene Prüfstrecke entwickelt. Bild 6 zeigt eine schematische Darstellung dieser Prüfstrecke.

Anstelle einer sonst für derartige Steuerungsaufgaben üblichen speicherprogrammierten Ablaufsteuerung wurde ein modulares Steuersystem verwendet, welches ähnlich einfach zu programmieren ist, jedoch einige, für die vorliegenden Steuerungsaufgaben wesentliche Leistungsmerkmale aufweist, die sonst nur bei größeren Prozeßrechnern zu finden sind:

Erfaßt werden die Eingangsparameter

- Temperatur in der Prüfstrecke (mehrere Meßpunkte)
- Methankonzentration
- Sauerstoffkonzentration
- Luftdruck
- Schieber- und Ventilator-Schaltzustände

Vorgegeben werden (über eine Tastatur) die Sollwerte für

- Gaskonzentrationen.

Angezeigt werden die jeweiligen Betriebsabläufe und Zustände der Prüfstrecke auf einem Farbmonitor.

Die Anlage kann in verschiedenen Betriebsmoden gefahren werden:

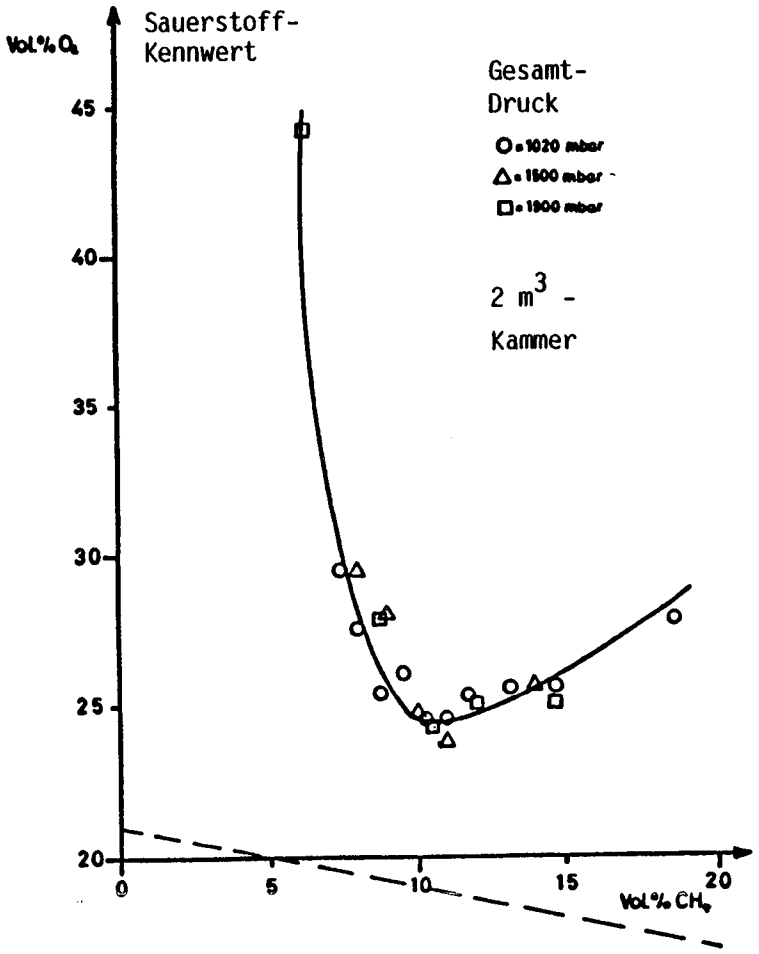


Bild 4 300 g W III freihängend
(Wandabstand 20 cm)

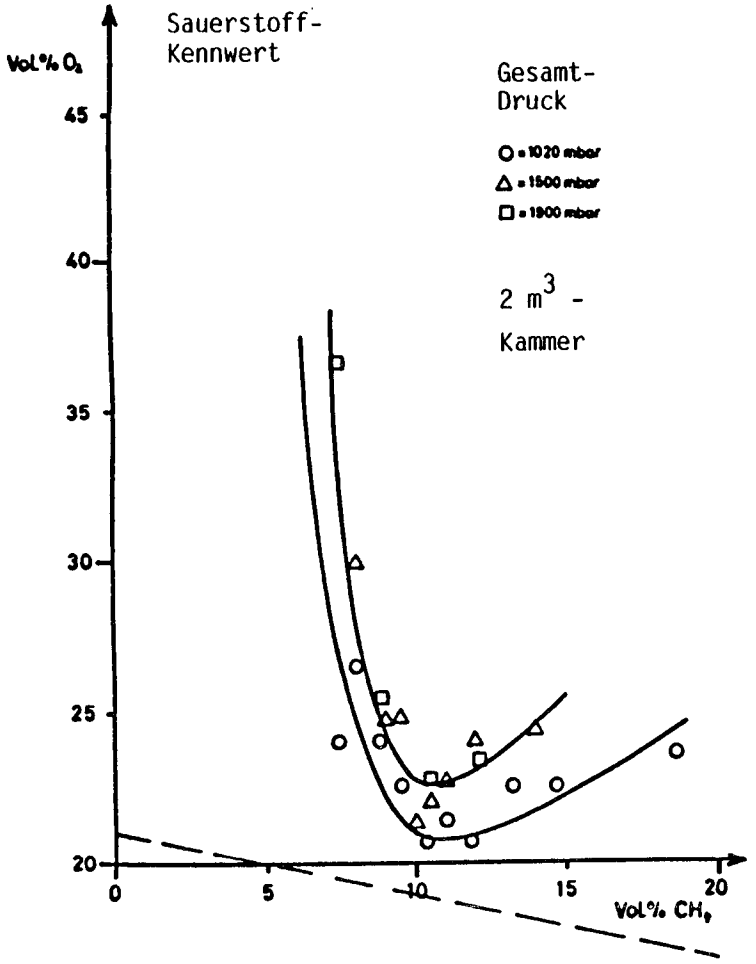


Bild 5 300 g W III im Bohrloch-
Mörser (Ø 50 mm)

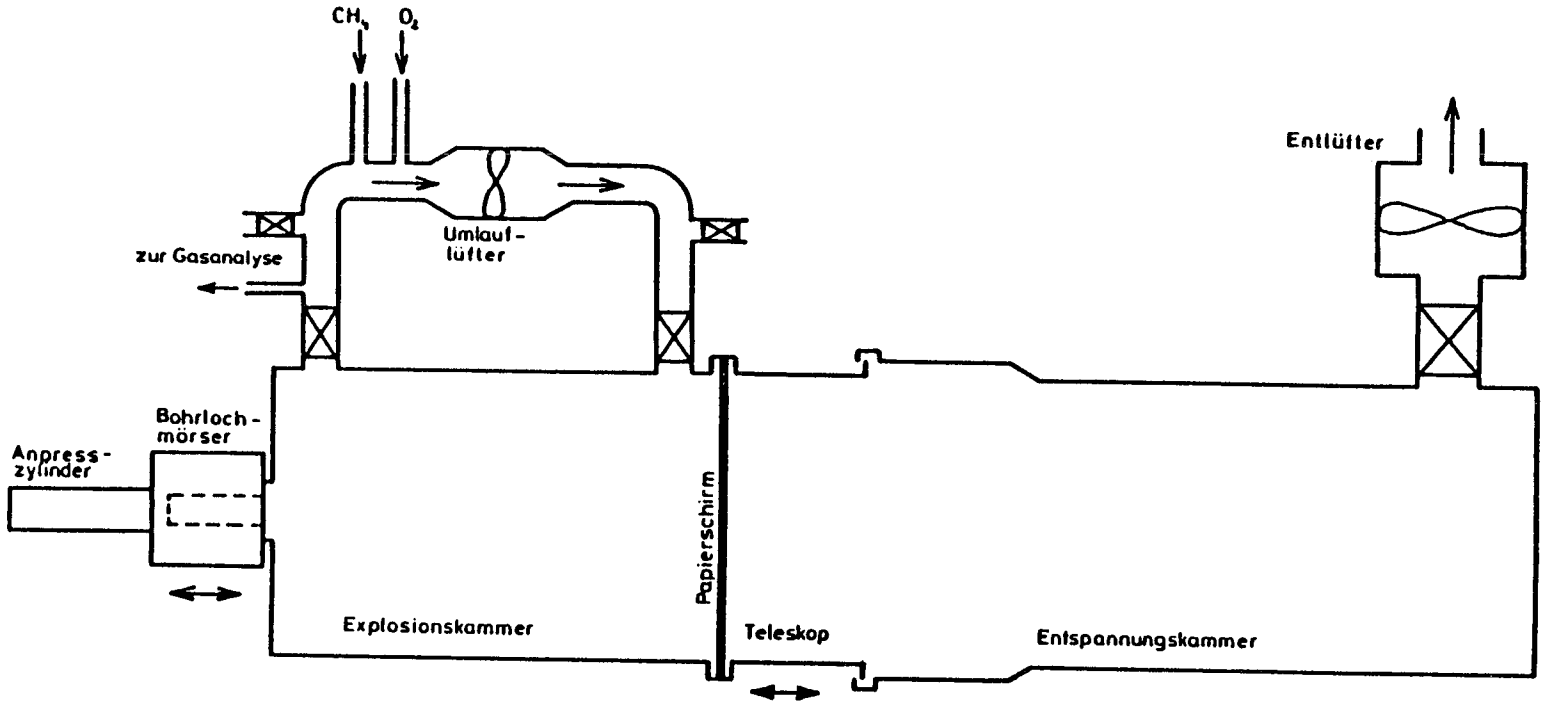


Bild 6: Schema der geschlossenen Prüfstrecke

a) manuell: Der Betriebsablauf kann über die Tastatur ähnlich wie über die Bedienungselemente einer Schalttafel gesteuert werden, wobei zur Vereinfachung der Bedienung mehrere Teilabläufe programmtechnisch verknüpft sind (der Befehl "Gasumlauf an" umfaßt z.B. die zeitlich abgestimmte Betätigung mehrerer Schieber und elektrischer Schalter für den Ventilator).

b) halbautomatisch: Aufgrund der Sollwertvorgaben werden die Prüfgase grob vordosiert; die Feindosierung erfolgt über die Tastatur "von Hand". Bei Erreichen der Sollwerte wird das Zündgerät entriegelt, so daß die Prüfladung gezündet werden kann. Die Schwadenabsaugung, das Öffnen der Prüfstrecke, sowie bei zu hoher Temperatur die Kühlung durch versprühtes Wasser und ähnliche vor dem erneuten Einbringen einer Prüfladung erforderliche Vorgänge erfolgen in Abhängigkeit von den gemessenen Parametern automatisch.

c) vollautomatisch (in Entwicklung): Aufgrund der in der Betriebsart b) gewonnenen Erfahrungswerte wird an einem mathematischen Modell gearbeitet, welches die exakte Dosierung der Prüfgaskonzentration vollautomatisch vornimmt und daraufhin die Zündanlage freigibt.

Die Programmierung des Systems erfolgt unter Verwendung von leicht verständlichen klartextähnlichen Steuerbefehlen, die ihrerseits wieder in der Programmiersprache FORTH erstellt werden. Die besonderen interaktiven Eigenschaften eines FORTH-Systems erlauben es, die beschriebenen drei verschiedenen Betriebsarten nebeneinander und abwechselnd zu verwenden, ohne die Steuerungssoftware jeweils neu laden zu müssen.

ERGEBNISSE DER SAUERSTOFF-TEST-METHODE AN W III-SPRENGSTOFF

Da nach den Ergebnissen in der 2 m^3-Kammer die höchste Zündempfindlichkeit bei leicht überstöchiometrischen Methan/Luft-Gemischen gefunden wurde, war es erforderlich, diese Ergebnisse bei Normaldruck in der Prüfstrecke mit den zur Prüfung benutzten Lademengen zu reproduzieren. Es wurde auf die Prüfung des W III-Sprengstoffes mit Ladungen von 10 Patronen bei 25 cm Prallwandabstand zurückgegriffen, die sich auch bei früheren Untersuchungen als die offenbar schärfste Prüfbedingung erwiesen hatte. Auch hier zeigen die in Bild 7 dargestellten Ergebnisse, daß die maximale Zündempfindlichkeit bei Methankonzentrationen zwischen 10 und 10,5 % zu erwarten ist. Der Kurvenverlauf ist im wesentlichen deckungsgleich mit den an der 2 m^3-Kammer erhaltenen Empfindlichkeitskurven.

In weiteren Versuchsreihen wurde der Einfluß von Verunreinigungen des zur Prüfung verwendeten Methans untersucht. Hierbei handelt es sich um die brennbaren Gase Ethylen und Wasserstoff. Ethylen ist im Prüfgas in Konzentrationen zwischen 2000 und 3000 vpm, manchmal jedoch auch darüber enthalten. Wasserstoff tritt normalerweise nicht auf, kann in einzelnen Chargen jedoch Werte bis zu 1000 vpm annehmen. Mit einem Wettersprengstoff der Klasse III wurde bei einer Konzentration von 10,5 Vol.% Methan durch Sauerstoffzusatz der Punkt einer gerade noch nicht zur Zündung führenden Empfindlichkeit des Gemisches eingestellt. Es wurde dann versucht zu ermitteln, bei welcher Zumischung von Ethylen bzw. Wasserstoff bei dieser Sauerstoffkonzentration eine sichere Zündung erreicht wurde. Die Ergebnisse in Bild 8 zeigen, daß dies erst bei Ethylenkonzentration von etwa 0,8 % bzw. Wasserstoffkonzentrationen von 1,9 % der Fall ist. Um diese Konzentrationen zu erreichen, muß das Methan die Fremdgase in 10-fach höheren Werten enthalten. Daraus kann geschlossen werden, daß die üblichen Verunreinigungen im Prüfgas noch keinen signifikanten Einfluß auf die Entflammhäufigkeit haben können. Man kann auch feststellen, daß die Zumischung von brennbaren Fremdgasen keineswegs zu so deutlich ausgeprägten Übergängen von Nichtentflammung zu Entflammung führt, wie sie die relativ kleinen prozentualen Änderungen der Sauerstoffvolumenkonzentration bereits bewirken.

ALLGEMEINES ÜBER KOHLENSTAUB-TESTS AN WETTERSPRENGSTOFFEN

Eine prüftechnisch ähnliche Situation, wie sie oben für die hochsicheren Wettersprengstoffe geschildert wurde, liegt für die meisten verwendeten Wet-

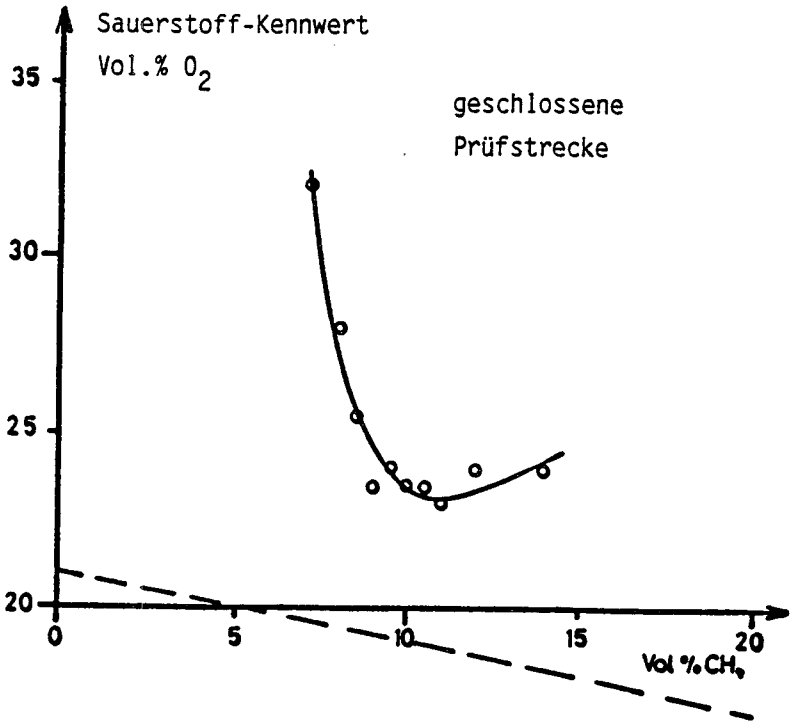


Bild 7 10 Patronen W III-Sprengstoff im Kantenmörser

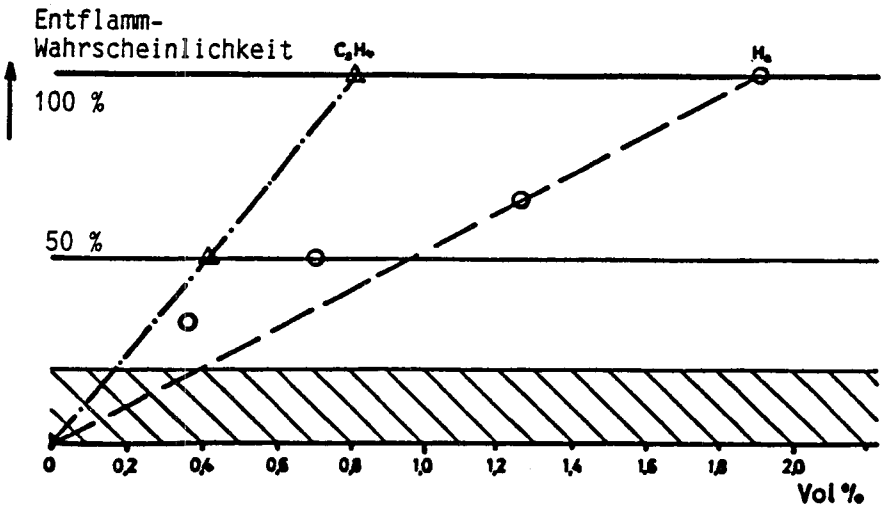


Bild 8 Einfluß von Verunreinigungen des Methans

tersprengstoffe hinsichtlich ihrer Kohlenstaubsicherheit vor. Es ist durchaus plausibel und entspricht der Erfahrung, anzunehmen, daß Sprengstoffe, die eine gewisse Schlagwettersicherheit erreichen, bezüglich ihrer Kohlenstaubsicherheit weit höheren Anforderungen genügen. Die Annahme ist jedoch keineswegs zwingend, da bei der Kohlenstaubzündung andere Mechanismen als bei der Schlagwetterzündung wirksam werden können. Dies haben Untersuchungen zur Kohlenstaubsicherheit im Kantenmörser an den früheren, nicht deflagrationsfesten W III-Sprengstoffen in den 60er Jahren bestätigt. Bei der Prüfung der Kohlenstaubsicherheit kann man zusätzlich zu den oben erwähnten unterschiedlichen Geometrien der Prüfanordnungen zwei weitere Varianten unterscheiden:

Bei der ersten Variante muß die zu prüfende Ladung mit ihrer Schwadenwolke selbst den Kohlenstaub aufwirbeln.

Bei der zweiten Variante erfolgt die Aufwirbelung eine kurze Zeit, meist 0,5 - 1,0 Sekunden, vor dem Zünden der Prüfladung.

Bei beiden Prüfanordnungen kann die Zumischung von Sauerstoff zur Prüfstreckenatmosphäre Unterscheidungsmerkmale liefern. Bei dem Prüfkohlenstaub handelt es sich meist um einen Steinkohlenstaub mit hohen Anteilen flüchtiger Bestandteile und einem geringen Asche- und Wassergehalt; der Staub wurde unter gleichbleibenden Bedingungen auf eine Kornfeinheit gemahlen, die durch Medianwerte von 10 - 30 µm gekennzeichnet ist.

ERSTE ERGEBNISSE DER SAUERSTOFF-TEST-METHODE BEI DER KOHLENSTAUBSICHERHEIT VON W I-SPRENGSTOFFEN

Bei den grundlegenden Untersuchungen zur Sicherheit der pulverförmigen Wettersprengstoffe der Klasse I, die Anfang der 80iger Jahre die gelatinösen Wettersprengstoffe der Klasse I abgelöst haben, war nicht nur beobachtet worden, daß die pulverförmigen Wettersprengstoffe in freihängender Anordnung gegen Schlagwetter wesentlich sicherer als die gelatinösen Sprengstoffe sind, sondern es wurde auch festgestellt, daß bei den Prüfungen auf Kohlenstaubsicherheit sogar unter den für Klasse II und III vorgesehenen Bedingungen keine Entflammungen mehr erhalten wurden (Zimmermann, 1981).

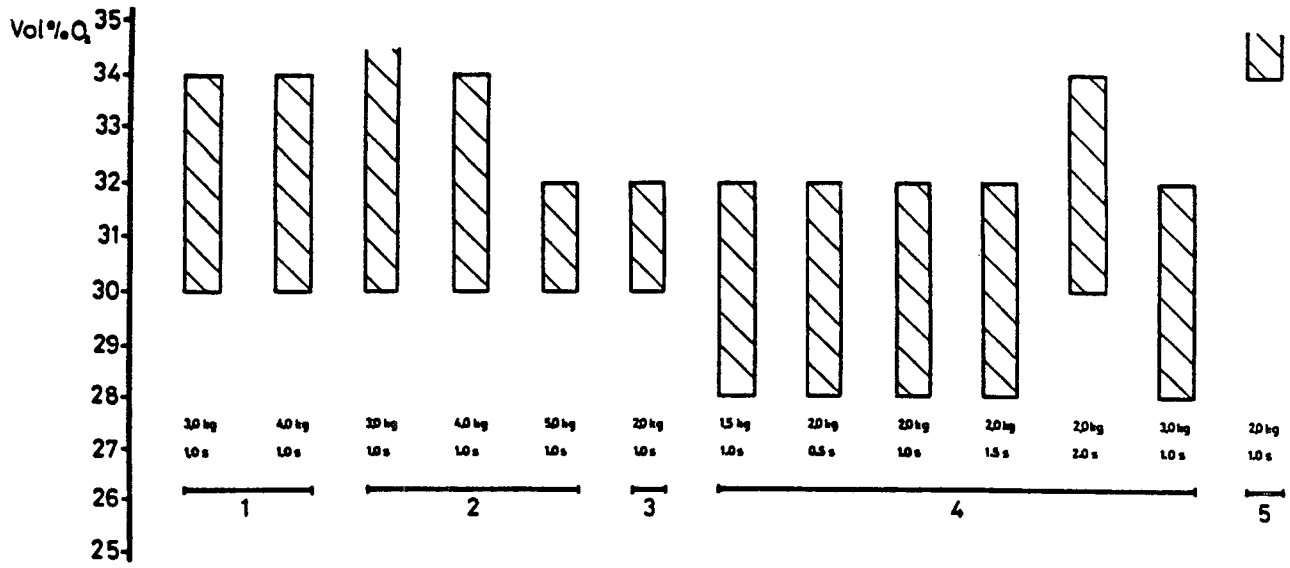
Um die Kenntnisse über die Sicherheit der W I-Sprengstoffe gegen verschiedene Arten von Kohlenstaub zu erweitern, wurden Versuchsreihen durchgeführt, ähnlich der offiziellen Prüfanordnung. In der vorgeschriebenen Prüfung wird eine Ladung von 60 cm Länge aus einem Mörser mit 60 cm langem Bohrloch in eine Kohlenstaubwolke gesprengt, welche eine kurze Zeit früher durch eine Hilfsladung desselben Sprengstoffs aufgewirbelt wurde. Die Prüfladung wird vom Tiefsten des Bohrlochs gezündet. Um die 50 %ige Zündwahrscheinlichkeit zu erhalten, wurde der Sauerstoffgehalt in der Prüfstreckenatmosphäre in Schritten von 2 Vol.-% erhöht.

In den Versuchsreihen wurden Steinkohlen- und trockene Braunkohlenstäube verwendet mit verschiedenen Mengen und Aufwirbelzeiten. Die Ergebnisse in Bild 9 zeigen jedoch keine signifikanten Unteschiede der Zündempfindlichkeit, sofern die Aufwirbelzeit zwischen 0,5 und 1,5 s bleibt und die Staubmenge zwischen 1,5 und 3 kg für Steinkohlenstaub und 3 und 5 kg für Braunkohlenstaub. Damit ist gezeigt, daß eine gute Reproduzierbarkeit der Versuchsergebnisse unterstellt werden kann.

Alle Kohlenstaubtests mußten in der offenen Prüfstrecke durchgeführt werden, da es z. Z. unmöglich erscheint, den Staub vollständig aus dem Ausdehnungsteil der geschlossenen Prüfstrekke zu entfernen.

ERSTE ERGEBNISSE ZUM VERGLEICH DER KOHLENSTAUBSICHERHEIT VERSCHIEDENER WETTERSPRENGSTOFFE

Die oben beschriebene Versuchsserie hat bestätigt, daß die optimalen Parameter für Aufwirbelzeit und Kohlenstaub gewählt wurden. Daraufhin wurde eine weitere Versuchsserie durchgeführt, um einen grundsätzlichen Vergleich der Kohlenstaubsicherheit aller Klassen der Wettersprengstoffe zu erhalten. Es wurde der vorgeschriebene Test für W II- und W III-Sprengstoffe benutzt. Bei diesem werden 2 m lange Ladungen, die aus 30 mm-Patronen von W I-, W II- oder W III-Sprengstoffen bestehen, in einem Mörser mit 2 m langem Bohrloch und 40 mm Durchmesser zur Detonation gebracht. Zusätzlich wurden



1 = Braunkohle A (Stolzenbach)
2 = Braunkohle B (Rheinbraun)
3 = Gasflammkohle (Ensdorf)
4 = Fettkohle (Dortmunder Normalstaub)
5 = Magerkohle (Ibbenbüren)

Bild 9 Ergebnisse bei Variation der Kohlenstaubmengen und Aufwirbelzeiten

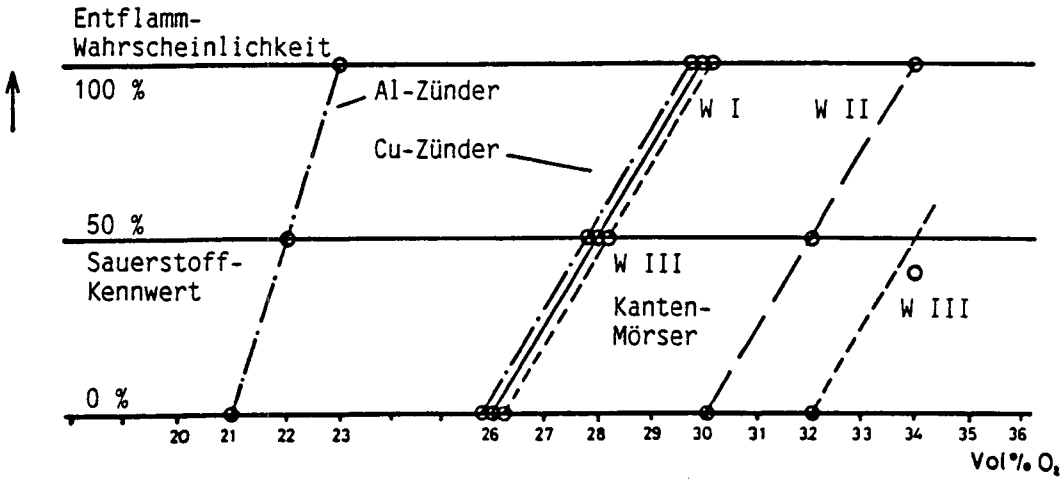


Bild 10 2 m - Ladungen verschiedener Wettersprengstoffe im Bohrloch-Mörser bzw. Kantenmörser sowie freihängende Zünder im Fettkohlenstaub

Versuche einbezogen mit gleichartigen Ladungen von W III-Sprengstoffen im Kantenmörser sowie mit frei aufgehängten Moment-Sprengzündern, sowohl in schlagwettersicherer Ausführung mit Kupferhülse als auch in nicht schlagwettersicherer Ausführung mit Aluminiumhülse. Hier zeigen die Ergebnisse in Bild 10 ausgeprägte Unterschiede sowohl in den Kennwerten der verschiedenen Wettersprengstoffe als auch der Sprengzünder. Bemerkenswert ist der stark verminderte Sauerstoffindex des W III-Sprengstoffs im Kantenmörser, obwohl eine solche Ladung nur einen kleinen Teil ihrer Gesamtenergie freisetzt.

Es muß hervorgehoben werden, daß in normaler Prüfstreckenatmosphäre die beschriebenen Unterschiede nicht erkennbar sind, da in keinem Fall Explosionen eintreten, mit Ausnahme des Aluminiumzünders, der Kohlenstaub mit einer gewissen Wahrscheinlichkeit zündet.

SCHLUSSFOLGERUNGEN

Mit den vorgelegten Versuchsergebnissen wird gezeigt, daß bei Prüfungen der Schlagwetter- oder Kohlenstaubsicherheit von Wettersprengstoffen durch Zusatz von Sauerstoff in einfacher Weise die Aussagekraft dieser Prüfungen von einer Ja/Nein-Entscheidung zu einer halbquantitativen Meßmethode erweitert werden kann. Diese erlaubt den Vergleich verschiedener Sprengstoffe unter gleichartigen Prüfmethoden auch bei weit auseinanderliegendem Zündrisiko sowie den Vergleich verschiedener Prüfmethoden mit dem gleichen Sprengstoff. Ferner erscheint damit eine Abschätzung des Sicherheitsabstandes bestimmter Prüfanordnungen zu Bedingungen der Praxis sowie eine Optimierung der Prüfparameter möglich.

Die Methode soll daher in der Bundesrepublik Deutschland im verstärken Umfang zur Abschätzung der Sicherheit von Sprengstoff-Neuentwicklungen und zum Vergleich dieser mit herkömmlichen Sprengstoffen eingesetzt werden. Darüber hinaus könnte die Methode in Versuchsgruben angewendet werden, um Schätzwerte des Sicherheitsabstandes von Sprengladungen zu erhalten, die unter gefährlichen Bedingungen in Schlagwettern oder Kohlenstaub abgetan werden.

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SECURITE GRISOUMETRIQUE DES EXPLOSIFS AUTORISES - INDICE D'OXYGENE UTILISE COMME CRITERE D'ESSAI

R. Zimmermann, M. Streich

Au cours des années 50, des expériences à petite échelle effectuées dans le Laboratoire du Bureau des Mines Américain ont permis d'établir que l'inflammabilité des mélanges méthane-air augmente graduellement si l'on ajoute de l'oxygène, sans avoir d'effet sur les composants chimiques réactifs.

Des expériences à grande échelle basées sur cette méthode ont récemment été effectuées à BVS dans une galerie d'essais fermée empêchant la libération d'ondes de choc d'air. Ces essais ont démontré que l'on peut établir la sûreté des explosifs autorisés - lors d'essais et autres tirages officiels - par un chiffre donné, obtenu par la concentration d'oxygène, pour une probabilité d'inflammation de 50%. Ce rapport examine les techniques de contrôle des paramètres d'essais nécessaires pour obtenir des résultats que l'on peut reproduire.

En suivant les méthodes décrites ci-dessus, des études supplémentaires sur l'inflammabilité ont été effectuées dans une pièce de $2m^3$ avec augmentation de la pression totale (simulant l'augmentation de la pression de l'air à des profondeurs plus importantes).

ГАЗОБЕЗОПАСНОСТЬ РАЗРЕШЕННЫХ К УПОТРЕБЛЕНИЮ ВЗРЫВЧАТЫХ ВЕЩЕСТВ, ОПРЕДЕЛЯЕМАЯ НА ОСНОВЕ КИСЛОРОДНОГО ИНДЕКСА

Р. Зиммерман и М. Штрайх

Эксперименты, проведенные в 50-е годы лабораторией Горного Бюро США установили, что способность смесей метана и воздуха воспламеняться может быть постепенно увеличена путем добавления кислорода, т.е. не изменяя компонентов химической реакции.

Этот метод был недавно использован в экспериментах, проведенных в крупном масштабе нашей лабораторией в закрытой испытательной штольне, предотвращающей высвобождение взрывных волн. Эти эксперименты показали, что безопасность разрешенных к употреблению взрывчатых веществ может быть определена по концентрации кислорода, необходимой для 50% вероятности воспламенения, как в официальных тестах, так и в других взрывных условиях. Метод проверки экспериментальных параметров, необходимых для получения воспроизводимых результатов, описывается в настоящем докладе.

Были также проведены дополнительные исследования воздействий других факторов на воспламеняемость с использованием того же метода в камере размером 2 кубических метра, в условиях повышенного общего давления (искусственно создавая повышенное глубинное давление воздуха).

许用炸药的沼气安全性 -- 以氧气指数作为试验准则

R. 切墨尔门 及 M. 溪特来许

五十年代时美国矿务局试验室曾由小规模的试验确定了甲烷及空气混合物的燃点可以用增加氧气比例的方法来渐渐提高，而不影响其化学作用的成份。

近年来矿业研究所曾在一个没有震荡波的封闭坑道内用这种方法作了实地试验。实验结果说明许用炸要的安全性 -- 在正式试验或其他情形之下 -- 可以用百分之五十引爆可能性的氧气浓度得到的特征数来测定。本文亦叙述到要得到有重复性的结果时对试验参数应该做的控制。

上述方法亦能在两立方米的容器内增加总压力后（模拟随深度增加的空气压力）进行关于对引爆影响的研究。

REGRESSION ANALYSIS OF SEISMIC EFFECTS OF QUARRY BLASTS

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ABSTRACT

Vibration produced by blasting works, expecially in the environment of surface mining, causes problems. This is why, it is important to determine the exact vibration velocity around quarries.

On the basis of data from measuring performed in the environment Hungarian quarries, the function of charge size, distance and the vibration frequency were determined for the calculation of vibration velocity.

The vibration velocity are determined by a more appropriate expression than the function suggested by Koch accepted internationally.

The calculated regression functions are given for some Hungarian openpits.

INTRODUCTION

Like everywhere else in the world, in Hungary also the breaking of solid rock is done by blasting. In most cases the blasting works are associated with undesirable vibrations. The vibrations caused by the blastings cause problems especially in the environments of open pits. This is because the human body is very sensitive to the vibrations, and qualifies even the very low intensity vibrations to be harmful if they are originated from blastings. Therefore, it is very important that we have to be able to calculate accurately and reliably the expectable vibration speeds in the surroundings of open pits and quarries - and we have to do it before the blastings are done.

The expectable value of vibration speed is usually determined by the Koch-formula in the international practice. On the basis of regression analysis of seismic measurements's data done in the environments of Hungarian quarries we have found a relationship for the calculation of the expectable vibration speed, which method is better than the Koch-formula.

THE REGRESSION ANALYSES OF SEISMIC MEASUREMENTS

As first step of the studies we started from examining the applicability of the Koch-relationship. Is is well known that the Koch-formula can be written by the following expression:

$$V = a\,(\ell/\sqrt{Q})^{-b}, \quad \text{mm/s} \qquad (1)$$

where a - depends on the conditions of the blasting and the geological structure of

the quarry,

ℓ - the distance between the place of blast and that of the object (real estate) to be protected;m,

Q - the mass of blasted charge (in some countries the blasted charge's mass blasted in one fire, in Hungary that of blasted in 100 msec, in other countries that of blasted in one delay stage); kg,

b - exponent, whose value varies between 1,0...1,4.

In practice, on the basis of many measurements the a and b values can be determined by quarries with regression analysis.

We examine the $V_x = f\ (\ell/\sqrt{Q})$ function by regression analysis, in several variations. These were the followings:

- the regression function of vibration speed in the form of $V_x = a\ (\ell/\sqrt{Q_{ö}})^{-b}$, taking into consideration the blasting charge in one fire and the distance;
- the regression function of vibration speed (rate as function of the charge blasted in 100 msec and the distance, in the form of

$$\left[V_x = a\ (\ell/\sqrt{Q_{100}})^{-b}\right];$$

- the regression function of vibration rate as function of the mass for one delay stage's charge and the " " distance, in the form of

$$\left[V = a\ (\ell/\sqrt{Q_f})^{-b}\right].$$

On the basis of measured data we listed the a and b parameters of the regression function as well as the qualitative parameters of correlation calculation in the Table 1.

From the locations of measuring points we came to the conclusion that another relationship also can be found, which is better than the Koch-formula. Our chosen function was the following:

$$V = a\ e^{-b\ (\ell/\sqrt{Q_i})} \quad (2)$$

where

Q_i - $Q_{ö}$, Q_{100} and $Q_{centigrade}$.

The parameters and qualitative characteristics of (2) relationship with the utilization of measurement data of plants indicated in Table 1. can be found in Table 2.

According to the data in Table 1. and 2. it can be seen, that the approach by the exponental function gives a better and more reliable relationship. The value of r correllation index is the highest if we substitute the charge mass for one delay stage in the relationship. Drawing up the $V = f\ (\ell/\ Q_i)$ functions on one curve those curves are going at the top which are determined by considering the charge for one delay stage (Figure 1.) This

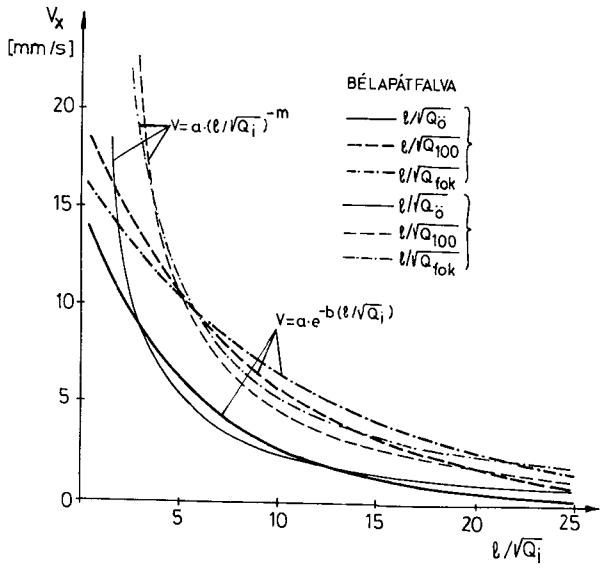


Fig.1.

means that, at a given $\ell/\sqrt{Q_i}$ the highest vibration rate can be expected at the relationship determined by considering $\ell/\sqrt{Q_{centigrade}}$, but this gives the vibration rate with the smallest standard deviation.

From the point of view for forming an opinion about the harmful effects of blasts on the environment, it is also an important question that, how the frequencies of generated vibrations alter the expectable vibration rates. We examined the effects of frequencies by multivariable regression analysis.

THE MULTIVARIABLE VIBRATION RATE FUNCTION AND ITS PARTIAL CORRELATION INDEXES

By the determination of partial correlation index the influences of all data recorded at seismic measurements on the vibration rate can be clarified. That parameter has the biggest influence, at which the partial correlation index value is nearest to 1 (one). We were searching our multivariable function in the form of

$$V = a_1 + b_1 x_1^{c_1} + b_2 x_2^{c_2} + \ldots + b_n x_n^{c_n} \qquad (3)$$

where:
- a, a_1, b_1, $b_2 \ldots b_n$ and c_1, $c_2 \ldots c_n$ are constant, and
- x_1, $x_2, \ldots x_n$ are variables.

For example, for the Beremend plant we got the following relationship for the (3) expression:

$$V = -1{,}17 + \frac{41{,}68\sqrt{Q_{centigrade}}}{\ell} - 0{,}0031\, f^2 + 0{,}036\, f \qquad (4)$$

where f - frequency of the vibration; Hz
r = 0,9730 (total correlation index)
DV = 1,13
DV% = $(\ell/\sqrt{Q_{centigrade}})$ = = 0,9702 (partial correlation index)
r (f) = 0,5500 (partial correlation index)

From the relationship (4) it can be seen that, with f increasing the V is decreasing, which means that, at higher frequencies the charge mass for one delaying stage can be higher, or a relatively large charge can be blasted even in a shorter distance. Also, according to the partial correlation indexes it is clear that, the generated vibration rate is a function of $\ell/\sqrt{Q_{centigrade}}$ and the frequency of the vibration.

THE EFFECT OF THE DISTANCE ON THE VIBRATION RATES

There are both Hungarian and foreign indications regarding that in time to time the seismic effects are increased at a higher distance from the location of the blasting. From the studied plants we examined the relationship between the distance and the vibration rate in Bélapátfalva.

We were searching for the function in the form of:

$$V = a\ell^2 + b\ell + c \qquad (5)$$

The resulting regression function and its qualitative parameters:

$$V = 4.10^{-6}\,\ell^2 - 1{,}36.10^{2}\ell + 11{,}91 \text{ mm/s} \qquad (6)$$

r = 0,9369; DV = 1,16; DV% = 54,13

The measurement points and the regression function are shown on Fig. 2. In the examined plant the vibration rate at ℓ = 1300...

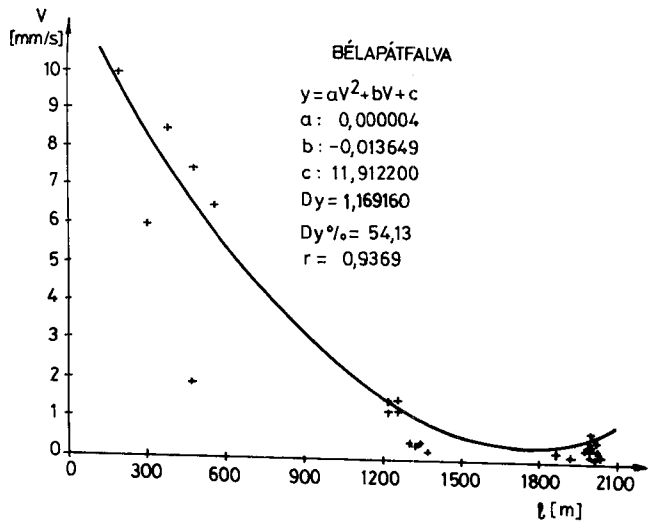


Fig. 2.

...1400 m is the same as in the distance of 2300...2400 m. From Fig.2. it can be seen, of course that, V < 1 mm/s. So what is the explanation of that this low rate can cause complaints sometimes? The reason of this is that the frequency of the generated vibration decreases into the band of natural frequancy of the human body, thus forming resonance situations even at vibrations of low intensity.

SUMMARY

On the basis of the performed examinations we can make the following statemets:

- the expectable seismic rate can be reliably determined by taking into consideration the mass of a charge for one delaying stage,
- the expectable vibration decreases at higher frequencies, therefore higher frequency vibarations have to be generated.
  For high frequency vibrations accurate, high stop number millisecond primers are necessary even for small charges.
- The expectable vibration rate can be estimated more accurately by a simple exponental relationship, which is more reliable than the Koch-formula.
- Under certain conditions the vibration rate can be increased at big distances from the location of blasting, but this is of a very low intensity.

I herewith gratitude to my colleauges Dr. Zoltán Buócz and László Mészáros, the employees of the Department of Mining, for the computerized processing of the measurement data. Part os the measurement data were supplied by the Instrument - and Measurement Office of the Hungarian Academy of Sciences, the Central Mining Development Institute, and the Research Division of Mecsek Coal Mines, for wich I also express my thankfulness towards them.

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Table 1.

Plants						
$\ell/\sqrt{Q_i}$	Parameters	Bélapátfalva	Nagyharsány	Tarcal	Beremend	Keszeg
$\ell/\sqrt{Q_ö}$	a	31,4	85,57	68,04	26,99	33,92
	b	- 1,12	- 1,29	- 1,16	- 1,23	- 0,85
	r	0,9178	0,8143	0,7513	0,9469	0,6757
	DV	1,30	2,58	1,27	1,51	3,62
	DV %	60,28	47,77	42,97	37,35	71,83
$\ell/\sqrt{Q_{100}}$	a	78,47	77,18	43,09	45,44	55,90
	b	- 1,20	- 1,14	- 0,85	- 1,24	- 0,84
	r	0,8897	0,7935	0,6589	0,9575	0,6457
	DV	1,49	2,70	1,45	1,35	3,75
	DV %	69,31	50,08	48,98	33,50	74,41
$\ell/\sqrt{Q_{fok}}$	a	70,15	175,37	139,25	45,44	209,92
	b	- 1,12	- 1,16	- 1,05	- 1,24	- 1,19
	r	0,9121	0,9471	0,9259	0,9575	0,851
	DV	1,34	1,43	0,72	1,35	2,58
	DV %	66,23	26,42	24,60	33,50	51,19

The parameters of regression function determined on the basis of the Koch-formula, and the qualitative indexes

Table 2.

Plants						
$\ell/\sqrt{Q_i}$	Parameters	Bélapát-falva	Nagyharsány	Tarcal	Beremend	Keszeg
$\ell/\sqrt{Q_{\ddot{o}}}$	a	15,07	32,91	11,41	24,15	15,74
	b	- 0,17	- 0,20	- 0,08	- 0,34	- 0,10
	r	0,9465	0,8518	0,7339	0,9449	0,7272
	DV	1,08	2,35	1,29	1,53	3,39
	DV %	49,96	43,54	43,85	37,99	67,39
$\ell/\sqrt{Q_{100}}$	a	19,61	29,68	8,04	25,48	14,39
	b	- 0,12	- 0,16	- 0,04	- 0,24	- 0,05
	r	0,9255	0,8414	0,5571	0,9591	0,6874
	DV	1,26	2,42	1,60	1,32	3,59
	DV %	58,62	44,91	54,08	32,87	71,30
$\ell/\sqrt{Q_{fok}}$	a	16,81	28,47	10,09	25,48	25,72
	b	- 0,09	- 0,08	- 0,03	- 0,24	- 0,07
	r	0,9590	0,9568	0,9338	0,9591	0,8757
	DV	0,95	1,30	0,69	1,32	2,39
	DV %	43,90	24,16	23,30	32,87	47,41

The parameters of regression function determined on the basis of the exponential function, and the qualitative indexes

ANALYSES PAR REGRESSION DES EFFETS SISMIQUES DE TIRAGES EFFECTUES DANS LES CARRIERES

J. Foldesi

Les vibrations produites par les tirages causent des problèmes, particulièrement dans les exploitations en surface. C'est pourquoi il est important de déterminer la vélocité exacte des vibrations au voisinage des carrières.

Sur la base de mesures effectuées dans des carrières hongroises, on a établi le rôle de la taille des charges, de la distance et de la fréquence des vibrations pour calculer la vélocité des vibrations.

Celle-ci a été calculée au moyen d'une formule plus appropriée que la fonction suggérée par Koch, acceptée dans le monde entier.

Nous présentons les calculs des fonctions par régression qui s'appliquent à certaines mines hongroises à ciel ouvert.

REGRESSIONSANALYSE DER SEISMISCHEN EFFEKTE VON SPRENGARBEITEN IN STEINBRÜCHEN.

J. Foldesi

Durch Sprengarbeiten in Tagebauen hervorgerufene Erschütterungen führen zu Problemen. Daher ist es wichtig, die genaue Vibrationsgeschwindigkeit in der Umgebung von Steinbrüchen zu bestimmen.

Auf Grund von Messungen in ungarischen Steinbrüchen wurde der Einfluß der Ladungsgröße, Entfernung und Vibrationsfrequenz zur Errechnung der Ausbreitungsgeschwindigkeit bestimmt.

Die Ausbreitungsgeschwindigkeit wird durch eine zutreffendere Formel bestimmt als die international anerkannte Gleichung von Koch.

Für einige ungarische Tagebaue errechnete Regressionsfunktionen werden vorgestellt.

РЕГРЕССИОННЫЙ АНАЛИЗ СЕЙСМИЧЕСКИХ ПОСЛЕДСТВИЙ ВЗРЫВНЫХ РАБОТ В КАРЬЕРЕ

Ж. Фолдези

Вибрация, являющаяся следствием взрывов, создает ряд проблем для поверхностных горных работ. Вследствие этого, важной задачей является определение скорости вибрации в районе карьеров. На основе измерений, проведенных в венгерских карьерах, была определена роль, которую играют величина заряда, расстояние и частота вибрации в расчетах вибрационной скорости. Скорость вибрации может быть вычислена по более точной формуле, чем повсеместно принятая функция Коха. Даются вычисленные регрессионные функции для некоторых открытых горных карьеров Венгрии.

矿场爆破地震作用的回归分析

J。福尔代锡

由于爆破产生的振动，特别在地面矿场，常会发生很多问题．所以在矿场处的振动速度的检定非常重要．根据在匈牙利矿场所得的测验结果，我们用炸药量，距离及振动频率来计算振动速度．振动速度的计算系用较国际上通用的柯霍函数更为适当的公式．本文给出了匈牙利露天矿场由计算得出的回归函数．

SOME RECENT WORK ON ANALYTICAL, EMPIRICAL AND STATISTICAL MODELLING IN BLASTING RESEARCH AT CMRS, INDIA

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ABSTRACT

In cognizance to the importance as well as necessity of incorporating basic theory based on physical concepts, the Blasting Research Laboratory of CMRS has undertaken some in-depth investigations wherein attempts have been made to evolve some analytical, statistical and computer modelling to deal some common problems involved in rock blasting. The present paper gives an overview idea of the work done so far.

The analytical study has been made to derive six explicit relationships for the prediction of peak particle velocity and peak air pressure which are functions of rock mass properties and blasting parameters. Statistical modelling which has been developed, is based on several statistical methods used to analyse blast vibration data. It incorporates some new concepts also. A modified predictor equation based on wave propagation theory has been developed while studying the predictive ability of all commonly used vibration predictors and their modified forms. In computer modelling, two compact package programmes have been developed in order to enhance data-based information in blast vibration analysis.

Brief discussion is made regarding the validity of the models from field data.

A MODIFIED BLAST INDUCED VIBRATION PREDICTOR BASED ON WAVE PROPAGATION LAW

An efficient blast vibration predictor based on wave propagation law has been established while considering the inelastic damping, i.e. reduction in the amplitude of vibration of the system due to dissipation of energy as proportional to $e^{-\alpha D/Q}$. Where, D is the distance of the measuring transducer from the source, Q is the charge per delay and α is the inelastic attenuation factor.

Inelastic attenuation of elastic waves is a characteristic that could be applied for the study of the geo-technical properties of rocks and to predict the change in the shape of a plane stress wave while passing through a rock mass. In the prediction of ground vibrations induced by blasting, numerous inelastic effects cause energy losses during wave propagation. Major parameters which affect propagation velocity of waves in rocks include rock type, texture, density, porosity, anisotropy, stress, water content, temperature and various geological discontinuities. It is quite unexpected to account all the above parameters in detail in a simple closed form expression. Thus, for better predictability it is required to search for dominant aspects of pulse initiation and propagation. The consideration of inelastic material damping by means of an explicit exponentially decaying function is a dominant factor in the vibration predictor.

Several causes of inelastic attenuation

having different degrees of influence include matrix inelasticity, attenuation due to fluid flow, dissipation in a fluid saturated rock, shearing flow of fluid layer, partial saturation effects, enhanced intercrack flow, stress induced diffusion, energy absorption during phase changes and geometrical effects. Many investigators considered internal friction as a factor causing attenuation and established that it varied directly with the attenuation. Although acoustic velocities and attenuation were determined extensively for metals and alloys, little systematic work was done on rocks. In some cases the attenuation coefficient was found to vary linearly whereas in other, with the second power of frequency.

Ground vibration induced by blasting is a complex mixture of various wave types. It is thus desirable to consider characteristics of geometrical spreading for various types of wave likely to be associated with blasting. In field measurements, the amplitude decay due to geometrical spreading is usually taken as $1/D^B$. Where, D is the distance of the measuring transducer from the source and B is a site constant depending on specific site data. For a homogeneous isotropic linearly elastic medium, the amplitude decay due to geometrical spreading for the dominant wave types may be specified by the following three values of B. They are

i) B = 1 for body waves within a (semi -) infinite medium

ii) B = 0.5 for Rayleigh waves

iii) B = 2 for body waves travelling along a free surface.

In addition to geometrical spreading, numerous inelastic effects cause energy losses during wave propagation through rock mass.

Field measurements show that inelastic attenuation can be better represented by an exponential decay function $e^{-\alpha D/Q}$. Inelastic material damping is thus dependend also on the source of the pulse, conventionally described by the maximum charge weight per delay. More charge would obviously improve the value of attenuation factor due to material damping. The proposed predictor takes the following form :

$$V = K (D/\sqrt{Q})^{-B} \cdot e^{-\alpha D/Q} \quad \ldots \quad \ldots \quad (1)$$

Another representative form of equation (1) is

$$Y = a_1 + b_1 x_1 + b_2 x_2 \quad \ldots \quad \ldots \quad (2)$$

where, $Y = \log V$, $a_1 = \log K$, $b_1 = -B$, $x_1 = \log (D/\sqrt{Q})$, $b_2 = -\alpha$, $x_2 = D/Q$ ... (2a)

Equation (2) is a functional relation of two independent variables. The constants a_1, b_1, b_2 could be obtained by the regression analysis of two independent variables. When a_1, b_1 and b_2 are known, the specific site constants K, B and α can be obtained from equations (2a).

The proposed equation fits the observed data better than the conventional predictors.

The improvement of the proposed method varies from site to site. This is quite reasonable as because α depends on rock at the site. The addition of pulse initiation in the attenuation factor makes the present predictor formula a non-linear in Q on a log-log plot. Its solution for Q at specific distance and vibration limit will be too difficult unless one uses a computer based solution package. For this purpose, a simple computer programme has been developed.

PREDICTION OF PEAK PARTICLE VELOCITY AND PEAK AIR PRESSURE FROM ROCK MASS PROPERTIES AND BLASTING PARAMETERS

When an explosive detonates at a certain depth below the earth's surface approximately 20 to 30 percent of its energy is utilised in fragmenting the surrounding rock and a good amount of energy, which is transmitted through the earth in the form of waves or vibrations, is lost. A part of the energy is also dissipated in the air which produces noise. Approximately 95 percent of the earthborne energy reaches the surface a few feet from the total displacement area and acts as surface waves. The detonation generated seismic waves are of two types-longitudinal and shear waves travelling with their respective velocities. When the seismic waves reach the surface of the earth, they are reflected thereby producing both a horizontal and vertical motion of the earth's surface. The vertical vibration of the earth's

surface produces an equivalent particle velocity in the air. The particle velocity in the air induces air vibration of the same frequency as the seismic waves.

From the theory of reflection of longitudinal waves at a free surface, the vertical particle velocity P_v of the free surface is determined as

$$P_v = P_R \frac{2A^2 \cos^2 2\beta \cos\alpha + 2 \cos 2\beta \sin 2\alpha \sin\alpha}{A^2 \cos^2 2\beta + \sin 2\beta \sin 2\alpha} \quad \text{.....(3)}$$

where,

$$A = \frac{V_P}{V_S} = \frac{\text{longitudinal wave velocity}}{\text{shear wave velocity}}$$

α = angle of incidence of the original longitudinal wave

β = angle of reflection for the reflected shear wave

Similar type of expressions have also been evolved for peak air pressure P_x, which have been generated in a circular ring around ground zero of radius x and width dx. These expressions, have been further simplified and presented in the form of three sets of different formulae for prediction of peak particle velocity and peak air pressure. These equations, which are the functions of dynamic elastic constants of rock and some blasting parameters involved in a detonation, have been derived to estimate PPV and PAP for inside, outside and at the boundary of the crater zone.

The parameters included are (i) longitudinal wave velocity, (ii) shear wave velocity, (iii)density of rock, (iv) characteristic impedance of air, (v) detonation pressure of explosive, (vi) depth of hole, (vii) radius of hole and (viii) distance of the measuring transducer.

(a) Outside the crater zone (x>D)

$$P_v = \frac{4\pi D_p D r_o^3 A^2 l}{\rho V_p x^2 \left[\frac{4D\sqrt{A^2-1}}{(A^2-2)} + (A^2-2)\right]} \quad \text{.....(4)}$$

$$P_L = \frac{4\pi D_p K D r_o^3 A^2 x^{-3} L^{-1} l}{\rho V_p \left[\frac{4D\sqrt{A^2-1}}{(A^2-2)} + (A^2-2)\right]} \quad \text{.....(5)}$$

(b) Within the crater zone (x<D)

$$P_v = \frac{4\pi D_p r_o^3 l}{\rho V_p D^2} \quad \text{.....(6)}$$

$$P_L = \frac{4\pi D_p K r_o^3 x L^{-1} l}{\rho V_p D^2} \quad \text{.....(7)}$$

(c) At the boundary of the crater zone (x=D)

$$P_v = \frac{\sqrt{2}\pi D_p r_o^3 A^2 l}{\rho V_p D^2 \left[(A^2-1) + \frac{\sqrt{(2A^2-1)}}{(A^2-1)}\right]} \quad \text{.....(8)}$$

$$P_L = \frac{\sqrt{2}\pi K D_p r_o^3 A^2 x L^{-1} l}{\rho V_p D^2 \left[(A^2-1) + \frac{\sqrt{(2A^2-1)}}{(A^2-1)}\right]} \quad \text{.....(9)}$$

Where,

P_v = Peak vertical particle velocity of the earth's surface at a distance x from ground zero

P_L = Peak air pressure which arrives at the gage position for each x distance where the air pressure generates

D_p = detonation pressure of explosive

D = depth of hole

r_o = radius of hole

A = ratio of longitudinal wave velocity to the shear wave velocity

l = length of column charge

ρ = density of the rock

V_p = longitudinal velocity of propagation in the rock

K = characteristic impedance of the air

= d.c

d = density of air

c = propagation velocity in the air

L = distance from ground zero to the gage position

The equations (4) to (9) were tested on seven competent and less competent rock types namely dunite, diabase, gabbro, dolomite, granite, limestone and sandstone based on their standard data.

The validation of the proposed formulation from field measurement was also tested by using data collected during controlled blasting experiments at Rajashree Cement Limestone mine in Karnataka and North Koel Hydroelectric Project in Bihar. In Rajashree Cement Project, the blasting was carried out in overburden which consisted of fissured and jointed

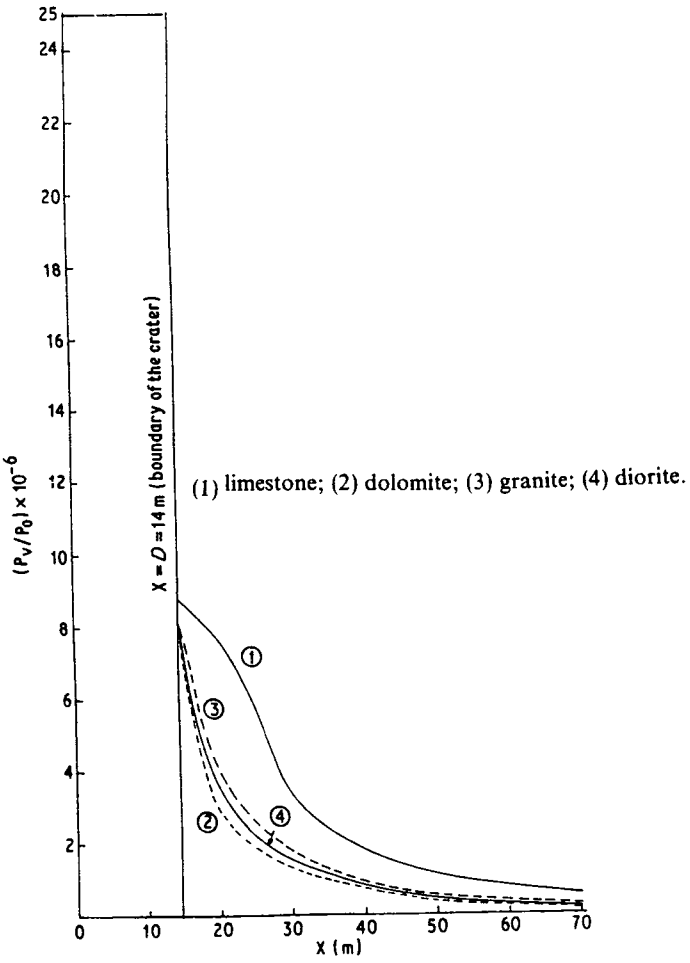


Fig. 1. Relative particle velocity amplitude Vs. distance

limestone whereas in North Koel Project the rock was hard and fresh granite.

For limestone the results obtained from the derived equation (4) were corroborated with those obtained from USBM predictor equation. However, for granite the results were comparable to those of USBM, Langefors-Khilstrom, and general type empirical relationships.

DYNAMIC EFFECTS IN VARIOUS ROCK MASS AND THEIR PREDICTIONS

One of the important dynamic effects due to blasting in rockmass is the ground vibration . Vastly expanded explosive usage in mining and quarrying industries, coupled with the urban sprawl encroaching on quarry and mine operations, has called for much greater control of blast produced ground vibrations and noise to contend with greater public environmental consciousness. The dimensions of the impacts have also enlarged markedly in recent years with the stepping up of mining activity, increased share of open-pit mining and resort to increasingly larger blast sizes for cost-effective rock excavation techniques.

One of the undersirable side effects of rock blasting producing structural damage is due to ground vibrations. It is one of the serious repercussions of blasting and creates great socio-economic problems for mine managements as well as for the people residing in its vicinity. Pre-operational planning is the key to efficiency, economy and environmental safety. Quite often questions are raised about the permissible charge for blasting which would not damage the surrounding structures. Solution to the problem calls for an in-depth understanding of the character of ground and stability of the structure involved.

Dynamic effects in terms of ground vibrations due to blasting are an undesirable side-effect of the use of explosives to fragment rock for mining, excavation, quarring and construction. When an explosive charge is detonated in a blast hole, large volume of gas is released instantaneously at high temperature and pressure producing shock waves and heaving effects during rock breakage. The energy liberated is only partially utilised for breaking the rock and a part of the energy is transmitted to the surrounding rock mass in the form of elastic and or seismic waves. The seismic energy or ground vibration is usually described as a time-varying displacement, velocity or acceleration of a particular point in the ground.

Large scale statistical analysis of data recorded during blasting in various rock mass in situ were conducted. It included limestone (fissured and highly jdinted), sandstone-alluvium, iron ore, basalt (closely jointed), sandstone (weathered), dolomite and granite(hard and fresh, soft and weathered , hard and soft mixed). Nine different forms of predictor equations were considered for analysing and comparing the outcomes. For better understanding of the relationships, all 'simple' and 'partial' regression coefficients as well as 'simple' and partial correlation coefficients were obtained. The following equations were considered :-

USBM predictor (USBM):

$$V = K(D/Q^{\frac{1}{2}})^{-B} \quad \ldots \quad (10)$$

Langefors - Kihlstrom predictor(LFKH):

$$V = K\left(\sqrt{Q/D^{3/2}}\right)^{B} \quad \ldots \quad (11)$$

Ambraseys - Hendron predictor (AMHEN):

$$V = K\,(D/Q^{1/3})^{-B} \quad \ldots \quad (12)$$

Indian Standard predictor (IS):

$$V = K\,(Q/D^{2/3})^{B} \quad \ldots \quad (13)$$

General type predictor (GENRL):

$$V = K\,D^{-\alpha}\,Q^{\beta} \quad \ldots \quad (14)$$

Modified USBM predictor (MUSBM):

$$V = K\,(D/\sqrt{Q})^{-B} \cdot e^{-\alpha D} \quad (15)$$

Modified lang. and Kihls. predictor (MLFKH):

$$V = K\left(\sqrt{Q/D^{3/2}}\right) e^{-\alpha D} \quad (16)$$

Modified Amb. and Hend. predictor (MMHWN):

$$V = K\,(D/Q^{1/3})^{-B} e^{-\alpha D} \quad (17)$$

Modified Ind. standard predictor(MIS):

$$V = K\,(Q/D^{2/3})^{B} e^{-\alpha D} \quad (18)$$

Table - 1: Site constants of USBM predictor equation for different rock mass in-situ.. R_{yx} is the correlation of $y = \log V$ and $x = \log(D/Q^{1/2})$

Rock	K	B	Ryx
Limestone (fissured and highly jointed)	3352.630	1.955	−.9425
Sandstone-alluvium	1483.460	1.980	−.9691
Iron ore	303.736	1.801	−.8723
Basalt (closely jointed)	114.677	1.020	−.7623
Sandstone (weathered)	50.424	0.826	−.7923
Dolomite	173.173	1.508	−.8778
Granite (mixed hard and soft)	470.639	1.557	−.8107
Granite (hard and fresh)	574.240	1.533	−.9161
Granite (soft and weathered)	176.540	1.631	−.8555

Table - 2: Site constants of Langefors - Kihlstrom predictor equation for different rock mass in-situ. R_{yx} is the corrlation coefficient of $y = \log V$ and $x = \log\left(\sqrt{Q/D^{3/2}}\right)$

Rock	K	B	Ryx
Limestone (fissured and highly jointed)	534.568	2.414	.8642
Sandstone-alluvium	310.583	2.525	.9449
Iron ore	30.0095	2.504	.8647
Basalt (closely jointed)	90.6920	1.355	.7758
Sandstone (weathered)	20.0687	1.024	.7960
Dolomite	112.084	1.994	.8744
Granite (mixed hard and soft)	237.457	1.877	.8306
Granite (hard and fresh)	255.804	1.807	.9269
Granite (soft and weathered)	104.896	2.049	.8880

Table - 3 : Site constants of Ambraseys - Hendron predictor equation for different rock mass in-situ . R_{yx} is the correlation coefficient of $y = \log V$ and $x = \log(D/Q^{1/3})$

Rock	K	B	Ryx
Limestone (fissured and highly jointed)	10520.400	1.831	−.9532
Sandstone-alluvium	7243.740	2.032	−.9845
Iron ore	2471.120	1.721	−.8767
Basalt (closely jointed)	141.813	1.016	−.7455
Sandstone (weathered)	144.409	0.886	−.7850
Dolomite	250.918	1.488	−.8717
Granite (mixed hard and soft)	588.746	1.568	−.7451
Granite (hard and fresh)	1265.18	1.691	−.8828
Granite (soft and weathered)	137.545	1.499	−.7588

Table - 4: Site constants of Indian standard predictor for different rock mass in-situ. R_{yx} is the correlation coefficient of $y = \log V$ and $x = \log(Q/D^{2/3})$.

Rock	K	B	Ryx
Limestone (fissured and highly jointed)	534.565	1.810	.8642
Sandstone-alluvium	310.584	1.894	.9449
Iron ore	30.010	1.878	.8647
Basalt (closely jointed)	90.692	1.016	.7758
Sandstone (weathered)	20.069	0.768	.7960
Dolomite	112.083	1.496	.8744
Granite (mixed hard and soft)	237.441	1.407	.8306
Granite (hard and fresh)	255.812	1.355	.9269
Granite (soft and weathered)	104.899	1.537	.8880

Table - 5 : Site constants of general type empirical relationship for different rock mass in-situ.

Rock	K	$\bar{\alpha}$	$\bar{\beta}$
Limestone (fissured and highly jointed)	7891.290	1.897	0.7387
Sandstone-alluvium	35015.600	2.042	0.3252
Iron ore	49587.400	1.588	0.0911
Basalt (closely jointed)	25.847	0.912	1.3641
Sandstone (weathered)	8.333	0.709	0.6033
Dolomite	163.204	1.509	0.7904
Granite (mixed hard and soft)	205.070	1.373	0.9583
Granite (hard and fresh)	186.766	1.284	0.9459
Granite (soft and weathered)	92.238	1.507	1.0463

Table - 6 : 'Simple' and 'Partial' correlation coefficients for different rock mass in-situ. y = log V, z = log D and w = log Q. The variable after the full stop is considered as constant.

Rock	Ryz	Ryw	Rzw	Pyz.w	Pyw.z
Limestone (fissured and highly jointed)	−.8642	−.3207	.70631	−.9511+	.8135
Sandstone-alluvium	−.9855	−.0574	.1531	−.9900+	.5577
Iron-ore	−.8790+	−.6784	.7804	−.7610	.0256
Basalt (closely jointed)	−.7009	.4722	−.1225	−.7351+	.5458
Sandstone (weathered)	−.7508+	.6806	−.6296	−.5663	.4051
Dolomite	−.8340	.0049	.3070	−.8779+	.4970
Grantte (mixed hard and soft)	−.4534	.5392	.2836	−.7508	.7813+
Granite (hard and fresh)	−.6745	.7566	−.1958	−.8210	.8627+
Granite (soft and weathered)	−.4381	.4555	.4943	−.8570	.8600+

Table - 7 : The values of inelastic attenuation factor α as calculated from equations (15) to (18).

Rock	α	Remarks
	8	
Limestone (fissured and highly jointed)	.003330 .004409 .002786 .004409	
Sandstone-alluvium	.010274 .014295 .003046 .014295	As α is positive, the modified curve, in each case, gives a slight downward bending of the best fit line
Iron ore	.000779 .000888 .000672 .000888	
Sandstone (weathered)	.004617 .003723 .005631 .003723	
Granite (hard and fresh)	−.009302 −.000076 −.025756 −.000076	As α is negative, the modified curve, in each case gives a slight upward bending of the best fit line.

The concept of partial correlation coefficient is important when there are more than two variables in the model. For three variables say y = log V, x= log ($D\sqrt{Q}$) and D (distance) in equation (15), it has been observed that the correlation coefficient between x and D i.e. R_{xD} is .881372 which is highly significant. This is of course a consequence of the nature of the process that leads to low values of x, will also tend to give low values of D and vice-versa. Thus, when we calculate the simple regression coefficient of y upon x, since the higher values of x will tend to be associated with higher values of D, the values of y for the larger values of x will tend to be larger than they would have been because they are also associated with the larger values of D.

CONCLUSIONS

1. Propagation equations proposed by Langefors-Kihlstrom(equation 11) and Indian Standard(equation 13) are identical. They also remain identically same when they are modified by introducing an inelastic damping factor, $e^{-\alpha D}$. The modified equations are represented by MLFKH(equation 16) and MIS(equation 18). Explosive charge per delay when calculated from the two equations for various distances and vibration limits were identical. The only difference in the physical appearance of the two equations is in the value of the constant B which is due to the manipulation of the powers of Q/D in the IS formula. It is found that $B_{IS} = 0.75 X B_{LFKH}$ (or $B_{MIS} = 0.75 X B_{MLFKF}$). The site parameter K and inelastic attenuation factor α remain identical for the two equations.

2. The explosive charge per delay when estimated using LFKH was the highest up to a certain distance, for soft to medium hard rock mass including limestone, sandstone-alluvium, iron ore. This limiting distance depended on the vibration limit. At larger distances (say exceeding 150 m) and at higher vibration levels(exceeding approximately 10 mm/sec) the explosive charge per delay for LFKH was the lowest.

3. For moderate to hard rock mass including dolomite and granite the charge per delay was the highest when estimated using AMHEN.

4. Analysis of observations indicated that when explosive charge per delay was estimated using USBM, the blast results were near optimum as compared to results obtained from other equations. The USBM was better applicable for blasting in soft to medium hard rock mass represented by limestone, sandstone, alluvium-sandstone and iron ore.

The blast results, for medium to hard rock mass represented by basalt, dolomite and granite, were near optimum when explosive charge per delay were estimated by using USBM and AMHEN.

5. In limestone, sandstone, alluvium-sandstone and iron ore the permissible charge per delay are low at shorter distances. However, for distances greater than say, 100 m the charge per delay sharply increases. The charges per delay when estimated using AMHEN for distances below 100m, are considerably low. It would be difficult to conduct any meaningful blasting with such low charges and therefore equation (12) is unsuitable for estimation of charges when the structures to be protected are within 100m distance of the blast site.

6. In sandstone and dolomite, the results indicated that the GENRL type of relationship was most acceptable and it showed consistent charge weight per delay.

DEVELOPMENT OF TWO COMPUTER PACKAGE PROGRAMMES FOR ANALYSING BLAST INDUCED VIBRATION DATA

For systematic analysis of blast vibration data, two computer package programmes have been developed, the first of which includes graphics part for diagrams. The inputs of both the programmes are V(measured peak particle velocity), D(distance of the measuring transducer) and Q(maximum charge per delay in a blast). Regression analysis for one and two independent variables are used to develop the programmes. It utilises seventeen built-in subroutine subprogrammes of HP-9000 series computer. The programmes would supply the information which are basically obtained from the USBM, Langefors - Kihlstrom, Ambraseys-Hendron and general type predictor equations. It determines the following:

a) site constants of the predictors

b) simple correlation coefficients(for the determination of the association of variables).

c) partial correlation coefficients(for the determination of partial effects)

d) standard error of estimates(for the accuracy of the regression equation)

e) coefficient of determination(for the overall accuracy of the predictor)

f) adjusted coefficient of determination (for the inclusion of the effect of number of independent variables in the model.)

g) suggested charge per delay for any combination of vibration and distance based on measured equations .

h) graphs of regression lines, plotting of field data and appropriate captions.

The programmes contain three parameters: input parameters, intermediate parameters, and output parameters. Input parameters are -

i) measured peak particle velocity(V)

ii)distance of the measuring transducer(D)

iii)maximum charge per delay (Q).

The intermediate parameters are those which are calculated in the process of determining output parameters. The output parameters are those which are used in the process of analysing and predicting ground vibration and disigning an optimum blast pattern to protect unde-sirable side effects of blasting. The programmes are based on several empirical and statistical relationships. For diagram and appropriate captions associated with the programmes seventeen built-in subroutine subprogrammes have been used.

CASE STUDIES

The programmes were tested on sixteen sets of field results obtained in several opencast and underground mines in

India. As an example, the outputs of both the programmes are shown in the appendix using data from project report EC/37/85, Rajshree Cement Project of karnataka, Limestone Mine. The results are self explanatory.

ACKNOWLEDGEMENT

The opinions, findings and conclusions expressed herein are of the authors and do not necessarily of the Institutes to which the authors belong.

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APPENDIX

Results of vibra.f and gen.f (data from EC/37/85)

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the u.s.b.m predictor formula

b= -.195481E+01

k= .335263E+04

the correlation coefficient= -.942493E+00

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suggested charge per delay in kg.

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	v=20(mm/s)	v=30(mm/s)	v=40(mm/s)	v=50(mm/s)
D(m)= 50.0	.132483E+02	.200596E+02	.269246E+02	.338299E+02
D(m)= 60.0	.190776E+02	.288859E+02	.387715E+02	.487150E+02
D(m)= 70.0	.259667E+02	.393169E+02	.527723E+02	.663066E+02
D(m)= 80.0	.339157E+02	.513526E+02	.689271E+02	.866045E+02
D(m)= 90.0	.429245E+02	.649932E+02	.872359E+02	.109609E+03
D(m)=100.0	.529933E+02	.802385E+02	.107699E+03	.135320E+03
D(m)=110.0	.641218E+02	.970886E+02	.130315E+03	.163737E+03
D(m)=120.0	.763103E+02	.115543E+03	.155086E+03	.194860E+03
D(m)=130.0	.895586E+02	.135603E+03	.182011E+03	.228690E+03
D(m)=140.0	.103867E+03	.157267E+03	.211089E+03	.265226E+03
D(m)=150.0	.119235E+03	.180537E+03	.242322E+03	.304469E+03
D(m)=160.0	.135663E+03	.205411E+03	.275708E+03	.346418E+03
D(m)=170.0	.153151E+03	.231889E+03	.311249E+03	.391073E+03
D(m)=180.0	.171698E+03	.259973E+03	.348943E+03	.438435E+03
D(m)=190.0	.191306E+03	.289661E+03	.388792E+03	.488503E+03
D(m)=200.0	.211973E+03	.320954E+03	.430794E+03	.541278E+03

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the langefors-kihlstrom predictor formula

b= .241363E+01

k= .534564E+03

the correlation coefficient= .864212E+00

---

suggested charge per delay in kg.

---

	v=20(mm/s)	v=30(mm/s)	v=40(mm/s)	v=50(mm/s)
D(m)= 50.0	.232289E+02	.325044E+02	.412544E+02	.496332E+02
D(m)= 60.0	.305352E+02	.427282E+02	.542303E+02	.652445E+02
D(m)= 70.0	.384787E+02	.538436E+02	.683380E+02	.822175E+02
D(m)= 80.0	.470119E+02	.657843E+02	.834930E+02	.100451E+03
D(m)= 90.0	.560967E+02	.784967E+02	.996274E+02	.119862E+03
D(m)=100.0	.657012E+02	.919364E+02	.116685E+03	.140384E+03
D(m)=110.0	.757988E+02	.106066E+03	.134618E+03	.161959E+03
D(m)=120.0	.863665E+02	.120854E+03	.153386E+03	.184539E+03
D(m)=130.0	.973841E+02	.136271E+03	.172954E+03	.208081E+03
D(m)=140.0	.108834E+03	.152293E+03	.193289E+03	.232546E+03
D(m)=150.0	.120701E+03	.168898E+03	.214364E+03	.257902E+03
D(m)=160.0	.132970E+03	.186066E+03	.236154E+03	.284117E+03
D(m)=170.0	.145629E+03	.203780E+03	.258636E+03	.311165E+03
D(m)=180.0	.158665E+03	.222022E+03	.281789E+03	.339021E+03
D(m)=190.0	.172069E+03	.240779E+03	.305594E+03	.367661E+03
D(m)=200.0	.185831E+03	.260035E+03	.330035E+03	.397066E+03

---

the ambraseys-hendron predictor formula

b= -.183095E+01

k= .105203E+05

the correlation coefficient= -.953208E+00

---

suggested charge per delay in kg.

	v=20(mm/s)	v=30(mm/s)	v=40(mm/s)	v=50(mm/s)
D(m)= 50.0	.435094E+01	.845488E+01	.135462E+02	.195256E+02
D(m)= 60.0	.751843E+01	.146100E+02	.234079E+02	.337402E+02
D(m)= 70.0	.119390E+02	.232002E+02	.371708E+02	.535782E+02
D(m)= 80.0	.178215E+02	.346312E+02	.554853E+02	.799768E+02
D(m)= 90.0	.253747E+02	.493088E+02	.790016E+02	.113873E+03
D(m)=100.0	.348076E+02	.676390E+02	.108370E+03	.156205E+03
D(m)=110.0	.463289E+02	.900275E+02	.144240E+03	.207909E+03
D(m)=120.0	.601475E+02	.116880E+03	.187263E+03	.269922E+03
D(m)=130.0	.764722E+02	.148603E+03	.238088E+03	.343182E+03
D(m)=140.0	.955119E+02	.185601E+03	.297367E+03	.428626E+03
D(m)=150.0	.117475E+03	.228282E+03	.365748E+03	.527191E+03
D(m)=160.0	.142572E+03	.277049E+03	.443883E+03	.639815E+03
D(m)=170.0	.171010E+03	.332311E+03	.532421E+03	.767434E+03
D(m)=180.0	.202998E+03	.394471E+03	.632013E+03	.910986E+03
D(m)=190.0	.238745E+03	.463936E+03	.743308E+03	.107141E+04
D(m)=200.0	.278460E+03	.541112E+03	.866958E+03	.124964E+04

the indian standard predictor formula

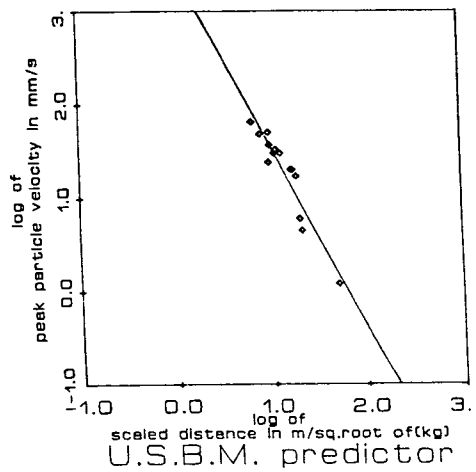
b= .181022E+01

k= .534562E+03

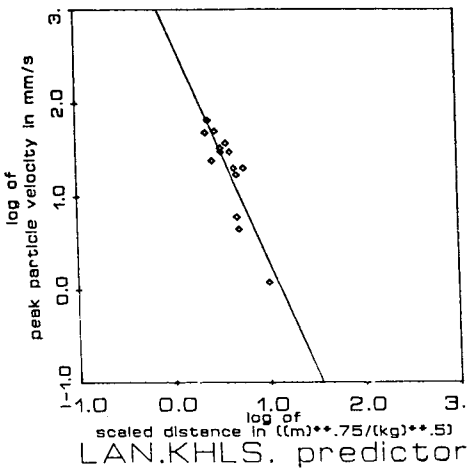
the correlation coefficient= .864211E+00

suggested charge per delay in kg.

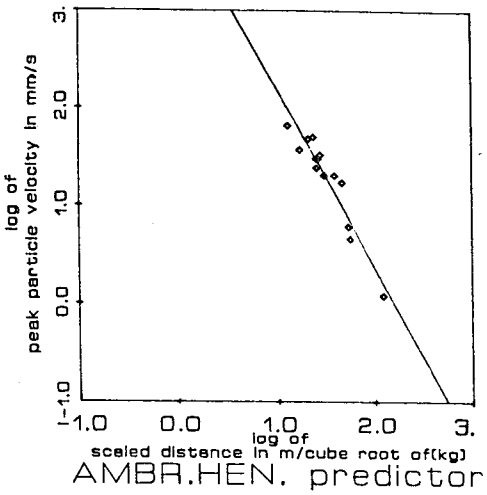
	v=20(mm/s)	v=30(mm/s)	v=40(mm/s)	v=50(mm/s)
D(m)= 50.0	.232289E+02	.325044E+02	.412544E+02	.496332E+02
D(m)= 60.0	.305351E+02	.427282E+02	.542303E+02	.652446E+02
D(m)= 70.0	.384787E+02	.538436E+02	.683380E+02	.822176E+02
D(m)= 80.0	.470119E+02	.657843E+02	.834930E+02	.100451E+03
D(m)= 90.0	.560966E+02	.784967E+02	.996274E+02	.119862E+03
D(m)=100.0	.657012E+02	.919364E+02	.116685E+03	.140384E+03
D(m)=110.0	.757988E+02	.106066E+03	.134618E+03	.161960E+03
D(m)=120.0	.863664E+02	.120854E+03	.153386E+03	.184540E+03
D(m)=130.0	.973841E+02	.136271E+03	.172954E+03	.208081E+03
D(m)=140.0	.108834E+03	.152293E+03	.193289E+03	.232546E+03
D(m)=150.0	.120701E+03	.168898E+03	.214364E+03	.257902E+03
D(m)=160.0	.132970E+03	.186066E+03	.236154E+03	.284117E+03
D(m)=170.0	.145629E+03	.203780E+03	.258636E+03	.311165E+03
D(m)=180.0	.158665E+03	.222022E+03	.281789E+03	.339021E+03
D(m)=190.0	.172069E+03	.240779E+03	.305595E+03	.367661E+03
D(m)=200.0	.185831E+03	.260035E+03	.330035E+03	.397066E+03



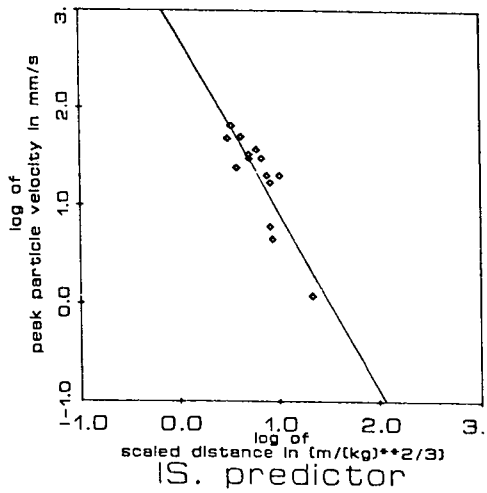
U.S.B.M. predictor



LAN.KHLS. predictor



AMBR.HEN. predictor



IS. predictor

----------GENERAL TYPE EMPIRICAL EQUATION----

v=k(D**b1)(Q**b2)

b1= -.189650E+01 b2= .738705E+00 k= .789032E+04

coeff. of determination= .914394E+00

standard error of estimate= .151214E+00

adjusted coeff. of determination= .898830E+00

Simple Corr. Coeff. Of log(D)&log(Q)= .706316E+00

Simple Corr. Coeff. Of log(V)&log(D)= -.864239E+00

Simple Corr. Coeff. Of log(V)&log(Q)= -.320717E+00

Partial Corr. Coeff. Of log(v)&log(D)taking log(Q) as constant= -.951095E+00

Partial Corr. Coeff. Of log(V)&log(Q)taking log(D) as constant= .813490E+00

F-TEST:F= .306875E+02

Suggested charge per delay in kg.

	V=20(mm/s)	V=30(mm/s)	V=40(mm/s)	V=50(mm/s)
D(m)= 50.0	.703838E+01	.121857E+02	.179880E+02	.243317E+02
D(m)= 60.0	.112398E+02	.194597E+02	.287255E+02	.388559E+02
D(m)= 70.0	.166967E+02	.289075E+02	.426719E+02	.577206E+02
D(m)= 80.0	.235243E+02	.407281E+02	.601211E+02	.813234E+02
D(m)= 90.0	.318304E+02	.551087E+02	.813490E+02	.110038E+03
D(m)=100.0	.417173E+02	.722262E+02	.106617E+03	.144217E+03
D(m)=110.0	.532825E+02	.922493E+02	.136174E+03	.184198E+03
D(m)=120.0	.666194E+02	.115340E+03	.170259E+03	.230303E+03
D(m)=130.0	.818176E+02	.141653E+03	.209101E+03	.282843E+03
D(m)=140.0	.989635E+02	.171338E+03	.252921E+03	.342117E+03
D(m)=150.0	.118141E+03	.204540E+03	.301933E+03	.408414E+03
D(m)=160.0	.139431E+03	.241400E+03	.356345E+03	.482013E+03
D(m)=170.0	.162913E+03	.282055E+03	.416356E+03	.563189E+03
D(m)=180.0	.188662E+03	.326636E+03	.482165E+03	.652206E+03
D(m)=190.0	.216755E+03	.375273E+03	.553961E+03	.749322E+03
D(m)=200.0	.247263E+03	.428093E+03	.631932E+03	.854789E+03

TRAVAUX RECENTS SUR DES MODÈLES ANALYTIQUES, EMPIRIQUES ET STATISTIQUES EFFECTUES DANS LE CADRE DE LA RECHERCHE SUR LES TIRAGES AU CMRS, EN INDE.

Dr. P. Pal Roy, Dr. B. Singh

Connaissant l'importance ainsi que la nécessité d'instaurer une théorie de base fondée sur des concepts physiques, le Laboratoire de Recherches sur les Tirages du CMRS a entrepris des études approfondies tentant de développer des modèles analytiques, statistiques et informatisés; ces modèles permettront de traiter certains problèmes courants rencontrés lors des tirages de roches. Le présent rapport donne une idée générale des travaux effectués jusqu'à ce jour.

L'étude analytique a permis de conclure à six rapports explicites entre le calcul de la vélocité de pointe des particules et de la pression de pointe de l'air et les propriétés de masse des roches et les paramètres des tirages. Le modèle statistique qui a été développé sur la base de plusieurs méthodes statistiques d'analyse des données de vibration des tirages, comprend également plusieurs concepts nouveaux. Une équation de calcul modifié, basée sur la théorie de la propagation des ondes, a été développée au cours de l'étude des capacités de calcul de tous les calculs de vibrations et de leurs formes modifiées utilisés habituellement. Trois programmes complets et compacts ont été développés dans les modèles informatisés pour traiter les données des analyses de vibrations des tirages.

La validité des modèles fabriqués à partir des mesures des gisements fait l'objet d'un bref examen.

NEUE ARBEITEN AN ANALYTISCHEN, EMPIRISCHEN UND STATISTISCHEN MODELLEN AM SPRENGFORSCHUNGSZENTRUM CMRS, INDIEN.

Dr. P. Pal Roy, Dr. B. Singh

In Kenntnis der Wichtigkeit sowie der Notwendigkeit der Einarbeitung einer auf physikalische Konzepte gegründeten Basistheorie, wurden im Spreng-Forschungslabor des CMRS einige tiefgreifende Untersuchungen durchgeführt. Dabei wurde versucht, analytische und statistische Computermodelle zur Lösung einiger allgemeinen Probleme des Gesteinsprengens anzuwenden. Der Vortrag gibt einen Überblick über die bisher geleistete Arbeit.

Eine analytische Studie hatte den Zweck, sechs eindeutige Zusammenhänge zur Vorhersage von maximalen Partikelgeschwindigkeiten und maximalen Luftdrücken abzuleiten, die Funktionen der Gesteinseigenschafen und der Sprengparameter sind. Das statistische Modell, welches entwickelt wurde, basiert auf verschiedenen statistischen Methoden, die zur Analyse der Daten der Spreng-Vibration dienen. Auch schließt es einige neue Konzepte ein. Eine abgewandelte Vorhersageformel, basierend auf der Theorie der Wellen-Fortpflanzung, wurde entwickelt. Dabei wurde die Vorhersagefähigkeit aller allgemein gebräuchlichen Vibrationsvorhersagemethoden und ihrer abgewandelten Formen untersucht. Im Computermodeling wurden drei kompakte Programme entwickelt, um die rechnergestützten Informationen für die Sprengvibrations-Analyse zu vergrößern.

Der Vortrag schließt mit einer kurzen Diskussion über die Gültigkeit der auf Meßergebnissen basierenden Modelle.

АНАЛИТИЧЕСКИЕ, ЭМПИРИЧЕСКИЕ И СТАТИСТИЧЕСКИЕ МОДЕЛИ ВЗРЫВНЫХ РАБОТ, НЕДАВНО РАЗРАБОТАННЫЕ НА ЦЕНТРАЛЬНОЙ ГОРНОЙ ИССЛЕЛОВАТЕЛЬСКОЙ СТАНЦИИ В ИНДИИ

Доктор П. Пал Рой и доктор Б. Сингх

Сознавая важность и необходимость использования теории, основанной на концепциях физики, лаборатория исследования взрывных работ предприняла серьезные попытки разработать аналитические, статистические и количественные модели, предназначенные для разрешения общих проблем ведения взрывных работ. Доклад представляет собой обзор выполненной работы.

В результате аналитического изучения проблемы было определено шесть взаимосвязей для предсказания максимальной скорости частиц и максимального давления воздуха, которые являются функциями от свойств горной массы и параметров взрывов. Статистическое моделирование основано на нескольких статистических методах, используемых в анализе данных взрывной вибрации. Оно также включает несколько новых концепций. Модифицированное уравнение прогноза, основанное на теории распространения волн, было выведено в процессе изучения прогнозной способности широко используемых вибрационных предсказателей и различных их вариантов. Три компактных программы для улучшения качества информации, полученной из данных анализа взрывной вибрации, были написаны как часть вычислительного моделирования.

Доклад заканчивается краткой дискуссией обоснованности моделей основанных на полевых измерениях.

印度中央矿业研究所（CMRS）在爆破方面的分析，试验及统计模型的最新研究结果

P． 派罗埃博士，B． 辛格 博士

认清了用物理观念作为基础理论的重要性和必要性后，已经对爆破作过深刻探讨的印度中央矿业研究所爆破研究室试图对岩石爆破过程中常见的问题建立一些分析，试验和统计的模型．本文将综述这方面研究工作至今所取得的成果．

我们用解析法推导出了六个显式函数式用来预测颗粒最大速度与空气最大压力，这些都是岩石体性质和爆破参数的函数．所建立的统计模型是以几个用来分析爆破震动数据的统计方法为基础的，同时也用了一些新概念．研究过所有常用的震动预测方程及其修正式的预测能力后，我们用震动波传播的理论 建立了一个修正的预测方程．在计算模拟方面，我们编了三套小型计算机软件用来增强数据库化的爆破震动数据的分析．

本文简要地讨论了根据现场测量结果对这些模型可靠性的评价．

LIGHTNING WARNING FOR EXPLOSIVES HANDLERS

L. D. Santis
Bureau of Mines, Pittsburgh Research Center, USA

ABSTRACT

This paper gives an overview of the methods that can provide warning of impending lightning to explosives handlers. The methods discussed range from the simplest, most inexpensive to the state-of-the-art. Warning that lightning will enter a sensitive area can be accomplished by visual or public techniques and instrumented techniques. Visual or public techniques include weather patterns, weather forecasts, and weather observations. Instrumented techniques include AM radio methods or spheric detectors, electric field measurements, interferometry, and network systems.

INTRODUCTION

If lightning strikes an explosive or explosive device, detonation is very probable, regardless of the precautions taken. Even a near miss could cause a detonation. Many studies have shown that a lightning strike can initiate standard electric blasting caps (2 amp all-fire) many kilometers away. Only a direct strike will initiate nonelectric initiation systems.

A number of industrial activities are sensitive to an unexpected lightning discharge. Mining operations and any other endeavors that involve explosive materials are particularly vulnerable. Because of the volatility of the materials, devastating financial or personnel losses could occur in the event of a lightning strike. A system that provides warning of the approach of a thunderstorm is essential to the safety and efficiency of such operations.

Lightning is a threat to any surface or underground operation using volatile materials. Although surface operations are more vulnerable, electric systems and spark-sensitive materials underground at any depth are susceptible to lightning. If lightning strikes conductors leading underground, like the headframe, the energy imparted into the grounding system is too much to handle. The current travels (possibly kilometers) underground, bleeding itself off and arcing along the way. Even if lightning strikes the ground above, dangerous currents can travel nearly 900 m deep in mountainous terrain and over 150 m deep in flat terrain (Berger 1977).

The unexpected detonation of explosives is just the beginning of the problem. Only part of the loaded shot detonates in 90% of all prematures caused by lightning. This creates not only safety problems but prolonged downtime and extensive wear and tear on the machinery that digs the poorly broken material. A typical surface coal mine would lose nearly $200,000 if lightning prematurely detonated a shot (Santis 1988).

There are two ways to reduce the vulnerability of an operation. The first is the implementation of a lightning warning system (LWS), and the second is the use of an initiation system that is less sensitive to lightning. The discussion in this paper is limited to lightning warning methods and procedures.

Thunderstorms develop in two ways. The first is the frontal type storm caused by a cold air mass overtaking a warm air mass. The second is the convective type storm caused by the solar heating of the Earth's surface or the right combination of wind direction and topographical features as shown in Figure 1 (Banta 1987).

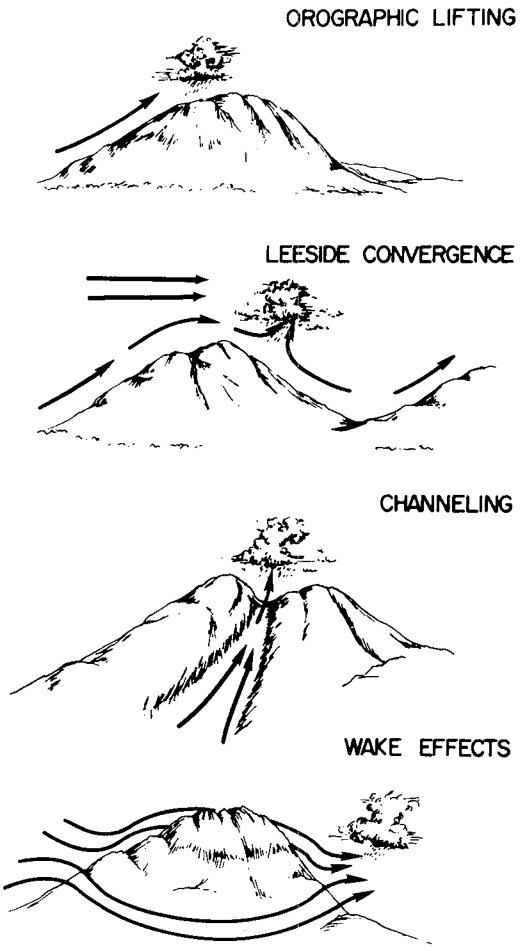


Figure 1.

In both cases, the cloud becomes electrified by the movement of water particles carried by the rising air.

In the past, lightning warning techniques offered only moderate improvements in safety and efficiency. Recently the ability to warn if lightning will enter a particular area has improved. It is relatively easy to predict where an active thunderstorm will travel during its lifetime. The problem of lightning warning, however, is a very difficult one because of the random behavior of lightning and site-dependent variables such as weather patterns, degree of vulnerability, warning time requirements, and schedule flexibility. It is particularly difficult to predict when and where the first strike will occur. The first strike is the very first cloud-to-ground discharge from a thundercloud.

Lightning behaves in such a haphazard manner that no single device or even array of devices will provide valid warning 100% of the time. Thus, the problem cannot be attacked simply by picking a system. Rather, it involves a program centered around lightning awareness.

Any effective lightning warning system will be a combination of three things. The first of these, visual or public techniques, involve making casual observations and using public information such as National Weather Service broadcasts and statistical data on local weather patterns. Secondly, spheric detectors, interferometers, and networks track active thunderstorms and allow for short-to-medium range localized forecasting. Finally, atmospheric electrostatic field (E-field) monitoring provides the only verified way to warn of a thundercloud building overhead.

VISUAL OR PUBLIC TECHNIQUES

Weather Patterns

On a global scale, lightning behaves haphazardly, but on a local scale, it tends to move into an area in a habitual fashion. It is very important to understand the tendencies of lightning in your location. Without this knowledge, solving your lightning problem would be like buying a set of rock drills without knowing what kind of rock or what size hole you needed to drill.

Determination of your local weather patterns may be obtained by analyzing the following--

1. Number of discharges per square kilometer per year. Operations in areas of high flash densities should incorporate the best LWS, whereas operations in areas of low flash densities may opt for a less expensive LWS. MacGorman, 1984 provides a good reference for the United States (U.S.).

2. Prevailing storm direction.

3. Type of storm, frontal or convective, that predominates.

4. Seasonal variations in lightning activity. Although lightning can occur at any time of year, summer months are the worst. Changery, 1981 provides a good reference for the U.S.

5. Time of day variations in lightning activity. Usually the hours of noon to 5:00 p.m. are the worst.

6. Any local topographical features that may be thunderstorm genesis zones.

If possible, schedule blasting activities at times of low lightning probability. U.S. Mine Safety and Health Administration (MSHA) records show that only 7% of lightning-caused premature initiations of explosives in mining operations occurred during the morning hours.

To supplement this information and gain a more individualized perspective, routinely record observations on days that lightning occurs, and look for patterns. Record such information as the date and time, the kind of storm, the direction it came from, how long it lasted, the weather conditions, how the warning (if any) was received, and if any strikes occurred nearby. This approach may also identify local thunderstorm genesis zones by examining the wind direction when convective storms develop.

Weather Forecasts

Weather forecasts have improved greatly recently. In fact, a 3-day forecast now is as reliable as a 1-day forecast 20 years ago. Still, weather forecasts alone are not reliable as a lightning warning technique. Like the variables mentioned above, they only indicate when trouble can be expected. A person charged with blasting activities should actually become an amateur meteorologist. Learn how to read a weather map and look for a cold front pushing a warm front (frontal-type storms). Convective type storms are much more difficult to predict but occur almost every day in some areas (Florida and mountainous areas) during certain times of the year.

If possible, subscribe to the Weather Channel or a local radar station through either cable, satellite, or closed circuit broadcasts to obtain up-to-the-minute visual information on storm activity. Level 3 or above on a radar plan position indicator usually means electrical activity is in progress or imminent. If neither of these broadcasts can be received, call the local National Weather Service (NWS) weather information service to receive updated information before sensitive activities begin. Also, the local NWS broadcasts 24-hour weather reports in the U.S. on a frequency between 162.40 and 162.55 MHz. Some radios can pick up this frequency but it can also be picked up with a special radio sold in department stores for about $15.

Observations

By making routine observations, you will have a better idea of what to expect, and will learn to recognize when blasting activities should stop. Prediction of thunderstorms for an area does not always mean that they will arrive. Even if

thunderstorms do arrive, it is possible that the vulnerable area is situated such that the storm activity will skirt around it. Conversely, thunderstorms may occur when they were not predicted.

Monitor the sky in the direction where storms usually develop or where they were predicted and look for cumulonimbus cloud formations, particularly anvil-type clouds, and darkening of the sky. Lightning begins to form when the cloud tops of summer convective storms reach about 8.5 km (Moore 1981). Winter thunderstorm tops develop at much lower altitudes and are harder to predict.

Keep alert for indicators that can give warning that a lightning discharge may occur nearby, like--

1. Illumination of distant clouds caused by lightning discharges.

2. Thunder. Thunder can be heard up to 13 km from the source (Pierce 1977). The distance to the strike in kilometers is equal to the time in seconds between visual observation and thunder arrival divided by three.

3. Rain or hail. A cloud giving as little as 3 mm/hour precipitation can produce lightning (Moore 1977).

4. Relative humidity and temperature. Moist air aids in the development of thunderstorms, and it is unlikely that lightning would form below certain limits (Pierce 1977) (see figure 2).

5. A sudden drop in barometric pressure or temperature, or a change in wind direction, especially a 180° change. These changes often indicate the approach of a cold front.

6. Luminescence around high or pointed objects (St. Elmo's fire).

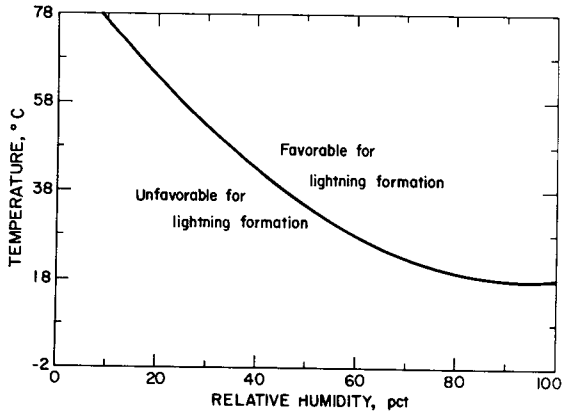


Figure 2.

Using the above-mentioned tactics with AM radio methods will provide a very inexpensive way to warn of potential lightning hazards. However, this approach has substantial drawbacks, such as:

1. The warning criteria are subjective.

2. Possible insufficient warning time to evacuate sensitive area.

3. A nearby lightning discharge may occur without any of the previously mentioned indicators being noticed.

INSTRUMENTED TECHNIQUES

AM Radio

The use of an ordinary AM radio is one of the simplest methods of detecting lightning. Most people are familiar with lightning's distinctive static crackling noises that disturb reception during thunderstorms. Every lightning discharge creates strong atmospherics (radio waves) throughout the AM radio band. By tuning the receiver to an unused or clear frequency in the lower end of the band, and listening for these static bursts, you can determine if lightning is occurring within 30 to 160 km, depending on the sensitivity of the receiver.

Lightning comes from an electrically active cell within a thundercloud. As a cell builds in intensity, lightning occurs more regularly. As the cell decays, so do the spheric bursts. Through the life of a cell, usually 1/2 to 1 hour, lightning flashes 2 to 3 times per minute. This rate varies greatly but usually peaks at about 10 discharges per minute. By listening to the radio and counting the number of bursts in 1 minute, every 5 to 10 minutes, one can get a feel for whether a storm is building or decaying. This approach is subject to large errors because two or more cells may be active at one time within listening range and there is no way to distinguish among them.

The loudness of the bursts give some idea of how close the activity is. If the static on an AM radio gets increasingly louder, the storm is probably moving closer. Again, this is subject to large errors owing to the overlap of other cells and the variability of lightning itself. The strength of the spheric bursts is a function of the distance to and the strength of the discharge. The current in a cloud-to-ground lightning bolt could peak anywhere between 1 and 200 kiloamperes; thus, a very intense strike far away would sound the same as a very weak close strike.

The advantages of using the AM radio method of lightning warning are that it is inexpensive, no training is required, and no special equipment is needed (most operations have an AM radio on site already).

There are significant disadvantages since the method--

1. Gives no indication of storm direction.

2. Gives poor indication of storm distance.

3. Has very subjective warning criteria.

4. Cannot filter out static that may sound like lightning but actually could be caused by electrical equipment, powerlines, dust storms, corona discharge, a faulty radio, or other sources.

5. Cannot operate automatically.

The main problem with the use of an AM radio as a lightning warning device is that it is very subjective. As with any subjective method, the operator must decide if conditions warrant a stoppage of operations. Underestimation leads to safety concessions; overestimation leads to unnecessary downtime. However, it is better than nothing, and when used with visual or public methods, it offers a cautious operator greatly improved safety.

Spheric Detectors

Spheric detectors also monitor radio waves to detect a discharge. These devices are merely AM radios with added circuitry to eliminate the human factor and unwanted signals. Some of the newer detectors are reasonably reliable as they use statistical analysis methods to monitor the growth and decay of a thunderstorm. Most devices have range settings and predetermined criteria for alarm. They work well in areas where only frontal-type storms develop (Johnson 1979).

Interferometry

This method involves the differentiation of the radio wave spectrum emitted from a thunderstorm. Various activities in a thunderstorm produce different frequencies. By monitoring, differentiating, and relating these signals, the devices determine the approximate distance to and the direction of the thunderstorm. One system incorporates radar precipitation images to determine if a cloud formation is electrically active. The French developed a system, SAFIR, that actually maps discharges in two or three

dimensions using three remote stations. These devices are very useful but are also very expensive.

Networks

The development of lightning detection networks (see figure 3) has made the biggest breakthrough in the area of lightning warning. Networks provide the best real-time, real-location detection of individual lightning strokes (Johnson 1979). Presently two methods are used: magnetic direction finding (MDF), and time of arrival (TOA) location finding.

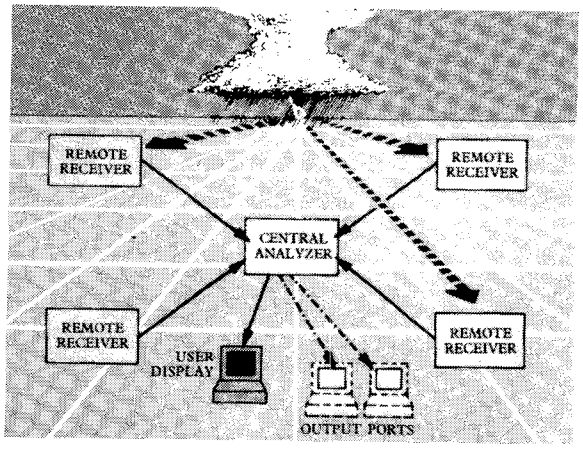


Figure 3.

The MDF network employs an array of two crossed loop antennas to determine an azimuth to the discharge. A central computer then triangulates between two or more remote stations to determine the location of the discharge. The network records and stores almost every (70%-90%) flash within the network's range. A lightning display system (LDS) can receive real-time or stored information on lightning activity. The network covers the contiguous United States, although detection efficiency is poor in the northern Midwest. The network can determine the location (within a few km), polarity, multiplicity, and peak current of a particular strike. It can compute flash rates and flash densities for a particular area. Flashes can be color coded chronologically. This allows easy determination of the direction the storm is moving and its velocity. Figure 4 is a typical display from the network. Dots are negative discharges, and plus signs are positive discharges. Although not very noticeable in black and white, six color codes in 1-hour increments provide a chronological record of each event.

The other network, TOA, uses an array of simple whip antennas to record the real-time (within microseconds) that the very low frequency (VLF) wave peak from a discharge arrives. Knowing how fast the spherics travel, the central computer uses hyperbolic intersection between two or more remote stations to find the location of the flash. Output from the TOA network is similar to that of the MDF network.

The TOA network offers a service called the Weather Sentinel Service (WSS). For a moderate monthly fee, the network computer will monitor a user-defined area within the network's range for lightning flashes. As soon as it senses a discharge in the defined area, the WSS computer notifies the subscriber via numeric or digital display, microprinter, or pager. Audible alarms notify the user of new messages. Perhaps the best definition of the warning area is a bullseye-like set of range circles at 25, 50, and 100 km radius. This would tell if a storm is moving closer to or away from the sensitive area.

The WSS also provides general weather information every morning that could be useful, such as predicted high and low temperatures, relative humidities and dew points, maximum possible rainfall, % chance of precipitation, % chance of severe weather, % of possible sunshine, and atmospheric inversion.

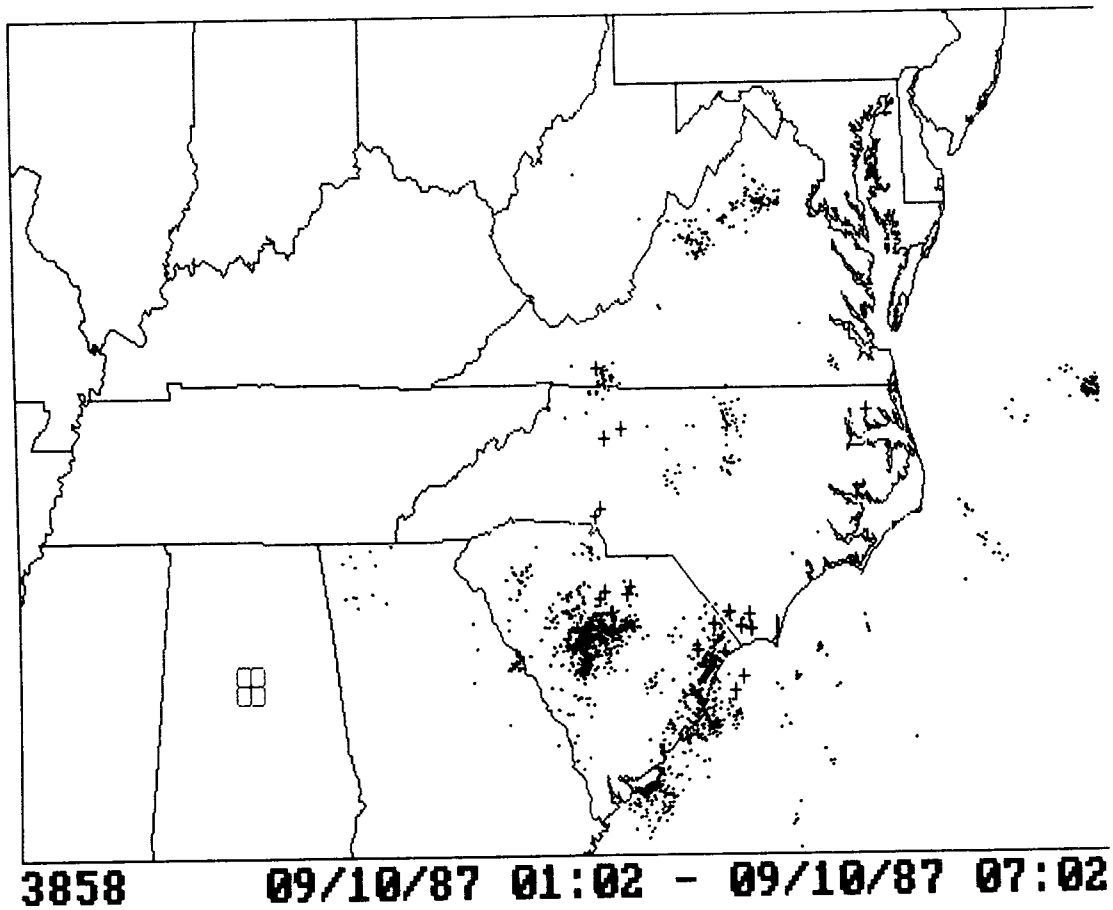


Figure 4.

Figure 5 shows the network's coverage area.

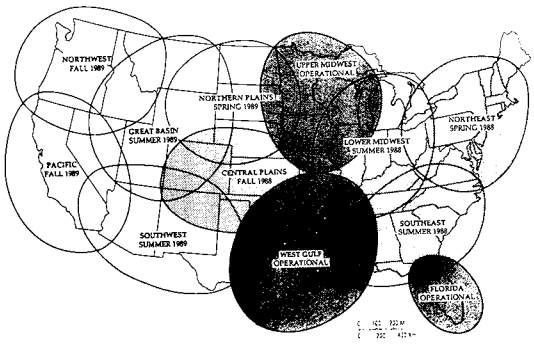


Figure 5.

A third network has been proposed. NASA is investigating the possibility of using satellites to detect the illumination of cloud tops caused by lightning. This project is still in the development stages. It does not appear to offer any significant advantages over the existing networks other than detecting intercloud discharges. These discharges occur up to six minutes before the first cloud-to-ground strike. This could give very short warning of the first strike.

Networks are in operation in other countries such as the United Kingdom, Japan, Brazil, Sweden, and the Netherlands using the MDF or TOA principle.

Atmospheric Electrostatic Field

This method monitors the electric (E) field between the cloud and the ground. The thundercloud environment is essentially a huge capacitor. The cloud acts as one plate, the earth acts as the oppositely charged other plate, and the lightning bolt acts as the discharge path.

By monitoring the electric field at the ground surface, a probability that a discharge will occur in the vicinity can be reached. Generally, fair weather fields are around +100 volts per meter, and lightning will form when the field reaches -2 to -5 kilovolts per meter.

This method provides the only verified way to warn of the first strike of a thundercloud building overhead. This quality is especially important in areas where convective type storms develop. However, operations should not rely upon electric field measurements as the sole method of warning because of the influence of space charges; also they may only allow minutes to take any action.

When an electrically active cloud moves in, oppositely charged ions flow from grounded objects into the air causing a masking or screening layer of space charge near the ground surface. This attenuates the magnitude of the E-field measured at ground level, which can lead to erroneous information on the location and magnitude of the charged area aloft. Even under fair weather conditions, there are many processes that introduce space charge into the atmosphere, including blowing snow or dust, splashing water, engine exhausts, industrial emissions, smoke, fog, and high-voltage powerlines. Wind can transport these space charges many kilometers.

Regardless, a particular site could arrive at criteria to stop sensitive operations and evacuate personnel when--

1. The E-field reverses polarity and/or steadily builds.

2. The E-field crosses the site-dependent threshold.
   a) Start with two kilovolts per meter.
   b) Through experience, determine if this value is suitable.
   c) Adjust threshold accordingly.

3. Jagged oscillations are noticed in the field. See figure 6. Notice how the E-field from the thunderstorm has almost straight up and down oscillations. These sudden collapses of the E-field are caused by lightning discharges.

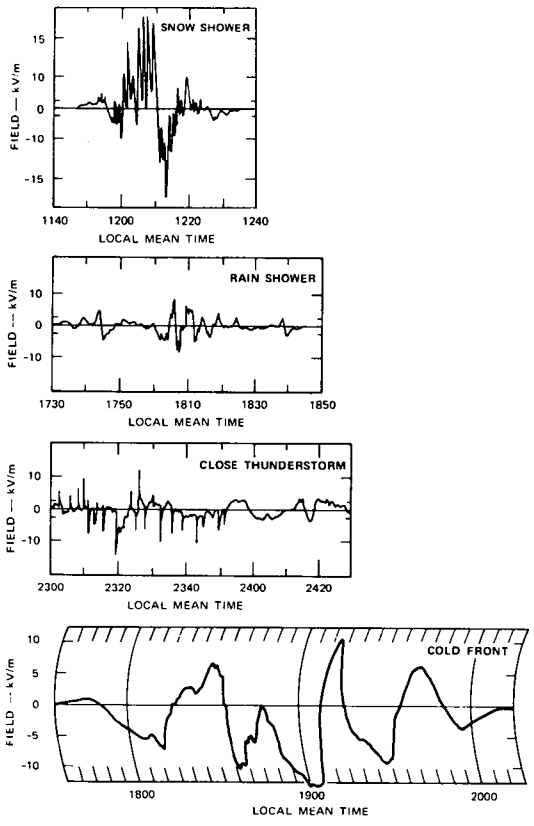


Figure 6.

EVACUATION PROCEDURES AND CRITERIA

Under certain conditions, lightning could set off electric blasting caps many kilometers away. From the existing literature on thunderstorm development (Cianos 1974), where D is equal to the range sensitivity of the operation to lightning in km, the following recommendations can be made for safe distances from known electrical activity at surface operations. When E-field measurements are available, the safe distance is D plus 8 km. When E-field measurements are not available, the safe distance is D plus 40 km. For operations where nonelectric initiation is used, D is 0 because a direct strike is necessary. For operations where standard electric blasting caps (2 amp all-fire) are used, D is 32 km in mountainous terrain and 8 km in flat terrain (Johnson 1979). Mountainous terrain is more hazardous because the strata typically have high resistivity values, allowing dangerous currents to travel farther.

Underground operations or those using special initiators need to determine their own unique D. Since underground operations would generally require a strike directly above, an acceptable D is 8 km in mountainous terrain and 0 in flat terrain. Nonelectric initiation systems underground are considered to be immune to lightning. However, in theory, any spark sensitive material underground could be initiated by lightning under certain conditions.

As soon as a hazard is present, immediately remove personnel from the vulnerable area. Employ an alarm system to notify the workers as soon as possible. Do not try to pick up explosives already unloaded. Resume operations only if 30 min has passed and there was no indication of lightning activity.

CONCLUSIONS

Because of the unpredictable behavior of lightning, a lightning warning system that provides valid warning 100% of the time has not been and probably will not be developed. However, an operation can obtain substantial improvements in safety and efficiency by using some form of LWS. The following conclusions can be drawn:

1. Base the decision as to what type of lightning warning system to install on the following criteria.
   a) Degree of protection desired.
   b) Warning time necessary.
   c) Type of storms (frontal or convective) that predominate.
   d) Affordable cost. In general, the effectiveness of a lightning warning system is proportional to its cost.

2. Any effective LWS will be a combination of--
   a) Short-to-medium range forecasting to avoid times of high lightning probability.
   b) Immediate hazard detection and prompt evacuation of the sensitive area.

3. The most effective instrumentation seems to be a combination of devices based on spheric detection and electric field measurement.

4. Network systems provide the most reliable spheric detection method.

5. Cost reductions could be made by using other spheric methods instead of networks at the expense of effectiveness.

6. E-field measurements provide the only verified way to warn of the first strike. This quality is especially important in areas where convective storms develop.

7. E-field measurements alone are not recommended as a LWS. Accompany them with a spheric method.

8. Always use visual or public methods.

LIST OF MANUFACTURERS

Spheric Detectors

Atmospheric Research Systems, Inc.
2350 Commerce Park Drive, N.E.
Suite 3
Palm Bay, FL 32905
(303) 725-8001

Nuclear Instruments Corp.
2345 W. Mill Road
Milwaukee, WI 53209
(414) 228-8800

Safety Devices, Inc.
7910-A Hill Park Ct.
Lorton, VA 22079
(703) 550-9899 or
(703) 339-6650

Thomas Instruments, Inc.
Rte. 9, P.O. Box 50
Spofford, NH 03462
(603) 363-4500

Signal Design, Inc.
3 Autry
Irvine, CA 92718
(714) 581-2870

Interferometry

Aviation Safety Systems/3M
6530 Singletree Dr.
Columbus, OH 43229
(614) 885-3310

Lightning Location and Protection, Inc.
1001 South Euclid Ave.
Tucson, AZ 85719
(602) 624-9967

ONERA
29 Avenue D'La Division Le'Clerk
92320 Chatillion, France

Sperry Aerospace Group
P.O. Box 21111
Phoenix, AZ 85036-1111
(602) 867-2311

E-field

3M/Static and EMC
P.O. Box 2963
Bldg. 590
Austin, TX 78769-2963
(512) 834-1800

Airborne Research Associates, Inc.
46 Kendal Common Road
Weston, MA 02193
(617) 899-1834

Atmospheric Research Systems, Inc.
2350 Commerce Park Drive, N.E.
Suite 3
Palm Bay, FL 32905
(303) 725-8001

Electroforces, Inc.
P.O. Box 523772
Miami, FL 33152
(305) 594-0304

Electronique 2000
8, rue Rere Champhine
38600 Fontaine, France
(76) 26 53 27

Environmental Sensing Technology
1054 Hawthorne Ave., East
St. Paul, MN 55106
(612) 776-9668

Interstate Electrostatics Corp.
8627 Guthrie Rd.
Box 216
Calhan, CO 80808
(719) 683-2419

Langmuir Labs
New Mexico Institute of Technology
Socorro, NM 87801
(505) 835-5503

Lightning Eliminators Consultants, Inc.
13007 Lakeland Rd.
Santa Fe Springs, CA 90670
(213) 946-6886

Mission Instruments Co.
5937 East Pima St.
Tucson, AZ 85712-9242
(602) 721-9242

Monroe Electronics, Inc.
100 Housel Ave.
Lyndonville, NY 14098
(716) 765-2254

Qualimetrics, Inc.
P.O. Box 41039
Sacramento, CA 95841
(916) 923-0055

Thunderstorm Technology
1911 Main Avenue
Suite 236
Durango, CO 81301
(303) 259-1800

MDF Network

Bureau of Land Management
3905 Vista Ave.
Boise, ID 83705
(208) 334-9880

National Severe Storms Laboratory
1313 Halley Circle
Norman, OK 73069
(405) 366-0405

State University of New York at Albany
Dept. of Atmospheric Science, ES216
1400 Washington, Ave.
Albany, NY 12222
(518) 442-4555

TOA Network and WSS

Atmospheric Research Systems, Inc.
2350 Commerce Park Drive, N.E.
Suite 3
Palm Bay, FL 32905
(303) 725-8001

R*Scan Corp.
Minnesota Supercomputer Center
1220 Washington Ave., South
Minneapolis, MN 55415-1258
(612) 333-1424

The preceding list of manufacturers does not claim to be complete. Nor is it an endorsement of a particular company or product. In addition, Blasters Tool and Supply Co., of Frankfort, KY, (800) 634-6250, and Ideal Supply of Asheville, NC, (800) 533-0144, market LWSs.

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Moore, C. B., and B. Vonnegut. The Thundercloud. Ch. in Lightning, ed. by R. H. Golde, Academic Press, v. 1, 1977, pp. 51-92.

Pierce, E. T. Lightning Warning and Avoidance. Ch. in Lightning, ed. by R. H. Golde, Academic Press, v. 2, 1977, pp. 497-518.

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ALERTES A LA FOUDRE POUR LES MANIPULATEURS D'EXPLOSIFS

Lon D. Santis

Ce rapport présente une vue générale des méthodes d'alerte à la foudre destinées aux manipulateurs d'explosifs. Elles vont des plus simples et des moins coûteuses aux méthodes de pointe. L'alerte de la pénétration imminente d'une zone sensible par la foudre est donnée par des systèmes visuels, publics ou instrumentaux. Les systèmes visuels ou publics comprennent les courbes du temps, les prévisions et les observations météorologiques. Les systèmes instrumentaux comprennent les méthodes par radio AM ou par détecteurs sphériques, les mesures des champs électriques, la différenciation des ondes ou l'interférométrie, ainsi que les systèmes de réseaux. Mention est également faite des procédures et des critères d'évacuation.

BLITZWARNUNG BEI SPRENGARBEITEN.

Lon D. Santis

Dieser Vortrag gibt einen Überblick über die Methoden, die zur Warnung vor drohenden Blitzschlägen bei Sprengarbeiten dienen können. Die beschriebenen Methoden umfassen die einfachsten und billigsten Anlagen, sowie solche auf dem höchsten Stand der Technik. Die Warnung vor einem Blitzschlag in einem gefährdeten Gebiet kann durch visuelle oder allgemeine Techniken, oder durch Instrumenttechnik gegeben werden. Unter visuellen oder allgemeinen Techniken versteht man die Beobachtung des Wettergeschehens, Wettervorhersagen und Wetterzyklen. Die Instrumententechnik umfaßt die AM - Radio Methode oder sphärische Detektoren, die Messung des elektrischen Feldes, Wellenverschiebung oder Interferometrie und Nachrichtensysteme. Methoden und Kriterien zur Evakuation werden erwähnt.

ПРЕДУПРЕЖДЕНИЕ ВЗРЫВНИКОВ О ПРИВЛИЖАЮЩИХСЯ ГРОЗАХ

Лон Д. Сантис

Настоящий доклад предлагает обзор методов, которые предостерегают взрывников о приближающихся грозах. Обсуждаемые методы варьируют от простейших и дешевых до основанных на последнем слове техники. Предостережения о приближающихся грозах могут быть даны общедоступными видимыми сигналами или с помощью приборов. Первые включают наблюдения за погодой и прогнозы погоды. Инструментальные методы включают использование радио и сферических детекторов, измерения электрического поля, распознавание волн или интерферометрию, и другие системы связи. Описываются также критерии и порядок эвакуации.

爆破工作者对闪电应警惕的事项

郎 D. 山铁士

本文概括了爆破工作者能预知临近的闪电的方法．涉及的方法包括从最简单，最廉价到最先进的．对闪电将进入敏感区域的预报可用目测法，通俗的办法和仪器测量法．目测法和通俗的办法包括气象观率，气象预报，和气侯观察． 仪器测量法包括调幅无线电法或球形集电器，电场测量法，波微分和波干涉测量法，以及网路系统法．文中提到了疏散步骤和规则．

MSHA'S NEW REGULATIONS FOR EXPLOSIVES USED IN COAL MINES

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Mine Safety and Health Administration, Approval and Certification Center, USA

ABSTRACT

The Mine Safety and Health Administration (MSHA) has made an extensive revision of its existing blasting regulations and approval standards for explosives used in underground coal mines. A proposed set of revised regulations were developed and provided to mine operators, union, trade associations, manufacturers and other interested parties for comment. Final rules were subsequently prepared and published in the U.S. FEDERAL REGISTER on November 18, 1988. A summary of the final rules is presented and discussed. The requirements for "qualified shotfirer" are specified. Requirements on the use of explosives and blasting equipment, means of transporting explosives and blasting procedures are addressed. New rules that apply to approval of explosives and sheathed explosive units are also discussed. In addition, data on accidents involving explosives are summarized.

INTRODUCTION

The Mine Safety and Health Administration (MSHA) has revised its existing safety standards for blasting and approval of explosives used in underground coal mines. The revisions update existing requirements consistent with current technology. Unnecessary and duplicative standards are eliminated and alternative methods of compliance are provided where possible. The final rule on Safety Standards for Explosives and Blasting, Part 75, and Requirements for Approval of Explosives and Sheathed Explosive Units, Part 15, were published in the Federal Register on November 18, 1988. The effective date of these final rules was January 17, 1989.

The use of explosives in underground coal mines presents a potential risk for serious injuries and death to miners. A major hazard associated with the use of explosives is the accidental ignition of methane and coal dust. In the United States, from 1982 through 1987, there were 10 fatalities and 126 injuries as a result of blasting accidents in underground coal mines (Coal Accident and Injury Abstracts). Table 1 shows the accident data for each of these years. The major causes of these accidents include premature blasts, blast area security, blasting fumes, and misfires.

Table 1: Underground coal mine explosive accidents, 1982-1987

Year	Fatalities	Nonfatalities
1982	3	19
1983	0	18
1984	1	16
1985	6	32
1986	0	18
1987	0	23

The new regulations provide increased safety protection for miners by including requirements to address identified hazards, by including more performance oriented requirements, and by reflecting advances in technology and modern blasting practices. The final rule, Part 15, for approval of explosives updates the existing regulations to reflect advances in technology that have led to the development of new types of explosives. The new regulations continue to require the use of explosives approved as permissible. Permissible explosives reduce the potential for igniting methane gas or coal dust that may be present where

blasting is performed.

BLASTING REQUIREMENTS, PART 75

The new regulations address the need for knowledgeable and experienced personnel to perform blasting functions. Only qualified persons or persons working in the presence of and under the direction of a qualified person may use explosives in underground coal mines. The final rule provides two alternatives for achieving qualification. A person certified to use explosives by the State in which the mine is located may be considered a "qualified person" if the State requires a demonstration of ability to safely use permissible explosives with respect to the new regulations. For States that do not certify persons to use explosives, a person may be considered a "qualified person" if he or she has at least one year of experience working underground on a coal-producing section of a mine where explosives are used. The person must demonstrate to an MSHA authorized representative the ability to use explosives safely.

The new regulations permit only permissible explosives, approved sheathed explosive units and permissible blasting units to be taken or used in underground coal mines. Black blasting powder, aluminum-cased detonators and detonators with aluminum leg wires and safety fuses are prohibited. Explosives, sheathed explosive units, and blasting units must meet certain design and performance requirements to be approved for use in underground coal mines.

The rules for transporting explosives and detonators underground require separate, substantially constructed containers made of nonconductive material. Transporting explosives and detonators in separate containers provides protection against accidental detonation of the explosives by the detonators. Nonconductive containers protect explosives and detonators from stray electric current during transportation.

The rules require that each container of explosives and detonators be marked with warnings to identify its contents. Trips using trolley locomotives to transport explosives and detonators must be separated from all other mantrips by at least a five-minute interval. The cars containing explosives or detonators on these trips must be separated from the locomotive by at least one empty car or car containing noncombustible materials.

Explosives must be separated from detonators by at least 50 feet when transported on conveyor belts. Conveyor belts must be stopped before explosives or detonators are loaded or unloaded.

Underground Magazines

The quantity of explosives kept underground must not be more than needed for 48 hours of use. Explosives and detonators taken underground shall be kept in separate closed magazines at least five feet apart or separated by a hardwood or laminated partition. The magazines must be at least 25 feet from roadways and any source of electric current. Magazine locations must be posted with warnings to indicate the presence of explosives.

Explosive Materials Outside of Magazines

The amount of explosives taken outside of a magazine for use in blasting must not exceed 100 pounds or must not exceed the amount needed to blast one round when more than 100 pounds is required.

Sheathed Explosive Units

The revised blasting regulations for underground coal mines permit the use of approved sheathed explosive units. The sheathed explosive unit, which was developed by the Bureau of Mines, U.S. Department of the Interior, can be fired outside of a borehole without the hazard of igniting gas or dust that can be in the mine atmosphere. Historically, the firing of unconfined shots has been prohibited in underground coal mines. However, there are situations where a safety advantage can be gained by using the sheathed explosive unit. Some situations are dislodging loose roof slabs or rock overhangs, boulder breaking and leveling of roof falls.

The sheathed explosive unit is designed to form a flame-prohibiting cloud when the unit is fired. This cloud will prevent flame caused by firing the sheathed explosive from igniting explosive gas or dust mixtures.

No more than three sheathed explosive units are permitted to be fired at one time. A separate instantaneous detonator must be used to fire each sheathed explosive unit. It is not permissible to fire sheathed explosive units in contact with each other. The sheathed explosive unit must be approved by MSHA under the new approval requirements included in Part 15. The major aspects of these approval requirements are discussed in a following section.

Boreholes for Explosives

With the exception of sheathed explosive units and shots fired in anthracite mines for battery starting or for blasting coal overhangs, all explosives fired underground must be confined in boreholes. Each borehole in coal for explosives must be at least 24 inches from any other borehole and from any free face, unless the thickness of the coal seam prohibits such spacing. Each borehole in rock must be at least 18 inches from any other borehole in rock, at least 24 inches from any other borehole in coal and at least 18 inches from any free face.

Blasting Preparation

Each borehole must be cleared and its depth and direction determined before loading with explosives. When priming explosives, a nonsparking punch must be used. Detonators must be inserted completely in and parallel to the length of the explosive cartridge; the leg wires must be secured around the explosive. The cartridge containing the detonator must be the first cartridge loaded in the borehole. Explosives of different brands and cartridges of different diameters are not permitted to be loaded in the same borehole. Tamping poles used for loading and tamping boreholes must be nonconductive and nonsparking.

Except when blasting solid rock in its natural deposit, not more than three pounds of explosive are permitted in any borehole in bituminous and lignite mines. Only noncombustible material is permitted for stemming boreholes. Only pliable clay dummies are permitted as stemming when blasting off the solid in bituminous and lignite mines. Water stemming bags can be used for boreholes in cut coal. However, water stemming bags must be constructed of tear-resistant and flame-resistant material.

Multiple-Shot Blasting

Not more than 20 boreholes are permitted to be fired in a round unless permission has been obtained from MSHA. Instantaneous detonators in combination with delay detonators are not permitted in the same blasting circuit. Only detonators with delay periods of 1,000 milliseconds (ms) or less are permitted in bituminous and lignite mines.

When blasting cut coal in bituminous and lignite mines, the first shot or shots fired in a round must be initiated in the row nearest the kerf, or the row or rows nearest the shear. After the first shot or shots, the interval between the delay periods of successive shots must be at least 50 ms but not more than 100 ms.

When blasting coal off the solid in bituminous and lignite mines, each shot in the round must be initiated in sequence from the opener hole or holes. After the first shot or shots, the interval between the delay periods of successive shots must be at least 50 ms, but not more than 100 ms.

Blasting Circuits

Blasting circuits must be protected from sources of stray electric current. Detonator leg wires must be shunted until connection is made into the blasting circuit. Blasting cables must be well-insulated, copper wire of a diameter not smaller than 18-gauge. Wire used between the blasting cable and detonator circuitry must be well insulated with a resistance no greater than 20-gauge copper wire and not exceed 30 feet in length. When 20 or fewer boreholes are fired in a round, the blasting circuit must be wired in a single

series. Blasting cables must be hunted until immediately before firing, except when testing for circuit continuity. Immediately prior to firing, all blasting circuits must be tested for continuity and resistance using a blasting galvanometer or other instrument designed for testing blasting circuits.

Determination for Methane

No shot is permitted to be fired in an area that contains 1.0 volume percent or more of methane. Immediately before shots are fired, the methane concentration must be determined by a person qualified to test for methane.

Examination After Blasting

After blasting, the blasting area must be examined for misfires, methane and other hazardous conditions. If a round has partially detonated, the qualified person must immediately leave the area and no person may reenter that area for at least five minutes. When misfires occur, only work by a qualified person shall be performed in the affected area. Misfires that are not found and disposed of properly can be accidentally detonated during loading or other operations. Because of varying characteristics of explosives, a particular procedure for handling misfires is not specified in the regulations. However, the regulations permit a qualified person and mine management to address these situations on a case-by-case basis and limit the exposure of persons to the hazard of accidental detonation.

APPROVAL OF EXPLOSIVES, PART 15

The final rule updates the existing approval regulations to reflect state-of-the-art technology and deletes obsolete provisions. The final rule contains general approval procedures, technical requirements for the testing and evaluations of explosives, and new requirements for the approval of sheathed explosive units.

The revised regulations for Part 15 were effective January 17, 1989. Explosives issued an approval under regulations in place prior to January 17, 1989, and in compliance with those regulations, can continue to be manufactured and marked as approved as long as no change is made to the explosive.

One revision in the approval requirements for explosives used in underground coal mines is the introduction of a modified gallery test. In this test, coal dust is predispersed prior to firing of an explosive charge into the gallery. This test provides for evaluation of explosives under conditions that are more representative of actual conditions in underground mines. Another change revises the existing requirements for toxic gases that are produced by detonation of explosives and a new method for calculating the equivalent volume of various gases relative to carbon monoxide is established. This method is based on threshold limit values published by the American Conference of Governmental Hygienists.

The test for air-gap sensitivity has been changed from a half-cartridge to a 7-inch cartridge method. Another modification includes marking the wrapper of each cartridge and each case of approved explosives with the test detonator strength and the minimum product firing temperature. A "test detonator" is an instantaneous detonator having a strength equivalent to a detonator with a base charge of 0.40-0.45 grams PETN. A test detonator is used to test explosives submitted for approval under the new requirements of Part 15. In addition, the lowest product temperature at which an explosive is approved for use under the newly revised Part 15 is defined as the "minimum product firing temperature."

Test Requirements for Explosives

Test requirements for approval of explosives consist of a rate-of-detonation test, an air-gap sensitivity test, a gallery test using methane, a gallery test using methane and bituminous coal dust, a pendulum-friction test, and a test for toxic gases.

The rate-of-detonation test is conducted on an unconfined 50-inch column of explosive. The test is intended to determine whether an

explosive has a tendency to misfire or partially detonate.

The air-gap sensitivity test is modified from a half-cartridge method to a 7-inch cartridge method. In this method, two 8-inch cartridges are used. One inch is cut off the end of each cartridge. The cartridges are placed in a paper tube with the cut ends facing each other with an air gap between them. The explosive must propagate across a 3-inch air gap when the test is conducted at a temperature between 68 and 86 degrees Fahrenheit. The explosive must also propagate across a 2-inch gap when the test is conducted at the minimum product firing temperature specified or 41 degrees Fahrenheit, whichever is lower.

Two gallery tests are conducted on an explosive to determine the incendivity when fired into a flammable atmosphere. One test is conducted by firing explosives from a steel cannon into a long gallery containing approximately 8 percent of natural gas. Another gallery test is conducted by firing explosives into a mixture of bituminous coal dust which is predispersed into a 4 percent natural gas atmosphere.

The pendulum friction test is conducted on a small sample of explosive. The explosive is placed on an anvil and the pendulum is adjusted and released from a height of 59 inches. The test demonstrates whether the explosive is unduly sensitive to impact and friction.

The test for toxic gases is made by firing a 1-pound explosive charge from a steel cannon into a large chamber. The gaseous products resulting from detonation of the explosive are collected and analyzed for toxic gases. The equivalent volume of each toxic gas produced, relative to carbon monoxide, is determined by multiplying the measured volume of the gas by a conversion factor. The conversion factor is equal to the threshold limit value, time weighted average in parts-per-million for carbon monoxide divided by the threshold limit value, time weighted average for the toxic gas. The total volume equivalent to carbon monoxide of the toxic gases produced by detonation of the explosive is the sum of the equivalent volumes of the individual toxic gases.

TEST REQUIREMENTS FOR SHEATHED EXPLOSIVE UNITS

The test requirements for the approval of sheathed explosive units are new. Each sheathed explosive unit must not contain more than 1-1/2 pounds of an approved or permissible explosive. This quantity of explosive is the maximum amount used in the prototype units designed and tested by the Bureau of Mines and found to be adequate for all situations where sheathed explosives are likely to be used. A drop test is conducted on sheathed explosive units to determine whether the contents of the units are susceptible to shifting or damage under conditions comparable to those likely to be encountered in underground mines.

Four gallery tests are conducted on sheathed explosive units to determine their incendivity in mixtures in methane and coal dust. In one gallery test, three units are placed two feet apart on a concrete slab in an explosive methane-air atmosphere. In this test, an evaluation under conditions simulating the blasting of fallen rock in an underground mine is made. A second gallery test using the same arrangement of sheathed explosive units is conducted in an explosive methane-air atmosphere in which bituminous coal dust has been predispersed. A third gallery test is conducted with three sheathed explosive units arranged in a triangular pattern in a simulated crevice formed by two concrete slabs. This test is made in an explosive methane-air atmosphere to simulate conditions of blasting an overhanging roof slab. A fourth gallery test is conducted with three sheathed explosive units arranged in a triangular pattern in a corner formed by three steel plates. This test is made in an explosive methane-air atmosphere to simulate conditions of blasting along a rib where compression and reflection of the explosive shock wave are produced. A detonation test is also made on sheathed explosive units when fired at the minimum product firing temperature established for the explosive used in the unit, or at

41 degrees Fahrenheit for units with previously approved explosives. The test evaluates the effect of temperature on the sensitivity of the explosive and the performance of the flame-inhibiting material in the unit.

SUMMARY

In the final rules, MSHA has reorganized, updated, and clarified existing regulatory provisions for the use and approval of explosives. MSHA believes that the clarification of these rules will facilitate better understanding and improve application of safe blasting practices in underground coal mines.

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NOUVELLES REGLEMENTATIONS DE LA MSHA CONCERNANT LES EXPLOSIFS UTILISES DANS LES MINES DE CHARBON

Harry C. Verakis

L'Administration de la Sécurité et de la Santé Minières (MSHA) a effectué une étude approfondie des conditions requises de tirages et des normes d'approbation des explosifs utilisés dans les mines de charbon souterraines; ces normes sont mentionnées dans les chapitres 75 et 15, Paragraphe A, du Code des Réglementations Fédérales des Etats-Unis. Les règlements proposés avaient déjà été annoncés dans le Registre Fédéral, Chapitre 75, Normes de Sécurité des Explosifs et des Tirages, ainsi que dans le Chapitre 15, Conditions Requises d'Approbations des Explosifs et des Eléments Explosifs sous Gaines, afin d'obtenir la réaction du public. Des audiences publiques ont également eu lieu concernant les règlements proposés. On a examiné et pris en considération les commentaires donnés sur ces règlements et ceux obtenus au cours des audiences publiques, pour la préparation des règlements finaux. Ce rapport présente et approfondit certains aspects majeurs des nouveaux règlements. Il précise les conditions d'utilisation des explosifs et des tirages par des personnes qualifiées. Il examine les types d'explosifs et d'équipements de tirages autorisés dans les mines de charbon souterraines, leurs conditions de transport et d'entreposage, ainsi que les procédures de tirages multiples.

Il examine également les réglementations des essais et des approbations des explosifs autorisés, ainsi que les nouvelles conditions concernant les explosifs sous gaines destinés à être utilisés hors des trous de forage.

Enfin, pour illustrer les incidents provoqués par les explosions,il présente un bref sommaire des données sur les accidents ayant eu lieu dans les mines de charbon.

NEUE MSHA VORSCHRIFTEN FÜR DEN GEBRAUCH VON SPRENGSTOFFEN IN KOHLEGRUBEN.

Harry C. Verakis,

Die Mine Safety and Health Administration (MSHA) hat eine ausführliche Überprüfung der Anforderungen und Zulassungsbestimmungen für Sprengstoffe, die für den Untertageeinsatz in Kohlegruben bestimmt sind, durchgeführt. Diese Bestimmungen sind in Teilen 75 und 15, Titel 30 des USA Code of Federal Regulations beschrieben. Die vorgeschlagenen Regeln wurden zuvor im Federal Register in Teil 75, und Sicherheitsstandards für Sprengstoffe und Spreng-Arbeiten unter Teil 15, Anforderungen für die Zulassung von Sprengstoffen und Sprengladungen, der Öffentlichkeit zur Meinungsäußerung vorgelegt. Öffentliche Anhörungen für die vorgeschlagenen Regeln wurden ebenfalls abgehalten. Alle Kommentare wurden in Betracht gezogen und bei der Ausarbeitung der endgültigen Regeln berücksichtigt. Hier werden die Hauptaspekte der Regeln vorgestellt und diskutiert. Die Vorschriften für Personen, die Sprengstoffe handhaben oder Sprengungen durchführen, werden in Einzelheit aufgeführt. Sprengstoffarten und Sprengausrüstungen, die für Untertageinsatz zugelassen sind, werden vorgestellt. Weiterhin werden Vorschriften für den Transport und die Lagerung von Sprengstoffen, sowie die Durchführung von Mehrschußzündungen beschrieben.

Ergänzende Regeln, die sich auf das Testen und die Zulassung von genehmigten Sprengstoffen beziehen, sowie neue Vorschriften für Spreng-Ladungen, die nicht in einem Bohrloch verwendet werden, werden erläutert.

Zur Illustration von Grubenunfällen im Zusammenhang mit der Benutzung von Sprengstoffen wird eine kurze Zusammenfassung der entsprechenden Daten gegeben.

НОВЫЕ ПРАВИЛА ДЛЯ ИСПОЛЬЗОВАНИЯ ВЗРЫВЧАТЫХ ВЕЩЕСТВ В УГОЛЬНЫХ ШАХТАХ, ПРИНЯТЫЕ УПРАВЛЕНИЕМ ПО БЕЗОПАСНОСТИ И ОХРАНЕ ЗДОРОВЬЯ НА ШАХТАХ

Харри К. Веракис

Требования и стандарты по отношению к взрывчатым веществам, используемым в подземных угольных шахтах, содержащиеся в 75-ой и 15-ой частях 30-ой статьи Кодекса правительственных постановлений США, были пересмотрены Управлением по безопасности и охране здоровья на шахтах (MSHA). Предложенные правила были предварительно опубликованы для всеобщего ознакомления и обсуждения в Ведомостях правительства США в части 75, "Нормы безопасности для взрывчатых веществ и взрывов", и в части 15, "Условия принятия взрывчатых веществ, включая взрывчатые вещества в оболочке инертных материалов". Предложенные правилы также обсуждались на слушаниях, открытых для публики. Замечания по исправлениям предложенных правил, полученные в процессе открытых обсуждений, были учтены при подготовке окончательного текста. В настоящем докладе обсуждаются основные аспекты новых правил, анализируются требования к квалифицированному персоналу, обсуждаются типы взрывчатых веществ и оборудование, разрешенное к использованию в подземных угольных шахтах. Также описываются требования по отношению к транспортировке и хранению взрывчатых веществ и порядок проведения групповых взрывов.

Обсуждаются требования, применимые к испытаниям и последующему разрешению к производству взрывчатых веществ и новые требования к взрывчатым веществам в оболочке инертных материалов, которые могут быть использованы не только в буровых скважинах.

В заключение, чтобы проиллюстрировать вызванные взрывчатыми веществами инциденты, дается краткое обобщение данных о несчастных случаях в угольных шахтах.

矿山安全和健康管理局（MSHA）对煤矿中使用的炸药的新规定

海瑞 C．弗来克士

矿山安全和健康管理局（MSHA）对美国联邦法规准则中第30条款第75项和第15项中的关于用于地下煤矿的炸药和爆破要求与审核标准进行了广泛的复议．第75项为以前公布在联邦登记册上所提出的对炸药和爆破标准的规则，以便吸取公众的意见；第15项则为炸药和筒装炸药包的审核要求．这些条款中也包括了公开听证会上所听取的公众意见．为了使这些法规最后定形，复议中重新审核了原稿的内容及公开听证会上所听取的公众意见．本文将叙述和讨论这些法规的最终形式的几个主要方面．这里述明了一个合格的炸药使用和爆破执行人员所必须具备的条件，也讨论了地下煤矿允许使用的炸药种类和爆破器材．文中还说明了炸药的运输与储存，及多药包爆破的步骤的规则．

本文还叙述了用于许用炸药的试验与审核的附属规定，以及不须填入炮眼中的筒装炸药包的使用的新规定．

此外，本文简要总结了煤矿中的失事记录，以说明因使用炸药引起的事故．

Session X

EXPLOSIONS, I

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REVIEW PAPER: PREVENTION AND SUPPRESSION OF COAL MINE EXPLOSIONS

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ABSTRACT

There has been a notable decline in the frequency and severity of mine explosions since the turn of the century. Among the major safety measures responsible for this decline are the use of rock dusting, development of permissible explosives and electrical equipment, improved ventilation and improved methods for detecting hazardous conditions. While these advances in mine safety are noteworthy, an examination of recent statistics indicate that the problem of mine explosion prevention is not completely solved. There is also growing concern over the number of frictional methane ignitions that occur annually; the frequency of methane ignitions has not as yet seriously impacted fatal accident statistics, but their potential for disastrous consequences is well known.

This paper reviews recent research aimed at explosion prevention. Particular emphasis is given to the present practice of rock dusting and the implications of recent experimental evidence on the overall effectiveness of rock dust standards. Developments in supplemental explosion protection including devices for monitoring the explosibility of mine dust and passive and triggered explosion barriers are also discussed. The paper concludes with a section on frictional face ignitions and methods for their control.

INTRODUCTION

There has been a notable decline in the frequency and severity of mine explosions since the turn of the century. Among the major safety measures responsible for this decline are the introduction of rock dusting, adoption of permissible explosives and electrical equipment, improved ventilation and improved methods for detecting hazardous conditions, for example, the methanometer. Although these advances in mine safety are noteworthy, an examination of recent statistics (table 1) indicates that the problem of mine explosion prevention is not completely solved. For example, in recent years major mine explosions have occurred in Turkey (Armutcuk 1983, 102 fatalities); South Africa (Hlobane 1983, 64 fatalities); Japan (Minami Oh Yubari 1985, 61 fatalities); France (Simon 1985, 22 fatalities); and the Federal Republic of Germany (Stolzenbach 1988, 51 fatalities). Other major mine explosions have recently occurred in Brazil (31 fatalities), Czechoslovakia (65 fatalities), Mexico (43 fatalities), Taiwan (90 fatalities), and Yugoslavia (39 fatalities). The high fatality rates in the United States for the years 1976 and 1981 were associated with the Scotia Mine Disaster in 1976 which claimed a total of 26 lives; the Dutch Creek explosion in April 1981 with 15 fatalities, the Atkins Mine Explosion on December 7, 1981 with 8 fatalities and the Grundy Mine Explosion on the next day which resulted in 13 fatalities (Richmond et al 1983).

Aside from these documented events there is also growing concern over the number of frictional methane ignitions that occur annually. Figure 1 shows the frictional ignition trend for the U.S. World-wide statistics on the frequency of frictional ignitions are not available, but the data presented in figure 1 indicate the magnitude of the problem. Strangely enough, the

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high frequency of methane ignitions by friction has not as yet seriously impacted fatal accident statistics, but their potential for disastrous consequences is well known. The mine disaster in Glace Bay, Nova Scotia in 1979, which claimed 12 lives and was attributed to frictional ignition, serves to illustrate this point (Canada 1979).

Table 1: Fatalities From Underground Coal Mine Explosions

Year	Worldwide[1]	United States[2]
1976	325	27
1977	342	4
1978	57	0
1979	119	0
1980	51	5
1981	204	36
1982	62	7
1983	291	8
1984	222	3
1985	223	3

[1]Michelis et al 1986
[2]USDOL-MSHA (annual reports)

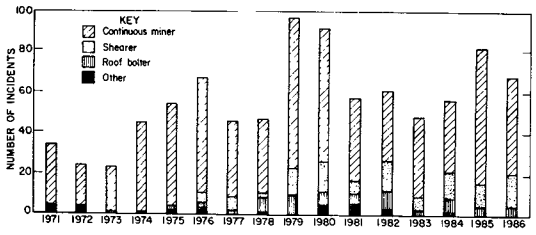


Figure 1. - Fatalities From Underground Coal Mine Explosions

ROCK DUSTING

Rock dusting of coal mines has been in use now for about half a century as a precautionary measure against the dust explosion hazard. It is generally agreed that the effectiveness of rock dust lies in its ability to be simultaneously dispersed with coal dust, and thus prevent flame propagation by serving as a heat sink. In the United States, Rice of the U.S. Bureau of Mines suggested the use of rock dusting for preventing or limiting coal dust explosions as early as 1911 (Rice 1911); he referenced experiments being conducted in Great Britain. Experiments at the Bureau of Mines confirmed that ignitions could not be obtained with mixtures of Pittsburgh coal dust and rock dust having an incombustible content of 64 percent. In 1927 (Rice 1927) and again in 1937 (Mine Safety Board 1937) the Bureau made recommendations concerning the use of rock dust; the specified level was to produce a total incombustible content of 65 percent. The 1953 Federal Mine Code called for 65 percent incombustible plus an additional 1 percent for each 0.1 percent methane present. This was further revised in the 1969 Health and Safety Act which raised the requirements for returns (U.S. Congress 1969).

The current regulations require that the incombustible content of the combined coal dust, rock dust and other inerts be not less than 65 percent, but the incombustible content in returns where the coal dust is finer shall be not less than 80 percent. Where methane is present, the required incombustible is increased 1.0 and 0.4 percent for each 0.1 percent methane where 65 and 80 percent incombustible, respectively, are required. The effectiveness of rock dusting in preventing mine explosions is illustrated by the decline in fatalities associated with mine explosions which in the U.S. declined from 300/year in the late 1920's to about 20/year in the late 1960's. Since the 1969 Act this number now averages about 8/year. Other safety measures, especially permissible cap lamps, explosives and electrical equipment together with improved ventilation contributed to this decline, but the general use of rock dust played a major role in reducing the frequency of coal dust explosions.

Most major coal producing countries have similar requirements, some more stringent and some less stringent

than those enforced in the U.S. A partial listing of these requirements is given in table 2.

Table 2: Summary of Rock Dusting Requirements for Various Countries

Country	TIC pct	VM pct	CH_4, pct
Australia[1]	65	---	>0.1
	75	---	>0.1
	60	---	<0.1
	70	---	<0.1
Belgium[2]	75	≥18	<1
	78		1-1.5
	80		>1.5
Canada (Nova Scotia)	75		<1
	80		>1
Czechoslovakia[3]	up to 80		
Federal Republic of Germany	80	---	
France[4]	55 (RD)	≤18	no gas
	65 (RD)	≤18	gas
	70 (RD)	>18	no gas
	80 (RD)	>18	gas
	50 (RD)	≤18	no gas
	60 (RD)	≤18	gas
	65 (RD)	>18	no gas
	70 (RD)	>18	gas
Japan[5]	76 (F)	35	>1
	83 (F)	35	>1
	71 (F)	35	<1
	78 (F)	35	<1
	.	.	.
	.	.	.
	.	.	.
	57 (F)	≤15	>1
	62 (C)	≤15	>1
	50 (F)	≤15	<1
	54 (C)	≤15	<1
Poland	70	>12	no gas
	80	>12	gas
Republic of South Africa	50	>14, 20	no gas
	55	>20, 22	no gas
	60	>22, 25	no gas
	65	>25, 27	no gas

Table 2: Summary of Rock Dusting Requirements for Various Countries Continued

Country	TIC pct	VM pct	CH_4, pct
South Africa Continued	68	>27, 30	no gas
	70	>32, 32	no gas
	72	>32, 35	no gas
	75	>35	no gas
United Kingdom	50	20	
	55	22	
	60	25	
	65	27	
	68	30	
	70	32	
	72	35	
	75	>35	
United States	65	>20	no gas
	75	>20	1
	80 retn	>20	no gas
	84 retn	>20	1
Soviet Union	60	>10	no gas
	80	>10	gas

[1] High values of TIC apply to face zone; low values apply to zone 200 m beyond face.

[2] For coals with <18 pct volatile matter (VM) lower values of TIC may be used on approval by authority.

[3] TIC set for particular mine based on explosibility of coal dust.

[4] French requirements are based on rock dust content (RD) not total incombustible content (TIC)

[5] In Japan the specific requirements depend on the ash, moisture and volatile content of the coal, the gassiness of the seam and the fineness of the rock dust used. Extreme values, see US Coals, are given in Table 3. (F) denotes fine rock dust (85 pct <74 μm) and (C) denotes coarse rock dust (45 to 55 pct 74 μm).

Table 3: Inerting Requirements (TIC) for Different Coals in Two Different Experimental Mines with No Methane Present[1]

Coal Type	Volatiles Percent	Total Incombustible Content, pct Pulverized Coal (80% < 200 mesh)		Mine Size Coal (20% < 200 mesh)	
		BEM	LLL	BEM	LLL
Pocahontas	18.0	60	>70	40	55 to 60
Pittsburgh	36.5	78	79 to 82	63	>68
Sunnyside	38.1	80	appr. 80	64	70 to 75

[1]Weiss, et al 1989

As will be noted some countries grade rock dust requirements according to the volatile content of the coal. Since rock dusting is the chief defense against dust explosions and probably will remain so in the foreseeable future, it seems appropriate to review the adequacy of the current standards in light of recent research findings. The U.S. regulations on rock dusting were developed on the basis of extensive experimentation in the Bruceton Experimental Mine (BEM) using "mine-sized" Pittsburgh seam coal (20 percent through 200 mesh). The BEM has a cross section of about 5.5 m^2 (2.9 m wide by 1.9 m high). This represented the typical mine entry size when the BEM was constructed in 1910. With advances in roof support, the width of mine entries gradually increased, and 7 m wide entries are now common in the United States and elsewhere. In order to conduct more meaningful experiments on coal dust explosion prevention the Bureau constructed the Lake Lynn Laboratory (LLL) which has entries that average 12 m^2 (6 m wide by 2 m high) (Mattes et al 1983). Since becoming fully operational in 1985, a number of important experiments that bear on the adequacy of current rock dusting practices have been conducted. The principal goal of those experiments was to determine the inerting requirements for coals of different particle size and volatility in large entries. The details of these experiments have been presented elsewhere, but the important findings are summarized in table 3.

For fine (pulverized) high volatile coals (Pittsburgh and Sunnyside) the inerting requirements in terms of total incombustible content and in the absence of methane are about the same in the BEM as at LLL. However for fine, low volatile coal (Pocahontas) significantly more inert is required to prevent propagation in the large entries at LLL than in the smaller BEM entries. For coarser coal dust significantly more inert is required at LLL than in the BEM for both high and low volatile coals, the differences being more pronounced with low volatile coals. In essence, coarse and low volatile coals were shown at LLL to be more reactive and require correspondingly higher inert content to prevent propagation. This effect is presumably associated with longer reaction zones (thermal fields) which apparently scale with entry size and reduced heat losses in the larger entry.

The recent results from LLL although preliminary in nature, point out the possible dangers of extrapolating small-scale laboratory data to full-scale. This is especially true when trying to set a limit for the volatility or on the particle size of coal particles that might enter into explosion reactions in large-scale explosions. The implications of these results regarding the current rock dusting standards in table 2 are obvious.

Inerting requirements for preventing the propagation of coal dust explosions is complicated by factors other than those associated with large entry sizes. The increased

utilization of high production, mechanized mining methods poses additional problems in terms of dust fineness and float dust deposition rates. There is strong evidence that the coal dust now deposited in mine roadways is much finer than the so-called "mine size dust" on which many current rock dusting requirements are based (table 4). For example, a dust survey taken in 1954 (Hartman et al 1954) showed that floor dust averaged 28 percent through 200 mesh; whereas surveys conducted in 1964 (Saltsman 1964) and again in 1982 (Courtney et al 1982) indicated that this value had increased to about 80 percent, roughly equivalent to the pulverized dust listed in table 3.

The explosibility of thin deposits of float coal dust is well documented and in fact, the most severe experimental dust explosion conducted by the Bureau involved float coal dust (Nagy 1981). There is little data regarding the rate of float dust deposition in mechanized mines, but there is no doubt that it correlates with production rates; with the increased utilization of longwall mining methods it represents a problem of growing concern. Recent experiments at the Bureau indicate that only the top layer (2 to 4 mm) of floor dust participates in a weak to moderate float dust explosion (Sapko et al 1987). This results in an enriched cloud of dispersed dust requiring much more inert in the substrate to effectively quench the float dust explosion than would be needed if thicker layers of the floor dust were dispersed. For example, a substrate containing 80 percent total incombustible was shown to be effective only for float dust deposits less than 0.08 mm thick, corresponding to a dispersed concentration of 50 mg/liter.

Heavier deposits of float coal dust required significantly higher values of total incombustible in the substrate to prevent propagation. Current sampling techniques which usually involve relatively thick layers of floor dust tend to overestimate the degree of protection against float dust explosions. Since rib and roof samples, which represent a small fraction of the total dust, are often combined with floor dust samples, the overall protection against float dust explosions is also overestimated.

Table 4: Dust fineness in the return entries of coal mines.

Year	Pct through 200 mesh	
	Rib-roof	Floor
1954[1]	49	28
1963[2]	50	40
1964[3]	--	83
1982[4]	--	72 - 100

[1]Hartmann, et al 1954
[2]Kawenski, et al 1963
[3]Saltsman 1964
[4]Courtney, et al 1982

Attempts to improve the effectiveness of rock dusting have not met with much success. Full-scale experiments with fluidized rock dust -- to increase dispersibility -- indicated that more fluidized rock dust was required to prevent propagation than ordinary rock dust (NCB 1967 and Greenwald 1948). This was attributed to the higher cohesiveness of ordinary rock dust which more effectively binds the coal dust particles. This is supported by some recent laboratory studies which showed that fluidized limestone dust was less effective than ordinary limestone dust in inerting coal dust. In addition, some large scale studies in the BEM with a phosphate dust, which was intrinsically more fluid than limestone dust, showed it to be less effective than ordinary limestone dust.

Salt encrustation is also utilized to combat float dust deposits and can effectively do so if properly maintained (Berg 1982). This

technique was examined for potential application in the U.S. mining industry (Foreman et al 1981). However, problems involving corrosion and maintaining effectiveness under conditions of low humidity were not resolved, and enthusiasm for the method waned.

The most promising methods for improving the quality of rock dust practice appear to lie in the recent development of practical methods for determining the inert content of mine dust samples, in-situ. The Bureau of Mines has developed an optical rock dust meter that can be used underground to give a direct and rapid read-out of the rock dust content of mine dust samples, thus eliminating the need for laborious and time-consuming laboratory analysis of rock dust content (Sapko et al 1985). The in-situ measurement allows for immediate corrective action and has high potential for improving mine explosion safety. The Bureau device is in the process of being commercialized. A similar device which utilizes nuclear radiation for measuring rock dust content is being developed in Poland and is also believed to be near commercialization (Lebecki 1975 and Krzystolik 1985). The principle of optical reflectance utilized in the Bureau rock dust meter appears to be applicable to the problem of quantitatively determining the build-up of float dust deposits and is being reduced to practice (Sapko et al 1988).

PASSIVE BARRIER

To provide supplemental protection against coal dust explosions, passive barriers have been deployed in most major coal producing countries. Conveyor roadways have received emphasis (NCB 1961). There are two approaches to the deployment of passive barriers --- concentrated and distributed. Concentrated barriers are placed at strategic locations within a mine and are designed to quench an explosion immediately on arrival at the location. Distributed barriers are placed throughout the length of an entry and are designed to moderate or quench an explosion over much longer distances. Table 5 summarizes requirements for various countries utilizing either concentrated rock dust or water barriers. The current trend is toward increased utilization of water barriers in lieu of rock dust barriers. Almost all countries that deploy water barriers use frangible plastic tubs which typically hold either 40 or 80 liters of water. A notable exception is Japan where flexible water bags are used (Ng et al 1986). For concentrated water barriers many countries have adopted a 200 liter/m^2 loading. However, for gassy mines some countries (Australia and Poland) (NSW 1984 and Ministry of Mining & Energy 1984) require a loading of 400 liter/m^2. In countries where distributed barriers are deployed a loading of 1 liter/m^3 has been adopted. Where water barriers are deployed they are in most situations suspended from the roof which in the case of roadways using yieldable steel arches does not significantly impede ground transport. The impact of the use of barriers for supplemental explosion protection on the local total incombustible or rock dust requirements varies from country to country. In the United Kingdom (The Coal Mine Regulations 1977) and in many other countries the level of rock dusting near barriers cannot be diminished. On the other hand, in France (Cerchar 1988) the rock dusting requirement is reduced by 5 percent in the zones where barriers are deployed.

Because sizeable pressure forces are required to activate passive barriers, they must be located at some distance from the probable ignition point (working face). At intersections, which provide considerable pressure relief, passive barriers can be easily defeated. This ease of defeat is the chief reason that passive barriers have not been adopted in the United States which still

Table 5: Summary of Passive Barrier Requirements in Various Countries

Country	Loading			
	Rock dust (kg/m^2)		Water ($liters/m^2$)	
	nongassy	gassy	nongassy	gassy
Australia[1]	200	400	200	400
Belgium[2]	---	400	---	200
Canada[3]	---	200	---	200
Czechoslovakia[4]	200	400	---	---
Federal Republic of Germany[5]	---	400	---	200
France[4]	---	400	---	200
Japan[6]	$0.1 m^3/m^2$ $0.3 m^3/m^2$ $0.4 m^3/m^2$	light heavy ext hvy	--- --- ---	100 light 300 heavy 400 ext hvy
Poland[7]	200	400	200	400
South Africa[8]	200	400	---	---
Romania[4]	---	400	---	200
United Kingdom[9]	---	200 light 400 heavy	--- ---	200 minimum
Soviet Union[4]	200	400	---	---

1 MSW 1984
2 Ministrere des Affaires Economiques 1982
3 Canada Labour Code 1979
4 Cybulski 1976
5 Michelis 1987
6 Ministry of International Trade and Industry
7 Ministry of Mining and Energy 1984
8 Republic of South Africa
9 The Coal Mine Regulations 1977

produces a significant fraction of its total coal in room-and-pillar mines.

The problem of erecting passive barriers to keep up with the changing location of the active face poses another problem. Erecting frames to support barriers suspended from the roof is a labor intensive operation. For wide entries, experimental studies on the deployment of barriers along the ribs have been conducted in several countries. Recent results at the LLL indicate (Sapko et al 1988) that stacked rib barriers are fairly effective and may have direct application in longwall mining.

There are several important examples of mine explosions where the value

of passive barriers in reducing the extent of propagation was convincingly demonstrated. At the Fenton Colliery (UK, 1963) (Eisner et al 1964) a methane explosion occurred at the face of an entry. The first shelf of the barrier installation was located 80 m from the face of the heading. There was clear evidence that the flames had actually reached the barrier. A transfer point was located just outbye the barrier where there were ample deposits of coal dust, and there was little doubt that without the operation of the barrier the explosion would have increased in intensity and travelled much further. In another incident at the Mainsforth Colliery in Durham (UK, 1963) an explosion occurred (Eisner et al 1964). A rock dust barrier on the mine intake airway 218 m from the face of the heading was completely discharged and without doubt arrested the spread of flame.

However, there have been other instances when the effectiveness of passive barriers was not so clearly demonstrated. In Belgium in March 1984 the efficacy of water barriers was proven to some extent during the explosion at the Eisden Mine in which seven men were killed (Jones 1988). The bottom gate to a face in an inclined seam was protected by distributed water barriers of four shelves, each containing a single 80-liter water trough, positioned 100 m behind the face, and then repeated outbye at 40 m intervals. The investigation indicated that the first five sets of distributed barriers operated, between 100 and 270 m from the face, and arrested the explosion. This was suggested as the reason why men who were located some 290 m from the face were not injured.

More recently, an explosion occurred at the Moura Mine in Australia which on July 16, 1986 killed 13 miners (Jones 1988). The inquiry report shows that a water barrier was installed in the belt heading as required by law and was in good working condition prior to the event. The report concludes that the explosion was quenched by the presence of the water barrier in the conveyor road, and by substantial quantities of water present in swills in other roadways.

TRIGGERED BARRIERS

Triggered explosion barriers are a relatively recent development as compared to passive barriers, and there is considerable research effort in the United States and elsewhere to develop and implement this technology.

There are two general types of triggered barriers: machine mounted devices for the rapid detection and extinguishment of face ignitions and fixed triggered barriers for quenching more or less fully developed methane or coal dust flames.

Table 6 summarizes the machine mounted technology for three countries. The Federal Republic of Germany has been using their BVS system in operating mines, and the United Kingdom has conducted tests with a Graviner system under conditions which simulate the mine environment. All three systems utilize ultraviolet flame detectors (1800 Å) which can distinguish methane or coal dust flames and are not prone to false triggering by artificial light sources (Furno 1985). Ammonium phosphate in its mono or dibasic form is the extinguishing agent of choice and has been demonstrated to be particularly effective against methane and coal dust flames (Michelis 1983 and Furno et al 1985). Expulsion of the extinguishing agent, which must occur in as short a time as possible, is affected with pressurized nitrogen or Halon 1301 (CF_3Br). The quantity of extinguishing agent varies with the application but typically runs from 16 to 48 kg depending on machine and entry geometry.

Table 6: Summary of Machine Mounted Ignition Suppression Practices in Various Countries

Country/ Type	Detector Type	Extinguishing Agent	Expulsion Mode	Vessels		
				Number	Shape	Loading, kg
Federal Republic of Germany[1] BVS System	UV	Tropolar Powder (ammonium phosphate)	Nitrogen at 120 bar + detonator activated valves	6	12.3 liter cyl.	48
United Kingdom Graviner System[2]	UV	Furex 770 ammonium phosphate	60 bar N^2 or Halon 1301	4-6 (seven diff pos)	app. 7 liter cyl.	16-24
United States PRC System[3]	UV	ABC Powder ammonium phosphate	linear shaped charge and Halon 1301 13.6 bar 13.6 bar	6	tubular cann. 0.76, 1.2, 1.8 m long x 5 cm dia	17

[1] Michelis 1983, Fabor 1987
[2] Browning et al 1985
[3] Furno et al 1985

There is general agreement that the use of machine mounted ignition suppression systems would greatly strengthen the strategy for mine explosion prevention since these systems are targeted at combating methane ignitions near their point of ignition. However, the systems are severely taxed in detecting and suppressing fast flames associated with near stoichiometric methane-air mixtures, and their effectiveness against turbulent flame has not been demonstrated. In addition, the systems are difficult to retrofit to existing mining machines in a way that would guarantee protection against all possible points of ignition. Components of machine mounted ignition suppression systems, especially the flame detectors, are rather delicate and will require dedicated maintenance in order to function properly. Noise associated with explosively activated systems (US) and material toxicity problems with Halon 1301 (US and UK) are also problems that have not been completely resolved.

Table 7 summarizes the characteristics of fixed location triggered barriers being developed or deployed in various countries. The Belgium system, which utilizes a length of detonating cord to expel water from a 2-m long by 0.25 m diameter cylinder of open pore polyurethane, is triggered by a thermo-mechanical device that is sensitive to both pressure and flame. This system is also uses in France. The system utilized in the Federal Republic of Germany also used detonating cord for dispersion, but the extinguishing agent (water) is contained in a PVC tub similar to that used for passive barriers; sensitive thermocouples are used to trigger the device. The system deployed in the United Kingdom consists of a steel cylinder containing 227 liters of water and a pressurized reservoir of nitrogen which is used to drive a piston to expel the water. The system is activated with a sensitive thermocouple coupled to a fast acting valve which releases the stored nitrogen. The system

Table 7: Summary of Fixed Location Triggered Barrier Practices for Various Countries

Country/ Type	Detector Type	Extinguishing Agent	Expulsion Mode	Vessels	
				Shape	Loading L/m^2
Belgium[1]	thermo-mechanical	water 90-100 liters/ unit	detonating cord	2 m long 25 cm dia open pore polyurethane foam	10
Federal Republic of Germany[2]	thermocouple	water 80 liters/unit	detonating cord	PVC water tub	80
France[3]	thermo-mechanical	water 90-100 liters/ unit	detonating cord	2 m long 25 cm dia open pore polyurethane foam	10
United Kingdom[4]	thermocouple	water 227 liters/unit	compressed N_2	long cylinder	45
United States new	pressure and UV (solar panel)	water or mono-ammonium phosphate 40 liters/unit	sheet explosive	ridged polystrene container	80

[1] Ministrere des Affairs Economiques 1982
[2] Michelis et al 1987
[3] Cerchar 1988
[4] HSE 1977

currently being examined in the United States uses a formed sheet of explosive to expel water (or monoammonium phosphate) from a 40-liter reinforced polystyrene container. The unit is activated with a novel flame sensor, an ultraviolet solar panel, which provides enough electrical energy to fire any reasonable number of electric detonators and is in a sense "explosion-powered".

All of these systems have successfully functioned with water loadings varying from 10 to 80 liters of extinguishing agent per square meter of entry in experimental mines. Total discharge volumes vary from 227 liters (UK) to 1600 liters (FRG) depending on entry size. Experiments in the U.S. center around total discharge volumes of 720 liters.

Triggered barriers for fixed locations offer significant advantage over machine mounted devices. First and foremost they do not have to withstand the rigors of the coal cutting environment at the face. In addition, they do not have to respond in the very short time frame necessary for effectively quenching face ignitions and as a consequence are far simpler in design. Furthermore, they do not appear to suffer from the noise and potential toxicity hazards that are associated with some of the machine mounted systems. They also offer advantages over passive barriers

inasmuch as they would appear to be applicable in low coal seams and in the room-and-pillar mine configurations.

FRICTIONAL IGNITION

In the U.S. the ignition of methane by coal cutting machinery outweighs ignitions by other sources, electrical, open flame, etc., by a wide margin, usually by three or four to one. Since there is no reason to believe that the U.S. is unique in this regard, a brief summary of the current efforts to mitigate this hazard is in order.

There has been a large amount of research on the subject of frictional ignition in general and the frictional ignition of methane by mining machines in particular; several excellent survey papers on the subject have been published (Hartman 1955, Powell et al 1975, Lobegko 1978, Watson 1981, and Browning 1988). The important findings are:

1. Most ignitions are caused by cutter bits striking rock in mines; siliceous or quartz bearing sandstones present the greatest hazard followed by pyrite inclusions.

2. The ignitions invariably start in methane air mixtures and not in coal dust. (There have been two or three reported instances of coal dust ignitions in allegedly "non-gassy" mines; however, they have not been verified.)

3. The principal source of ignition is hot or molten rock created through friction -- not the elevated temperature of the bits or flying sparks.

4. All bit materials can cause ignitions, but some materials are more incendive than others.

5. The tendency for ignition to occur greatly increases with bit wear. (This is due to an increase in the contact area associated with bit wear and the subsequent exposure of the steel shank material which is more incendive than the tungsten carbide cutting tips.)

6. Reducing the cutting speed appears to reduce the tendency for ignitions to occur. (Recent research at the Bureau indicates that the benefit of machine speed reduction is marginal at best.)

These observations suggest practical ways of reducing the frequency or severity of ignition, i.e., improved ventilation, a rigid bit inspection program with frequent bit changes, improved cutter bits, and the use of water to cool the hot spots responsible for ignitions.

Observations on the relative incendivity of various metals used in the construction of coal cutting bits indicate that there is some room for improvement in the design of bits to reduce their tendency to ignite methane. For example, Blickensderfer's early work showed (Blickensderfer et al 1974) that the steel used in the construction of bit shanks was significantly more incendive than the tungsten carbide cutting tip.

This suggested that any modification in bit design that would prevent early exposure of the shank material to frictional abrasion would serve to moderate the tendency for bits to cause ignitions. On this basis, a considerable amount of experimental work was conducted with conical bits used on continuous miners (Cheng et al 1983). One experimental design having a "mushroom" tip or capped carbide, as it is referred to in the bit industry, was particularly effective in protecting the steel shank from early exposure during the cutting operations. Laboratory testing of a commercial version of

the "mushroom" bit also showed a strong relationship between ignition probability and bit attack (and clearance) angle. The optimum attack angle corresponding to minimum ignition probability, was found to be near 57° for the particular bit used (Cheng et al 1987). On the basis of this research an extensive field evaluation was conducted under a joint cooperative effort between the Bureau and Jim Walter Resources Incorporated (JWR), Brookwood, Alabama together with Hughes Mining Tools, a Division of Hughes Tool Company, Bristol, Virginia (McNider et al 1987). An experimental cutter head laced to accommodate a capped carbide bit operating at a 57° attack angle was constructed and tested in JWR's Blue Creek #3 Mine in an area prone to ignitions. The machine mined over 50,000 tons in an ignition prone area without incident. When it was removed for internal rebuilding, a miner with a 50° attack angle replaced it. Before 10,000 tons were mined, two ignitions had occurred. Consequently, all miner heads scheduled for rebuilding were equipped with the capped carbide bits fixed to operate at a 57° attack angle. In the past four years over ten mining machines have been modified in this manner resulting in a dramatic reduction in the number of ignitions at Jim Walter Resources. In fact, there has not been a single ignition traceable to a modified machine that was within specifications.

Another promising method for controlling frictional ignitions would appear to be the use of machine mounted water sprays to cool the hot spots responsible for ignition. There is overwhelming laboratory evidence that properly designed water sprays are effective for this purpose (Watson 1981 and Browning 1988). All of these studies agree that water sprays in back of the bit are much more effective than sprays in front of the bit. Our experience is that front sprays are almost totally ineffective in preventing ignition although Browning (Browning 1988) indicates that front sprays do have some effect. This is probably associated with bit tip cooling and the lubricating effect of water in reducing frictional forces. Browning also cites work that indicates that bits with water through the body, with the water released immediately behind the carbide tip, were ineffective in eliminating ignition. However, other work indicates (Morris et al 1984) reasonable effectiveness of "through-the-bit" water sprays.

Although there is no disagreement that back sprays can be used to control frictional ignitions, there is no consensus on the optimum spray configuration. Courtney found that a 30° full cone spray nozzle operating at 6.9 bar and 1.9 lpm helped when used with a relatively small chisel bit (Courtney 1987). Browning recommends (Browning 1988) a 65° hollow cone spray delivering about 2.3 lpm. A recent bulletin from the British Coal Board recommends (British Coal - Pvt comm) a 25° full cone spray operating at between 1.9 and 2.3 lpm. Recent work at the Bureau showed (Cecala et al 1988) that jet sprays operating at 6.9 bar and 6.1 lpm were effective in preventing ignition with a large chisel bit. However, lower flow rates were ineffective in quenching the hot spot responsible for ignition. This indicates that the water flow requirements are probably sensitive to bit tip size and geometry which seems reasonable. Other factors complicate the identification of optimum spray parameters for behind-the-bit spray systems. For example, besides hot spot cooling, the water spray may serve as an inerting agent for the methane containing air volume in the vicinity of the pick, thus reducing the possibility of an ignition. Aerodynamic effects which would sweep the methane-air mixture from around the cutting tip may also be important. In any case, the back sprays should be carefully oriented

to sweep the area immediately behind the carbide cutting tip.

It should be pointed out that there is little documentation of the effectiveness of machine mounted back sprays under actual mining conditions. Most laboratory research does not involve the actual cutting of coal, and the effect of the production of large amounts of cuttings in the vicinity of the back spray is not known. Recent field experiments indicate (Cecala et al 1988) that the effectiveness of back sprays demonstrated in the laboratory experiments are indeed carried over to real mining conditions.

CONCLUSIONS

Notwithstanding the effort devoted to their prevention, coal mine ignitions and explosions continue to occur with unsettling frequency. The increased utilization of high production, mechanized mining methods in deeper and gassier coal seams will place additional demands on those responsible for mine safety particularly in explosion prevention. Rock dusting is still the chief defense against coal dust explosions and probably will remain so in the foreseeable future. Research has raised some question regarding the overall effectiveness of present rock dust practice. Recent experiments in a large entry indicate that coarser and less volatile coal dusts participate in explosion reactions more readily than in smaller entries and require more rock dust for inerting than anticipated under current regulations. In addition, high production mining methods lead to the generation of larger quantities of fine dust which is difficult to inert especially in the form of float dust deposits. Except for the development of techniques for monitoring the quality of rock dusting, in-situ, there is little on the research horizon.

Passive explosion barriers have proven to be effective for supplemental explosion protection in arched roadways with few intersections but are difficult to apply in low coal and the room-and-pillar mining configuration. Triggered explosion suppression systems for entry protection are being developed and may have wider application than passive barriers if problems of reliability, maintenance, noise, and potential health hazards (in some cases) can be overcome. Triggered ignition suppression systems are also being developed for use on continuous miners and longwall shearers for controlling frictional ignitions, but these too suffer from the same problems associated with triggered explosion barriers. In addition, they are difficult to retrofit to existing mining machines and will be difficult to maintain in operating condition. Results of field studies on the effectiveness of improved bit designs and cutting geometries in reducing the frequency of frictional ignitions are encouraging; the same is true for "water sprays" mounted behind the cutting bits for the purpose of cooling the hot spots responsible for ignition.

The overall direction in research on mine explosion prevention seems to be headed in the right direction, but it is disappointing that quantum improvements in the practice of rock dusting do not appear imminent.

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COMPTE-RENDU: PREVENTION ET SUPPRESSION DES EXPLOSIONS DANS LES MINES DE CHARBON

M. Sapko, N. Greninger et R. Watson

Les méthodes de sécurité utilisées : le poudrage minéral, les explosifs autorisés, les équipements électriques intrinsèquement sûrs, l'amélioration de la ventilation et l'amélioration des méthodes de détection des risques sont en grande partie responsables de la diminution remarquable du nombre des explosions minières. Toutefois, selon de récentes statistiques, le problème de la prévention des explosions minières subsiste toujours. Il y a également une préoccupation croissante concernant le nombre d'inflammations par an provoquées par la friction du méthane. Les recherches récentes sur la prévention des explosions portent tout particulièrement sur le poudrage minéral dont l'efficacité générale a été récemment mise en évidence. D'autres méthodes de protection contre les explosions comprennent la surveillance de l'explosibilité des poussières minières, ainsi que les protections anti-déflagrantes passives et actives. Nous examinons les inflammations par friction des fronts de taille ainsi que les moyens de les contrôler.

VERHÜTUNG UND UNTERDRÜCKUNG VON EXPLOSIONEN IN KOHLEGRUBEN

M. Sapko, N. Greninger, R. Watson

Zu den wichtigsten Sicherheitsfaktoren, die für den bemerkenswerten Rückgang der Explosionen in Kohlenzechen verantwortlich sind, gehören solche Maßnahmen wie das Einstauben mit Steinstaub, Verwendung von zugelassenen Sprengstoffen, eigensichere Elektroanlagen, verbesserte Bewetterung und eine Verbesserung in der Erkennung von Gefahrensituationen. Trotzdem zeigen neuere Statistiken, daß die Explosionsverhütung immer noch ein Problem ist. Auch wächst die Besorgnis wegen der jährlichen Anzahl von durch Reibung verursachten Methanentzündungen. Bei einer Betrachtung der neuesten Explosionsverhütungs-Forschung wird besonders auf das Einstauben und die kürzlich erhaltenen Beweise der Effektivität von Einstaubungs-Normen geachtet. Zusätzliche Explosions-Verhütungsmittel bestehen in Einrichtungen zur Überwachung der Explosivität des Grubenstaubes und passiv sowie aktiv ausgelöste Explosionssperren. Entzündungen durch Oberflächenreibung und ihre Beherrschung werden diskutiert.

ОБЗОРНЫЙ ДОКЛАД: ПРЕДОТВРАЩЕНИЕ И УМЕНЬШЕНИЕ СИЛЫ ВЗРЫВОВ В УГОЛЬНЫХ ШАХТАХ

М. Сапко, Н. Гренинжер и Д. Вотсон

Среди главных факторов, способствующих значительному сокращению взрывов в шахтах, можно отметить запыление каменной пылью, применение рекомендованных взрывчатых веществ, взрывобезопасное электрическое оборудование, усовершенствованную вентиляцию и улучшенные методы слежения за возникновением опасных условий. Однако, согласно последним статистическим данным, предотвращение взрывов попрежнему остается проблемой. Все большее опасение внушает ежегодное количество воспламенений метана, вызванных трением. Особое значение во всех рассмотренных авторами исследованиях придается способу каменного запыления и недавно полученным экспериментальным доказательствам эффективности стандартов каменного запыления. Дополнительная защита от взрывов включает приспособления для слежения за взрывоопасностью пыли в шахтах, а также пассивные и быстродействующие противовзрывные барьеры. Обсуждается проблема воспламенения при трении и способы контроля.

评论文：煤矿爆炸的预防和抑止

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减少煤矿爆炸主要的安全因数为：撒石粉，使用准许用的炸药，本质上安全的电力设备，改进的通风设备，及改进的测定危险情况方法。但从最近的统计数字来看，煤矿爆炸仍是一个问题。而且美年甲烷摩擦引燃的事件也为数不少。最近防止爆炸的研究多着重在撒石粉和石粉标准与防止有效性的关系。另外爆炸的保护方法包括侦察煤尘的爆炸性，及无源和触发的爆炸珊栏。摩擦的表面引燃和它的控制亦在本文里讨论。

IMPROVEMENTS ACHIEVED FOLLOWING THE MOST DISASTROUS EXPLOSION IN TURKISH COAL MINES

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ABSTRACT

On March 7, 1983, a great explosion occurred at the Kandilli Colliery of the Armutçuk District of Turkish Hardcoal Enterprises (TTK) in Zonguldak,Turkey. 102 miners were killed and 86 were injured. The main reason of the disaster was ignition of methane accumulated in a band roadway. Also, incidents of a secondary coal dust explosion were detected.

Recently, the Safety Services of TTK developed various projects including rehabilitation of mining method and monitoring of the underground environment. In this paper, a brief information about the Armutçuk District of TTK is given with special attention to the critical factors which cause the risky conditions. Then, special precautions undertaken to control the mine environment and to minimize the ignition of possible methane accumulations are described. Finally, the most recent modifications in working method such as introduction of hydraulic stowing are explained.

INTRODUCTION

Zonguldak is the only hardcoal basin in Turkey. İt is situated on the North-Western Anatolia and the coal is underground mined at five districts in twenty-two workable coal seams of gassy nature (Fig.1). Due to the intense tectonic disturbances, extensive folding and faulting are the dominant geological features of the region. Presently, four million tonnes of saleable coal produced yearly. Pneumatic hand picks and explosives are the only means of coal excavation at the faces supported exclusively by timber. The primary reason which has hindered the application of mechanized extraction in the mines is the adverse geological conditions existing throughout all the districts.

The Armutçuk District is located on the most western part of the Zonguldak Coal Basin and covers an area of 38.5 km^2. Total reserves of the district are 94 million tonnes and avarage annual production is about 300 000 tonnes of saleable coal. The most productive coal seam in the district is the Büyük seam. The coal which is very liable to spontaneous combustion and of gassy and friable nature has a mean calorific value of 6400 kcal/kg.

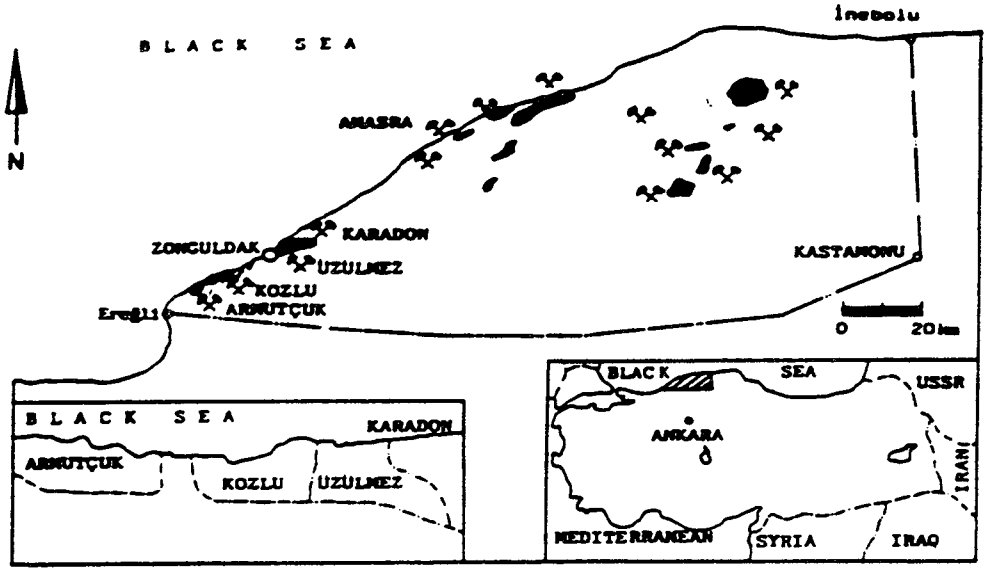


Figure 1.

The avarage moisture content is 8%, ash content 11%, sulphur 1% and volatile matter 33%.

The overlying formation is Westphalian A while the underlying strata are of Namurian age. In general, the dip of the strata is very steep (65 to 85°) in the south part of the district and at upper levels. The coal seams form a syncline at lower levels and turn towards north with 10 to 22° of dip (Fig.2). The axis of this syncline dips towards west. The thickness of Büyük seam is about 7-8 m in south and 3.5-4 m in north.

The whole mine is planned on a variation of horizon mining with 50 and 100 m level intervals. Presently, the main working level is -400 m. The main mining method which was applied in the district before 1984 was a variation of sub-level caving (i.e. sub-level caving with flat shortwall method) and it is shown in Fig.3. In the past, this method caused some fires and explosions since 40% of the coal was being left in the gob area.

The great explosion which occurred on March 7, 1983, resulted in 102 deaths and 86 injuries. The location of the

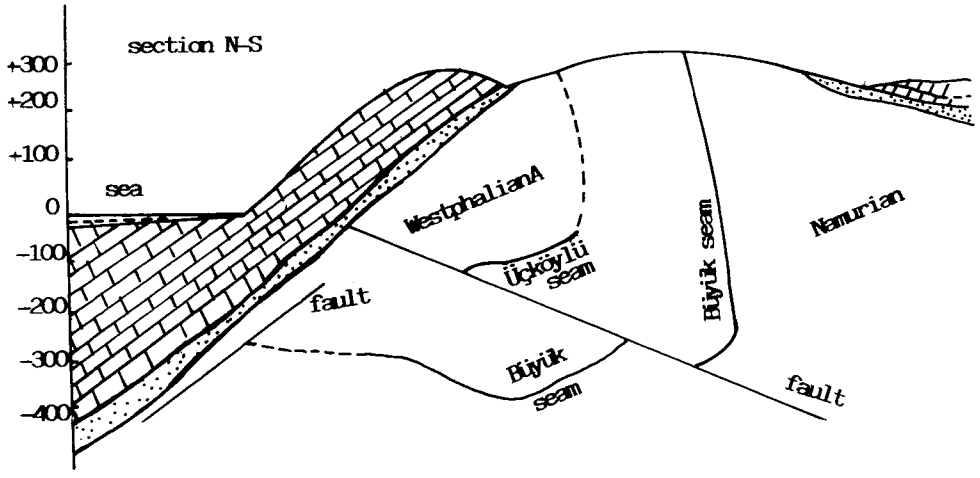


Figure 2.

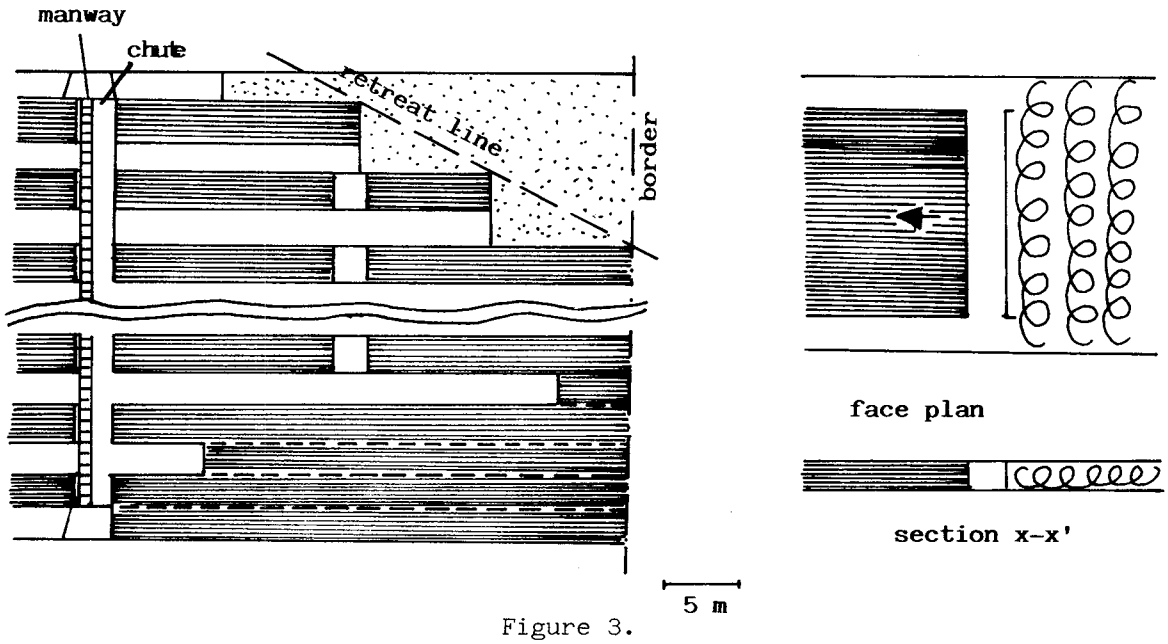


Figure 3.

12003 panel in which this incident took place is shown in Fig.4. Varying views were expressed during the inquiries after the explosion. However, it seemed quite possible that a fire-damp accumulation in 12002A (-350/-375) band roadway caused the first explosion and after that the stirred up dust cloud caused a second explosion in 12200 (-400) main roadway (Bilgin 1985).

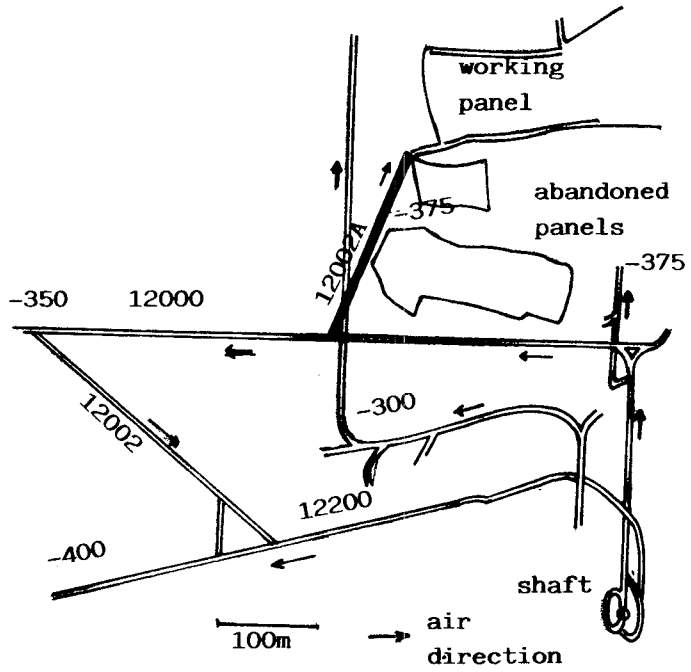


Figure 4.

Following this most disastrous explosion in Turkish mines, various projects have been developed to improve the safety conditions in underground workings of the Zonguldak Coalfield. Kandilli Colliery of the Armutçuk District was one of the collieries selected for the pilot applications.

In this paper, the nature of these projects and initial improvements achieved by the pilot applications are described.

PROJECTS INITIATED FOLLOWING THE DISASTER

TTK has established a Research and Development group on Health and Safety. Projects which have been commenced by this group can be listed as below:

1. Determination of in situ gas contents and methane drainage possibilities of the coal seams of gassy nature.
2. Installation of a modern central gas analysing laboratory.
3. Installation of continous gas monitoring systems for underground workings in the collieries where the seams liable to spontaneous combustion are being worked.

4. Development of precautions to cope with the dust explosions.
5. Improving mining methods to reduce the risk of fires.
6. Rehabilitation and concentration of the certain pilot collieries for more productivity with better safety conditions.
7. Education programs in foreign countries for colliery managers and technical staff.

The initial results of these projects are given in the following chapters.

Measurement of Methane Contents of the Coal Seams and Methane Drainage

A direct gas measuring method similar to that used by Cerchar of France has been developed in cooperation with a Canadian company. Routine measurements are being carried out in each colliery.

Recently, a pilot drainage system has been installed in the Kandilli Colliery of the Armutçuk District. The initial results are quite satisfactory.

Gas Analysis Laboratories

Three full-gas analysis sets (one as a mobile unit) have been installed in 1988. Each set has a double beam luft type infrared gas analyser, an oxygen analyser and a hydrogen/carbonmonoxide chromatographic analyser. Gas samples are collected from collieries in special tubes by hand-pumping and analyzed to obtain the methane, carbondioxide, carbonmonoxide, hydrogen and oxygen proportions. The resuls of the analyses are evaluated by a computer. Also, Graham Index and explosibility of the mixtures are determined. The initial trials are still being carried on.

Continous Gas Monitoring Systems

Computerized central methane monitoring systems of telemetric type were installed in the Armutçuk and Kozlu Districts. The methane and carbonmonoxide gases in underground workings have been continously measured and recorded since September 1986.

Precautions Against the Coal Dust Explosions

Several attempts are being made to set up a routine measuring system for the explosibility of dust and to provide a sufficent stone dust supply for the collieries.

Improvement of the Mining Method

Formerly applied mining method at the Armutçuk District caused serious production losses and safety problems. Recently, a version of top slicing (i.e. top slicing with mat) is being applied. In this method, the thick coal seam is extracted by caving longwall faces from upper slice to the lower ones. Three or four slices are projected depending upon the thickness of the seam in the panel (Fig.5). A wire mesh is placed at the floor of the first slice. This method which minimizes the production losses and provides a better ventilation inside the panels has greatly reduced the spontaneous combustion incidents at the district.

Also, a pilot panel have been installed in which the slices are extracted in upwards manner (Fig.6) and hydraulic stowing is applied (Birön et al.1988). The expected improvements of this method have not been realized yet.

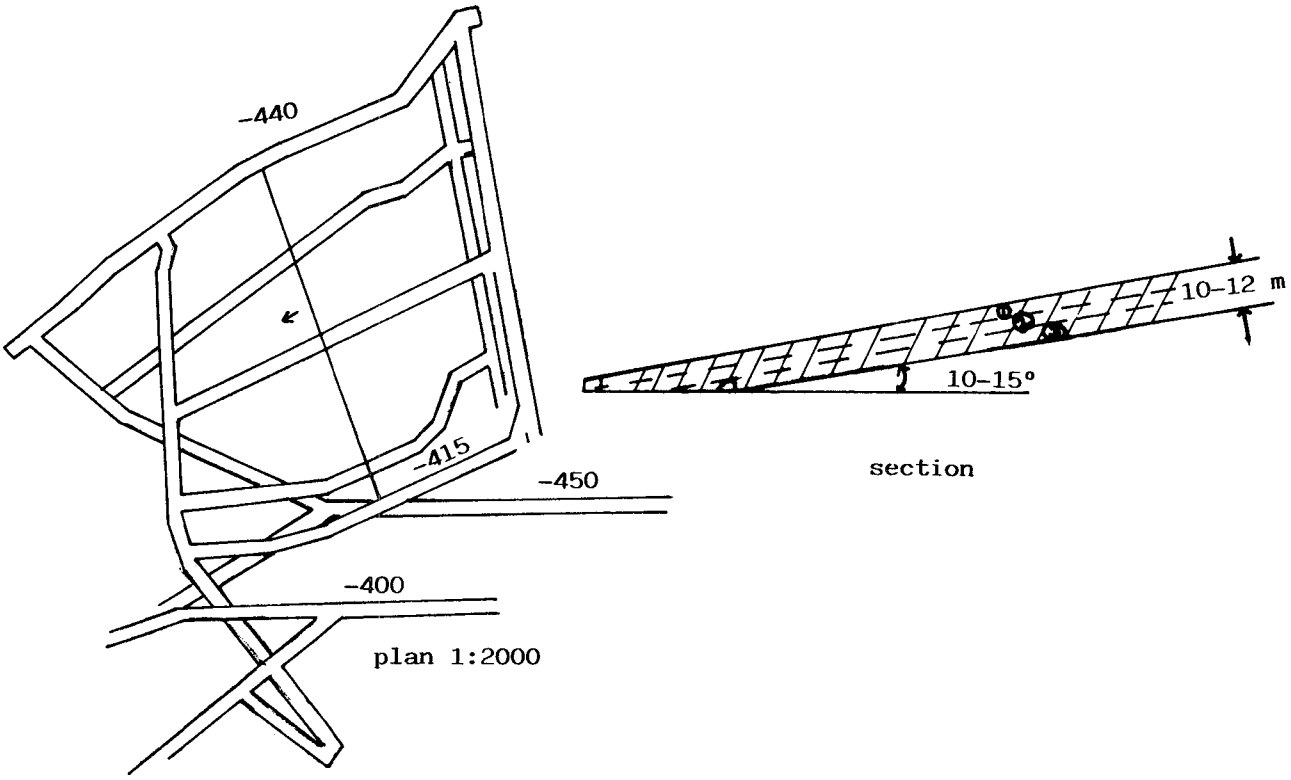


Figure 5.

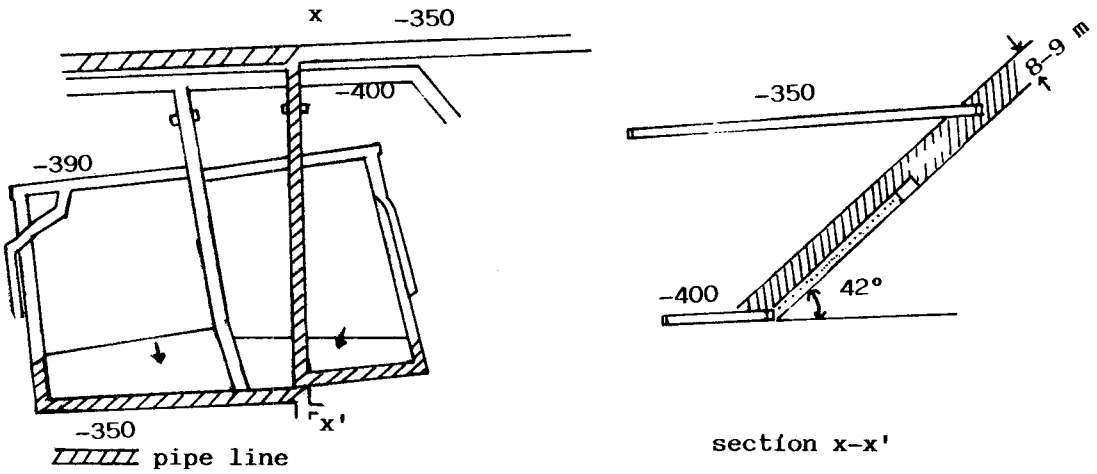


Figure 6.

Rehabilitation Projects

Firstly, three collieries including the Kandilli Colliery of the Armutçuk District have been selected for pilot application of concentration and modernization of the mining activities. The projects commenced at the end of 1988 and are still being carried on. Consulting groups consisting domestic companies, foreign firms and Universities have taken part in the preparation of these rehabilitation projects.

Education Programs

During 1986 and 1987, 18 technical staff of the TTK were educated on the subjects of ventilation, mine fires, methane drainage and mine environment monitoring in foreign countries as a part of a project financed by ILO. This program will be repeated in the coming years.

CONCLUSIONS

Considerable improvements have been achieved in the safety conditions of TTK Collieries in a relatively short period following the great disaster in 1983. Almost all the initial trials have been realized successfully and they have given satisfactory results so far. This encourages more widespread applications of such improvements throughout the Zonguldak Coalfield.

ACKNOWLEDGEMENTS

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PROGRES QUI ONT SUIVI UNE EXPLOSION DESASTREUSE DANS LES MINES DE CHARBON TURQUES

Vedat Didari, Mesut Özturk

Le 7 mars 1983, une explosion importante s'est produite dans la houillère d'Armutçuk appartenant aux Entreprises de Charbon Dur Turc (TTK) à Zonguldak, en Turquie. 102 mineurs furent tués et 86 blessés. Ce désastre était dû principalement à une inflammation de méthane accumulé dans une voie. On a également détecté des incidents dûs à une explosion secondaire de poussières de charbon. Les Services de Sécurité de TTK ont développé récemment un certain nombre de projets de réhabilitation de méthodes d'excavations, d'amélioration des systèmes de ventilation et de surveillance de l'environnement souterrain.
Ce rapport, qui fournit de brèves informations sur la houillère d'Armutçuk de TTK, insiste particulièrement sur les facteurs cruciaux provoquant les risques. Il examine ensuite en détail les précautions spéciales de contrôle de l'environnement minier et celles visant à minimiser l'inflammation de l'accumulation possible de méthane. Enfin, il passe en revue les modifications les plus récentes des méthodes de travail, telles que le remblayage hydraulique.

NACH DER VERHEERENDSTEN GRUBENEXPLOSION IN DER GESCHICHTE DER TÜRKEI EINGEFÜHRTE VERBESSERUNGEN.

Prof. Dr. Dr. Vedat Didari, Mesut Öztürk

Die Armutcuk Zeche des türkischen Kohleunternehmens (TTK) in Zonguldak in der Türkei wurde am 7. 3. 1983 von einer schweren Schlagwetterexplosion betroffen. Dabei verloren 102 Bergleute ihr Leben und 86 wurden verletzt. Die Hauptursache für die Explosion war die Entzündung einer Methangas-Ansammlung in der Bandstrecke, außerdem wurden Anzeichen einer sekundären Kohlestaubexplosion entdeckt.

Der Sicherheitsdienst der TTK hat in letzter Zeit neue Projekte entwickelt, die unter anderem die Überarbeitung der Abbaumethoden, Verbesserungen des Bewetterungssystems und die Überwachung der gesamten Grubenwetter einschließen.

Der Vortrag gibt eine kurze Beschreibung der Armatcuk Grube von TTK mit besonderer Betonung der kritischen Faktoren, welche Gefahren verursachen. Anschließend werden die speziellen Vorsichtsmaßnahmen zum Zweck der Kontrolle der Grubenwetter und der Minimierung der Zündgefahr von etwaigen Methangas-Ansammlungen beschrieben. Abschließend werden die neuesten Änderungen der Arbeitsmethoden, wie z. B. : das hydraulische Hinterfüllen, erklärt.

УСОВЕРШЕНСТВОВАНИЯ, ДОСТИГНУТЫЕ НА ОДНОЙ ШАХТЕ ПОСЛЕ НАИБОЛЕЕ КАТАСТРОФИЧЕСКОГО В ИСТОРИИ ТУРЕЦКОЙ УГОЛЬНОЙ ПРОМЫШЛЕННОСТИ ВЗРЫВА

Ведат Дидари и Мезут Озтурк

Седьмого марта 1983 года произошел огромный взрыв на угольной шахте Армутцук в Зонгулдаке, Турция, принадлежащей Турецкой Антрацитовой компании (ТТК). Погибло 102 человека, и 86 человек получили увечья. Главной причиной катастрофы было возгорание метана, скопившегося в изогнутых участках внутришахтных подъездных путей. В добавление были отмечены вторичные по характеру взрывы угольной пыли. Недавно служба техники безопасности ТТК произвела различные улучшения, включающие модернизацию методов добычи, усовершенствование вентиляционной системы и введение постоянного наблюдения за состоянием окружающей среды под землей. В настоящем докладе содержится краткая информация о шахте Армутцук, при этом особое внимание уделяется критическим факторам, обусловливающим угрожающую обстановку в этой шахте. Детально описываются специальные меры предосторожности, принятые в этой шахте, меры по контролю окружающей среды и уменьшению возможности возгорания скоплений метана. В заключение предлается объяснение таких недавних изменений в методах проведения горных работ, как применение гидравлического заполнения.

土耳其煤矿最惨重的爆炸事故后的改进

费达特 的达瑞, 墨苏 奥士土耳克

１９８３年３月７日土耳其宋古达土耳其硬煤公司（ＴＴＫ）的阿莫特士克煤矿发生了一起大爆炸。其中１０２名矿工死亡，８６人受伤。事故主要是由于聚在夹层巷道中的甲烷点燃所致。次级的煤尘亦曾产生。

近来ＴＴＫ的安全处进行了采矿方法的重整，通风设备的改进和地下矿环境的侦测等等方案。

本文简单地叙述了ＴＴＫ阿莫特士克煤矿里的情形，特别着重于导致危险状态的关键因素。然后详细地说明了控制煤矿环境和尽量减少甲烷积聚引燃的预防措施。最后解释近来开采的改进办法例如水力充填等。

СОВЕРШЕНСТВОВАНИЕ СПОСОБОВ БОРЬБЫ С ПЫЛЬЮ И ПЫЛЕВЗРЫВОЗАЩИТЫ НА УГЛЕДОБЫВАЮЩИХ ПРЕДПРИЯТИЯХ СССР

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Аннотация

В угольной промышленности СССР непрерывно совершенствуются способы и средства борьбы с пылью с целью защиты шахтеров от заболеваний, пылевзрывозащиты и охраны окружающей среды на подземных и открытых горных работах. Выполненные научные исследования позволили усовершенствовать средства предварительного увлажнения угольного массива, создать пылевзрывозащитные системы орошения для выемочных и проходческих комбайнов, развить пенный способ пылеподавления, разработать схемы нагнетательно-всасывающего проветривания тупиковых выработок и др.

На открытых горных работах наметились технические решения по снижению пылевыделения при всех технологических процессах добычи угля в различных климатических условиях СССР, реализация которых в промышленность является первоочередной задачей для создания "экономически чистых" угольных разрезов.

Необходимость обеспечения уровней эффективности комплекса противопылевых мероприятий на угольных шахтах 99% и более потребовало широких исследований механизма связывания, осаждения и улавливания пыли, результаты которых послужили основой для разработки принципиально новых решений по технологии и средствам борьбы с пылью.

Из множества научных результатов следует отметить:

разработку методологии оценки пылевыделяющей способности угольных пластов и вмещающих пород и установление области применения различных способов борьбы с пылью в очистных и подготовительных забоях;

разработку теоретических основ взаимодействия жидкостей с угольными массивами и пылевыми частицами углей различной степени углефикации на основе изучения их физико-химических и электрофизических свойств, что позволило создать новые высокоэффективные пылесвязывающие вещества и усовершенствовать технику и технологию предварительного увлажнения угольных пластов, орошения источников пылевыделения, пенного способа пылеподавления, безыскровой подзарядки воды и др.;

разработку научных основ борьбы с пылью при передвижке секций механизированных крепей, которые явились базой для новых технических решений;

научное обоснование экранирования различных источников пылевыделения, создание средств пылеотсоса и пылеулавливания;

установление влияния концентрации парамагнитных центров в угольной пыли различных размеров, вещественного состава и ста-

дии метаморфизма на пневмокониозоопасность горных работ, что позволяет изыскать принципиально новые методы активного воздействия на носителей фиброгенной активности угольной пыли - неспаренные делокализованные электроны с целью снижения их агрессивности и повышения эффективности профилактических мероприятий.

С целью повышения эффективности предварительного увлажнения угольного массива проведены комплексные исследования фильтрационных и хрупко-пластических свойств, разрушаемости и увлажняемости углей различной стадии метаморфизма, в результате которых выявлены три группы угольных пластов и установлены особенности их увлажнения:

I группа - пласты с углями низкой стадии метаморфизма (с содержанием летучих веществ $V^r > 30\%$ и удельным водопоглощением пластов $\Psi_0 = 0{,}03-0{,}06$ см/МПа.с), отличающиеся высокой прочностью, пониженной хрупкостью, повышенной пластичностью, слабым раскрытием прирожденных трещин и низкой водопроницаемостью; для эффективного увлажнения необходимо применять режим с длительным контактом нагнетаемой жидкости с массивом без его разрыва одиночными трещинами, что возможно обеспечить путем многократного нагнетания жидкости в пласт насосными установками с автоматическим регулированием параметров нагнетания при давлениях, исключающих гидроразрыв пласта;

II группа - пласты с углями средней стадии метаморфизма (с $V^r = 10-30\%$ и $\Psi_0 = 0{,}04-0{,}08$ см/МПа.с), отличающиеся низкой прочностью, высокой пластичностью, хорошо развитой трещиноватостью и высокой водопроницаемостью, требуют низконапорного увлажнения при пониженных расходах жидкости, что во многих случаях обеспечивается безнасосной подачей воды непосредственно от противопожарно-оросительного водопровода, или насосными установками с автоматическим регулированием параметров нагнетания;

III группа - пласты с углями высокой стадии метаморфизма (с $V^r < 10\%$ и $\Psi_0 = 0{,}01-0{,}05$см/МПа.с), отличающиеся высокой прочностью и хрупкостью, низкой пластичностью, хорошо развитой и легко раскрывающейся трещиноватостью, низкой водопроницаемостью; требуют высоконапорного увлажнения путем непрерывного нагнетания жидкости в пласт под давлением на пределе упругих деформаций или многоциклического нагнетания жидкости в пласт с переменным темпом и периодическим повышением давления до гидроразрывных величин с последующим длительным насыщением жидкостью растрескавшегося массива. Для повышения эффективности предварительного увлажнения угольного массива необходимо применять водные растворы поверхностно активных веществ, повышающих смачиваемость углей. Разработана методика дифференцированного выбора жидкостей с учетом эффективности их взаимодействия с различного типа углями и установлены требования к свойствам жидкостей для предварительного увлажнения углей различной стадии метаморфизма (поверхностное натяжение, постоянная капиллярного впитывания и интенсивность испарения).

На основании этих исследований освоено серийное производство насосных установок УНР с автоматическим регулированием параметров нагнетания и малотоксичных смачивателей "синтанол ДТ-7" в жидкости и СТС в твердом виде.

Шахтные испытания показали, что применение новых режимов нагнетания и рекомендованных смачивателей обеспечивают снижение запыленности воздуха в очистных забоях после увлажнения угольного массива до 70-80%.

Орошение источников пылевыделения является основным способом борьбы со взвешенной угольной и породной пылью. Преимуществами способа орошения являются простота конструкции, малые габариты оросителей и высокие эксплуатационные качества. Существенным недостатком орошения является недостаточная эффективность (80–90%).

При комбайновой выемке угля нашли применение низконапорное орошение, пневмогидроорошение, высоконапорное орошение, орошение низконапорными и высоконапорными эжекторами.

Перспективными являются электризующие оросители, которые подзаряжают безискровым способом орошающую жидкость и при взаимодействии с электрозаряженными пылевыми частицами повышают эффективность пылеподавления. Созданы оросители: ФЭБПВ с электретными покрытиями поверхности, промышленные испытания которых подтвердили повышение эффективности в 1,5–2,5 раза по сравнению с обычными форсунками.

Пневмогидроорошение с аппаратурой автоматического управления подачей жидкости сжатого воздуха в систему орошения комбайна широко и эффективно применяется на шахтах Карагандинского бассейна, где в забоях имеется сжатый воздух.

Решающее значение для совершенствования систем орошения выемочных комбайнов для пластов мощностью более 1,6 м имеет переход на высоконапорное орошение (давлением 8–10 и более МПа). Даже при применении только внешней системы высоконапорного орошения с давлением до 10 МПа эффективность орошения повышается с 88 до 98% при снижении на 30% удельного расхода воды на пылеподавление.

Дальнейшим шагом по пути снижения запыленности воздуха и пылевзрывозащиты в очистных забоях на пластах мощностью более 1,6 м является переход на высоконапорное орошение с подачей жидкости на след резца и на применение принципиально новых шнековых исполнительных органов механогидравлического типа, в которых водяные струи играют роль не только пылеподавителя, но и режущего инструмента.

На тонких пологих пластах, где орошение в традиционном виде приводит к переувлажнению почвы и кровли и ухудшает гигиенические условия труда, лучшие результаты получены при применении эжекторов, которые являются пылеотсасывающе-оросительными аппаратами.

Применение на тонких выемочных комбайнах двухсторонних эжекторов, которые имеют два раструба с оросителем в каждом при одном отсасывающем окне и в которых запыленный воздух с места выгрузки угля на конвейер засасывается под действием факелов оросителей в раструбы, смачивается и в виде пылеводо-воздушного аэрозоля подается на работающие шнеки, где осаждает образующуюся пыль, обеспечило эффективность при низконапорном орошении (1–1,2 МПа) на комбайне РКУ-13 – 92,3% и при высоконапорном орошении (8 МПа) на комбайне 2К-52м – 97,7%. В последнем случае удельный расход воды снизился на 20–25% по сравнению с заводской (типовой) системой орошения.

Борьба с пылью на механизированных крепях осуществляется применением апробированных решений по технологии передвижки секций уплотнению межсекционных пространств, смачиванию перекрытий и орошению обрушаемой породы.

На тонких пологих и крутых пластах (мощностью менее 1,3 м) хороший эффект пылеподавления получен при применении пенного способа. Для этих целей используется пена кратностью 100-200 единиц, что достигается при расходе пенообразователя 0,1-0,2 кг на 1 т добываемого угля. Для обеспечения способа созданы малотоксичные пенообразователи "Углепен" в жидкой и "Углепен-Т" в твердой товарной форме, а также оборудования для их дозирования и пенообразования.

В комбайновых подготовительных забоях нормальные санитарно-гигиенические условия труда обеспечиваются путем комплексного применения орошения и нагнетательно-всасывающего проветривания подготовительных выработок.

Все проходческие комбайны оснащаются системами орошения с подачей жидкости давлением 1,5-2 МПа под резцы. Создаются системы высоконапорного пылевзрывозащитного орошения с контактной подачей жидкости к резцам под давлением 10 МПа и выше.

Теоретические и экспериментальные исследования пылеаэродинамики при различных технологических процессах подземной угледобычи послужили основой для разработки аэродинамических способов борьбы с пылью с использованием нагнетательно-всасывающих схем проветривания тупиковых забоев применением различных типов укрытий, аэродинамических завес и экранов, отсосом и очисткой воздуха от пыли в малогабаритных пылеуловителях.

С вводом в эксплуатацию мощных угольных разрезов в районах с резкоконтинентальным климатом и при наличии вечномерзлых пород приобрела остроту проблема борьбы с пылью при различных технологических процессах открытой добычи угля. Сложность ее решения обусловлена огромными единичными объемами перерабатываемой горной массы, отсутствием во всем мире каких-либо приемлемых технических решений по этому вопросу, специфическими свойствами горных массивов и тяжелыми климатическими условиями, ограничивающими потребление воды для пылеподавления.

Впервые в мире разработаны комплексные системы пылеподавления для высокопроизводительных роторных эскаваторов (производительностью 2500-5250 м^3/ч), способы и средства покрытия экскавируемой горной массы искусственным снегом и автодорог пылесвязующими веществами. Все кабины горнорабочих оснащены системами пылеочистки и кондиционирования.

Созданы уникальные системы аспирации, осаждения и улавливания пыли на сложных узлах погрузки угля в железнодорожные полувагоны и на приемных ямах выгрузки угля при автомобильном транспорте.

Наряду с этим задача еще полностью не решена. Экспериментальные работы направляются на изыскание эффективных способов предварительной обработки горных массивов и разрыхленной горной массы, предупреждения перехода пыли в атмосферу при взрывных работах, погрузочно-разгрузочных операциях, транспортировании и складировании угля и вскрышных пород.

Наметилось несколько технических решений по снижению пылевыделяющей способности углей: размораживание мерзлых углей паром, токами высокой частоты, увлажнение разрыхленной горной массы, применение связующе-смачивающих химических реагентов, заблаговременное предварительное увлажнение

угольного массива в летний период, гравитационное насыщение разрыхленной горной массы и т.п. Промышленные испытания в ближайшее время покажут их эффективность и экономическую целесообразность.

Одним из эффективных способов борьбы с пылью на разрезах при отрицательных температурах воздуха является пылеподавление искусственным снегом.

Как показали исследования на разрезе "Нерюнгринский" в Якутии, пылеподавление искусственным снегом может осуществляться путем воздействия на взвешенную в воздухе пыль, и экранирования разрыхленной горной массы путем покрытия ее снегом перед экскавацией и погрузкой. Испытания передвижной снегогенераторной установки показали, что запыленность воздуха в рабочей зоне экскаватора ЭКГ-8И при ее применении снизилась на 96,5% (с 2000 до 70 мг/м^3 по общей массе пыли). Этот способ пылеподавления должен найти широкое применение на угольных разрезах Якутии, Забайкалья и Дальнего Востока в условиях малоснежных морозных зим.

Большие уровни запыленности воздуха наблюдаются на углевозных автодорогах. Для борьбы с пылью в этих условиях рекомендованы пылесвязывающие средства на основе компонентов, являющихся промпродуктами нефтеперерабатывающих заводов: универсин-А(летний), универсин-З (зимний при $t > -30^{\circ}C$) и универсин-С (северный при $t < -30^{\circ}C$).

Промышленное применение универсина-А на разрезах Кузбасса показали, что при оптимальном расходе универсина (0,5-2 л/м^2) запыленность воздуха при движении автомобильного транспорта снижается до санитарных норм на срок 5-I5 суток. При применении универсина-З на разрезах"Центральный" и "Богатырь" ПО"Экибастузуголь" эффективный срок обеспыливания автодорог составил I5-20 суток при удельном расходе универсина 0,8-I,2 л/м^2. Испытания универсина-С на разрезе "Нерюнгринский" показали следующие результаты: запыленность воздуха у обочины автодорог снизилась с 90-I60 до 3-I2 мг/м^3, удельный расход универсина-С составил I,5-2 л/м^2, срок эффективного действия колебался в пределах 5-I5 суток в зависимости от интенсивности движения автотранспорта.

Приведенные результаты свидетельствуют о целесообразности применения различных видов универсина для борьбы с пылью на технолс - гических автодорогах. Наряду с этим нельзя отказываться от гидрообработки автодорог, что в ряде случаев может быть оправдано с точки зрения эффективности экономичности.

На ряде угольных разрезов ("Бородинский" ПО"Красноярскуголь", "Восточный" ПО"Экибастузуголь" и др.) предусмотрено применение магистральных конвейерных линий значительной протяженности с большим количеством перегрузочных пунктов, которые могут быть интенсивными источниками пылевыделения. В этих условиях необходимо укрывать конвейерные линии легкоразборными укрытиями.

На пунатах перегрузки наряду с их укрытием должно применяться орошение с автоматически регулируемым расходом воды, или пылеотсос и сухое пылеулавливание.

Для обеспечения сухого пылеулавливания найдены оригинальные технические решения, которые осуществляются в настоящее время в экспериментальные модели.

Таким образом, практически все пылящие технологические узлы открытой угледобычи охвачены кругом исследований и для большинства из них (за исключением взрывных работ) найдены приемлемые технические решения пылеподавления. Ближайшая задача - довести эти решения до практической реализации и широкого применения на угольных разрезах.

IMPROVEMENT OF DUST CONTROL AND DUST-EXPLOSION PROTECTION TECHNOLOGY AT USSR COAL MINES

Yu. Smorchkov, I. Ishchuk

Increased face outputs, high-productive equipment, and complicated geological and mining conditions require steady improvement of dust control.

Dust control techniques based on fluid use involve water or wetting-agent solution infusion of coal seams, spraying at sites of dust generation, rock mass treatment with foams, and dust collection. Automatic pump systems control fluid pumping parameters.

Spray systems, installed on coal-winning and heading machines, with operating pressures of 1.5 to 2.0 MPa, have proved inadequate to combat dust hazards. Spray systems with pressures as high as 10 to 15 MPa are being developed, with water directed to behind where the bit strikes. Field tests of water-jet-assisted miners have been completed. Their introduction will lower dust concentrations at faces.

New types of roof supports feature 'skin-to-skin' setting and positive roof contact during the advance cycle. Dust deposited on canopies and generated during roof caving is wetted.

Based on studies of aerodynamic dust properties, dust control techniques have been developed involving combined blowing/exhausting ventilation, various sealing arrangements, as well as air treatment in dust collectors.

Operation of 'clean' surface coal mines, having no negative impact on the environment, is a major challange facing opencast coal mining. Developments already achieved include integrated dust control systems for high-capacity bucket-wheel excavators, methods for covering the rock mass with synthetic snow and techniques for applying dust suppressants to road surfaces. All operators' cabs are equipped with dust protection systems and air conditioners. Nevertheless, the challange of providing 'clean' opencast coal mines remains. Methods are being developed for the pretreatment of rock mass and broken rock, as well as to prevent airborne dust from blasting.

AMELIORATION DE LA TECHNOLOGIE DU CONTROLE DE LA POUSSIERE ET DE LA PROTECTION CONTRE LES EXPLOSIONS DE POUSSIERES DANS LES MINES DE CHARBON D'URSS

Yu. Smorchkov, I. Ishchuk

L'augmentation du rendement des fronts de taille, les équipements à productivité élevée, ainsi que des environnements géologiques et miniers complexes exigent l'amélioration régulière du contrôle des poussières.

Les techniques de contrôle des poussières à base de fluides comprennent l'infusion des couches de charbon par de l'eau ou par une solution d'agent mouillant, l'arrosage aux endroits de formation des poussières, le traitement des masses rocheuses par des mousses, ainsi que la collection des poussières. Des systèmes de pompes automatiques contrôlent les paramètres de pompage des fluides.

Les dispositifs d'arrosage montés sur les machines d'extraction de charbon et sur les machines d'avancement, fonctionnant à des pressions de 1,5 à 2,0 MPa, se sont révélés insuffisants pour combattre les risques dûs aux poussières. On est en train de développer des dispositifs d'arrosage fonctionnant à des pressions allant jusqu'à 10 ou 15 MPA, dont les jets d'eau sont dirigés vers l'arrière du taillant. Des essais sur le terrain ont été achevés, avec des mineurs assistés de jets d'eau. L'introduction de ces méthodes diminuera la concentration des poussières sur les fronts de taille.

Les nouveaux types de soutènements des toits comportent des montages "surface contre surface" et des contacts de toits positifs au cours du cycle d'avancement. On mouille la poussière qui s'est déposée sur les toitures et qui s'est formée au cours des sous-cavages des toits.

On a développé des techniques de contrôle des poussières à partir de l'étude de leurs propriétés aérodynamiques; ces techniques comportent des systèmes de ventilation combinée par soufflage/aspiration, divers systèmes d'étanchéité ainsi que le traitement de l'air dans les collecteurs de poussières.

L'exploitation de mines de charbon à surfaces "propres", qui n'a pas d'impact négatif sur l'environnement, est un important aspect de l'exploitation des mines à ciel ouvert. Parmi les développements déjà opérationnels, citons les systèmes intégrés de contrôle des poussières pour des excavatrices à godets à haute capacité, les méthodes de couverture de la masse rocheuse par de la neige synthétique et les techniques d'application de suppresseurs de poussières sur les surfaces des voies. Toutes les cabines d'opérateurs sont équipées de systèmes de protection contre la poussière et de régulateurs d'air. Toutefois, essayer d'avoir des mines de charbon à ciel ouvert "propres" est encore problématique. On développe à l'heure actuelle des méthodes pour pré-traiter la masse rocheuse et les roches fracturées, ainsi que pour empêcher les poussières en suspension dans l'air de sauter.

VERBESSERUNGEN IN DER STAUBBEKÄMPFUNG UND IN DEM SCHUTZ GEGEN STAUBEXPLOSIONEN IN KOHLEZECHEN DER UdSSR

Yu. Smorchkov, I. Ishchuk

Gesteigerte Förderleistungen, hochproduktive Ausrüstungen und komplizierte geologische und bergbauliche Bedingungen erfordern eine ständige Verbesserung der Staubbekämpfung.

Die auf der Verwendung von Flüssigkeiten basierenden Staubbekämpfungs-Maßnahmen beinhalten das Tränken mit Wasser oder Benetzungsmitteln, Sprühdüsen am Ort der Stauberzeugung, die Behandlung des Gebirges mit Schäumen und die Staubsammlung. Automatische Pumpsysteme kontrollieren die Parameter der gepumpten Flüssigkeit.

Sprühsysteme, die auf Kohlegewinnungsmaschinen und Vortriebsmaschinen angebracht sind und mit einem Druck von 1.5 bis 2.0 MPa arbeiten, erwiesen sich als nutzlos im Kampf gegen den Staub. Es wurden Sprühsysteme entwickelt, die mit einem Druck von bis zu 10 bis 15 MPa arbeiten und bei denen das Wasser in die Schneidspur des Meißels gerichtet wird. Einsatzversuche mit Gewinnungsmaschinen mit Hochdruckwasserstrahl wurden bereits abgeschlossen. Ihre Einführung wird die Staubkonzentration im Streb senken.

Neue Typen von Schildausbauen weisen das sogenannte 'Skin to Skin' -Setzen und einen positiven Kontakt zum Hangenden während des Rückvorganges auf. Staubablagerungen auf dem Kappendach und der Staub, der während des Hereinbrechens des Alten Mannes erzeugt wird, werden befeuchtet.

Auf Grund von Studien über die aerodynamischen Eigenschaften des Staubes wurden Staubbekämpfungsmaßnahmen wie die kombinierte saugende und blasende Bewetterung, verschiedene Abdämmeinrichtungen sowie die Luftaufbereitung in Entstaubern entwickelt.

Der Betrieb eines 'sauberen' Tagebaues ohne negative Einwirkung auf die Umwelt stellt eine große Herausforderung für Tagebaue dar. Die Ergebnisse der bisherigen Forschung beinhalten integrierte Staubbekämpfungs-Systeme für Hochleistungs-Frontlader, Methoden zur Abdeckung der Gesteinsmassen mit künstlichem Schnee und die Behandlung von Straßenoberflächen mit Staubbindemitteln. Alle Fahrerkabinen sind mit Staubschutzsystemen und Klimaanlagen ausgestattet. Trotzdem verbleibt die Herausforderung eines 'sauberen' Tagebaues bestehen. Es werden Methoden zur Vorbehandlung von Steinmassen und gebrochenem Stein sowie zur Verhütung von flugfähigem Staub bei Sprengarbeiten entwickelt.

苏联煤矿中尘埃控制的改进及煤尘爆炸的保护方法

Yu。司莫契可夫，I。益希契克

由于开采出产的增进，应用高产量的设备，和复杂的地质及煤矿条件，尘埃控制便更须逐渐改进。用液体的尘埃控制方法包括用水或浸湿剂溶液浸入煤层，喷溅在起灰的地方，岩石体泡沫处理，及尘埃的聚积。自动泵系统可以控制抽吸参数。喷溅系统装在采煤机上，作用压力为1。5到2。0兆帕斯卡，对于尘埃危害不足以阻抑。压力高达10到15兆帕斯卡的喷溅系统亦曾使用过，这里水引到钻头后面。用水喷射辅助的开矿机曾做个实地试验。它们的应用可以减少采矿面尘埃的浓度。新的坑顶支撑在开采过程中用“面层对面层”安置及肯定的坑顶接触。积在篷顶上的尘埃和坑顶塌落时产生的尘埃都经润湿。根据尘埃的空气动力性质，尘埃控制包括联合吹抽通风方法，各种封闭设备及积尘器里的气体处理。对于环境无坏影响的“清洁”的地面矿场仍是露天矿场一个困难的问题。目前已经达到的成果包括高产量的丰轮挖掘机的整体的尘埃控制系统，用人造雪遮盖岩石体，及用尘埃压抑剂撒在路面。所有操作员的轿厢都有防尘装置及空调设备。然而“清洁”的露天矿场仍不能实现。岩石体和破损的岩石的预先处理以及防止空中尘埃爆炸的方法正在研究发展中。

FLAMMABILITY AND EXPLOSION PROPAGATION OF METHANE-COAL DUST HYBRID MIXTURES

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ABSTRACT

This paper presents the results of research work into the influence of small percentages of methane in the atmosphere on the risk of coal dust explosion. A study is made of the variation in explosibility parameters and the extent of propagation in coal dust explosions due to increasing the methane content in the dispersion air is analysed.

1. INTRODUCTION

Explosion risks in coal mines are becoming increasingly better studied and well-known. As a result of research, new explosion prevention, detection and extinction systems are appearing every year (Winter 1987) (Faber 1985), meaning that our mines are becoming safer.

However, with the current trend to increase production and operate new mines - frequently at greater depths - the risks grow and new dangers appear related with the use of larger capacity equipment (Marchán 1987).

The new equipment, which is more powerful, faster and with a higher mining capacity effectively causes greater gas seepage due to the rapid rate of heading in coal seams, as well as a higher possibility of encountering unforeseen faults and hard ore inclusions, a greater difficulty of providing adequate ventilation at the mine face and, lastly, greater production of coal fines.

This consequently favours the creation of potentially explosive atmospheres, due to the presence of an inflammable gas - firedamp - and a combustible dust - coal.

The situation is even more serious due to the mutual interaction between the coal dust and the methane, which jointly give rise to extremely dangerous hybrid mixtures (Nagy 1961) (Lebecki 1984) (Lunn 1987).

It is consequently necessary to continue safety research in order to reach common criteria and face up to the new risks generated by new technology.

2. COAL DUST CHARACTERISTICS

Four different Spanish coals and coal from Pittsburgh were selected for this study. The specification and analysis of each one is shown in Tables 1 and 2. The particle size was below 74μm in all of them. It can be seen that they all have a high volatiles content, despite being different coal types from different sources.

TABLE 1. Coal specifications

SAMPLE	SOURCE	TYPE
B-1	Asturias	Bituminous coal
B-2	Asturias	Bituminous coal
B-3	Pittsburgh (USA)	Bituminous coal
L-1	Teruel	Lignite
L-2	Teruel	Lignite

TABLE 2. Coal analysis

SAMPLE	CARBON (%)	HYDROGEN (%)	SULPHUR (%)	VOLATILES (%)	ASH (%)	MOISTURE (%)
B-1	81.00	5.50	0.45	33.55	6.05	0.60
B-2	70.10	4.72	1.75	37.65	9.70	0.00
B-3	77.25	5.00	1.00	35.90	6.50	1.60
L-1	60.45	3.95	7.35	33.90	12.45	18.45
L-2	54.20	3.65	6.60	32.30	15.40	7.95

Laboratory test were performed to analyse the behaviour of these coals, determining the explosibility parameters. Explosion propagation tests were also performed in a small scale propagation tube.

2.1. Explosibility parameters

The following explosibility parameters were determined: minimum ignition temperature (MIT), minimum ignition energy (MIE), minimum explosive concentration (MEC), maximum explosion pressure (MEP) and maximum rate of pressure rise (MRPR). The MIT was determined in a Godbert--Greenwald furnace, the MIE and MEC in a Hartman tube and the

MEP and the MRPR in the 20 litre sphere. The new dust dispersor developed by Ciba-Geigy (IEC 1988) was used for the sphere tests. The results of these tests can be seen in table 3.

Grouping the five samples together on the basis of coal type (B-1, B-2 and B-3 for bituminous type coals and L-1 and L-2 for lignite), it can be seen that the parameters are similar for each of the two groups. This fact confirms the similarity of the samples within the groups, a similarity which can also be appreciated when analysing the data of table 2.

Thus it can be seen that although the lignites have a lower MIT than the bituminous coals, for inflammation they need sparks with a higher energy level. In turn, their MEC figures are higher than the ones of the bituminous coals.

At any event, the low level of the MEC and MIE, both for the bituminous coals and lignites, indicates that all of these coals are easily inflammable.

With regard to the results obtained in the 20 litre sphere, the lignites give slightly higher explosion pressures compared to the bituminous coals. In contrast, the maximum gradients are of the same order for all of the samples. Both the maximum pressures and the highest gradients were obtained with dust concentrations of between 500 and 750 g/m^3. These dust concentrations are higher than the stoichiometric concentration, which means that a surplus of combustible material is required to obtain the highest pressure and gradient figures.

TABLE 3. Explosibility parameters

	MIE (mJ)	MEC (g/m^3)	MIT (°C)	MEP (bar)	MRPR (bar/s)
B-1	120	31	560	6.5	340
B-2	66	24	540	6.9	340
B-3	40	24	560	6.8	305
L-1	310	40	460	7.6	263
L-2	100	62	470	8.5	365

2.2. Explosion propagation

The propagation tests were performed in the propagation tube of Laboratorio Oficial Madariaga (Torrent 1987). This device consists of methacrylate tubes with an interior diameter of 192 mm. and a length of two meters. Overall tube length can be increased by coupling several tubes together in series. In the tests carried out this

overall length was four meters. The explosion ignition source was a two litre chamber with a mixture of 40% hydrogen and air. This can be considered as a relatively low ignition source. The samples tested were those classified as L-2 and B-2.

The minimum dust concentration which is capable of propagating an explosion along the full length of the tube was determined. The proportion of steriles ($CaCO_3$) necessary for this propagation not to take place was also determined.

The results show that for a nominal concentration of 340 g/m^3, B-2 coal gives rise to highly powerful explosions, while the explosions of L-2 coal are much weaker. The latter is incapable of total propagation under 250 g/m^3. In contrast, with B-2 coal the dust concentration has to be reduced to 120 g/m^3 before partial propagations occurs.

The greater ease of B-2 coal to propagate explosions is also reflected in the amount of sterile required to impede propagation. Thus, for sample B-2, 35% sterile is enough to impede propagation, while only 20% is required for sample L-2. These two percentages were obtained by using the dust concentration in the tube which gave the most powerful explosions.

Nevertheless, it must be taken into account that these figures are considerably lower than the theoretical values, due to the use of a relatively low energy explosion ignition source.

3. HYBRID MIXTURE CHARACTERISTICS

3.1. Explosibility parameters

The samples tested were those mentioned previously. In order to find out the influence of methane on these explosibility parameters various percentages of this gas were used. In the MIT, MEC and MIE tests, a series of tests with 1%, 1.5% and 2% methane were performed, while percentages of 1%, 2% and 3% were used for the sphere tests.

Figure 1 shows for the three samples of the B series how MEP increases as the methane content rises. The fact that the MEP figures in the absence of methane are not too high indicates that most of the coal dust does not participate in the combustion reaction. Consequently this progressive increase in the MEP means that the presence of methane makes more of the coal react and participate in the combustion.

In contrast, sample L-2 with 0% methane gives rise to a very high MEP, which does not rise as the methane is added. Nevertheless, this does not mean that reactivity has fallen. In contrast, this has also risen, which can be seen by looking at the curves of fig. 2. This figure shows the maximum gradient trend as a function of the methane content for several coals.

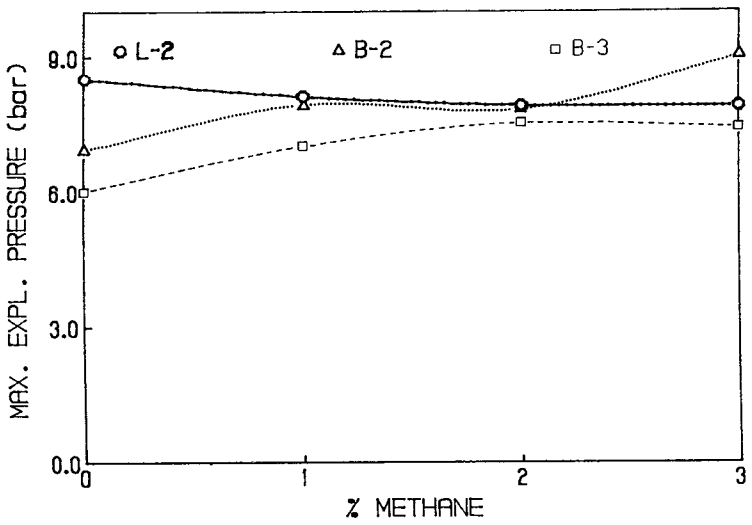
Figures 3 and 4, obtained with the results of tests performed on coal B-2, show how explosion pressure and gradient vary with methane content for dust concentrations of between 20 and 750 g/m^3. These figures also reflect the increase in reactivity mentioned above as

the methane content rises. It can clearly be seen how the maximum pressure and maximum gradient are obtained each time with lower dust concentrations, closer to the stoichiometric concentration. It can also be seen how maximum pressure is obtained at a concentration somewhat lower than that giving the highest gradient. Lastly, for concentrations below that giving the maximum figure - which are concentrations between 20 and 250 g/m³, both the pressure and the gradient increase significantly as the methane content rises.

Another very important parameter is the MIE. The dependence of this parameter on the methane content is shown by figure 5. It is clear that a major reduction occurs when rising from 0% methane to 2%. Thus, e.g. sample L-1 changes from having an MIE of 310 mJ to a figure of only 20 mJ. Another example is that of B-1 coal, where the MIE falls from 120 mJ with air to only 0.8 mJ with 2% methane.

It can be seen that the reduction which takes place is exponential, so that small percentages of methane are sufficient to reduce the MIE by an order of magnitude.

FIGURE 1



3.2. Propagation of the explosion

This section analyses how the amount of sterile needed to impede explosion propagation varies as a function of the methane content in the propagation tube. Coal B-2 was used for these tests. Two series of tests were performed: the first one maintaining constant an amount of coal corresponding to a concentration of 170 g/m³; the second one with an equivalent concentration of 350 g/m³.

The influence of 1% and 2% methane was analysed. The results are shown in figure 6. As the methane content rises, the amount of sterile required also rises. In this way for 170 g/m³ of coal with

FIGURE 2

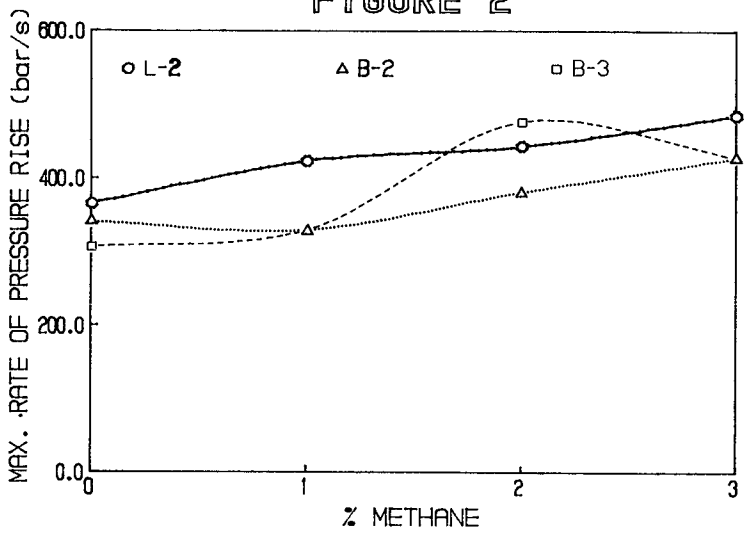


FIGURE 3

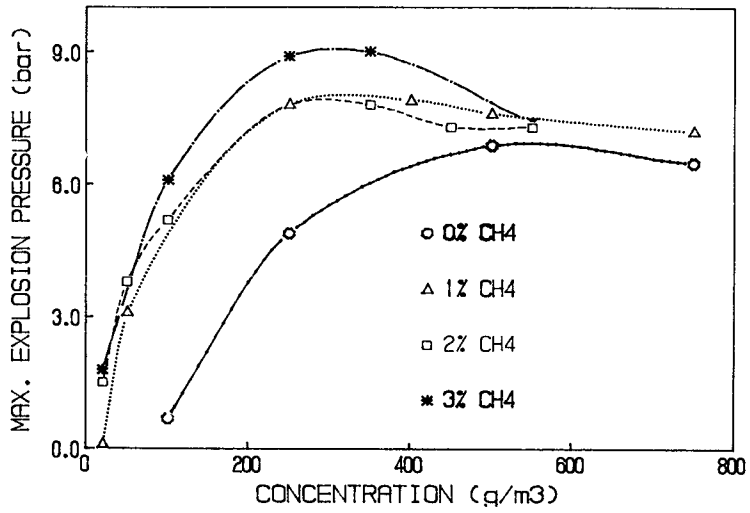


FIGURE 4

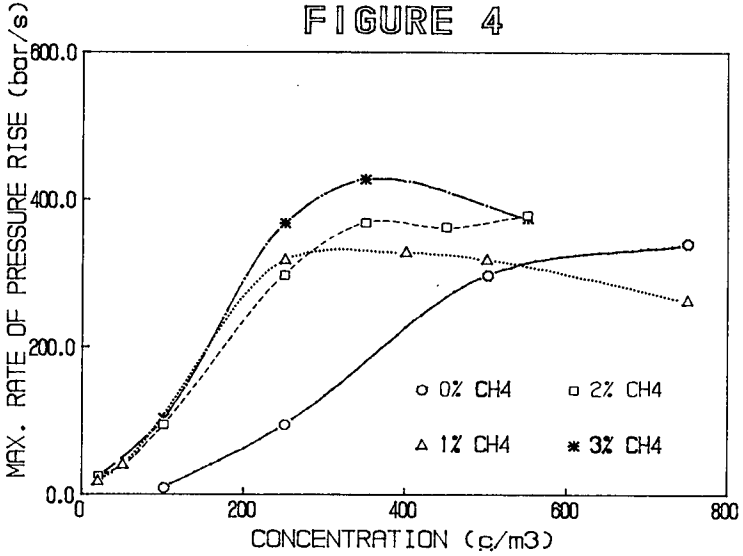
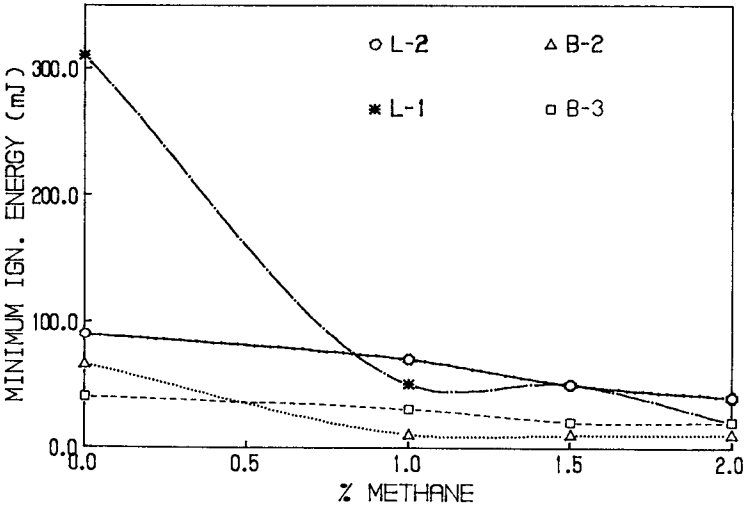


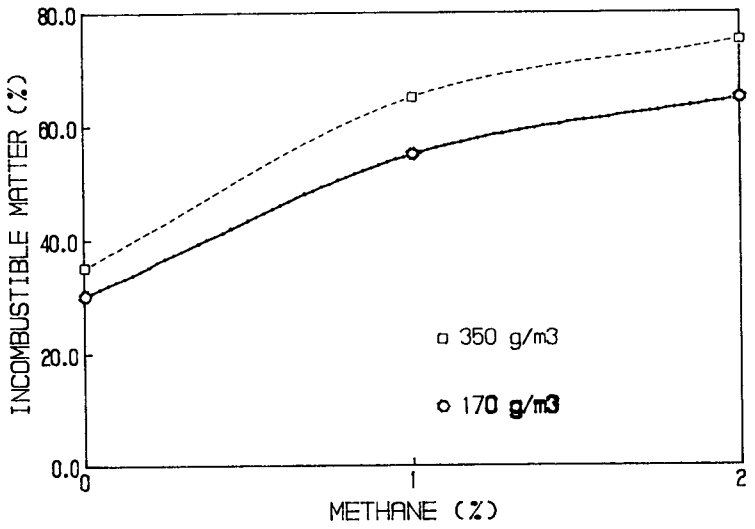
FIGURE 5



1% methane, 55% sterile is enough to disable any propagation. On rising to 2%, the sterile figure goes up to 65%. In the second series of tests with 350 g/m³, with 1% methane 65% sterile was required, whereas with 2% methane, 75% sterile was necessary. If it is taken into account that in the absence of methane in the first case 30% sterile (35% in the second one) was enough, it can be seen that the presence of methane significantly increases the ease with which coal can propagate. This is even more so the higher the methane content present.

Consequently, when designing protection measures to avoid the propagation of an explosion, the possibility of methane being present must be taken into account, or, put another way, the possibility of a coal-methane hybrid mixture being present.

FIGURE 6



4. DISCUSSION AND CONCLUSIONS

The presence of small quantities of methane significantly modifies coal dust behaviour, considerably increasing both sensitivity to inflammation and the severity of the explosion. This occurs for both lignitic and bituminous coals.

It was confirmed that laboratory-scale test equipment can be used to determine explosibility parameters when a certain percentage of methane is used in the dispersion air. Specially noteworthy is the use of the 20 liter spherical unit for the study of hybrid mixtures with the new dust dispersion system.

The propagation tube previously used for research into the degree of propagation of coal dust is still valid for the study of hybrid mixtures, although some components must be remodelled. Subsequent research work will increase explosion ignition source energy in order to analyse its influence on the

degree of propagation and on the percentage of non-combustible matter which must added to the coal dust to avoid propagation of an explosion.

In order to obtain final conclusions it will be necessary to carry out a greater number of experiments and to study other types of coals.

ACKNOWLEDGEMENTS

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INFLAMMABILITE ET PROPAGATION DE MELANGES HYBRIDES DE METHANE-POUSSIERES DE CHARBON

Garcia Torrent, J.; Cantalapiedra Fuch J. (Lom)

La présence de faibles quantités de méthane dans l'air augmente l'inflammabilité des mélanges de poussière de charbon dans l'air. Certains appareils de laboratoire, tels le tube de Hartmann et la sphère 20 1, ont servi à étudier les paramètres d'explosivité. Nous avons analysé plus particulièrement l'énergie de combustion minimum et la concentration explosive minimum.

Puisqu'il est également nécessaire d'étudier comment de faibles teneurs en méthane augmentent le risque de propagation d'une explosion, nous avons conçu et construit un tube de propagation, ou une galerie à petite échelle. Plusieurs paramètres ont été examinés, tels que l'effet de l'énergie de la source de combustion et le pourcentage de matière inerte nécessaire pour éviter la propagation d'une explosion de poussières de charbon ou d'un mélange de poussières de charbon-méthane.

Les résultats ont été comparés à ceux des essais à grande échelle. Puis, nous avons examiné le bien-fondé de la méthode et des appareils d'expérimentation et avons mis en évidence l'importance et les avantages des essais de propagation à petite échelle.

ENTFLAMMBARKEIT UND FORTPFLANZUNG VON METHAN - KOHLESTAUBMISCHUNGEN.

Garcia Torrent, Cantalapiedra Fuchs

Die Anwesenheit von kleinen Methangasmengen in der Luft erhöht die Entflammbarkeit von Kohlestaubmischungen in Grubenwettern. Laboreinrichtungen wie die Hartmann Röhre und die 20 l Kugel werden zum Studium von Explosivitätsparametern benutzt. Dabei wird die Analyse der niedrigsten Zündenergie und der kleinsten explosionsfähigen Konzentration mit besonderer Aufmerksamkeit durchgeführt.

Zur Untersuchung des Einflusses von prozentual geringen Mengenerhöhungen von Methangas auf die Gefahr einer möglichen Explosionsfortpflanzung wurde eine Fortpflanzröhre, oder kleine Strecke, entworfen und gebaut. Verschiedene Parameter, wie der Einfluß der Zündquellenenergie und der Anteil inerter Materie, die zur Verhütung einer Kohlestaubverpflanzung oder Kohlestaub - Methangemisch - Explosion notwendig sind, wurden verglichen.

Ein Vergleich der Ergebnisse von Labor - und Großversuchen wird angestellt, wobei die Wirksamkeit der angewandten experimentellen Methoden und Einrichtungen unter besonderer Betonung der Wichtigkeit und der Vorteile von Fortpflanzungstests in kleinen Testeinrichtungen diskutiert wird.

ВОСПЛАМЕНЯЕМОСТЬ И РАСПРОСТРАНЕНИЕ СМЕСИ МЕТАНА И УГОЛЬНОЙ ПЫЛИ

Гарсия Торрент и Канталапиедра Фук

Присутствие небольших концентраций метана в воздухе увеличивает воспламеняемость смеси воздуха с угольной пылью. Для изучения параметров взрывоопасности были использованы некоторые лабораторные усовершенствования, такие, как трубка Гартмана и 20-ти литровый шар. Особое внимание обращалось на определение наименьшего количества энергии, приводящего к воспламенению, и на нахождение нижнего предела взрывоопасных концентраций. Поскольку было необходимо установить, насколько низкий процент содержания метана в воздухе увеличивает риск распространения взрывов, были спроектированы и сооружены трубки распространения или штольни лабораторного масштаба. Были изучены такие параметры, как влияние энергии источника возгорания и процентное содержание инертного материала, необходимое для остановки распространения взрывов угольной пыли или смеси угольной пыли с метаном. Результаты лабораторных испытаний сравниваются с результатами испытаний, проведенных в естественных условиях, и обсуждаются преимущества и полезность лабораторных исследований. Особо подчеркивается важность и преимущества испытаний на способность огня к распространению, проверенных в лабораторных условиях.

甲烷煤尘混合气体可燃性及其传播

加息亚 托闰德，康塔拉比德拉 福克

马达雷亚加国家实验室

空气中微量甲烷的存在会增加煤尘混合气体的可燃性．一些实验仪器，如哈特曼管和 20 升球，曾被用来研究爆炸性参数．在分析最低点火能量和最小爆炸物浓度时我们曾特别谨慎．因为又要研究甲烷微小的成分是如何会引起爆炸传播的，我们曾设计并建造了一个传播管道或称为小尺度巷道，并对一些参数例如：点火源能量的影响，惰性物质的百分比以避免煤尘或煤尘甲烷混合气体爆炸的传播都曾加以研究．

研究结果曾与实尺度的试验结果相比较．本文并且讨论到实验方法及仪器的优点，并强调以小尺度装置试验传播性的重要及长处．

COAL DUST EXPLOSION RESEARCH IN THE REPUBLIC OF SOUTH AFRICA

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ABSTRACT

Concern about the safety in underground coal mining operations recently led to the establishment of the Kloppersbos research facilities where a giant cylindrical steel gallery, which measures 200 metres in length and 2.5 metres in diameter has been constructed.

The design of this tunnel, which can stand static pressures of up to 2 MPa, as well as its instrumentation and data systems will be discussed.

The research programme includes inter alia

- the development of a locally applicable "standard" test procedure for coal dust explosibility and correlation with results obtained in a 40 litre explosion vessel;
- ranking of South African coals in terms of explosibility index;
- correlating coal properties with explosibility;
- evaluating the efficiency and required level of stone dusting.

The paper will be concluded with a five minute video of a demonstration simulating the effect of an underground explosion hazard.

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Manuscript was not received in time to be included in Proceedings.

RECHERCHES SUR LES EXPLOSIONS DE POUSSIERES DE CHARBON EN REPUBLIQUE D'AFRIQUE DU SUD

J.J. Bensch

Les préoccupations concernant la sécurité des exploitations de mines de charbon ont abouti récemment à l'établissement des installations de recherches de Kloppersbos, où une galerie cylindrique géante a été construite, mesurant 200 mètres de long sur 2,5 mètres de diamètre.

Le rapport examine la conception de ce tunnel, qui peut supporter des pressions statiques allant jusqu'à 2 MPa, ainsi que les systèmes d'instrumentation et de données.

Le programme de recherches comprend, entre autres :

- le développement d'une procédure d'essais "standard" applicable localement, sur l'explosivité des poussières de charbon et la corrélation avec les résultats obtenus dans un récipient d'explosions de 40 litres;
- la classification des charbons sud-africains en termes d'indice d'explosivité;
- la corrélation entre les propriétés du charbon et l'explosivité;
- l'évaluation de l'efficacité et du niveau de poudrage minéral exigés.

Notre rapport se conclut par une vidéo de démonstration de 5 mn, simulant l'effet d'un risque d'explosion souterraine.

UNTERSUCHUNG VON KOHLESTAUBEXPLOSIONEN IN DER REPUBLIK SÜDAFRIKA

J. J. Bensch

Sorge um die Sicherheit in Untertage-Kohlegruben führte vor kurzem zu der Gründung der Kloppersbos Forschungsanlage, wo eine enorme, zylindrische Stahlstrecke von 200 Meter Länge und 2,5 Meter Durchmesser gebaut wurde.

Der Vortrag behandelt die Konstruktion dieser Röhre, die einem statischen Druck von bis zu 2 MPA widersteht, sowie ihre Instrumentierung und Datensysteme.

Das Forschungsprogramm umfaßt unter anderem:

- Die Entwicklung eines örtlich anwendbaren 'standard' Testverfahrens für Kohlestaubexplosivität, sowie ihre Übereinstimmung mit Ergebnissen aus einem 40 Liter großen Explosionsbehälter;
- Bewertung der südafrikanischen Kohle in Relation zu dem Explosionsindex;
- Wechselbeziehung zwischen Kohleneigenschaften und Explosivität;
- Auswertung der Effektivität und der erforderlichen Häufigkeit des Einstaubens mit Steinstaub.

Der Vortrag schließt mit einem fünf Minuten dauernden Videofilm, der die Demonstration des simulierten Effektes einer untertagigen Explosionsgefahr beinhaltet.

ИЗУЧЕНИЕ ВЗРЫВОВ УГОЛЬНОЙ ПЫЛИ В ЮЖНО-АФРИКАНСКОЙ РЕСПУБЛИКЕ

Дж.Дж. Бенш

Необходимость решения проблемы безопасности работы в угольных шахтах привела к необходимости создания исследовательского центра в Клоппесбос, где была сооружена гигантская цилиндрическая стальная штольня длиной 200 м и диаметром 2,5 м. В настоящем докладе обсуждается конструкция этого туннеля, который может выдержать давление до 2 МПа, а также вычислительная система и приборы, которыми он оборудован. В программу исследований, среди прочего, включены:
- разработка применимых в Южно-Африканских условиях "стандартных" испытаний на взрывоопасность угольной пыли и корреляция с результатами, полученными во взрывном контейнере объемом в 40 литров;
- классификация угля, добываемого в Южной Африке, по степени взрывоопасности;
- корреляция между свойствами угля и взрывоопасностью;
- рассмотрение эффективности и необходимого уровня концентрации каменной пыли.

В заключение доклада будет показан пятиминутный видеофильм, демонстрирующий моделирование эффекта подземного взрыва.

南非共和国煤尘爆炸的研究

J . J . 班许

由于地下煤矿操作安全问题的考虑，南非设立了克罗帕斯波士研究设备，里面建造了 200 米长，直径为 2．5 米的巨型园柱形钢卷道．

本文将讨论这隧道的设计，仪器设备和数据计录系统．隧道可以承受 2 百万帕斯卡静压力．

研究方案特别包括：

- 发展了可在本地应用的煤尘爆炸性 "标准" 试验程序，及研究它与从 40 升爆炸容器所得结果的相关性；
- 按爆炸性指数排列南非产煤的次序；
- 研究煤性质与爆炸性的相关性；
- 评定其效率及撒岩粉除尘的需要程度．

本文结束时有五分钟录相机影带作模拟地下矿爆炸危害的示范．

ANALYSIS OF BARRIERS BEHAVIOUR IN COAL DUST EXPLOSIONS IN MINES

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ABSTRACT

In underground mines two kinds of protection against coal dust explosion propagation are used:

- neutralization of dust deposits by addition of incombustible dust,
- stone dust and water barriers.

The conditions of use of barriers are presented and their effectiveness is analysed on example of some different real explosions in mines.

It is proved that stone dust barriers, properly used are the most effective mean of protection against coal dust explosion in mines.

The water barriers are as effective as stone dust barriers but sometimes, with weak explosions are less efective. It is detaily presented analyse of behaviour of stone dust barriers during explosion on the base of experiments and investigation after explosion in mines.

INTRODUCTION

Nearly 20 years are over since the book of Prof.Cybulski - director of many years of the Experimental Mine Barbara - was edited, where he included his work and summarized his investigations on coal dust explosions and their suppression.

On the basis of those research results Polish safety regulations have been elaborated regarding prevention and suppression of coal dust explosions.

In the author´s opinion it is time events to evaluate if the protection system introduced into the mines against coal dust explosion passed the examination.

From the point of view of mining pracitce this system is very good.

This is made clear by mining regulations, which in the last 20 years were actualized.
In the chapter of regulations relating to coal dust explosions combating no essential changes were made. Changes concern only two questions;
the potential source of explosion initiation is not only the coal face.
Greater attention is paid to combating dust formation during mining operations, which is connected with the fact, that 20 years ago mechanization has been not so wide defeloped as it is today. The argument, that the mines protection system against coal dust explosion is regarded as effective comes from the analysis of coal dust explosions, which occured in the discussed period.
The barriers passed the examination despite of the faults of the human factor, which was the cause of any analyzed accident. Therefore it is purposeful to describe the behaviour of barriers in disasters of coal dust explosions, which occured in the last period of twenty years.

The effectiveness of barriers prior to this period has been presented in different places of his work by Prof.Cybulski (1).

PHILOSOPHIE OF PROTECTION OF POLISH COAL MINES AGAINST COAL DUST EXPLOSIONS

Prevention of coal dust explosions corresponds with methods employed in the world mining industries.

The methods of prevention of coal dust explosions propagation and their stopping is based mainly on stonedusting and water and stonedust barriers.

As a protection of workings minimum 200 m long stonedusted or washed zones are used in any direction from faces, places with shotfiring operations, or from places where great dust accumulations are caused by mining machines and also from active mine fires fields.

In gassy fields and in regions of increased bumps hazard the regulations mention several cases, where stonedusted zones should be used. There is no need to employ stonedusting zones at drivage of headings in rocks, in extraction workings, where the coal dust is naturally protected by water or incombustibles or in workings where dust accumulations do not exceed 30 g/m3.

The protection zones should contain minimum 70% incombustibles in non-gassy fields and 80% in gassy fields.

At increased gassiness the incombustibles should be added up to 85% in the protection zone and up to 50% in other workings beyond this zone.

The last line of defense against coal dust explosions are water and stonedust barriers.

The main barriers are located on the inlet and outlet of every ventilation district or in all other workings connecting ventilation regions.

Subsidiary barriers are located within the ventilation districts. These barriers protect exploitation faces and headings, places where are installed mining machines producing dust accumulations, and active mine fire fields.

Additionally in gassy fields barriers are installed every 200 m on the whole lenght of air ducts, in workings with electrical equipment where methane concentrations exceed 0.5%, in headings, with zones of great dust accumulations over 0.5 kg/m3.

Subsidiary water or stonedust barriers should be located 60-200 m from the place where the initiation is possible

The heading on the whole lenght of the barrier should be stonedusted minimum up to 80% incombustibles. In the case of the water barrier the heading should be wetted or stone dusted also minimum up to 80% incombustibles.

Stonedust and water barriers are used with 200 kg/m2 in non-gassy mines and 400 kg/m2 in gassy mines.

The Polish mining industry uses 154.000 tonnes of rockdust per year.

The mines in the Rybnik-Jastrzębie region are using about 45.000 tonnes of rockdust for stonedusting of protection zones and 28.000 rock-dust for barriers.

In this mines are installed:

2790 stone-dust barriers,
231 part-width barriers,
13 distributed barriers,
127 water barriers.

The lenght of stone-dusted roadways is:

554 km up to 50% incombustibles,
29 km up to 70% incombustibles,
890 km up to 80% incombustibles,
56 km up to 85% incombustibles.

For 1000 tonnes of output 1.9 t of stone-dust is used.

The maximum settlement of dust in this mines is 60 g/m2/24 hours which corresponds to 40 g/m3/24 hours,

(the rate of dust quantity per 24 hour fits with the lower limit of explosibility).

ANALYZIS OF BEHAVIOUR OF BARRIERS IN COAL DUST EXPLOSIONS IN POLAND IN THE LAST TWO DECADES

The restraint of this paper does not allow to analise wider the behaviour of barriers in explosions, which occured in Poland in the last two decades.

Every accident was followed by a searching inquiry and post explosion investigations of a commission called by the Chief Inspector of Mines. In that inquiry participated always representatives from the Experimental Mine Barbara. Explosions were investigated separately by the Experimental Mine Barbara and the results make an integral part of the end protokoll of the commission.

There exists a wide material under investigation from the analyzed period, therefore the behaviour of barriers in occured explosions have been listed hereunder.

It was resigned from presentation the course of explosions on figures., because it is impossible to show the way of the flame in colors. This will be possible during the presentation of this paper at the Conference.

From this analysis conclusions can be drawn, that the barriers are an effective means for stopping coal dust explosions, despite of not fulfilling other requirements of regulations. Without doubt in several cases the range of the flame could be limitted as well as fatalities if the protection zones had the proper percentage of incombustibles and the barriers were properly located and installed.

It should be underlined that as a result of analysing particular accidents several new requirements of regulations have been introduced at once. These requirements have been dealing not only with elimination of explosion initiators but also with the strategy of their application. As an example can serve the initiation of a coal dust explosion by the cable box at the "1 Maja" colliery in 1974 in a 600 m distance from the face! Proper requirements were introduced presented in the foregoing chapter.

The obtained experiences have initiated several researches. As an example can serve a device for the measurements of incombustibles - Inflabr (3).

The introduction into the Polish mines a device which enables the estimation if the protection zones are properly dusted, goes far towards the effective combating of coal dust explosions.

The introduction into the mines pneumatic charging of long blastholes enabled in several cases to eliminate the detonating cord independently from rigoruos regulations in this field.

The two last explosions have been caused by welding operations. This was the reason for considerable elimination of welding operations from underground.

ANALYSIS OF COAL DUST EXPLOSIONS OCCURED IN POLAND IN THE LAST TWO DECADES

1. Date of accident: 22.10.1972
   Colliery: "Szczygłowice"
   Place of initiation: face of the longwall intake
   Seam: non gassy
   Ventilation: circulating ventilation
   Kind of explosion: coal dust explosion
   Initiator: blasting of rock explosives and detonating cord beyond the blasthole
   Path of the flame: 200 m
   Number of fatalities: none
   Evaluation of the operation of barriers: positive
   Remarks: The explosion stopped by the barrier despite of low incombustible content in the heading

2. Date of accident: 18.8.1973
   Colliery: "Wieczorek"
   Place of initiation: face of pillar heading
   Seam: gassy
   Ventilation: auxiliary
   Kind of explosion: coal dust explosion
   Initiator: detonating cord fired beyond the blasthole
   Path of the flame: 87 m
   Number of fatalities: none
   Evaluation of the operation of barriers: explosion didn't reach the barrier
   Remarks: weak explosion, low dustines, fading of explosion because of high content of incombustibles.

3. Date of accident: 30.1.1974
   Colliery: "1 Maja"
   Place of initiation: longwall intake
   Seam: gassy
   Ventilation: circulating
   Kind of explosion: coal dust explosion
   Initiator: explosion of a cable box 600 m from the longwall
   Path of the flame: 645 m
   Number of fatalities: 4
   Evaluation of the operation of barriers: positive
   Remarks: great quantities of not protected dust at the ventilation stopping

4. Date of accident: 7.11.1974
   Colliery: "Anna"
   Place of initiation: longwall
   Seam: gassy
   Ventilation: circulating
   Kind of explosion: methane and coal dust explosion
   Initiator: probably fire as a result of short circuit and a delayed methane explosion
   Path of the flame: 1940 m
   Number of fatalities: none
   Evaluation of the operation of barriers: positive
   Remarks: upon a strong shock and fractures followed ventilation disturbances, heavy methane emissions, the remained barriers limitted the range of the flame.

5. Date of accident: 25.3.1976
   Colliery: "Mysłowice"
   Place of initiation: heading face
   Seam: gassy
   Ventilation: auxiliary
   Kind of explosion: coal dust explosion
   Initiator: detonating cord fired beyond the blasthole
   Path of the flame: 289 m
   Number of fatalities: 3
   Evaluation of the operation of barriers: positive

6. Date of accident: 5.12.1978
   Colliery: "Dymitrow"
   Place of initiation: heading face
   Seam: gassy
   Ventilation: auxiliary
   Kind of explosion: coal dust explosion
   Initiator: detonating cord fired beyond the blasthole
   Number of fatalities: 3
   Evaluation of the operation of barriers: positive
   Remarks: strong explosion stopped by the first barrier.

7. Date of accident: 10.10.1979
   Colliery: "Dymitrow"
   Place of initiation: heading face
   Seam: gassy
   Ventilation: auxiliary
   Kind of explosion: coal dust explosion
   Initiator: deflagration of detonator
   Path of the flame: 1900 m
   Number of fatalities: 34
   Evaluation of the operation of barriers: not all basriers operated positive
   Remarks: long flame range as a result of incomplete barriers and not properly stone-dusted protection zones

8. Date of accident: 11.2.1985
   Colliery: "Niwka-Modrzejów"
   Place of initiation: coal container
   Seam: gassy
   Ventilation: circulating
   Kind of explosion: coal dust explosion

Initiator: welding operations caused fire in the coal container
Path of the flame: 900 m
Number of fatalities: 4
Evaluation of the operation of barriers: positive
Remarks: the first barrier too close to the container, which was installed between ventilation stoppings. The next barriers stopped the weakened explosion by the first barriers.

9. Date of accident: 4.2.1987
Colliery: "Mysłowice"
Place of initiation: crossing of longwall and intake
Seam: gassy
Ventilation: circulating
Kind of explosion: methane and coal dust explosion
Initiator: welding operations initiated a CH_4 explosion
Path of the flame: 1574 m
Number of fatalities: 18
Evaluation of the operation of barriers: positive
Remarks: Coal dust explosion initiated at the subsidiary barrier stopped by main barriers, Bad protection zones.

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ANALYSE DES VERHALTENS VON EXPLOSIONSSPERREN BEI KOHLESTAUBEXPLOSIONEN IN KOHLEGRUBEN.

Pawel A. Krystolik

Die Ausbreitung von Kohlestaubexplosionen in Untertagegruben wird auf zwei verschiedene Weisen verhütet:
- Neutralisierung von Staubablagen durch nichtbrennenden Staub;
- Steinstaub - und Wassertrogsperren.

Der Vortrag behandelt die Bedingungen für den Einsatz von Sperren, die anhand der Analyse von tatsächlich in Gruben aufgetretenen Explosionen auf ihre Effektivität untersucht worden sind.

Es ist bewiesen, daß Steinstaubsperren bei richtiger Anwendung die wirksamste Methode zum Schutz gegen Kohlestaubexplosionen darstellen.

Die Wassertrogsperren sind ebenso wirksam wie die Steinstaubsperren, allerdings sind sie bei schwachen Explosionen manchmal weniger effektiv. Das Verhalten von Steinstaubsperren während einer Explosion wird auf der Basis von Experimenten und Untersuchungen nach einer Explosion im einzelnen beschrieben.

ANALYSE DU COMPORTEMENT DES PROTECTIONS DANS LES EXPLOSIONS MINIERES DE POUSSIERES DE CHARBON

Pawel A. KRZYSTOLIK

Dans les mines souterraines, on utilise deux types de protections contre la propagation des explosions de poussières de charbon :
- neutralisation des dépôts de poussières par l'apport de poussières incombustibles;
- poussières minérales et écrans liquides.

Nous présentons les conditions d'utilisation des protections et étudions leur efficacité en prenant pour exemple diverses explosions minières réelles.

Il s'est avéré que, utilisés correctement, les écrans de poussières minérales constituent le moyen de protection le plus efficace contre les explosions minières de poussières de charbon.

Les écrans liquides sont tout aussi efficaces, sauf parfois dans le cas d'explosions plus faibles.

Le rapport présente le détail d'une analyse du comportement des protections de poussières minérales au cours d'explosions, sur la base d'essais et d'études effectuées à la suite d'explosions minières.

АНАЛИЗ ПОВЕДЕНИЯ ЗАСЛОНОВ ВО ВРЕМЯ ВЗРЫВОВ УГОЛЬНОЙ ПЫЛИ В ШАХТАХ

Павел А. Кржыстолик

В подземных шахтах используются два вида защиты от распространения взрывов угольной пыли:
- нейтрализация пыли путем добавления невоспламеняемой пыли;
- заслоны с инертной пылью и водяные заслоны.

В докладе описаны условия использования заслонов и проанализирована их эффективность на примере различных взрывов, действительно происшедших на шахтах.

Было доказано, что правильно использованные заслоны с инертной пылью являются наиболее эффективным способом защиты от взрывов угольной пыли в шахтах.

Водяные барьеры столь же эффективны, как и барьеры с инертной пылью, но в случае слабых взрывов они менее эффективны. На основе лабораторных исследований и изучения послевзрывных условий в шахтах, детально анализируется поведение заслонов с инертной пылью во время взрывов.

煤矿煤尘爆炸阻隔物效果的分析

巴凡尔 A．克罗席思托立克

在地下矿井防止煤尘爆炸传播用下列两种办法：
- 加不可燃尘末抵消尘埃堆积
- 岩粉和防水煤柱

这里叙述运用阻隔物的条件和它们的有效性，并从矿里不同的真实爆炸例子加以分析．

这里证明岩粉棚如果使用适当是防止煤尘爆炸最有效的办法．

防水煤柱与岩粉棚同样有效，但遇到较弱的爆炸时效力较差．

本文根据实验和煤矿爆炸后的调查对岩粉棚在爆炸时的作用详加分析．

RECENT RESULTS OF DUST EXPLOSION STUDIES AT THE LAKE LYNN EXPERIMENTAL MINE

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ABSTRACT

The U.S. Bureau of Mines, as part of its continuing research program for evaluating coal dust explosion hazards, has recently completed a series of tests in the Lake Lynn experimental mine (LLEM) to study the effect of entry size on the flammability limits of pure coal and coal dust/rock dust mixtures. Data will be presented that compare measurements from explosions of mixtures of coal dust, methane, and rock dust. There are significant differences in dust explosion propagation characteristics between the LLEM and the smaller entry Bruceton experimental mine (BEM). The larger entry (LLEM) results indicate a larger rock dust percentage is required to prevent flame propagation than in the BEM. Lower flammable limits of coarse and fine Pittsburgh seam coal dusts were obtained in the LLEM. A lower volatile mine-size coal required about 40 pct total incombustible content (TIC) to prevent propagation in previous BEM experiments, whereas, recent tests with a similar volatile (18 pct) Pocahontas seam mine-size coal required 60 pct TIC in the larger entry of the LLEM to prevent flame propagation.

As one means of preventing these coal dust explosions, 80-L water tub barrier units were tested in the larger entry of the LLEM. The relative efficiencies of roof-mounted and rib-stacked barriers were determined for suppressing fully developed coal dust explosions in a single entry. Barrier sets were evaluated at various locations (40 to 160 m from the face) and configurations in the entry. Units were suspended near the roof and triply stacked at the ribs. Stacked barrier units which are portable, simple to install, and inexpensive to erect provided protection against coal dust explosions in the LLEM. In wide mine entries, they pose less of an impedance problem for men and equipment than do roof-mounted barriers. For some mines having high dust deposition rates, stacked barriers would offer additional protection beyond that afforded by generalized rock dusting.

INTRODUCTION

All of the Bureau's previous large-scale dust explosion data have been gathered from tests conducted in the narrow entries of the BEM (Bruceton, PA). The knowledge obtained from these explosion tests, dating back to 1911, has led to a better understanding of the fundamental explosion processes. Pressure and flame travel relationships have been examined under a variety of experimental conditions in the 1.8-m-high by 2.1-m-wide BEM. These studies have resulted in the implementation of new safety regulations in the mining industry which have significantly reduced the number of injuries and fatalities that have plagued the industry each year due to underground mine fires and explosions.

Over the past several decades, the average entry size of a typical U.S. coal mine has steadily increased due to the many technological advances in the mining industry. Many of these advances are in the areas of roof control, mining machinery, and ventilation. Current U. S. operating mines with high seam height typically have entry widths of 6.1 m. In late 1979, the Bureau commenced development of an

experimental underground facility which is more representative of modern mining geometries. Completion of the LLEM provided the opportunity for comparison studies in entries having approximately twice the cross-sectional area of that of the BEM. A comparison of the entry lengths and cross-sectional areas of the BEM and the LLEM is shown in figure 1. Since the LLEM geometry closely matches that typically found in current U.S. operating mines with high seam height and wide room width, the LLEM research findings on the propagation and suppression of coal dust explosions have more direct applicability to the U.S. mining industry than those from experimental mines with small cross-sectional areas.

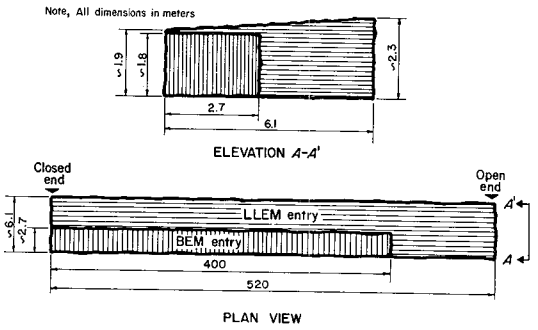


Figure 1. - Comparison of entry sizes for the BEM and the LLEM.

In a recent report by Sapko et al (1987 A), comparisons of single entry gas and dust explosion test results in the BEM and LLEM revealed some interesting differences between the two facilities. The flame accelerates faster in the BEM than in the LLEM for all methane (CH_4) - air mixtures due to a higher surface area to volume ratio. The local roughness in the BEM generates a greater degree of turbulence per unit volume of the unburned gas mixture ahead of the flame than in the LLEM. Also, the pressure pulse produced by the face ignition of CH_4-air decayed less rapidly with distance in the larger LLEM entry than in the BEM, presumably because of the smaller surface to volume ratio. The same factors that cause higher acceleration of the flame front in the BEM appear to be responsible for the more rapid decay of the pressure pulse. A minimum of 8 m^3 of 10 pct CH_4-air was required to create a sufficient aerodynamic disturbance to disperse and initiate a coal dust explosion in the 12.8-m^2-cross-section LLEM as compared to 4 m^3 10 pct CH_4-air in the 4.9-m^2-cross-section BEM.

This report will discuss the results of current coal dust explosion research in the LLEM and study the explosion size-scaling characteristics relative to the BEM. Experimental results show the influence of coal dust particle size and volatility on the flammability limits of pure coal and coal/rock dust mixtures. Dust explosion propagation characteristics of three coals are compared between the BEM and LLEM. Finally, this report will briefly describe recent Bureau research with passive water barriers and evaluate the suppression performance of rib stacked barriers and roof-mounted barriers in the wide LLEM entry.

EXPERIMENTAL PROCEDURE

The coal dust explosion and passive barrier suppression tests were conducted in the single entry D-drift of the LLEM.

The entry is 518 m long with a cross-sectional area ranging from 11.2 to 13.0 m^2 (ave. 12.8 m^2). D-drift is instrumented with pressure transducers and optical flame sensors located at the face and approximately 10, 15, 30, 45, 60, 90, 120, 150, 180, and 230 m from the face. The mine has four parallel drifts -- A, B, C, and D. Drifts C and D are connected by E-drift, a 152 m long entry which simulates a longwall face. D-drift, for these studies, was isolated from E-drift by means of an explosion-proof movable bulkhead.

The dust in the BEM tests was loaded 1/3 on the roof, 1/3 on the side shelves, and 1/3 on the floor for a distance of 120 m from the closed

end. In the LLEM tests (including the barrier tests), the dust loading was 1/2 on the roof and 1/2 on the floor for a distance ranging from 76 to 95 m from the face. During these tests, the LLEM ignitor was held constant at 152 m^3 of a 10 pct CH_4-air mixture or 12 m length of entry, while the BEM consisted of either a 76 m^3 of 10 pct CH_4-air (C-50, 15 m length of entry) or 46 m^3 of 7.0 pct CH_4-air mixture (C-30, 9 m length of entry). Table 1 shows the properties of the ignition sources used to compare explosion limits between LLEM and BEM. The total energy released at LLEM is about twice that used in BEM but the energy per volume of flame extension is about the same, i.e. 165 to 166 kg-cal/m^3. To obtain the ignitor intensity, divide the energy density per volume of the ignitor flame extension, 166 kg-cal/m^3, by the time it takes the flame to get to 61 m, or 0.78 s for the LLEM. As can be seen in table 1, the ignitor intensities for both experimental mines are similar for the given ignitor zones. In both mines, the CH_4-air mixture was ignited near the center of the face (closed end). The ignitor flame extends to 61 m producing 0.8 bar overpressure at LLEM, whereas the flame extends 76 m (in 0.68 s) producing 2.7 bar at BEM(C-50 ignitor). The maximum ignitor flame speed at BEM is much higher than at LLEM (Sapko et al 1987 A). From early BEM research, the C-50 (10 pct CH_4, single or double entry testing) and the C-30 (7 pct CH_4, single entry) ignitors had similar ignitor intensities and produced similar inerting data.

COAL DUST EXPLOSION RESULTS AND DISCUSSION

The effectiveness of limestone rock dust in preventing coal dust explosions has been demonstrated both by experiments in the BEM and in practice by limiting flame propagation and explosion damage in certain accidental coal mine explosions (Richmond et al 1975). The exact mechanism by which rock dust quenches the flame has not been fully explained, but it is believed to be the simple absorption of thermal energy from the flame front. Some decomposition may take place in the flame front, but rock dust is not a chemical inhibitor. Table 2 shows the average proximate and ultimate analyses of the three major coal dusts used in tests at the LLEM. The key difference in these coal dusts is in the percentage of volatile matter. The Pittsburgh and Sunnyside coals, volatilities of 36.5 and 38.1 pct respectively, have over twice the volatile matter as the Pocahontas coal which has 18.0 pct. The effect of the coal's volatility will be discussed later in this paper.

Particle size effect on a coal dust explosion:

Table 3 lists the size distribution and average particle diameters (D_s and D_w) for the various dusts tested at LLEM. The pulverized (80 pct coal particles <74 μm) and the mine-size (100 pct <840 μm; 20 pct <74 μm) coal dusts are standard sizes that have been used for decades at the Bureau. The requirement of 65-pct-incombustible for intake entries in commercial operating coal mines is based on earlier research (Rice et al, 1929 and 1933) when the typical size dust found in intake entries of mines at that time was approximately 20 pct < 74 μm. The 80-pct-incombustible requirement for return entries is based on research by Nagy et al (1965) in the BEM, using finer coal dust particles which incluced up to 81 pct <74 μm.

A series of tests were recently conducted in the LLEM to determine the effect that coal dust particle size has on the limits of flame propagation. Figure 2 shows the pressure/flame data from three tests using 20, 45, and 80 pct coal fraction less than 74 μm. In each of these tests, a 200 mg/L coal concentration was combined with rock dust to provide a mixture of 68 pct TIC. The dust was loaded on the floor and roof from the edge of the

Table 1: Ignitor intensity

	Gas volume m^3	Max static pressure, bar	Flame length, m	Total energy, kg-cal	Energy density per volume of ignitor flame, kg-cal/m^3	Ignitor intensity, kg-cal/m^3-s
BEM:						
C-50	76	2.7	76	63k	165	243
C-30	46	2.0	30	26K	170	244
LLEM	152	0.8	61	126k	166	213

Table 2: Average proximate and ultimate analyses of coal used in tests.

	Pocahontas coal[1]	Pittsburgh coal[2]	Sunnyside coal[3]
	As received, pct	As received, pct	As received, pct
PROXIMATE ANALYSES			
Moisture.....	0.5	1.7	1.2
Volatile matter....	18.0	36.5	38.1
Fixed carbon.....	75.7	55.6	54.1
Ash.....	5.8	6.2	6.6
Total.....	100.0	100.0	100.0
ULTIMATE ANALYSES			
Hydrogen.....	4.3	5.4	5.5
Carbon.....	84.7	77.4	75.5
Nitrogen.....	1.0	1.5	1.5
Oxygen.....	3.4	8.1	10.2
Sulfur.....	0.8	1.4	0.7
Ash.....	5.8	6.2	6.6
Total.....	100.0	100.0	100.0
	[1]14,709 Btu/lb.	[2]13,803 Btu/lb.	[3]13,632 Btu/lb.

Table 3: Average size analysis data of coal used in tests.

DUST TYPE		-20 M, pct	-200 M, pct	D_s, μm	D_w, μm
Pocahontas	pulverized	100	78	38	55
	mine-size	89	21	74	391
Pittsburgh	pulverized	100	81	31	50
	mine-size	84	19	97	435
Sunnyside	pulverized	100	83	36	52
	mine-size	85	22	88	402
Limestone rock dust		100	76	13	45

ignitor zone (12 m) out to 76 m from the face. The dashed line in each of the traces represents the pressure profile exhibited by the ignitor without any dust. The ignitor-only flame travels approximately 61 m from the face which is the reason the dusted zone extends at least 76 m from the face.An important aspect to note from this study was that the resultant explosion propagated in all three tests. However, as the particle size decreased, the overpressures and flame travel beyond the dusted zone increased significantly. Since the coal dust inerting is strongly dependent on coal particle size, it is very important to know the size distribution of the coal deposit being protected by rock dust.

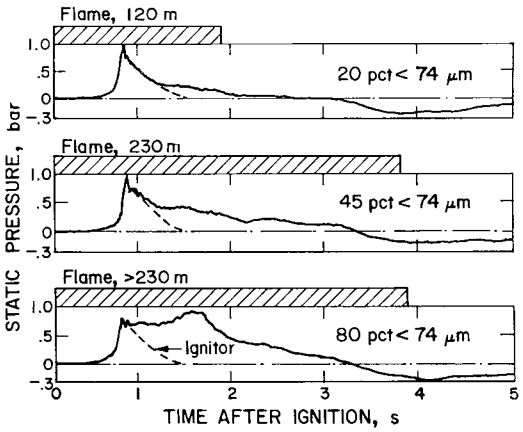


Figure 2. - Pressure profiles of LLEM tests for coals with 20, 45, and 80 pct <74 μm. Hatched areas represent flame travel only.

Lean limit explosion concentrations in the BEM and LLEM:

In both the BEM and LLEM, the initiation of a near stoichiometric CH_4-air zone serves as the ignition source for determining the lean limit concentrations of the various coal dusts. The pure coal dust is loaded entirely on roof shelves since dust on elevated surfaces is dispersed much more readily closer to the face by the pressure pulse generated from the ignition zone than dust on the floor. The lean limit concentrations are often used to compare coal dusts and assess the flammability hazards of the dusts. In the BEM, the minimum nominal loading for the Pittsburgh seam pulverized coal (PPC) dust explosions ranged from 50 to 70 mg/L using the C-50 or C-30 ignitors (Nagy 1981 and Richmond et al 1975). Note that a nominal loading assumes a complete and uniform entrainment of the dust throughout the entire cross-section of the mine entry during the explosion process. In the wider LLEM entry, a 40 to 50 mg/L minimum loading was sufficient to produce a propagating coal dust explosion with PPC (table 4). For mine-size Pittsburgh seam dust; the nominal loading was 80 to 100 mg/L in the BEM as compared to 50 to 60 mg/L at the LLEM. As can be seen from this data, both fine and coarse dust limits in the LLEM are lower than those in the BEM.

Table 4 provides a listing of the lean limit tests conducted at the LLEM. The lean limits for the three pulverized coal dusts, regardless of the volatile matter, range from 40 to 50 mg/L. This would seem to indicate that the coal particle size and surface area reactivity are important factors, as opposed solely to the coal's volatility determined by the ASTM method. Preliminary analyses have shown that the surface area (as measured from CO_2 adsorption) on the lower volatile Pocahontas pulverized coal averaged 165 m^2/g as compared to 112 m^2/g for the pulverized Sunnyside coal and 118 m^2/g for the PPC. The Pocahontas coal has a higher surface area per gram of dust than the other two coals and also has the lowest Hardgrove number. The sieve analysis of these three coals (table 3) shows about the same surface weighted mean diameter (D_W and D_S). However, the measured surface area indicates that the Pocahontas dust has a more porous structure than that shown by the sieve analysis. Even though all of the pulverized coals are characterized by having approximately 80 pct of the particles pass through a 200 mesh screen, the Pocahontas coal has a larger surface area per gram of coal than the others. Therefore, the Pocahontas coal has more surface area for enhancing the rate of devolatilization, or surface reactions, when exposed to the flame front than the higher volatile coals. Based on volatility only, in order to produce about the same lower explosive limit, Pocahontas dust would have to produce about the same amount of volatile as Pittsburgh and Sunnyside coal dusts do.

Flammability Limits of Coal/Rock Dust Mixtures:

Early research by Rice and Greenwald (1929 and 1933) in the BEM shows

Table 4: Effect of coal volatility and particle size on the lean limit and inerting requirements of three bituminous coals from LLEM tests.

COAL	POCAHONTAS volatility, 18.0 pct		PITTSBURGH volatility, 36.5 pct		SUNNYSIDE volatility, 38.1 pct	
	pulverized	mine-size	pulverized	mine-size	pulverized	mine-size
LEAN LIMIT, mg/L	----	50 NP	30 NP	40 NP	----	50 NP
	50 M	60 M	40 M	50 M	50 M	60 M
	60 P	75 P	50 P	75 P	60 P	-----
TIC TO INERT, pct	----	60 NP	82 NP	-----	80 NP	75 NP
	70 M	55 M	79 M	-----	-----	70 M
	65 P	50 P	77 P	68 P	75 P	65 P

NP - Nonpropagation (flame travel ending within the dusted zone).
M - Marginal propagation (flame travel slightly beyond dusted zone).
P - Propagation (flame travel >15 m beyond dusted zone).

that pulverized coal dust having volatile ratios less than 0.2 require less than 65 pct incombustible to prevent flame propagation for mine-size coal dust (figure 3 solid line). The volatileratio is calculated from the proximate analysis as follows:

$$VR = \frac{VC}{VC + FC},$$

where VR is the volatile ratio (or moisture-ash-free volatility), VC is the volatile content, and FC is the fixed carbon. The volatile ratio gives a value independent of the added incombustible in the coal. The inherent ash and moisture in the coal is added to the rock dust to give the TIC value of the coal/rock dust mixture. Figures 3 and 4 show that coal dusts (mine-size or pulverized) having volatile ratios of less than 0.12 do not propagate explosions in BEM tests. As the volatile ratio increases from 0.12 to 0.20, a sharp rise in the pct TIC is required in the dust mixture to prevent propagation. In the case of pulverized coal dusts (figure 4), this increase rises from 20 to 65 pct TIC. With VR from 0.2 to 0.45 for pulverized coals, the increase is less sharp for 65 to 80 pct TIC. As can be seen in these figures, each coal, depending on its volatile ratio and particle size, requires a specific value of incombustible to prevent flame propagation. These are the data upon which current

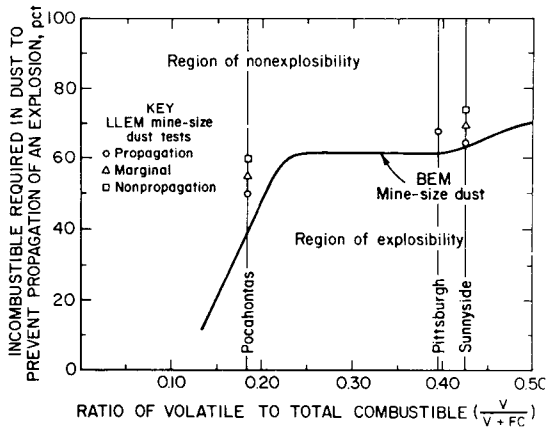


Figure 3. - Comparison of BEM and LLEM sincle entry explosion data for incombustible requirements as a function of the volatile ratio for mine-size coal dusts.

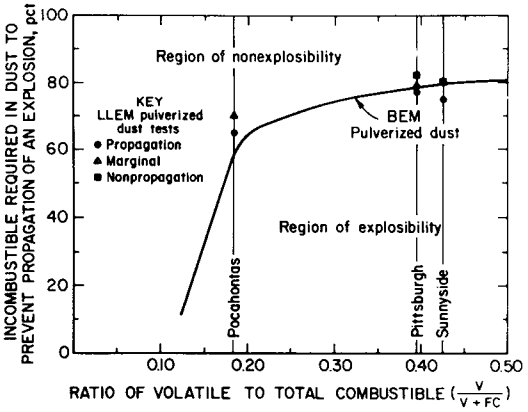


Figure 4. - Comparison of BEM and LLEM single entry explosion data for incombustible requirements as a function of the volatile ratio for pulverized coal dusts.

Federal regulations (30 CFR Part 75.403) are based. The current Law requires 80 pct TIC in returns in the absence of methane and 65 pct TIC elsewhere in all underground workings except anthracite mines. A series of coal/rock dust flammability tests have been recently conducted to assess the validity of these rock dusting requirements in the larger LLEM entry.

The effect of rock dust in limiting flame travel is shown in figure 5 for recent dust explosion tests in the LLEM. Figure 5 shows the flame front positions as a function of time after ignition for the LLEM. In these tests, the PPC (see table 3) nominal concentration was held constant at 200 mg/L and the rock dust varied to give from 64 to 2 pct TIC. The ignition of a 68-pct-TIC dust mixture at the LLEM (figures 2 and 5) results in flame propagation of about 215 m/s over 230 m from the closed end. Increasing the rock dust content continues to slow the flame propagation rate and, as a consequence, decreases the explosion overpressure. At 75-pct-TIC, the LLEM propagation is over twice the length of the dusted zone. Finally, at 82-pct-TIC, the flame propagation at the LLEM is about the same as the ignitor itself. Therefore, the amount of incombustible required to limit flame propagation in the LLEM is 79- to 82-pct-TIC (table 4).

Experiments were also conducted with a lower volatile Pocahontas seam coal using mine-size dust (20 pct <74 μm). In previous BEM tests, a coal/rock dust mixture of 40 pct TIC was sufficient to prevent flame propagation of this Pocahontas mine-size coal (figure 3). However, in recent LLEM tests, a 60 pct TIC dust mixture was required to prevent the flame propagation through and beyond the dusted zone. For pulverized Pocahontas coal dust, 60 pct TIC was required at the BEM but over 70 pct TIC was needed at the LLEM to prevent flame propagation.

Based on this data for Pocahontas mine-size coal tests at the LLEM, the Federal rock dusting regulation offers some margin of safety with respect to the current 65 pct TIC requirement for intake airways in the absence of any float coal dust accumulation (Sapko et al 1987 B). Moreover, for higher volatile mine-size Pittsburgh and Sunnyside seam coal particles, tests show that at least 75 pct TIC is required to prevent flame propagation in the larger LLEM entries whereas, 65 pct TIC was sufficient at BEM. This indicates that the present regulation of 65 pct TIC established using mine-size dust is not adequate for preventing flame propagation of the same dust distribution in larger mine entries.

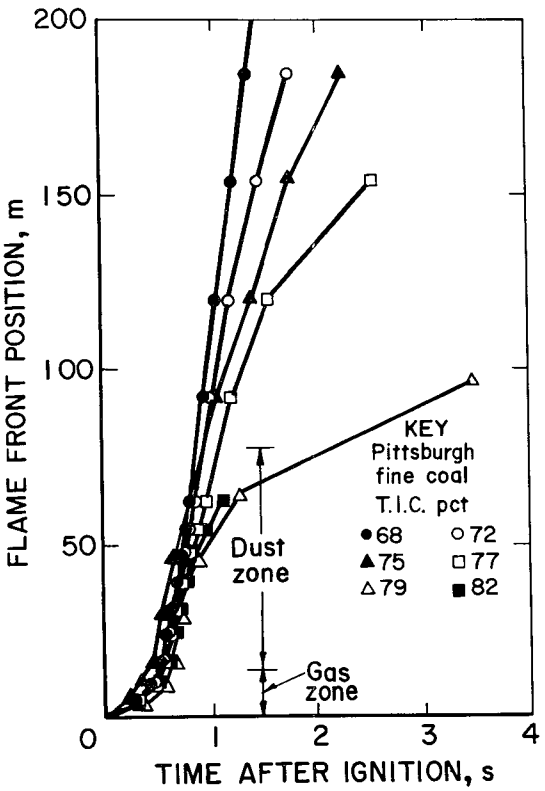


Figure 5. - Flame front position as a function of time after ignition for various PPC/rock dust mixtures at the LLEM.

PASSIVE WATER BARRIER RESULTS AND DISCUSSION

Experiments were recently conducted in D-drift to determine the effectiveness of conventional

roof-mounted barriers and floor-stacked barriers deployed at different distances from the face (Sapko et al, 1988). Note that all of the barrier results are from single entry experiments and may not be applicable to multiple entry scenarios. Explosions were initiated by igniting near stoichiometric CH_4-air mixtures confined to the first 12.2 m from the closed end (face) by a plastic diaphragm. A mixture of rock dust and PPC was spread on the floor and on the chemical foam roof shelves in the dust zone (edge of the gas zone at 12.2 m up to 131 m from the face) to provide a PPC nominal dust concentration near 200 mg/L. This standard test coal dust was mixed with limestone rock dust to provide mixtures ranging from 58.6 to 67.8 pct TIC in order to vary the explosion intensity. In explosion tests in the single entry without barriers the flame propagated about 3 to 4 times the length of the dusted test zone (>230 m from the face).

The water tub barriers used in this study were standard German-manufactured troughs made of polyvinyl chloride (PVC). They are designed to hold 80-L of water. They are approximately 29 cm in height, 48 cm in width, and 75 cm in length. These water barrier experiments were conducted at 40- to 160 m, in 20 m increments, from the closed end.

Figure 6A shows the experimental configurations for the 80-L PVC tubs suspended from the roof. The roof barrier tests were conducted with 4 tubs using zones B, C, D, and E. Figure 7 shows a typical view of a roof-mounted water tub barrier in D-drift. The tubs were supported by metal frames secured to the roof by bolts. Small diameter wires (break wires) were attached to the tubs to indicate when the tubs would start to rupture or tip during the explosion. The roof installation requires support framework, roof bolts, plates, and suspension hooks.

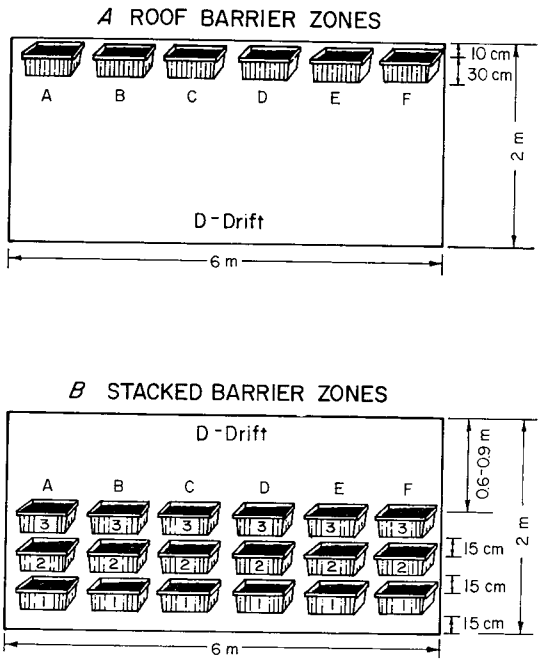


Figure 6. - Barrier deployment configurations.

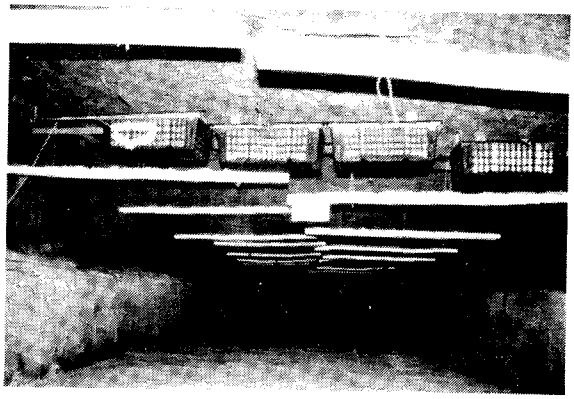


Figure 7. - Roof-mounted water tub barrier.

Considerable fabrication and installation time are involved. Roof-mounted tubs are usually deployed in high roof regions (such as where yieldable steel arches are used) in order not to impede transportation.

The experimental configurations for the stacked barriers are shown in figure 6B. The stacked barrier tests were conducted by erecting a triply stacked barrier unit along each rib (zones A and F) for a total of 6 tubs per test. Figure 8 shows

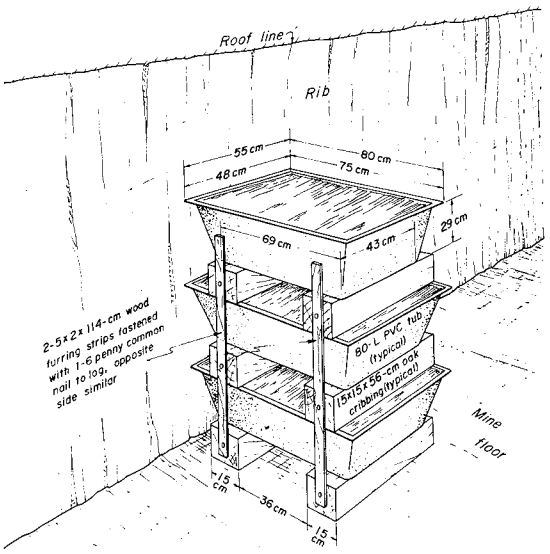


Figure 8. - Stacked water tub barrier

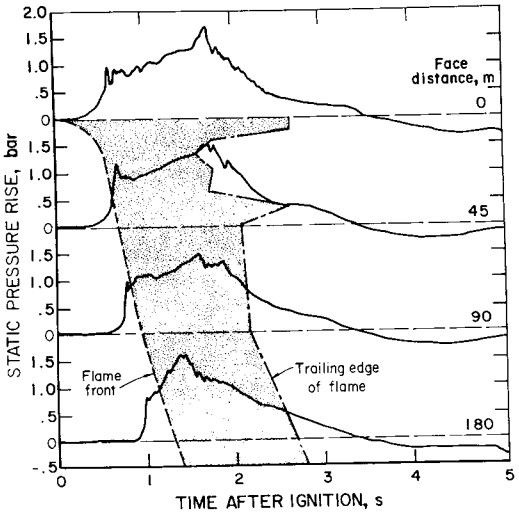


Figure 9. - Explosion wave diagram without a barrier.

a typical stacked water tub barrier unit. In the stacked configuration, the first 80-L tub was set on two cribbing timbers which lie on the mine floor. The second barrier tub straddled the first using two more timbers and so on. To help distribute the weight, vertical wood strips were nailed to the exposed end of the horizontal timbers. Break wires were also attached to the stacked tub units to indicate when the tubs would start to rupture or tip during the explosion.

A typical pressure/flame diagram of a coal dust explosion (67.8 pct TIC) without a barrier system is shown in figure 9. The figure shows the wall or static pressure history recorded at various distances from the face. The pioneer wave moves toward the open end at the speed of sound.

Flame travels past 230 m maintaining a rather constant speed. Notice that there is little attenuation of pressure with distance over much of the dusted zone (Sapko et al 1987 A). Superimposed on this plot are the leading edge of the flame and the end of the hot radiation zone (trailing edge). At a particular station along the entry, the interval between the leading edge of the flame front and the trailing edge of the flame represents the duration of the flame at that station.

The precursor wave, generated from the ignition of the CH_4/dust mixture, initiates the barrier operation depending on the strength of the barrier mount. Roof-mounted barriers start to operate at as low as 0.1 bar overpressure -- this pressure is acting at about the time of the rupture of the break wire. For tests in which suppression occurred, flame speeds at the barrier ranged from 153 to 211 m/s. It is observed that the roof-mounted barrier effectiveness decreases at large distances from the face. Non-suppression occurred when the barriers were located at 120, 140, and 160 m from the face. The water dispersed early due to the rupture of the barriers by the pressure wave and the flame was not able to interact with the water cloud. The time between barrier operation and flame arrival ranged from 0.32 - 0.44 s during which time a significant fraction of the water dropped out of suspension. At a location of 60 m from the face, the roof-mounted barriers suppressed the explosion propagating through a zone of 67.8 pct TIC; however, they failed to suppress the explosion when it propagated along a 63.2 pct TIC zone. The roof-mounted barrier effectiveness also decreased at small distances from the face (<60 m).

Figure 10 shows a pressure/flame diagram illustrating suppression of the coal dust explosion by stacked barrier units and its effect on the flame and the pressure development. The figure shows that the barrier stacks begin to tip when the flame has propagated about 33 m from the face, allowing about 0.02 s of water discharge prior to the flame arrival. Suppression occurs at 53 m which is just beyond the barrier site of 40 m.

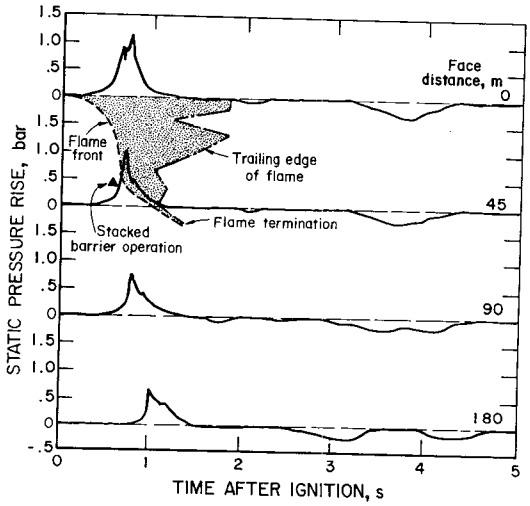


Figure 10. - Explosion wave diagram with a stacked barrier unit along each rib at 40 m from face (closed end).

Figure 11 shows a comparison of flame propagation between an explosion propagation without a barrier and one with a stacked barrier unit located at each rib at 40 m from the face. The stacked barriers suppress the explosion propagating through the 67.8 pct TIC zone. A triply stacked barrier unit along each rib at 80 m from the face can suppress an explosion in a dust mixture as low as 58.6 pct TIC. In tests which suppress, the stacked barriers start to operate at as low as 0.8 bar and the flame speeds at the barriers range from 150 to 230 m/s.

To better understand the action of the stacked barrier units at the 40-m location, high speed motion pictures were taken. These pictures showed (1) the interaction between the stacked water tubs and the air blast preceding the ignitor flame and (2) the formation of the water cloud. The pictures revealed that initially, some water was aspirated off the top of the tubs. Under the influence of the air blast, the wooden structure (made from cribbing and furring strips) which supported the tubs pivoted; and the balance of the water was dispersed. Approximately 5 m from the barrier site, the water had dispersed, extending a water cloud boundary almost from the floor to the roof.

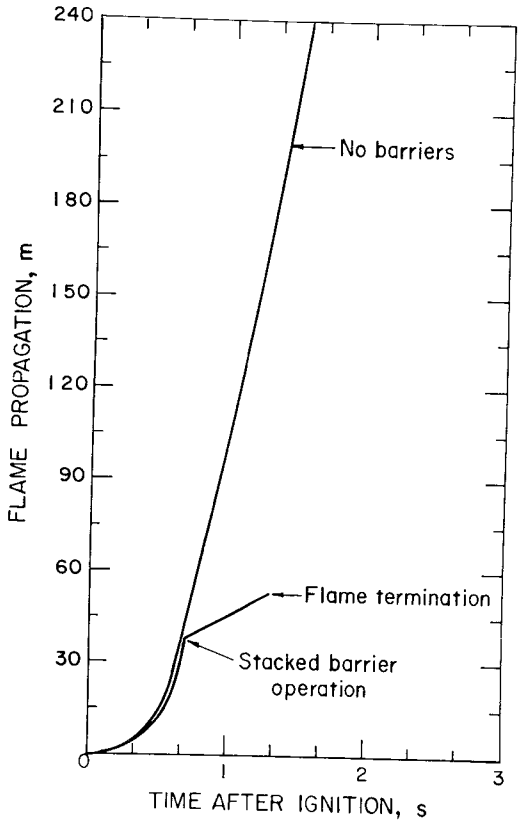


Figure 11. - Effect of a stacked barrier unit along each rib at 40 m on the explosion flame generated from a 68 pct TIC test.

Although some water was dispersed through aspiration, it appears that the bulk of it dispersed following the pivoting action of the barrier unit. A barrier tub design which

facilitates pivoting of the individual unit might provide earlier dispersion, and this would promote more effective barrier action when the barrier site is closer to the point of ignition.

The Bureau's explosion suppression tests with water barriers show that stacked tub units located along the ribs are quite effective in suppressing dust explosions and are significantly less expensive, in terms of materials and man-hours, to install than roof-mounted barriers. Optimally, a triply stacked water barrier unit would be placed near each rib in 60 to 100 m increments from the advancing face.

CONCLUSIONS

A size-scaling comparison between single entry coal dust explosion characteristics from early research in the BEM and recent experiments in the larger LLEM facility shows that a larger percentage of incombustible is required with the mine-size (20 pct of particles <74 μm) coal dusts to prevent flame propagation in the LLEM entry. For mine-size Pittsburgh and Sunnyside seam coal dusts, at least 75 pct TIC is required to prevent a propagating coal dust explosion when ignited by a primary explosion flame produced by the combustion of a 12 m long, 10 pct CH_4-air zone in the LLEM; whereas, only 65 pct TIC was required in the BEM when using an ignition zone with an ignitor intensity similar to the LLEM ignitor. This pct TIC necessary to prevent flame propagation in the larger entry is higher than the 65 pct TIC required by current Federal rock dusting standards (based on early BEM tests with mine-size dust) for intake airways.

Based on large scale Bruceton experimental mine tests, the 80 pct TIC regulation for return airways is adequate in the absence of any float coal dust accumulation. In the early BEM testing, 80 pct TIC dust mixtures were sufficient to prevent flame propagation with the Pittsburgh pulverized coal dust (PPC, 80 pct <74 μm). In the recent LLEM tests, 79 to 82 pct TIC PPC/rock dust mixtures are required to prevent flame propagation in the larger entry.

Recent large scale explosion testing of a lower volatile Pocahontas mine-size coal dust show that 60 pct TIC is required in the LLEM to prevent flame propagation where earlier tests in the BEM show 40 pct TIC was sufficient. Therefore, the Federal regulation of 65 pct TIC for intake airways offers some margin of safety over the 60 pct TIC required with this low volatile coal to prevent explosions in the large entry of the LLEM.

Preliminary results from the LLEM show that the coal dust particle size and porosity (surface area) is as an important factor to consider as volatility when studying coal dust explosion hazards. An 18.0 pct volatile Pocahontas seam coal dust (pulverized) requires over 70 pct TIC (58 pct TIC from early BEM tests) to render the dust mixture inert at the LLEM as compared to about 80 pct TIC for the 36.5 pct volatile PPC. Analysis of the Pocahontas coal has shown that it has a significantly larger surface area, enhancing devolatilization in the flame front.

Passive water barriers are an effective method of suppressing coal dust explosions in the single entry of the LLEM (representative of roadways typical of that found in current operating U.S. mines). Both roof-mounted and stacked barrier units are quite effective in stopping moderate strength single entry explosions in 63.2 and 67.8 pct TIC dust zones when placed at 60 to 100 m from the face.

A stacked water tub barrier unit located at each rib at 40 m from the face suppressed an explosion propagating through a 67.8 pct TIC zone. At 60 m from the face, a stack of three tubs located near each rib were able to suppress a

coal dust explosion in dust mixtures of 63.2 pct TIC. At 80 m from the face, this stacked water tub configuration can suppress a single entry coal dust explosion in a mixture as low as 58.6 pct TIC. Stacked water tub barrier units are portable, simple, quick, and relatively inexpensive to erect. Stacked water tub units optimally located near the ribs would provide reasonable clearance for most vehicular traffic expected in mines and have the potential to provide an additional gas and dust explosion protection in certain areas that are difficult to protect solely by generalized rock dusting.

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RESULTATS RECENTS D'ETUDES D'EXPLOSION DE POUSSIERES EFFECTUEES DANS LA MINE EXPERIMENTALE DU LAC LYNN

Eric S. Weiss, Nevin B. Greninger, Michael J. Sapko

Dans le cadre de son programme continu de recherches sur l'évaluation des risques d'explosions de poussières de charbon, le Bureau des Mines Américain a effectué une série d'essais dans la mine expérimentale du Lac Lynn (LLEM) afin d'étudier l'effet de la taille de l'entrée sur les limites d'inflammabilité du charbon pur et des mélanges de poussières de charbon/poussières de roches. Les données présentées comparent les mesures effectuées au cours d'explosions de mélanges de poussières de charbon, de méthane et de poussières de roches. Il existe d'importantes différences de caractéristiques de propagation d'explosions de poussières entre LLEM et BEM, la mine expérimentale de Bruceton, dont les dimensions de l'entrée sont plus petites. Les résultats donnés par l'entrée de LLEM indiquent qu'un pourcentage plus important de poussières de roches y est nécessaire pour empêcher la propagation des flammes que pour BEM. LLEM a donné des limites d'inflammabilité moins élevées pour les poussières grossières et fines de charbon de Pittsburgh. En ce qui concerne le charbon de Pocahontas, à volatilité plus faible (17 pct), il a fallu une quantité incombustible totale (TIC) de 40 pct environ pour empêcher la propagation dans BEM, et de 60 pct pour l'entrée plus grande de LLEM.

Pour empêcher ces explosions de poussières de charbon, on a procédé dans l'entrée de LLEM à des essais d'écrans d'eau dans des récipients de 80 litres. On a établi l'efficacité relative des protections montées sur le toit par rapport à celles empilées sur les parois pour supprimer les explosions complètes de poussières de charbon. On a évalué des protections situées à divers emplacements (40 à 160 m du front de taille) et dans diverses configurations d'entrées. On en a placé sur le sol de l'entrée, à mi-hauteur, et on en a suspendu près du toit; on les a souvent empilées en trois couches sur les parois. Les écrans portatifs à empiler, simples à installer et peu onéreux à monter, ont fourni la meilleure protection contre les explosions de poussières de charbon dans LLEM. Dans les larges entrées de mines, ils posent moins de problèmes d'impédance pour les hommes et les équipements que les écrans montés sur les toits. Pour certaines mines à taux de dépôts de poussières élevés, les écrans à empiler fournissent une protection supplémentaire, meilleure que celle donnée par le poudrage minéral généralisé.

NEUE ERGEBNISSE VON STUDIEN MIT STAUBEXPLOSIONEN IN DER LAKE LYNN VERSUCHSGRUBE.

Eric S. Weiß, Nevin B. Greninger, Michael J. Sapko

Das US Bureau of Mines hat, als Teil seines fortlaufenden Forschungs-Programms zur Bestimmung der Gefahren von Kohlestaubexplosionen, eine Testserie in der Lake Lynn Versuchsgrube (LLEM) abgeschlossen, in welcher der Einfluß der Streckengröße auf die Entflammbarkeitsgrenzen von reiner Kohle und Kohle/Steinstaubmischungen untersucht wurde. Vergleichsdaten von Explosionen mit Kohlestaub, Methangas und Steinstaub werden angegeben. Die Charakteristiken der Staubexplosionsfortpflanzung der LLEM und die in der Bruceton Versuchsgrube (BEM), die eine kleinere Strecke besitzt, unterscheiden sich deutlich voneinander. Die Ergebnisse der LLEM mit der größeren Strecke beweisen, daß ein größerer Prozentsatz an Steinstaub zur Unterdrückung der Flammenfortpflanzung nötig ist als in der BEM. Versuche mit groben und feinen Pittsburgh Kohlestäuben erbrachten geringere Entflammbarkeitsgrenzwerte für die LLEM. Pocahontas Kohle, die weniger flüchtige Bestandteile hat (17%), erfordert im Ganzen etwa 40% Anteile von unbrennbarem Inhalt (TIC) zur Verhütung der Flammenfortpflanzung in der BEM im Vergleich zu 60% TIC, die auf der längeren Strecke der LLEM erforderlich waren.

Eine Verhütungsmethode für diese Kohlestaubexplosionen wurde in der größeren Strecke der LLEM mit 80 Liter Trogsperren getestet. Der Wirkungsgrad der an der Firste angebrachten Sperren zur Unterdrückung einer vollentfalteten Kohlestaubexplosion wurde getestet und mit dem von an den Stößen aufgestapelten Sperren verglichen. Gruppen von Sperren wurden an verschiedenen Stellen (40 bis 160 Meter Entfernung vom Streb) und in verschiedenen Anordnungen in den Strecken getestet. Die Sperren wurden auf der Streckensohle angebracht, in mittlerer Höhe oder nahe der Firste aufgehängt und oft dreifach an den Stößen gestapelt. Gestapelte Sperren erwiesen sich als der beste Schutz gegen Kohlestaubexplosionen in der LLEM, außerdem sind sie tragbar, leicht aufzustellen und billig in der Anlage. In Gruben mit breiten Strecken bereiten sie weniger Behinderung für Mann und Maschine als die an der Firste aufgehängten Sperren. In Gruben mit hohen Staubablagerungsraten bieten gestapelte Sperren zusätzlichen Schutz über den des allgemein durchgeführten Einstaubens hinaus.

РЕЗУЛЬТАТЫ НЕДАВНИХ ИССЛЕДОВАНИЙ ПЫЛЕВЫХ ВЗРЫВОВ В ЭКСПЕРИМЕНТАЛЬНОЙ ШАХТЕ НА ОЗЕРЕ ЛИН.

Э.С. Вейсс, Н.Б. Гренинжер и М.Д. Сапко

С целью изучения влияния размера нарезной выработки на пределы воспламеняемости чистого угля и смеси угольной пыли с минеральными частицами, Горное Бюро США завершило серию испытаний в экспериментальной шахте на озере Лин (ЭШОЛ). Этот проект представляет собой часть продолжающихся исследований по оценке взрывоопасности угольной пыли. Представлены сравнительные показатели, полученные при взрывах смесей угольной пыли, метана и минеральных частиц. Найдены значительные различия между характеристиками распространения пылевого взрыва на ЭШОЛ и на экспериментальной шахте Брюстон (ЭШБ), которая имеет нарезную выработку меньшего размера. Результаты показывают, что для предотвращения распространения пламени на ЭШБ требовалось более высокое процентное содержание пылевых частиц, чем на ЭШОЛ. Более низкие пределы воспламеняемости крупной и мелкой пыли от питсбургского угля были получены на ЭШОЛ. Чтобы предотвратить распространение пламени на ЭШБ с менее летучим покахонтским углем (17%), общее содержание несгораемого материала не должно было превышать 40%. Однако на шахте в нарезной выработкой большего размера содержание несгораемой составляющей должно было быть 60%. В качестве одного из методов предотвращения взрывов угольной пыли, в нарезной выработке большего размера на ЭШОЛ были испытаны 80-литровые водяные заслоны. Заслоны, установленные на кровле, были сравнены с заслонами, установленными на стенке выработки, для подавления полностью сформировавшихся взрывов угольной пыли. Барьеры были испытаны на расличных расстояниях (от 40 до 60 метров) от плоскости забоя и в различных положениях. Они были поставлены на дне забоя, подвешены посередине, а также непосредственно под кровлей; часто они устанавливались в три ряда на стенке выработки. Составленные один на другой переносные заслоны, простые и дешевые в установке, представляли собой наилучший способ защиты от взрывов угольной пыли на ЭШОЛ. В широких забоях они являются меньшей помехой для шахтеров и для оборудования, чем заслоны, установленные на кровле. Составленные один на другой заслоны предоставляют дополнительную защиту в шахтах с высоким уровнем отложений пыли.

陵湖实验煤矿尘末爆炸最近的研究结果

埃里克 S． 魏思，内文 B．克兰宁格 及迈克尔 J．沙布柯

作为估计煤尘爆炸危害的持续研究方案的一部分，美国矿业局曾在陵湖实验煤矿（LLEM）完成了一系列的试验研究水平坑道的大小对纯煤和煤尘/岩粉混合物可燃性极限的影响．煤尘，甲烷和岩粉混合物的爆炸测值数据将在这里提出并做比较．LLEM 与水平坑道较小的布鲁斯顿实验煤矿（BEM）尘埃爆炸传播的特征有显著的不同．在较大水平坑道（LLEM）里的结果指明如要防止火焰传播（较 BEM）需要较大的岩粉百分比．LLEM 里粗粒和细粒匹兹堡煤尘的可燃性极限都比较低．低级挥发性玻卡鸿塔斯煤（百分之十七）在 BEM 需要百分之四十的完全不燃烧物量（TIC）来防止爆炸的传播，在较大的水平坑道矿里则需要百分之六十的 TIC．作为另一种防止煤尘爆炸方法，LLEM 试验用 80 升水盆阻隔设备．坑顶固定式和煤壁堆积式的阻隔设备压抑充分发展的煤尘爆炸的相对效率也在这里试验过．阻隔设备在水平坑道里不同的位置（离工作面 40 至 160 米）和构型也作了测定．这些设备有时放在坑道地上，有时在半高位置，有时靠近坑顶，常常三套都堆积在煤壁．堆积式的阻隔设备可以搬动，安装简单，建造费低廉，在 LLEM 防止煤尘爆炸最为适宜．在宽的水平坑道里它们较装在坑顶上的设备对人员和器材阻碍较少，在有些尘埃沉积较高的矿里，堆积式的阻隔设备较普遍用的岩粉有更好的防止作用．

Session XI

EXPLOSIONS, II

DEVELOPMENT OF COMPONENTS, DESIGNED TO RESIST EXPLOSION PRESSURES OF APPROXIMATELY 1 MPa, FOR USE IN VENTILATION STRUCTURES IN UNDERGROUND MINES

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ABSTRACT

Explosions in underground coal mines can be avoided by preventive safety measures, controlled by protective methods of construction and the propagation of the explosion can be impeded. This is done by explosion barriers and stoppings. As stoppings form a hermetic seal against part of the mine working, it is necessary, from the operational aspect, to provide openings for man-riding, haulage or ventilation. In order to make this compatible with the technical requirements relating to explosions, the Tremonia Experimental Mine, between 1983 and 1987, developed explosion-proof equipment such as doors in manways and haulage roads and also safety gates for conveyor belt and air locks. These components had to withstand a static explosion pressure of up to 1 MPa (10 bar).

INTRODUCTION

Despite all the safety measures taken against the occurrence and propagating of explosions in underground mines, a large number of incidents still occur each year, causing recurring cases of injury and death to miners. According to an unofficial statistic, explosion alone caused the death of more than 1500 miners throughout the world in the years between 1980 and 1987.

Constructive measures for explosion prevention, to supplement preventive measures, may take the form of explosion barriers or explosion-proof components (fig. 1.).

The use of explosion barriers throughout the whole of the mine workings is meant to ensure that the effects of an explosion remain limited to the smallest possible area. When correctly used, explosion barriers can extinghuish the explosion flame either within the barrier zone or some metres behind it. Although the explosion pressure is likewise reduced by the explosion barriers it still requires a "run-out" distance of several hundred metres, or even kilometres, until it reaches a stage where it no longer causes any damage (Michelis 1983).

Explosion-proof components are, primarily, stoppings. These seal off, hermetically, parts of a mine working. This isolation provides the most effective explosion protection, as, in the case of an explosion, it prevents the propagation of the mechanical, thermal and toxic effects.

However, from the operational point of view, such a hermetic seal does not always represent the optimum solution. In particular cases, an explosion-proof stopping has yet to be such that man-riding, haulage and ventilation can still be maintained. This is especially so when explosion-proof stoppings also assume the function of operational structures; this, for example, may be so in the case of ventilation equipment. If ventilation structures are subject to pressure and are damaged by explosions, the ventilation system may be affected and this can have serious conse-

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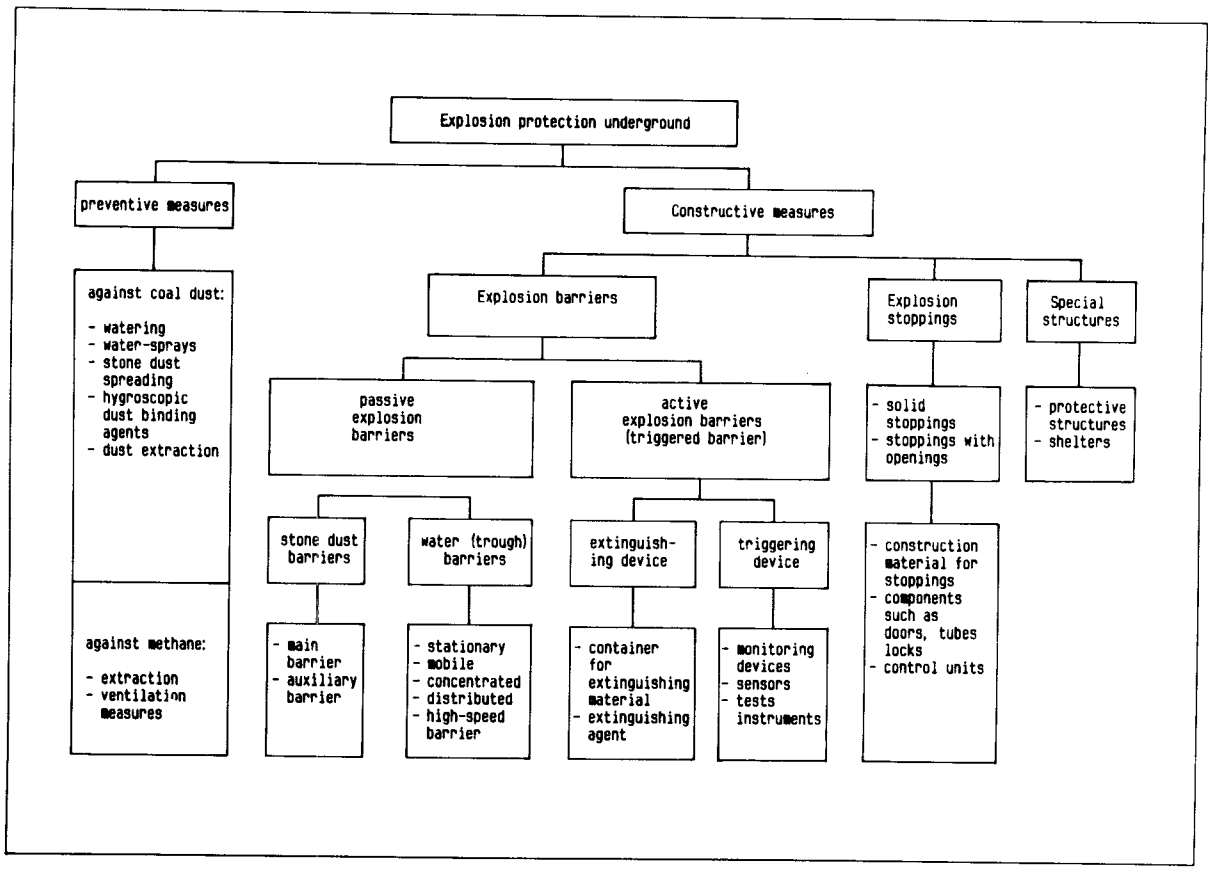


Figure 1. List of measures for explosion protection underground

quences for the workers in even distant parts of a mine. In exceptional cases, therefore, it should be possible to make a ventilation structure explosion-proof. This is always so when the mine region to be protected represents an exceptional case from the technical/ventilation point of view and when, consequently, this region can be supplied with fresh air only via the roadways to be protected - such, for example, is the case when work is being done under the main haulage level or in combined mines, in which shaft plant are inter-connected on the same level or in ventilation districts where it is not possible to maintain the required distance of 200 m between the start and end of a ventilation district.

The former explosion-proof ventilation structures used in practice were, primarily, based on data derived from theoretical studies and sometimes also on experience gained from explosion tests at the Tremonia Experimental Mine.

Exploratory explosion tests with "explosion-roof" ventilation structures were carried out by the Versuchsgrubengesellschaft mbH (Hibernia Experimental Mine) about forty years ago but did not result in any solutions that would be considered practical today (N.N. 1944).

GENERAL REQUIREMENTS FOR EXPLOSION-PROOF COMPONENTS

When explosion-proof stoppings are being installed, some corresponding requirements must be established for

- the structure of the stopping and
- the components.

In West German coal mines the official "Directives for the construction of stoppings" require the installation of explosion-proof stoppings. These regulations prescribe the materials to be used for constructing the stopping and also the maximum static explosion pressure that these must withstand. This is fixed at 0.5 MPa

(5 bar) and allowing for a corresponding safety margin, is set at approximately 1 MPa (10 bar) for the tests carried out in the Tremonia Experimental Mine. In dimensioning the thickness of the stopping in the case of so-called explosion-proof short stoppings, one takes into account the largest dimension of a roadway and, above all, the bending strength of the material used for the stopping. The calculation is based on the formula

$$l_D = \frac{0.7 \times a}{\sqrt{\sigma_{bz}}} \qquad (1)$$

In this formula:

l_D = length of stopping in level or inclined mine workings, in m

a = greatest width or height of roadway at the site of the stopping, in m

σ_{bz} = bending strength of the stopping material in N/mm².

For a ventilation structure to be described as "explosion-proof", the components must logically be exposed to the same pressure as the explosion-proof stopping itself, i.e. in the explosion tests, they are, likewise, exposed to a static pressure of approximately 1 MPa.

Apart from these technical requirements as to explosion, the operational requirements must also be taken into account. This means that the equipment for haulage and man-riding must, as far as possible, be designed to existing standard dimensions and must be adapted to the normal operational events.

Therefore, in the developments carried out by the Tremonia Experimental Mine in the years 1983 - 1987, special attention was paid to the operational feasability of the components.

EXPLOSION-PROOF COMPONENTS FOR VENTILATION STRUCTURES

In order to adapt an explosion-proof ventilation structure to the necessary operational routine, the following components were developed and tested, as to their applicability, in explosion tests:

- doors in manways
- doors in haulage roads
- safety gates for conveyor belts
- air locks.

The function required of the listed components are, in some respects, extremely different. Whereas the doors must always be closed in normal working conditions and may be opened only in emergence, safety gates for haulage and ventilation must always be kept open. In the case of an explosion, however, all openings in the stopping must be closed. This means that different structural requirements are set for the individual components.

All the test explosions carried out in order to test the components took place in the 16 m² (approx.) R 2 explosion gallery on the 4th level of the Tremonia Experimental Mine.

EXPERIMENTAL CONDITIONS

In each case, for trying out the explosion-proof components, we installed explosion-proof stoppings, in the form of so-called short stoppings approximately 1.50 m thick.

The pressure loading was produced by both firedamp and coal dust explosions. Fig. 2 shows the site of the experiment.

Depending on the length of the relevant fuel zone and on the fuel concentration all-round (static) pressures of between 0.2 to 1.0 MPa were obtained at the stopping.

The measuring technique used in the explosion tests was based on the standard equipment in the explosion gallery, that used to record pressure, flame, temperature and CH_4 concentration and we also used supplementary equipment in the zone adjacent to the stopping, such as, high-speed and video cameras and CO analysers among other things.

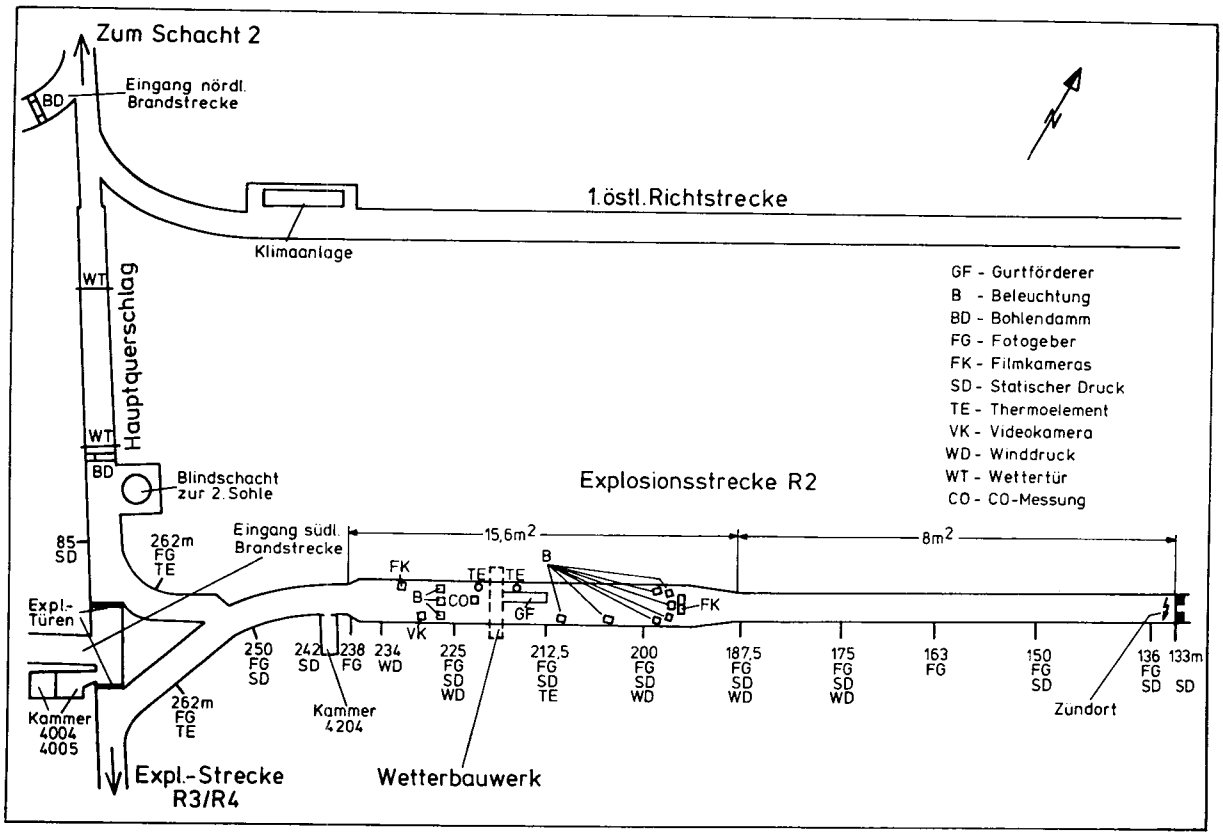


Figure 2. Technical measuring equipment in the R 2 explosion gallery

EXPLOSION-PROOF DOORS IN MANWAYS

The doors in manways used in underground mines in West Germany are in accordance with the dimensions specified in the German Standard (DIN). According to this standard, the inside width may be between 600 mm and 800 mm and the inside height between 1800 mm and 2000 mm. In the development of explosion-proof doors for manways, the dimensions 700 mm x 1800 mm and 800 mm x 1800 mm were taken as a basis.

The development of explosions-proof manway doors proved to be not so difficult as that of haulage-way doors, the main reason for this being that it was possible to refer back to the explosion-proof doors used in the measuring chamber at the Experimental Mine; the material used for these was 30 mm thick. Doors of this kind and of the aforementioned size are well able to resist explosion pressures of up to 2 MPa (fig. 3). In the course of the development, the changeover to

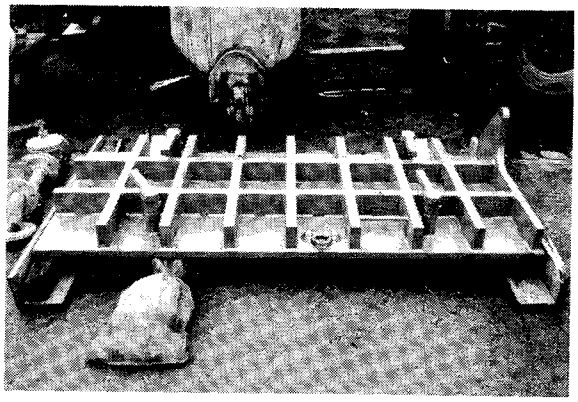


Figure 3. Explosion-proof manway door (flat), internal dimensions 700 mm x 1800 mm - Thickness of material panel 30 mm - Weight of door panel 930 kg

a domed door panel showed that - as in the case of the haulage-way doors - the maximum requirements as to pressure could be fulfilled with considerable savings in material (fig. 4). Of course, a door panel that can

be made "plan", is much easier to manufacture.

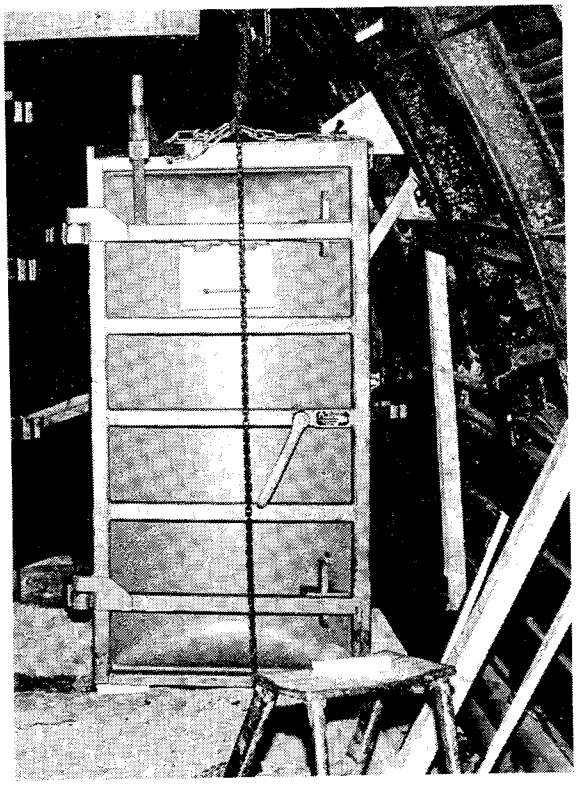


Figure 4. Explosion-proof manway door (domed), internal dimensions 800 x 1800 mm - Thickness of material of door panel 4 mm - Weight of door panel 240 kg

The manway doors were tested, in each case, with a completely equipped and fully operational pneumatic control device for opening and closing the door (fig. 5). Also in the case of the manway doors, there was always an auxiliary opening device for the situation where, following explosion pressure effects, the door could no longer be opened pneumatically.

EXPLOSION-PROOF HAULAGE ROAD DOORS

The haulage-way doors are also subject to the DIN Standard. Accordingly, the inside width of the door may vary between 1400 mm and 2600 mm and the inside height between 2000 mm and 2300 mm. The main types used in the workings are the so-called, balanced double-air doors.

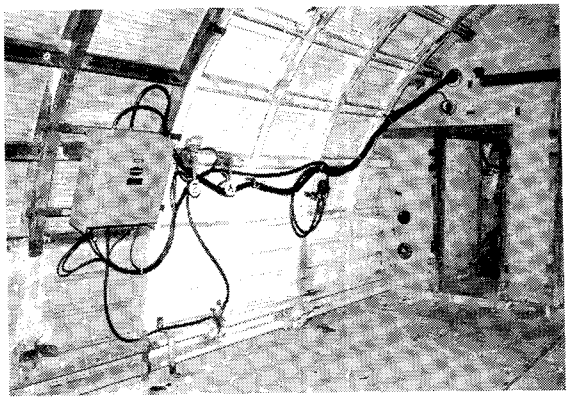


Figure 5. Pneumatic control device for air doors (facing away from explosion)

In order, however, to take the explosion/technical requirements into account, the explosion-proof doors had to be of the single-leaf type. Because of the given 15.6 m² road-section in the R 2 explosion gallery and the need to be able to use, simultaneously, a haulage-way door and a manway door, 1600 mm x 2000 mm were selected as the inside dimensions for the haulage-way door. At a later stage in the tests we also used a haulage-way door with inside dimensions of 1800 mm x 2300 mm (Michelis, Kleine 1986).

The types of door used in the explosion tests carried out in the Tremonia Experimental Mine were all pneumatically or manually operated. With the development of a domed door-panel construction we had a haulage-way door, which took into account both the explosion/technical and also the operational requirements (fig. 6). Plane door panels, even when reinforced by extra ribbing did not withstand the stipulated pressure of 1 MPa. Using doors measuring 1800 mm x 2300 mm we finally obtained and tested a door (domed) suitable for large transport units.

EXPLOSION-PROOF BELT CONVEYOR LOCKS

As, in normal operation, a belt conveyor gate must always be kept open, it must be ensured by constructive measures, that in the case of an explosion, no explosion effects can get

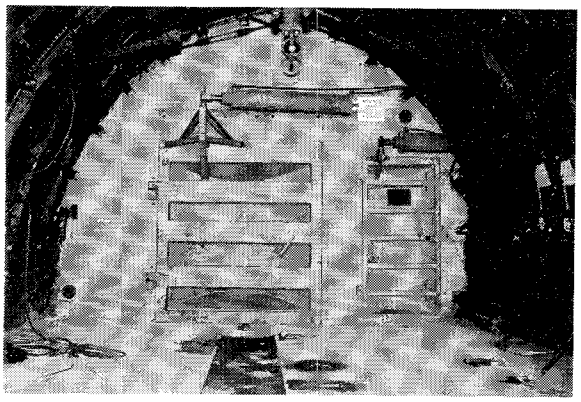


Figure 6. Explosion-proof haulage-way door (domed), internal dimensions 1600 mm x 2000 mm - Thickness of material of door panel 7 mm - Weight of door panel 1200 kg

through this opening and into the zone to be protected.

In order to achieve this, different measures were tried out in the course of the investigations. These were based on the use of a belt conveyor with a belt width of 1400 mm - this is the greatest belt width currently used in underground mines in West Germay.

Tests with corresponding components in the gate aperture in the form of rubber - or chain-mesh aprons - were as unsuccessful as stacks of plates. On the one hand, these components would no longer have permitted the continuous conveyance of coal and, on the other hand, the explosion effects were fully propagated through the opening in spite of these obstacles.

We, therefore, developed a technique, by which, with the help of a slide valve, the opening in the gate could be closed briefly in the case of an explosion. Fig. 7 shows a schematic representation of the fitting of an explosion-proof belt conveyor gate with automatic slide valve. The conditions which, in addition to the explosion-proof mode of construction, had to be fulfilled by this slide valve in order to prevent the transmission of an explosion, are, among other things:

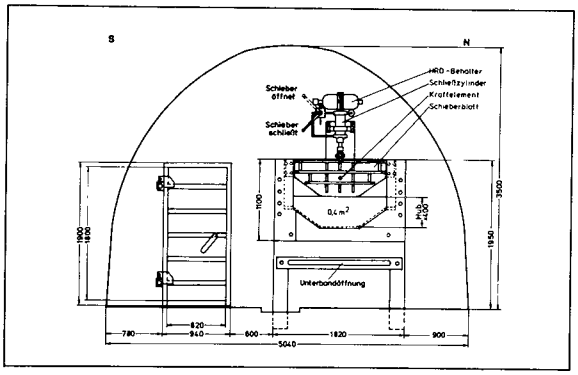


Figure 7. Schematic diagram of a belt conveyor gate with automatic slide valve plate

- closing time 100 ms
- cross-section of aperture 0.4 m²
- closing process without destroying the belt conveyor.

In order to achieve this objective, the valve plate had to be moved by a separate power source. This was provided by a cylinder which was pressurised by compressed air from an HRD tank. This tank contained compressed air at an overpressure of 4 MPa. In the resting position, the valve plate was held by an element known as the power element.

In the case of an explosion, the actual event must first of all be recognized. This is done by the detectors already used in triggered barriers and which operate on a thermal or optical basis (Michelis 1985). Fig. 8 clearly shows the time sequence of the closure process.

After the explosion flame arrives at the site of the detector (t_1), the trigger signal activates the initiating elements in the HRD tank and in the power element (t_2). This takes a maximum of approximately 2 to 3 ms. By this means a pressure of 4 MPa is applied to the cylinder; at the same time the holding element releases the slide valve plate (t_3). This process lasts for a maximum of 10 ms. The closing of the valve plate itself takes about 60 to 70 ms (t_4). The valve plate is stopped by buffer elements about 10 mm above the belt conveyor, in order to prevent destruc-

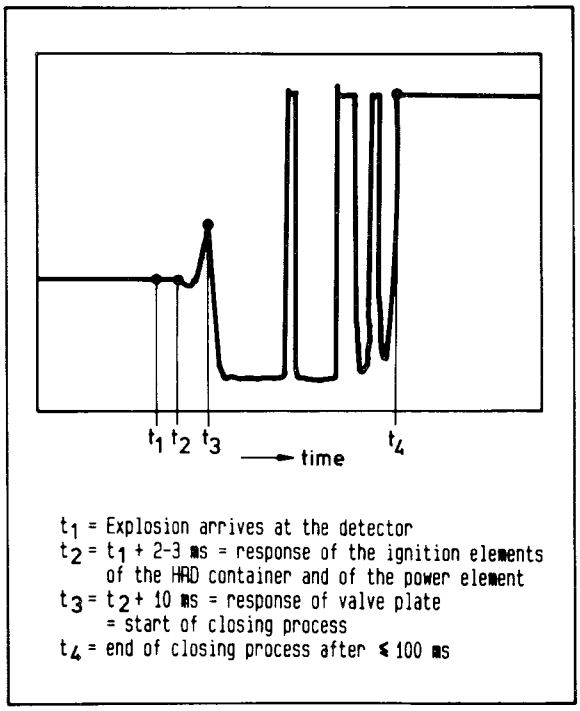


Figure 8. Closing process of an automatic belt conveyor gate as a function of time

tion of the belt conveyor. In order to prevent foreign bodies in the form of coarse coal-, rock- and wood-particles from hindering the closing process and perhaps preventing the gate aperture from closing properly, the valve plate was "whetted". With the construction described developing explosions could be successful controlled with a static pressure of up to about 1 MPa.

EXPLOSION-PROOF AIR LOCK

When explosion-proof ventilation structures are used some partial ventilation must, at last, be provided between the structures if an auxiliary ventilation system is not to be used. An explosion-proof version was likewise developed for this air split. This could, for the most part, be based on the already developed explosion-proof lay-out of the belt conveyor gate. The only modification to this, but one which, however, had a great influence on the closing time, was a horizontal method of closing the slide valve, because it cannot always be guaranteed that valves will work vertically. Fig. 9 shows the combined use of both valves. The working principle is the same in both valves. Constructive measures can be taken to ensure that even with the horizontally closing valve the closing times are about 100 ms.

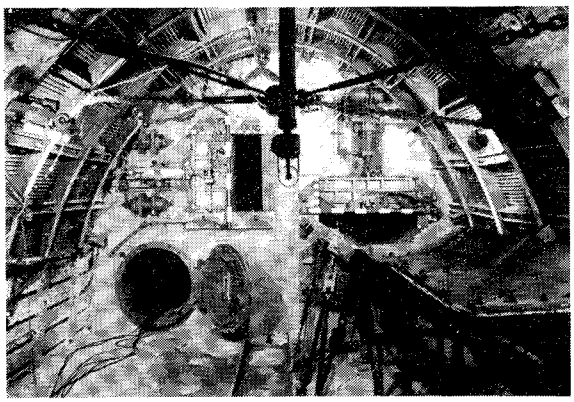


Figure 9. Combined installation of automatic gates for a belt conveyor and for partial ventilation

CONTROL DEVICES

When explosion-proof ventilation structures are being fitted with the necessary control devices, a series of safety measures, not necessary for the normal type of ventilation structures, must be adopted. These include:

- Functional capacity of the components and of the control devices to be maintained even after an explosion
- Preventing accidental opening of the air doors during the explosion by means of the control devices
- Laying of temperature-proof and mechanically resistant control lines
- Installation of the control switchboard on the side of the stopping facing towards the explosion
- Control units, like manually controlled contacts, should be small in area so that they cannot be activated by the explosion pressure

- Use of detectors already tried out in underground explosions and which work selectively, in addition to the relevant necessary control devices.

OTHER MATTERS

The requirements for an explosion-proof ventilation structure, to resist an explosion pressure of approximately 1 MPa, ca be fulfilled only when the structure is installed in solid rock and is firmly connected with the latter.

In addition to the constructive measures devised in order to make the components explosion-proof, it was found necessary, considering the strength of the structure, to give increased attention to the stopping. In the course of the tests, it was found that the strength of the structure could be guaranteed if the following two measures were adopted:

1. Depending on the road cross-section, we had to ensure that the stiffeners between the separate components in the stopping had a certain minimum thickness. With a road cross-section of approximately 16 m², the minimum thickness of the individual stiffeners had to be at least 0.6 m. In the case of road cross-sections of 28 m², the stiffeners had to be at least 1.00 m. Furthermore, the stopping had to be set into the solid rock for a distance of approximately 0.5 m.

2. The formula, referred to in the general requirements, for calculating the thickness of the stopping had to be modified so that, for these particular structures, it ran:

$$l_D = \frac{0.95 \times a}{\sqrt{\sigma_{bz}}} \qquad (2)$$

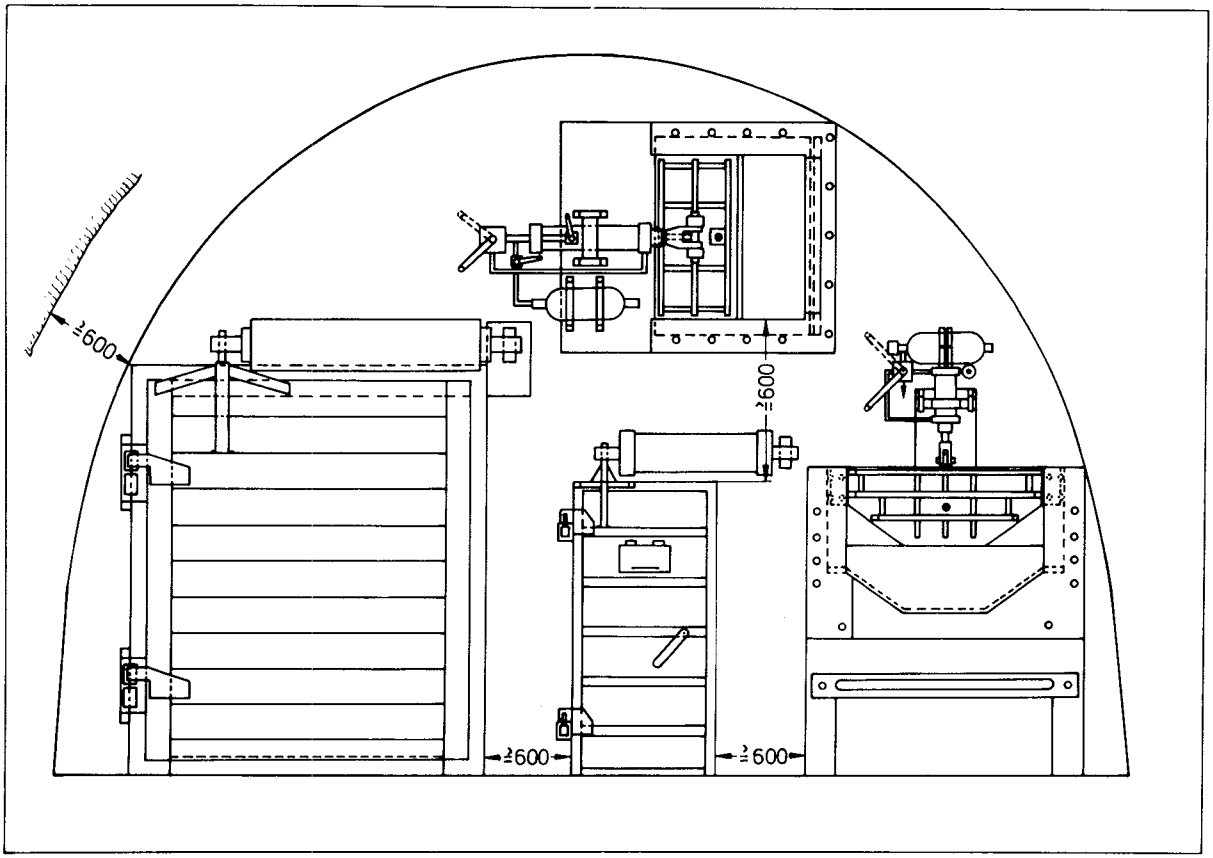


Figure 10. Schematic diagram of a group of explosion-proof components for haulage, man-riding and ventilation in a stopping

Ventilation structures of the form described here are special structures which are of very great value locally as regards safety. As the components cannot be installed in large quantities, they are correnspondingly expensive. At present, a ventilation structure fitted with all the components shown in fig. 10 would cost about 75 000 to 100 000 DM. This includes the components as well as the control gear and trigger device for the slide valves but does not include the costs of installation.

SUMMARY

In the years 1983 to 1987, the Tremonia Experimental Mine developed explosion-proof components for ventilation structures, designed to resist a maximum static pressure of approximately 1 MPa. The explosion/technical requirements were also fulfilled, resulting in a decisive contribution to the safety of personnel in the event of a mine underground.

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ENTWICKLUNG VON EINBAUTEN FÜR WETTERBAUWERKE IM BERGBAU UNTER TAGE, DIE EINEM EXPLOSIONSDRUCK VON RUND 1 MPa STANDHALTEN

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ZUSAMMENFASSUNG

Explosionen im Kohlenbergbau unter Tage können durch präventive Schutzmaßnahmen vermieden, durch konstruktive Schutzmaßnahmen bekämpft oder in ihrer Ausbreitung behindert werden. Dies erfolgt durch Explosionssperren und -dämme. Da Dämme eine hermetische Abriegelung eines Teiles des Grubengebäudes bedeuten, wird betrieblicherseits gefordert, daß Öffnungen für Fahrung, Förderung oder Bewetterung vorhanden sein sollten. Um dies mit den explosionstechnischen Anforderungen in Einklang zu bringen, wurden bei der Versuchsgrube Tremonia in den Jahren von 1983 bis 1987 explosionsfeste Einbauten wie Fahr- und Förderwegtüren sowie Gurtförderer- und Bewetterungsschleusen entwickelt. Diese Einbauten mußten einem statischen Explosionsdruck bis zu 1 MPa (10 bar) standhalten.

EINLEITUNG

Trotz aller Sicherheitsmaßnahmen gegen die Entstehung und Ausbreitung von Explosionen im Bergbau unter Tage treten jährlich doch noch zahlreiche Ereignisse ein, so daß immer wieder verletzte und tödlich verletzte Bergleute zu beklagen sind. Nach einer inoffiziellen Statistik kamen im Zeitraum von 1980 bis 1987 weltweit alleine durch Explosionsereignisse mehr als 1500 Bergleute ums Leben. Die Anwendung konstruktiver Explosionsschutzmaßnahmen in Ergänzung zu Präventivmaßnahmen kann in Form von Explosionssperren oder explosionsfesten Einbauten erfolgen (Bild 1).

Der Einsatz von Explosionssperren im gesamten Grubengebäude soll sicherstellen, daß die Auswirkungen einer Explosion auf einen möglichst kleinen Raum beschränkt bleiben. Explosionssperren sind, bei richtiger Anwendung, in der Lage, die Explosionsflamme im Sperrenbereich oder einige Meter dahinter abzulöschen. Der Explosionsdruck wird zwar ebenfalls durch die Explosionssperren verringert, jedoch bedarf es zum Teil einer Auslauflänge von mehreren hundert Metern, wenn nicht sogar Kilometern, bis der Bereich erreicht ist, an dem der Explosionsdruck keinen Schaden mehr anrichtet (Michelis 1983).

Explosionsfeste Einbauten sind vorrangig Dämme. Mit ihnen wird eine hermetische Abriegelung von Teilen eines Grubengebäudes bewirkt. Diese Abriegelung stellt den wirkungsvollsten Explosionsschutz dar, da durch sie im Falle einer Explosion die Ausbreitung der mechanischen, thermischen und toxischen Auswirkungen verhindert wird.

Jedoch stellt aus betrieblicher Sicht eine solche hermetische Abriegelung nicht immer ein Optimum dar. In Sonderfällen sollen durch einen explosionsfesten Damm bestimmte Betriebsabläufe wie Fahrung, Förderung und Bewetterung jedoch weiterhin aufrechterhalten bleiben. Dies trifft besonders dann zu, wenn explosionsfeste Dämme auch die Aufgabe von Funktionsbauwerken übernehmen, wie dies bei Wetterbauwerken zum Beispiel der Fall sein kann. Werden Wetterbauwerke durch Explosionen beaufschlagt

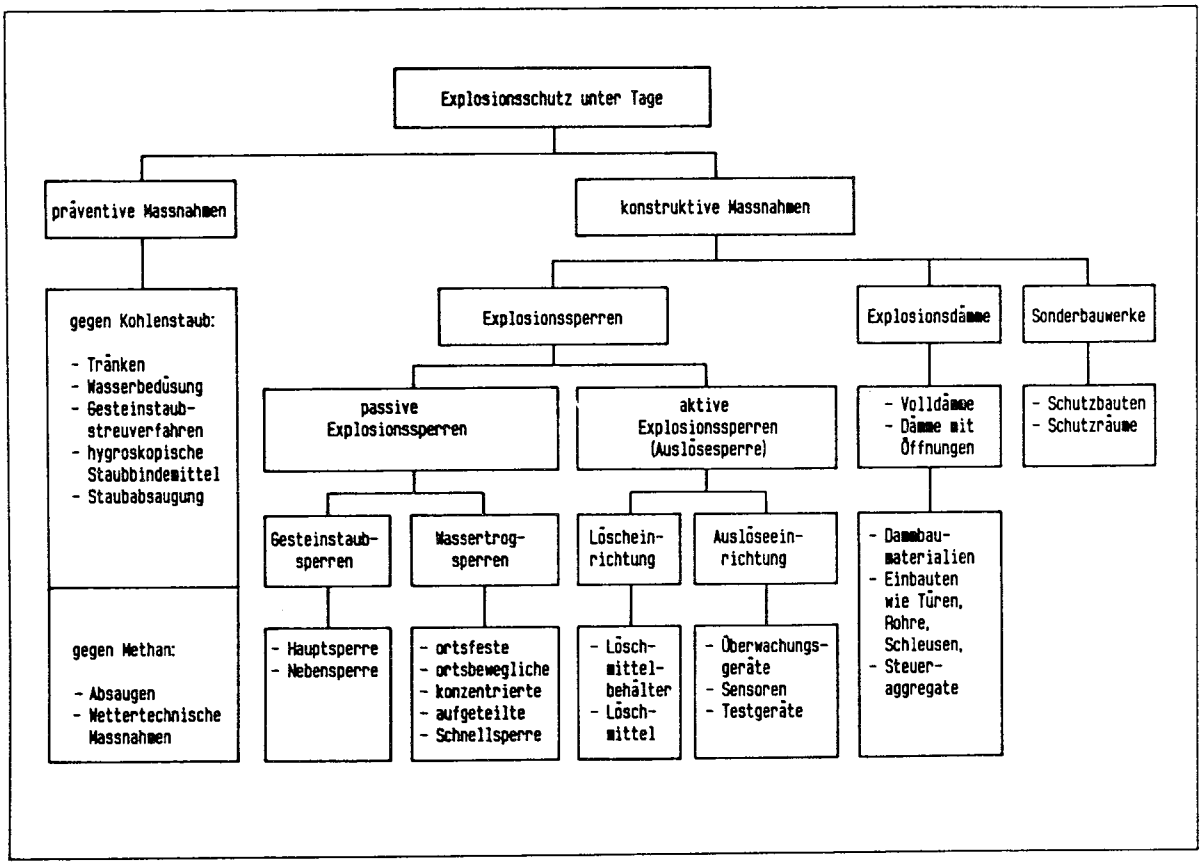


Bild 1: Gliederung der Maßnahmen für den Explosionsschutz unter Tage

und beschädigt, können wettertechnische Auswirkungen eintreten, die schwerwiegende Folgen für die Belegschaft in weiten Teilen eines Grubengebäudes haben können. In Ausnahmefällen sollte daher ein Wetterbauwerk explosionsfest erstellt werden können. Dies wird immer dann der Fall sein, wenn es sich bei dem zu schützenden Bereich des Grubengebäudes um eine wettertechnische Ausnahmesituation handelt und demzufolge eine Frischwetterversorgung dieses Bereiches nur durch die zu schützenden Strecken möglich ist, wie es zum Beispiel beim Unterwerksbau der Fall ist, oder bei Verbundbergwerken, bei denen Schachtanlagen auf gleichem Sohlenniveau miteinander in Verbindung stehen oder in Wetterabteilungen, bei denen der geforderte Abstand von 200 m zwischen Anfangs- und Endpunkt einer Wetterabteilung nicht eingehalten werden kann.

Den bisher in der Praxis erstellten explosionsfesten Wetterbauwerken lagen vorrangig Erkenntnisse aus theoretischen Betrachtungen und zum Teil auch Erfahrungen aus Explosionsversuchen bei der Versuchsgrube Tremonia zugrunde.

Orientierende Explosionsversuche mit "explosionsfesten" Wetterbauwerken sind bereits schon vor rund 40 Jahren bei der Versuchsgrubengesellschaft mbH (Versuchsgrube Hibernia) durchgeführt worden, ergaben aber keine, nach heutigen Vorstellungen brauchbaren Lösungen (N.N. 1944).

ALLGEMEINE ANFORDERUNGEN AN EXPLOSIONSFESTE EINBAUTEN

Bei der Errichtung explosionsfester Dämme mit Einbauten müssen entsprechende Anforderungen an

- das Dammbauwerk sowie an
- die Einbauten

gestellt werden.

Explosionsfeste Dämme müssen im westdeutschen Steinkohlenbergbau nach den bergbehördlichen "Dammbau-Richtlinien" erstellt werden. Hierdurch sind sowohl die zu verwendenden Dammbaumaterialien vorgeschrieben als auch der maximale statische Explosionsdruck, dem ein Damm standhalten soll. Dieser ist auf 0,5 MPa (5 bar) festgelegt, und beträgt unter Berücksichtigung einer entsprechenden Sicherheitsreserve bei den Prüfversuchen der Versuchsgrube Tremonia rund 1 MPa (10 bar). Die Dimensionierung der Dammstärke für sogenannte explosionsfeste Kurzdämme berücksichtigt neben der größten Abmessung einer Strecke vor allem die Biegezugfestigkeit des verwendeten Dammbaumaterials. Die Berechnung erfolgt nach der Formel

$$l_D = \frac{0{,}7 \times a}{\sqrt{\sigma_{bz}}} \qquad (1)$$

hierbei bedeuten:

l_D = Dammlänge in söhligen oder geneigten Grubenbauen in m

a = größte Breite oder Höhe der Strecke an der Dammbaustelle in m

σ_{bz} = Biegezugfestigkeit des Dammbaumaterials in N/mm²

Damit ein Wetterbauwerk als "explosionsfest" bezeichnet werden kann, müssen folgerichtig auch die Einbauten den gleichen Druckbelastungen ausgesetzt werden können wie der explosionsfeste Damm selbst, das heißt, sie werden ebenfalls bei den Explosionsversuchen einem statischen Druck von rund 1 MPa ausgesetzt.

Außer diesen explosionstechnischen Anforderungen muß auch den betrieblichen Anforderungen Rechnung getragen werden. Das bedeutet, daß die Einrichtungen für Förderung und Fahrung sich möglichst an bereits bestehenden Normmaßen orientieren und sich dem normalen Betriebsgeschehen anpassen müssen.

Bei den in den Jahren von 1983 bis 1987 von der Versuchsgrube Tremonia durchgeführten Entwicklungen wurde daher besonderes Augenmerk auf die betriebliche Durchführbarkeit der Einbauten gerichtet.

EXPLOSIONSFESTE EINBAUTEN FÜR WETTERBAUWERKE

Um ein explosionsfestes Wetterbauwerk den notwendigen Betriebsabläufen anpassen zu können, wurden folgende Einbauten entwickelt und in Explosionsversuchen auf ihre Verwendbarkeit hin getestet:

- Fahrwegtüren
- Förderwegtüren
- Gurtförderschleuse
- Bewetterungsschleuse

Die Aufgabenstellungen der aufgezählten Einbauten weichen zum Teil extrem voneinander ab. Während die Türen im Betriebsfall ständig geschlossen sein müssen, und nur im Bedarfsfalle geöffnet werden, sind Schleusen für die Förderung und Bewetterung ständig offen zu halten. Im Falle einer Explosion müssen aber alle Öffnungen am Damm geschlossen sein. Daraus resultieren unterschiedliche Konstruktionsanforderungen an die einzelnen Einbauten.

Alle Explosionsversuche, die zum Testen der Einbauten durchgeführt wurden, fanden in der rund 16 m² großen Explosionsstrecke R 2 auf der 4. Sohle der Versuchsgrube Tremonia statt.

VERSUCHSBEDINGUNGEN

Zur Erprobung der explosionsfesten Einbauten wurden jeweils explosionsfeste Dämme als sogenannte Kurzdämme mit einer Dammstärke von rund 1,50 m erstellt.

Die Druckbeaufschlagung erfolgte durch Explosionen, die sowohl als Schlagwetter- als auch als Kohlenstaubexplosionen ausgelegt waren. In Bild 2 wird der Versuchsort dargestellt.

In Abhängigkeit von der Länge der jeweiligen Brennstoffzone und der Brennstoffkonzentration ließen sich allseitige (statische) Drücke am Damm zwischen 0,2 bis 1,0 MPa erzielen.

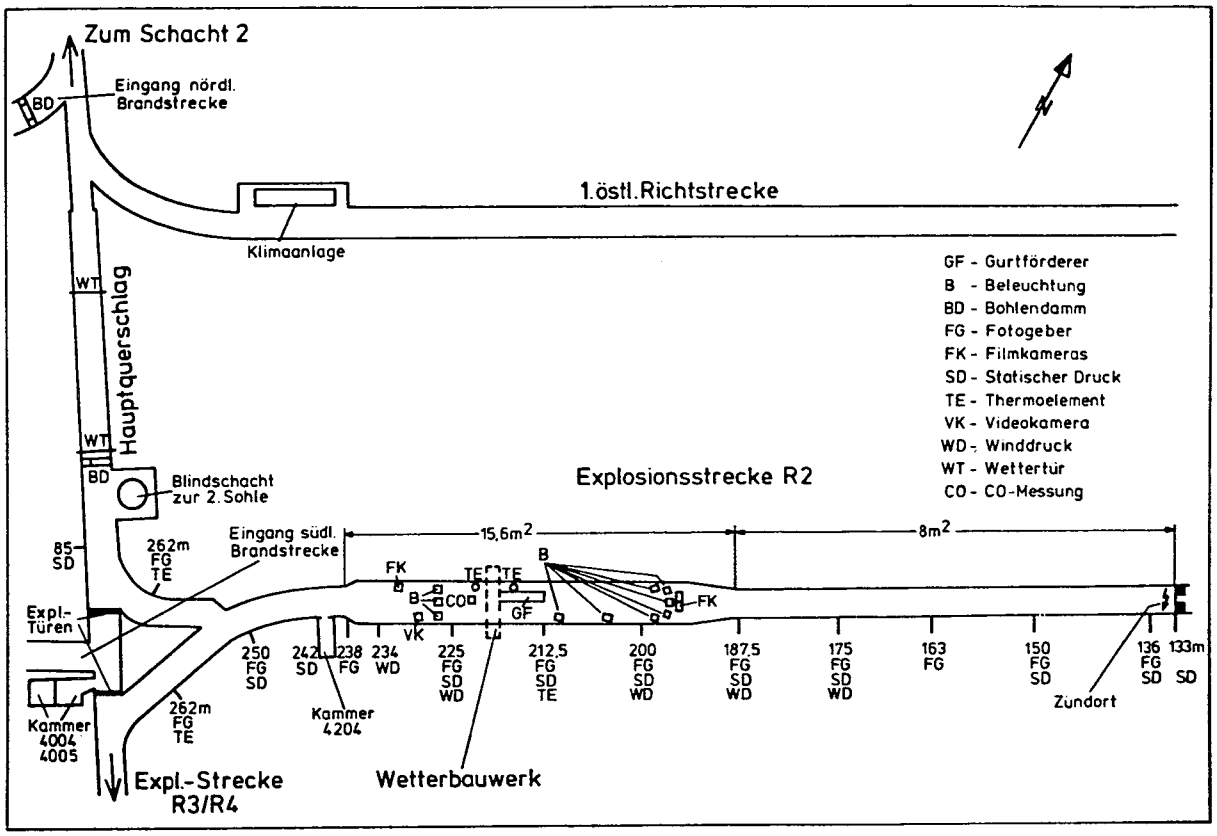


Bild 2. Meßtechnische Ausrüstung der Explosionsstrecke R 2

Die bei den Explosionsversuchen eingesetzte Meßtechnik ergab sich aus der Standard-Ausrüstung der Explosionsstrecke zur Erfassung von Druck, Flamme, Temperatur und CH_4-Konzentration sowie aus ergänzenden Ausrüstungen im Nahbereich des Dammes, wie unter anderem Hochgeschwindigkeits- und Videokameras sowie CO-Analysatoren.

EXPLOSIONSFESTE FAHRWEGTÜREN

Fahrwegtüren für den westdeutschen Steinkohlenbergbau unter Tage sind nach der Deutschen Norm (DIN) in ihren Abmessungen festgelegt. Danach kann die lichte Breite zwischen 600 mm und 800 mm betragen und die lichte Höhe zwischen 1800 mm und 2000 mm. Bei der Entwicklung von explosionsfesten Fahrwegtüren wurden die Maße 700 mm x 1800 mm und 800 mm x 1800 mm zugrunde gelegt.

Die Entwicklung explosionsfester Fahrwegtüren erwies sich als nicht so schwierig wie die der Förderwegtüren, zumal hier vor allem auf die bei der Versuchsgrube Tremonia eingesetzten explosionsfesten Meßkammertüren mit einer Materialstärke von 30 mm als Vorgabe zurückgegriffen werden konnte. Türen dieser Art sind in der oben aufgeführten Größe durchaus in der Lage, Explosionsdrücken bis zu 2 MPa standzuhalten (Bild 3). Der im Laufe der Entwicklung vorgenommene Übergang zu einem gewölbten Türblatt zeigte, daß - ähnlich wie bei den Förderwegtüren - mit erheblichen Materialeinsparungen die maximalen Druckanforderungen erfüllt werden konnten (Bild 4). Allerdings ist ein Türblatt, das "plan" hergestellt werden kann, erheblich leichter zu fertigen.

Die Erprobung der Fahrwegtüren erfolgte jeweils mit einer komplett ausgerüsteten und voll funktionierenden pneumatischen Steuereinrichtung zum Öffnen und Schließen der Tür (Bild 5). Zusätzlich wurde bei den Fahrwegtüren stets eine Öffnungshilfe gefordert für den Fall, daß sich die Tür nach Explosionsdruckeinwirkungen pneumatisch nicht mehr öffnen läßt.

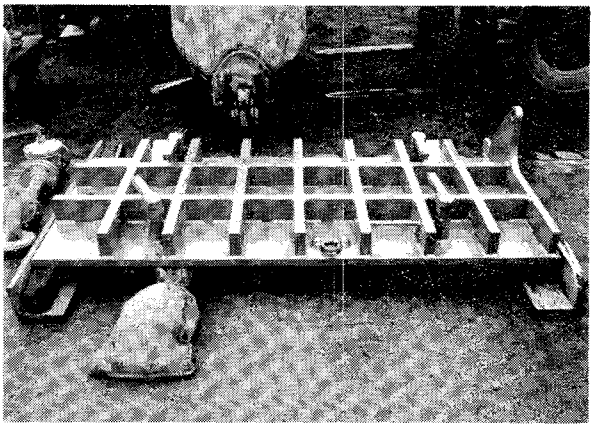


Bild 3. Explosionsfeste Fahrwegtür (flach) mit dem lichten Maß von 700 mm x 1800 mm - Materialstärke des Türblattes 30 mm - Gewicht des Türblattes 930 kg

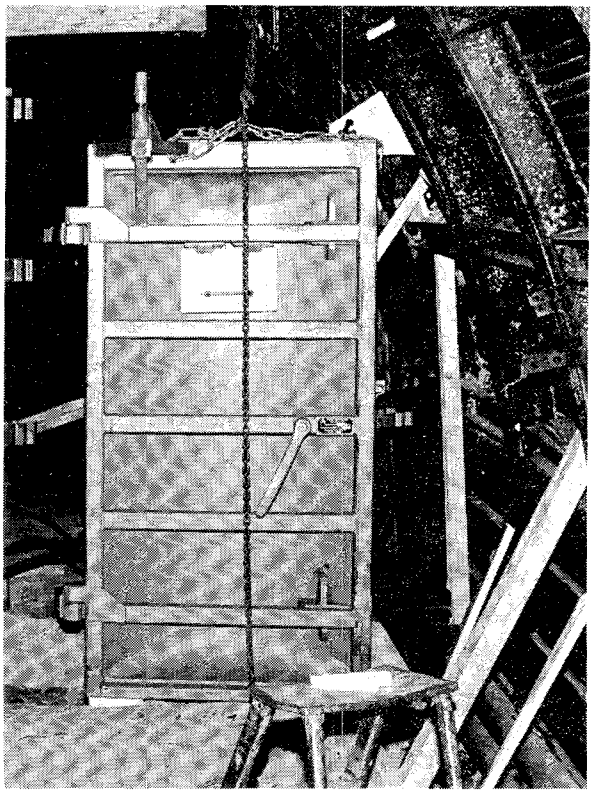


Bild 4. Explosionsfeste Fahrwegtür (gewölbt) mit dem lichten Maß von 800 mm x 1800 mm - Materialstärke des Türblattes 4 mm - Gewicht des Türblattes 240 kg

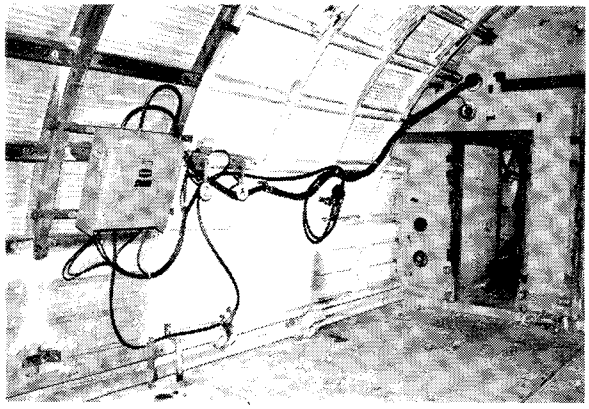


Bild 5. Pneumatische Steuervorrichtung für Wettertüren (explosionsabgewandt)

EXPLOSIONSFESTE FÖRDERWEGTÜREN

Auch die Förderwegtüren sind nach DIN genormt. Danach kann das lichte Maß für die Türbreite zwischen 1400 mm und 2600 mm variieren und für die Türhöhe zwischen 2000 mm und 2300 mm. Vorrangig werden in den Betrieben sogenannte zweiflügelige, druckentlastete Wettertüren eingesetzt.

Um den explosionstechnischen Anforderungen jedoch gerecht werden zu können, mußten die explosionsfesten Türen einflügelig konstruiert werden. Wegen des vorgegebenen Streckenquerschnittes von 15,6 m² in der Explosionsstrecke R 2 und der Vorgabe, bei den Untersuchungen jeweils eine Förder- und Fahrwegtür gleichzeitig einsetzen zu können, wurden für die Förderwegtür die lichten Maße von 1600 mm x 2000 mm gewählt. In einem späteren Versuchsabschnitt kam auch eine Förderwegtür mit den lichten Maßen von 1800 mm x 2300 mm zum Einsatz (Michelis, Kleine, 1986).

Die im Rahmen der bei der Versuchsgrube Tremonia durchgeführten Explosionsversuche zum Einsatz gebrachten Türtypen waren alle pneumatisch oder von Hand zu betätigen. Mit der Entwicklung einer gewölbten Türblattkonstruktion konnte eine Förderwegtür gefunden werden, die sowohl den explosionstechnischen als auch den betrieblichen Anforderungen gerecht

wurde (Bild 6). Plan gearbeitete Türblätter, auch wenn sie durch eine zusätzliche Verrippung verstärkt worden waren, hielten den Druckanforderungen von 1 MPa nicht stand. Mit dem Einsatz eines Türmaßes von 1800 mm x 2300 mm konnte schließlich auch eine Tür (gewölbt) für große Transporteinheiten erfolgreich erprobt werden.

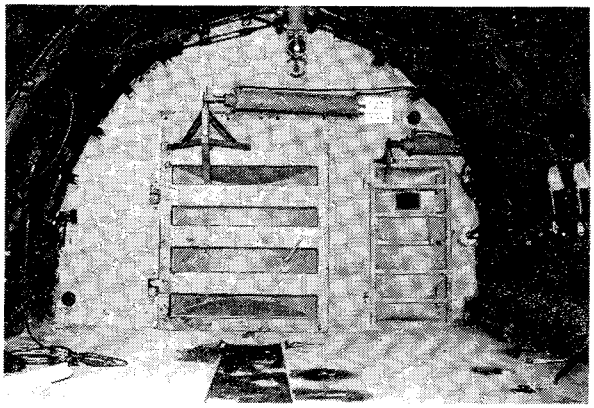


Bild 6. Explosionsfeste Fahrwegtür (gewölbt) mit dem lichten Maß von 1600 mm x 2000 mm - Materialstärke des Türblattes 7 mm - Gewicht des Türblattes 1200 kg

EXPLOSIONSFESTE GURTFÖRDERERSCHLEUSEN

Da eine Gurtfördererschleuse im Normalbetrieb stets geöffnet sein muß, ist durch konstruktive Maßnahmen zu gewährleisten, daß im Falle einer Explosion durch diese Öffnung keine Explosionsauswirkungen in den zu schützenden Bereich gelangen.

Um dieses Ziel zu erreichen, wurden im Rahmen der Untersuchungen verschiedene Maßnahmen erprobt. Ausgangsbasis war der Einsatz eines Gurtförderers mit einer Bandbreite von 1400 mm, der größten zur Zeit im westdeutschen Steinkohlenbergbau unter Tage eingesetzten Gurtbreite.

Versuche mit entsprechenden Einbauten in der Schleusenöffnung in Form von Gummi- oder Kettenschürzen führten ebenso wenig zum Erfolg wie Plattenpakete. Zum einen hätten diese Einbauten eine kontinuierliche Kohlenförderung nicht mehr ermöglicht, zum anderen schlugen trotz dieser Hindernisse die Explosionsauswirkungen voll durch die Öffnung.

Es wurde daher eine Technik entwickelt, bei der mit Hilfe eines Schiebers die Schleusenöffnung im Falle einer Explosion kurzfristig geschlossen werden konnte. In Bild 7 ist die schematische Darstellung des Einbaus einer explosionsfesten Gurtfördererschleuse mit automatisch wirkendem Schieber wiedergegeben. Die

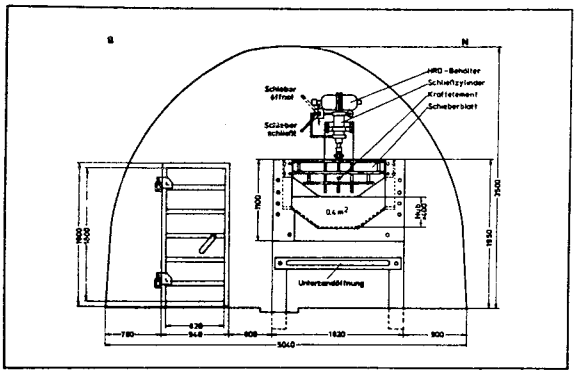


Bild 7. Schematische Darstellung einer Gurtfördererschleuse mit automatisch arbeitendem Schieberblatt

Anforderungen, die, neben der explosionsfesten Bauweise, an diese Schieberkonstruktion gestellt wurden, um einen Explosionsdurchschlag zu verhindern, lauteten unter anderem:

- Schließzeit 100 ms
- Öffnungsquerschnitt rund 0,4 m²
- Schließvorgang ohne Zerstörung des Gurtförderers.

Um diese Zielsetzung erreichen zu können, mußte das Schieberblatt mit Hilfe einer Fremdenergie bewegt werden. Dies geschah durch Einsatz eines Schließzylinders, der über einen HRD-Behälter mit Druckluft beaufschlagt wurde. Im HRD-Behälter befand sich Druckluft unter einem Überdruck von 4 MPa. Im Ruhezustand wurde das Schieberblatt durch ein Halteelement, ein sogenanntes Kraftelement, gehalten.

Im Falle einer Explosion muß als erstes das Ereignis erkannt werden. Dies geschieht mit den bereits für Auslösesperren verwendeten Detektoren, die auf thermischer oder optischer Basis arbeiten (Michelis 1985). Aus Bild 8 wird der zeitliche Ablauf des Schließvorganges deutlich.

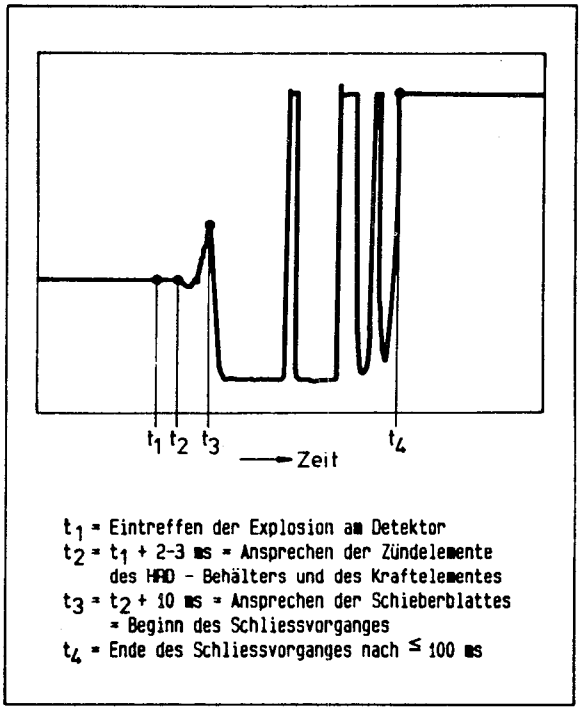


Bild 8. Zeitlicher Ablauf des Schließvorganges einer automatisch arbeitenden Gurtfördererschleuse

Nach Eintreffen der Explosionsflamme am Detektorstandort (t_1) bewirkt das Triggersignal die Auslösung der Zündelemente in dem HRD-Behälter und im Kraftelement (t_2). Hierfür werden maximal etwa 2 bis 3 ms benötigt. Dadurch werden 4 MPa Druck auf den Schließzylinder ausgeübt; gleichzeitig gibt das Halteelement das Schieberblatt frei (t_3). Dieser Vorgang dauert maximal rund 10 ms. Das Schließen des Schieberblattes selbst vollzieht sich in rund 60 bis 70 ms (t_4). Das Schieberblatt wird rund 10 mm oberhalb des Gurtförderers durch Pufferelemente aufgefangen, um eine Zerstörung des Gurtförderers zu verhindern. Damit Fremdkörper in Form von größeren Kohle-, Gesteins- oder Holzstücken den Schließvorgang nicht behindern und gegebenenfalls die Schleusenöffnung nicht ordnungsgemäß geschlossen werden kann, wurde das Schieberblatt "angeschärft". Mit der vorgestellten Konstruktion konnten Laufexplosionen mit einem statischen Druck bis zu rund 1 MPa erfolgreich aufgehalten werden.

EXPLOSIONSFESTE WETTERSCHLEUSE

Beim Einsatz explosionsfester Wetterbauwerke muß zumindest für eine Teilbewetterung zwischen den Bauwerken gesorgt werden, wenn nicht eine Sonderbewetterung eingesetzt werden soll. Für diese Teilbewetterung wurde ebenfalls eine explosionsfeste Version entwickelt. Dabei konnte man sich größtenteils auf die bereits entwickelte explosionsfeste Auslegung der Gurtfördererschleuse stützen. Die einzige Änderung hierzu, die aber einen starken Einfluß auf die Schließzeit haben konnte, war ein waagerechter Schließvorgang des Schiebers, da betrieblicherseits nicht immer gewährleistet sein kann, daß die Schieber senkrecht arbeiten können. In Bild 9 ist der kombinierte Einsatz beider Schieber dargestellt. Das Arbeitsprinzip ist bei beiden Schiebern gleich. Durch entsprechende konstruktive Maßnahmen konnte erreicht werden, daß auch bei dem waagerecht schließenden Schieber die Schließzeiten bei rund 100 ms lagen.

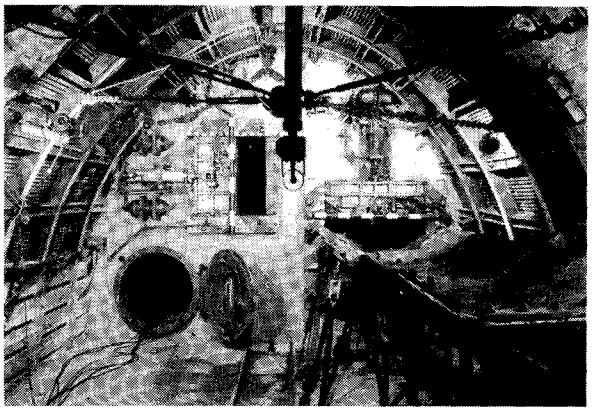


Bild 9. Kombinierter Einbau automatisch arbeitender Schleusen für einen Gurtförderer und für eine Teilbewetterung

STEUEREINRICHTUNGEN

Bei der Ausrüstung explosionsfester Wetterbauwerke mit den notwendigen Steuereinrichtungen ist eine Reihe von Sicherheitsmaßnahmen zu beachten, die für übliche Wetterbauwerke nicht notwendig sind. Dies sind unter anderem:

- Funktionsfähigkeit der Einbauten nebst Steuereinrichtungen auch nach einem Explosionsereignis
- Verhindern eines unzeitigen Öffnens der Wettertüren während der Explosion durch die Steuereinrichtungen
- Verlegen temperaturbeständiger und mechanisch widerstandsfähiger Steuerleitungen
- Errichten der Steuerschränke auf der explosionsabgewandten Seite des Dammes
- Steueraggregate, wie Handsteuerkontakte kleinflächig auslegen, damit sie durch den Explosionsdruck nicht betätigt werden können
- Einsatz von in Explosionen unter Tage bereits erprobten Detektoren, die entsprechend selektiv arbeiten, nebst den dazu notwendigen Steuereinrichtungen.

SONSTIGES

Die Anforderungen, die an ein explosionsfestes Wetterbauwerk für einen Explosionsdruck von rund 1 MPa gestellt werden, sind nur zu erbringen, wenn das Bauwerk in standfestem Gebirge erstellt wird und fest mit diesem verbunden ist.

Neben den konstruktiven Maßnahmen, um die Einbauten explosionsfest zu entwickeln, hat es sich als notwendig gezeigt, dem Damm bei den Festigkeitsbetrachtungen erhöhte Aufmerksamkeit zu schenken. Durch zwei Maßnahmen, die sich im Laufe der Versuchsdurchführungen ergeben haben, konnte die Standfestigkeit des Bauwerkes gewährleistet werden. Dies waren

1. In Abhängigkeit vom Streckenquerschnitt mußte eine Mindeststärke der Dammrippen zwischen den einzelnen Einbauten gewährleistet sein. Bei einem Streckenquerschnitt von rund 16 m² mußte die Mindeststärke der einzelnen Dammrippe mindestens 0,6 m, bei Streckenquerschnitten um 28 m² mußte die Dammrippe mindestens 1,00 m betragen. Des weiteren mußte der Damm rundum etwa 0,5 m ins feste Gebirge gesetzt werden.

2. Die bei den allgemeinen Anforderungen aufgeführte Formel zur Berechnung der Dammstärke mußte modifiziert werden, so daß sie für diese Sonderbauwerke lautete:

$$l_D = \frac{0{,}95 \times a}{\sqrt{\sigma_{bz}}} \qquad (2)$$

Wetterbauwerke der hier vorgestellten Bauweise stellen Sonderbauwerke dar, die in ihrer Aufgabenstellung einen hohen sicherheitlichen Stellenwert haben. Da die Einbauten in keinen hohen Stückzahlen erstellt werden können, sind sie entsprechend teuer. Nach dem derzeitigen Stand würde ein Wetterbauwerk, das mit allen, wie in Bild 10 dargestellten Einbauten ausgerüstet ist, rund 75.000 DM bis 100.000 DM kosten. Dies beinhaltet die Einbauten, nebst Steuergeräten und Auslöseeinrichtung für die Schieber, schließt aber nicht die Erstellungskosten ein.

ZUSAMMENFASSUNG

Die Versuchsgrube Tremonia hat im Zeitraum von 1983 bis 1987 explosionsfeste Einbauten für Wetterbauwerke entwickelt, die einem maximalen statischen Druck von rund 1 MPa standhalten. Dabei ist es gelungen, neben den explosionstechnischen Forderungen auch den betrieblichen gerecht zu werden. Es konnte somit ein entscheidender Beitrag zur Erhöhung der Sicherheit der Belegschaften bei Explosionsereignissen im Bergbau unter Tage geleistet werden.

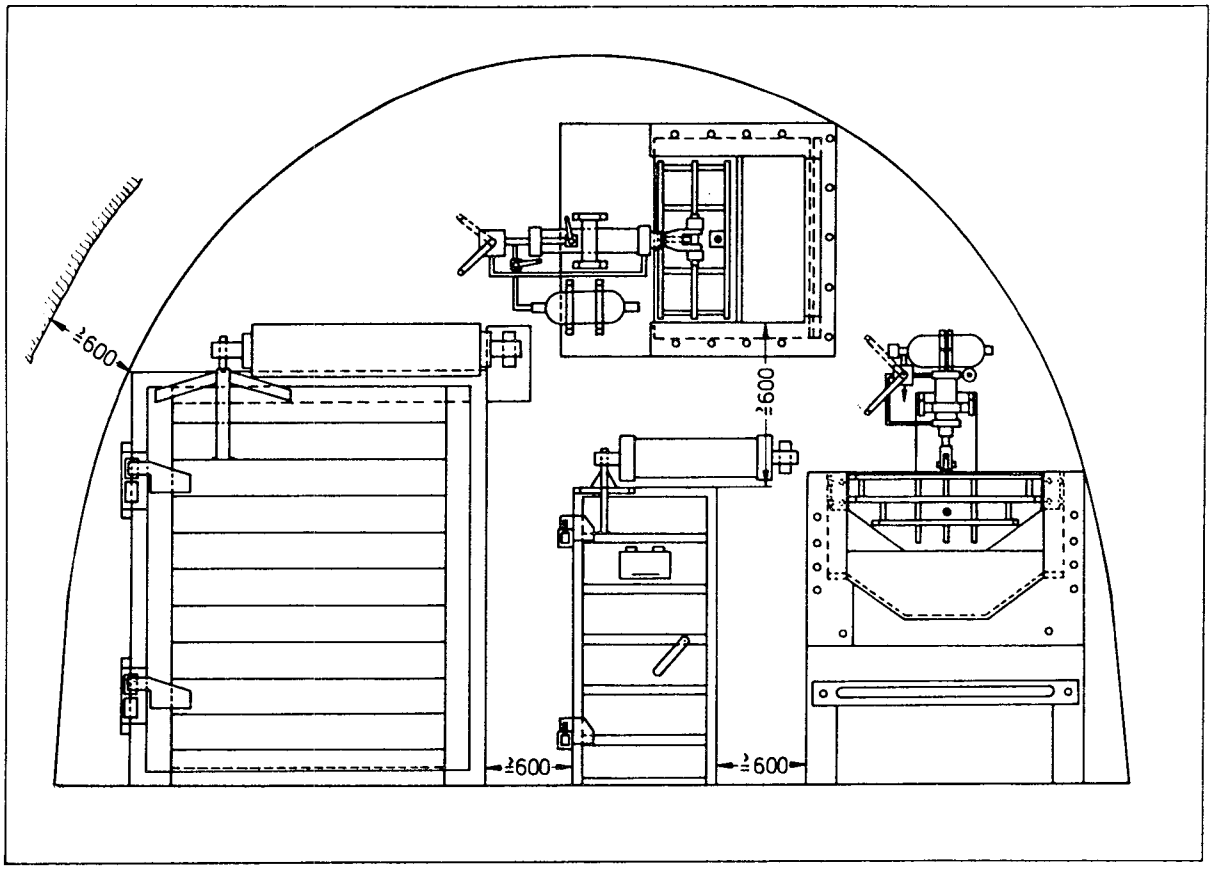


Bild 10. Schematische Darstellung einer Zusammenstellung von explosionsfesten Einbauten für Förderung, Fahrung und Bewetterung in einem Damm

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DEVELOPPEMENT D'ELEMENTS CONÇUS POUR RESISTER A DES PRESSIONS EXPLOSIVES DE 1 MPA ENVIRON, DESTINES A ETRE UTILISES DANS LES SYSTEMES DE VENTILATION DES MINES SOUTERRAINES

J. Michelis et W. Kleine

Dans les mines d'anthracite d'Allemagne de l'Ouest, la propagation des explosion se limite à de petites parties des mines, grâce à des fosses à eau constituant des barrières déflagrantes. Pour empêcher par exemple la destruction des principales structures de ventilation par les effets mécaniques d'une explosion, il faut que ces structures soient anti-déflagrantes, conformément aux normes de construction de barrages de l'Allemagne de l'Ouest; elles doivent également pouvoir résister à des pressions statiques de 0,5 MPa maximum. Les éléments qui composent ces structures de ventilation doivent pouvoir résister à des pressions similaires. Dans le cadre d'un projet de recherches, on a développé un moyen de hissage, des ouvertures de transports anti-déflagrants, ainsi que des bandes de convoyeurs et des soupapes de ventilation également anti-déflagrants. Sur la base de l'expérience acquise au cours de ces essais, on a établi une liste des exigences qui servent de directives à l'industrie minière, aux fabricants et aux organismes d'Inspection des Mines.

РАЗРАБОТКА КОМПОНЕНТОВ ВЕНТИЛЯЦИОННЫХ СТРУКТУР, СПОСОБНЫХ ВЫДЕРЖАТЬ ДАВЛЕНИЕ ОТ ВЗРЫВНОЙ ВОЛНЫ В 1 MPa

Ж. Михелис и В. Кляйн

В западногерманских антрацитовых шахтах распространение взрывов ограничивается с помощью водяных канав-барьеров, делящих шахту на маленькие участки. Для предотвращения разрушения взрывом главных вентиляционных систем, они должны быть спроектированы таким образом, чтобы противостоять взрывам, в соответствии с западно-германской "Инструкцией по сооружению дамб", и они должны быть способны выдержать максимальное статическое давление в 0.5 MPa. Отдельные компоненты этих вентиляционных структур подчиняются тем же правилам. В процессе исследований разрабатывались взрывостойкие способы откатки и пути транспортировки на поверхность, а также взрывостойкие конвейерные ленты и вентиляционные клапаны. На основании результатов испытаний был составлен список необходимых требований, который послужит руководством для горной промышленности и инспекционного управления шахт.

地下煤矿中能耐约1兆帕斯卡爆炸压力的通风设备另件设计

J。密歇力斯 及 W。克郎耐

西德硬煤矿中我们用水槽防止爆炸的璀栏以致爆炸的传播仅限于矿中的一小部分。为防止爆炸对通风设备由于机械作用所造成的损坏，这些设备必须据西德“挡墙建造规格”作防爆设计，使它能承受0。5兆帕斯卡的最大静压力。通风设备另件也必须能承受同样的压力。防爆搬运设备，防爆皮带输送机，及通风阀均在一项研究项目中研究发展完成。从这些研究和试验的经验里我们订出一些规定和准则以作煤矿工业，制造商及煤矿检查机关的参考。

COMPUTER SIMULATION OF EXPLOSIONS

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ABSTRACT:

The authors are developing computer simulation codes to predict the type of explosions that occur in mines. The simulation relies on solving the equations of motion and combustion. To model combustion a simple one-step mechanism is assumed. Turbulence effects are incorporated and a Finite Volume method is used to descretise the eqations which are solved using a modified Runge-Kutta integration proceedure. Many examples of computations are presented, with and without combustion including a simulation of a gas explosions in the US Bureau of Mines Lake Lynn Facility. The results of all the cases considered are compared with experimental data. Further developments to include the effects of partical combustion and radiation will be briefly described.

INTRODUCTION:

The Australian Coal Mining Industry has a high fatal incident rate compared to other industries. About one tenth of the deaths over the last twenty years have been due to explosion. Such incidents, although infrequent, have occurred on average every five years with a death toll between 5 and 15 per accident. While much has been done to reduce the number of accidents by trying to eliminate potential ignition sources and through the use of inerting, the explosions at Moura in 1986 and Appin in 1979 demonstrated the need for a better understanding of explosion development and control procedures in the mine roadway systems used throughout Australia.

Much of our understanding of coal mine explosions comes from experimental tests undertaken in explosion galleries throughout the world. The majority of these galleries are single entry, a few having double entries with one or two crosscuts linking the entries. The cross-sections conform to the mining roadways in the respective country. Although these galleries bear little resemblance to mining practices in Australia, such experiments have clearly demonstrated that:

1) Explosions are more difficult to start in larger cross-sectional roadways.

2) Developing explosions are more difficult to suppress in larger cross-sectional roadways

3) Turbulence generated by obstructions accelerate flame propagation and can lead to detonation.

4) Turbulence generates higher pressures.

5) Stone dusting must exceed a critical concentration to suppress the development of a coal dust explosion for a given coal dust.

6) Suppression systems must exceed a critical concentration and a critical time parameter to successfully suppress an explosion for a given cross-section.

In practice, however, mining operations are not confined to the simple geometries associated with explosion galleries. Equipment, particularly in development headings, in the longwall and in belt roadways can cause substantial reduction in cross-section. Very little data exists on how even single obstacles accelerate an explosion.

Thus, questions arise as to how explosions will develop in more complicated geometries and with obstructions in roadways, without considering the further complications of siting and reliability of suppression systems in such geometries.

In order to address these outstanding problems in explosion development and control, the centre is undertaking a research program with the following objectives:

1) Develop 2 and 3-dimensional gas and coal dust explosion models.

2) Use the developed explosion models to study explosion development in various mine geometries subjected to constrictions and blockage.

3) Review the literature on barrier operation and develop computer models of their performance.

4) Use the barrier models to determine the effectiveness of different barrier systems relative to mine geometries.

The 3-dimensional explosion problem requires extensive computing time. Estimates of about 3 months CPU for one problem on the main-frame at Wollongong for a typical mine showed that a parallel computer would be necessary to reduce the problem to a tractable time. Considerable effort has gone into producing a code that can be run on Transputers to overcome this time constraint. It is now estimated that a full mine problem will run in 10 hours on a PC using 12 transputers.

THEORY:

An explosion involves a propagating pressure front with gases at high pressure and high temperature following it. The temperatures may be high enough to ignite the gas and may give rise to a flame front as well. Hence an explosion simulation will have to include prediction of fluid flow and combustion. The fluid flow itself is a complex phenomenon with the effects of convection and diffusion brought about by viscosity and turbulence. Combustion adds to this complication and requires chemical kinetics to handle it.

Fluid Flow model.

Any general flow is governed by the Navier Stokes equations (Fletcher,1988) and when writen in what is called the 'Conservative Form' will automatically capture shock-like gradients. In flows with combustion, the energy at any location is the sum of the internal, kinetic energies and the heat of combustion.

Effects of turbulence are included through the use of a two equation model of turbulence, the K - Epsilon model. This necessitates the solution of two additional differential equations for the turbulent kinetic energy and the dissapation rate.

In order to compute combustion one needs a good model for chemical kinetics. It is possible to write equations for each of the species involved in a chemical reaction and this warrants the solution of as many as 50 equations for a typical Methane-Oxygen system. The computational overhead for such an approach is almost prohibitive even for a general two dimensional flow. One alternative is to use a four step mechanism (Peters, 1985). This reduces the computational requirement considerably but in the present work a simple one step mechanism is used;

1 kg. fuel + s kg. oxygen = (1+s) kg. products.

Here, s is the stoichiometric ratio for the particular fuel.

For this simple scheme three additional equations need to be solved for the mass fraction of the fuel, the mixture fraction, and the variance of the mixture fraction.

The reaction rate is given by the Magnussen and Hjertager model (1976). This is a modification of the Spalding model (1971) and the justifications for it are available in the literature. Further, the above model demands that the chemical kinetics be fast. A switch is introduced based on

the chemical times and eddy life times in the usage of this model for the reaction rate.

To close the model, one also needs an equation of state. In actual usage the perfect gas equation is modified for a mixture of species (reactants, oxygen and products) and not a single gas.

In effect, we are solving 8 (9)equations of the same form for a two (three) dimensional flow. The various components of these equations are given in Green and Srinivas, 1988.

Solution Methods.

In the realm of Computational Fluid Dynamics many methods exist to solve the fluid flow problem without combustion. But it seems that only a couple of methods have found application to flows with combustion.

One of the methods is that employed by Hjertager (1985) and is based on the SIMPLE method due to Patankar and Spalding (1972). This is an implicit method which in principle can accommodate any value of the time step. This claim is justified for flows with a steady state solution. When it comes to transient flows, there is always a time step limitation. In fact, in the computation of the flow in an ignition tube, Hjertager has used a Courant number of only 0.5.

Another method is the FCT (Flux Corrected Transport) used by Boris et al (1974) and Green et al (1985). This is an explicit method and therefore the usable time step size is limited. It must be pointed out that the chemical kinetic model used by Boris et al. is different from the one used by us or by Hjertager.

The method chosen here differs from the two described above. The equations developed have been integrated in time by employing a modified Runge - Kutta scheme. The region of interest is divided into finite volumes (these degenerate into finite areas for a two dimensional flow). Without loss of generality we can describe the method for a two dimensional flow. For each of the areas the governing equations are applied and simplified using the Green's theorem. The governing equations now take the form;

$$\frac{dWA}{dt} + Q(W) = 0$$

where Q is an operator, A is the finite area.

Note that we have reduced the governing Partial Differential Equations to Ordinary Differential Equations. Many methods are available in literature to solve these. One of them is the Runge - Kutta method.It is a four level scheme and has very desirable properties;

a) The scheme is explicit and as a consequence the boundary conditions are easily implemented.

b)The allowable Courant number for the scheme is 2.8 whereas the other explicit schemes allow a Courant number of 1.0 only.

c) The scheme renders itself to parallel processing. This feature can make an explicit scheme yield solution far quicker than an implicit scheme even though the latter accepts a higher time step size.

Explosions are accompanied by large pressure gradients and additional diffusion is required to handle them. This is provided through the artificial dissipation terms. The one used here is a blend of second and fourth order terms and is adapted to the flow in the manner suggested by Jameson and Schmidt (1984).

CODE VALIDATION.

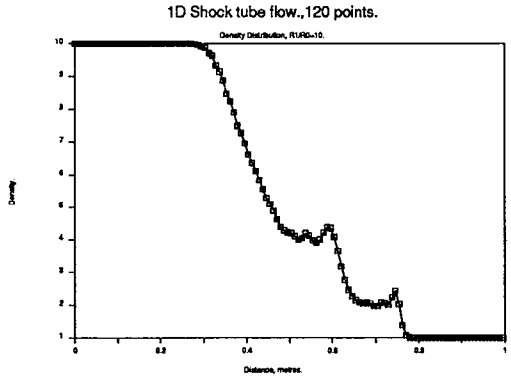
In order to validate the code a number of flows have been computed. Some of these flows are computed under one dimensional approximations and some under two dimensional approximations. Further, a few of the flows are treated ignoring the effects of viscosity and combustion. A brief description of the results obtained follows.

One Dimensional Shock tube flow.

This is a standard shock tube flow where initiallys a high pressure gas is separated by a

diaphragm from a low pressure gas in a tube. At t=0, the diaphragm is burst allowing the high pressure gas to expand. The result is a very familiar flow field consisting of a shock wave advancing into the low pressure gas at rest. Behind the shock is a contact surface, which in turn is followed by a rarefaction wave.

This example is computed with 120 cells to cover the tube and an initial pressure ratio of 10 across the diaphragm. The resulting density and pressure distributions are shown in Fig. 1. A sharp definition of the shock wave (spread around 4 cells) is evident. The contact discontinuity is smeared but this seems to be beyond control with the numerical methods available. It is to be noted that all the features of the shock tube flow are predicted well.



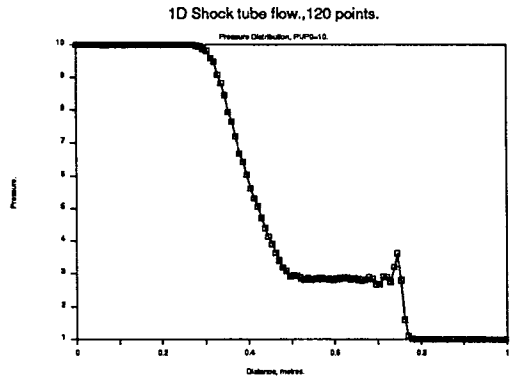


Fig. 1 Computed Shocktube Density and Pressure.

Two Dimensional Inviscid flow: Oblique Reflection of a Shock wave at a solid wall.

This two dimensional example is illustrated in Fig. 2. An oblique shock with flow upstream of it at a Mach number of 2.0 impinges on a solid

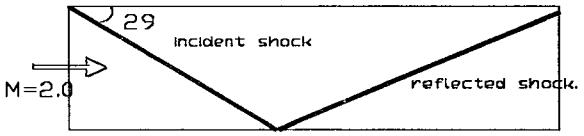


Fig. 2 Oblique Shock.

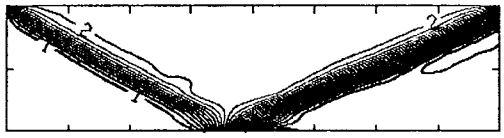


Fig. 3 Computed Pressure Contours.

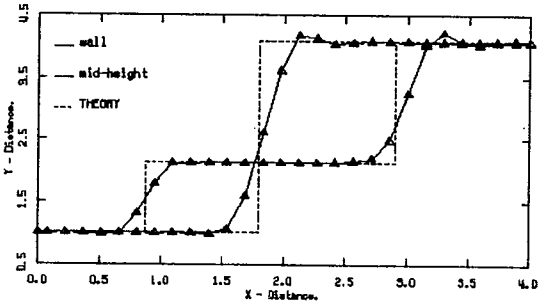


Fig. 4 Computed Pressure Distribution.

wall making a 29 degree angle. The shock gets reflected off the wall. The incident and reflected shocks as well as the flow behind them are calculated.

This problem has a steady state solution which was approached using time marching. The grid used was 30 by 15 and the grid spacing was uniform in both the directions. At the inlet uniform conditions for a Mach number of 2.0 were prescribed whereas at the downstream boundary extrapolation conditions were employed. On the wall the normal velocity was forced to vanish and the shock conditions were imposed on the freestream boundary.

On the NEC APC computer the convergence (density changes per time step almost negligible) was obtained in 20 minutes of CPU time. The same problem took less than 2 minutes with a transputer board using one T800 chip.

Fig. 3. shows the computed pressure contours. A clear definition of the incident and the reflected shocks is observed. A uniform flow region prevails between the two shocks and this is also predicted well. Shown in Fig. 4. are the pressure distributions on the wall and on a horizontal plane at mid height from the wall. It may be seen that the pressure distribution agrees very well with the theory. The shocks are smeared but the smearing is limited to within 3 or 4 cell widths. This can be further reduced by mesh refinement.

Flow in an Ignition Tube

This is a flow with combustion and was computed both with one and two dimensional approximations. A tube 10m long and 2.46m in diameter with the upstream end closed is considered. The tube is filled with a stoichiometric mixture of methane and air and at t=0 the mixture is ignited near the closed end. Combustion of methane sets up a pressure wave which travels down the tube. A flame front also develops and propagates behind the pressure wave. The complex flow resulting as a consequence of the propagation of the pressure front and the flame front is calculated.

The Ignition tube flow is first studied under one dimensional approximations. Turbulence is absent in a 1D flow and as such one cannot use the Magnussen and Hjertager model for the reaction rates. An Arhenius formula was used instead.

The tube length was divided into 50, 100 and 200 cells. Solid wall conditions were imposed on the left boundary which is that of the closed end. On the right (open) boundary, atmospheric pressure was imposed. Initially a reaction rate was imposed at the closed end to set off ignition. The flow was followed up to about 0.2 s from ignition.

Fig. 5. shows the distribution of fuel mass fraction as a function of time at a few locations. These imply a speed of 250 ms^{-1} for the combustion wave. The overpressure histories are plotted in Fig. 6. The initiation of the pressure

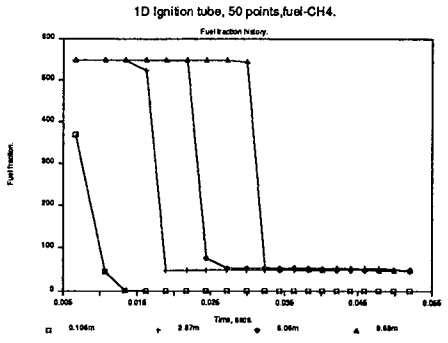


Fig. 5 Computed Fuel Mass Fractions.

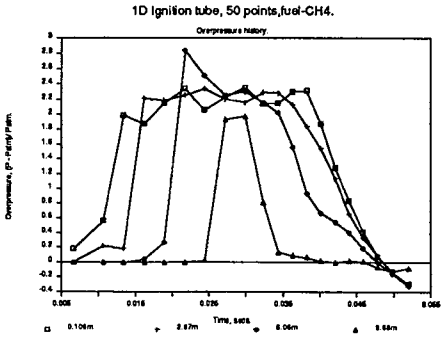


Fig. 6 Computed Overpressures.

pulse and its gradual decay are evident. The magnitude of the overpressures are clearly high. This is a consequences of the use of Arhenius formula which is known to give high reaction rates.

The calculations have revealed a grid dependence of the solution. It appears that the combustion wave travels faster on a coarse grid than on a fine grid. The overpressures predicted are smaller on a finer grid. The possible remedies to remove this grid dependence are being investigated.

Flow in an Ignition Tube fitted with rings

To validate the code for a more practical example, flow in an ignition tube fitted with rings to enhance turbulence levels was considered. The tube geometry is sketched in Fig. 7. There are five rings placed at uniform intervals and these block 30% of the free tube area. It may be pointed out that the enhanced turbulence levels help to accelerate the flame speed.

Exploiting the symmetry of the flow, only the top half of the tube geometry is computed. The inlet and outlet boundary conditions are identical to those use for the one - dimensional flow above. At the lower boundary, being one of symmetry, a no flow condition is imposed normal to the boundary. At the top boundary, which is a solid wall, velocities are set to zero.

The grid used has 30 cells in the flow direction and 18 in the normal direction. The starting conditions are that the pressure is uniform at atmospheric pressure and that close to the left wall, 10% of the fuel has been burnt. A stoichiometric composition of methane - air is present throughout the tube. Furthermore, a small reaction rate is imposed near the left to start with. This is to simulate ignition. The reaction rates were calculated using the Magnussen and Hjertager model.

The set of equations (9 in number) were 'stiff' and the stable solutions were possible only for a limited range of k and e values. This range of values predicted very low overpressures compared to experimental ones. To make the scheme accomodate a wider range of k and e values, dissipation levels in the sckeme had to be increased.

Further it was found that the prescribed initial distribution of k and e had a considerable effect on the rateat which burning occurred and hence on the overpressure behaviour. The initial dis-

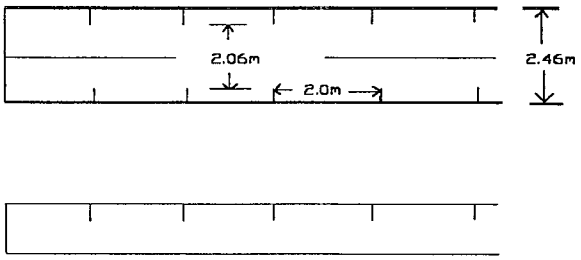


Fig. 7 Ignition Tube with Rings.

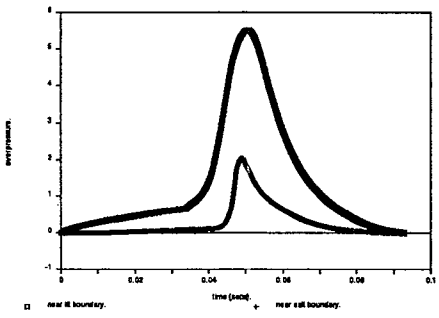


Fig. 8 Computed Fuel Fractions along Wall.

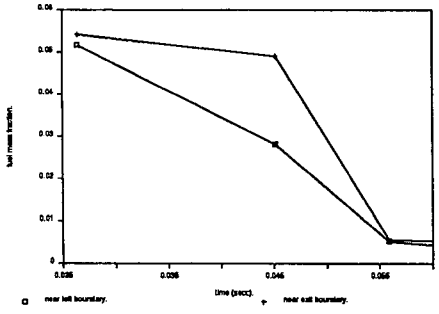


Fig. 9 Computed fuel fraction along centreline.

tribution was adjusted until the overpressure history matched the experomental data.

Fig. 8 shows the overpressure histories for locations close to the left boundary (ie. near to the ignition point) and near the open end of the tube. The magnitude and time at which peak overpressure occurs near the left boundary, closely compare with experiment (Hjertager, 1985). Close to the exit, the overpressure predicted is lower. Our computations revealed that the overpressure at this location has a strong dependence on the exit boundary conditions. Here, atmospheric pressure has been imposed on the exit plane and this boundary condition seems to inhibit the full growth of the overpressure at this location.

The fuel fraction histories are shown in Fig. 9 for locations near to the left and exit boundaries. A rapid burning at the latter boundary is evident. The fuel fraction contours, 0.06 s after ignition is shown in Fig. 10. Isolated burning at the position close to the rings is evident. Furthermore, the burning on the top and bottom boundaries lag behind ones inbetween. there seems to be a pronounced deceleration of flame on the centreline.

SIMULATION OF EXPERIMENTS AT LAKE LYNN.

The US Bureau of Mines have undertaken a series of experiments with methane in their Lake Lynn facility and compared the results with the Bruceton facility (Watson et al., 1987). The "D" heading at Lake Lynn has been represented in two dimensions on a 20 x 60 grid. The grid is finer at the wall and near to the closed end of the heading where combustion occurs.

The pressure histories at several locations are presented in Fig. 11. The general shape is well predicted but the absolute pressure and arrival times appear to be dependent on the coupling of the reaction rate and turbulence model.

The magnitudes of overpressures at the downstream locations tend to be lower than the experimentally observed ones and the flame is predicted to travel faster. Further work is in hand to understand this interaction.

Fig. 12 shows the history of fuel fraction at a few typical locations. Clearly, the mass of fuel burnt at the downstream locations is smaller than that



Fig. 10 Fuel Fraction Contour 0.06s after Ignition.

pressures.

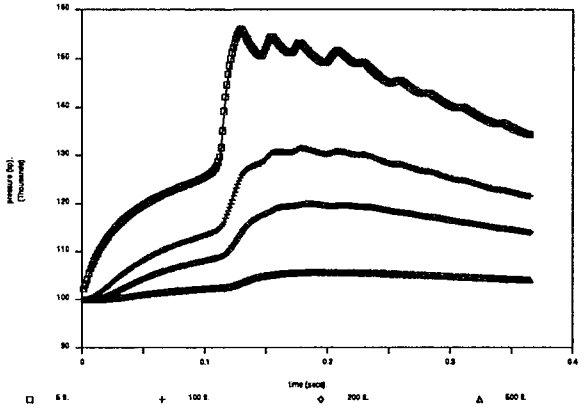


Fig. 11

fuel fractions.

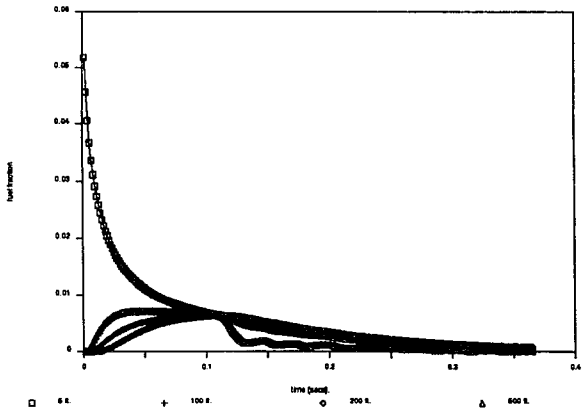


Fig. 12

near the left end, contributing to somewhat lower overpressures at these locations.

CONCLUSIONS.

Although further work is needed to refine the predictions, gas explosions can be realistically modeled and agree reasonably well with experiment. The computing time for these computations is about two hours using one T800 transputer chip. Although the code has been written in a parrallel form, there are still problems in farming out the computation to more than one processor and obtaining a faster computation.

The present simulation is being extended to include particles. The method uses a Lagrangian approach to track particles. In the first instance the particles will be non-combusting and the simulations will be toward the effect of particles (stone dust or water droplets) in quenching an explosion.

The second stage will be to add combustion and radiation exchange in order that dust explosion can be studied.

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SIMULATION D'EXPLOSIONS PAR ORDINATEUR

A.R. Green, K. Srinivas, I. Piper, R.W. Upfold,

Les auteurs mettent au point à l'heure actuelle des codes de simulation par ordinateur destinés à calculer les explosions du type de celles qui se produisent dans les mines. Ces simulations sont basées sur la solution d'équations de mouvements et de combustion. Pour simuler une combustion, on prend l'hypothèse d'un mécanisme simple à une étape. On incorpore les effets de turbulence en résolvant deux autres équations différentielles. On utilise une méthode de Volume Fini pour définir les équations, que l'on résout par une procédure d'intégration Runge-Kutta modifiée. A partir des conditions présentes au début d'une explosion, on calcule le développement du front de l'explosion et de l'écoulement à son arrière.

Nous présentons de nombreux exemples de calculs - avec ou sans combustion. Le débit d'un tube de choc unidimensionnel et l'interaction d'un front de choc oblique avec le sol ont été calculés sans inclure les effets de combustion, de turbulence et de visquosité. Un exemple tient compte de tous ces effets: l'accélération de la flamme dans un tube d'allumage muni d'anneaux produisant des turbulences. Nous présentons les résultats de tous ces cas et nous les comparons avec les expériences effectuées.

Nous décrivons brièvement un autre développement comprenant les effets des radiations et de la combustion partielle.

COMPUTERSIMULIERUNG VON EXPLOSIONEN.

A. R. Greeen, K. Srinivas,I. Piper, R. W. Upfold

Die Autoren entwickeln Computersimulierungscoden zur Berechnung von Explosionen, wie sie in Gruben vorkommen. Die Simulation beruht auf der Lösung der Gleichung von Bewegung und Verbrennung. Zur Darstellung der Verbrennung wird ein einfacher, aus nur einem Schritt bestehender Mechanismus angenommen. Die Turbulenzeffekte werden durch die Lösung von zwei weiteren differentialen Gleichungen einbezogen.

Eine Finit-Volumen Methode wird benutzt, die Gleichungen zu 'deskretieren', ein modifiziertes Runge - Kutta Integrationsverfahren, um sie zu lösen. Zunächst werden die zu Beginn einer Explosion auftretenden Bedingungen, später dann alle nachfolgenden Entwicklungen der Explosionsfront, sowie die Strömung dahinter berechnet.

Zahlreiche Beispiele für die Berechnung werden aufgeführt - mit und ohne Verbrennung. Die eindimensionale Stoßwellenströmung und die Wechselbeziehung zwischen einer versteckten Stoßwelle und dem Boden wurden unter Vernachlässigung der Verbrennungseffekte, Turbulenz und Viskosität berechnet. Ein Beispiel, welches alle diese Einflüsse berücksichtigt, ist die Flammen-Beschleunigung in einer Explosionsröhre, die mit Ringen zur Erzeugung von Turbulenz ausgestattet ist. Die Ergebnisse für alle diese Fälle werden vorgestellt und mit den experimentellen Ergebnissen verglichen.

Eine weitere Entwicklung, welche die Effekte der Strahlung und der Teilchenverbrennung einbezieht, wird kurz beschrieben.

МОДЕЛИРОВАНИЕ ВЗРЫВОВ НА ЭВМ

А.Р. Грин, К. Сринивас, И. Пайпер и Р.У.Алфолд

Авторы работают над созданием системы моделирования, позволяющей вычислить определенные типы взрывов, которые могут произойти в шахтах. Моделирование основано на решении уравнений движения и сгорания. Для расчета сгорания принимается простой одноступенчатый механизм. Эффекты турбулентности рассчитываются путем решения двух дополнительных дифференциальных уравнений. Метод конечных обьемов используется для расчленения уравнений, и модифицированная процедура Рунге-Кутта используется для их решения. Моделирование начинается с условий, возникающих в начале взрыва, затем рассчитываются последующее развитие взрывного фронта и воздушного потока позади него. Будет представлено множество примеров вычислений, включающих и исключающих сгорание. Был рассчитан одномерный взрывной поток и взаимодействие наклонного взрывного фронта с землей, которое не принимает во внимание эффекты сгорания, турбулентности и вязкости. Дается пример, включающий эти эффекты: ускорение пламени в трубке зажигания, которая подогнана при помощи колец так, чтобы вызвать турбулентность. Будут представлены все перечисленные варианты, а также будет проведено сравнение с результатами экспериментов. Кратко описаны дальнейшие разработки, включающие эффекты радиации, а также сгорание частиц.

爆炸的计算机模拟

A．R．格林，K．史力尼凡斯

I．派帕儿，R．W．厄不复得

作者们发展了计算机模拟程序以计算能在矿里发生的爆炸类型．模拟是根据运动方程和燃烧方程的解法．燃烧的模拟中假定一个单阶模型．紊流作用以解另两个微分方程来处理．方程式先用有限体积法来分立化，然后用改进的隆盖库塔积分布骤求解．从爆炸起始作为开端按步计算爆炸面和后面的流动．

这里给出很多计算例子——有些包括燃烧，有些不包括燃烧，我们计算了一度激波管流动及斜激波阵面与地面相互作用，但忽略了燃烧，紊流和粘度的影响．包括这些作用的例子是装有环的点火管（以产生紊流）里的火焰的加速度计算．所有这些题例均将在本文列出，并且与实验值相比较．

包括辐射和颗粒燃烧的推导将在这里简单报告．

SULPHIDE ORE DUST EXPLOSION RESEARCH IN CANADA

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ABSTRACT

In recent years, the Canadian hard-rock mining industry has become aware of the need to eliminate sulphide ore dust explosions, which are initiated by blasting operations. The major source of injuries comes not from the explosions themselves but from the toxic sulphur dioxide gas produced. The mining industry, other related industries, academia and governments have started a collaborative program to carry out research on the causes and possible control methods of these sulphide dust explosions. Equipment is being developed to measure the quantity of sulphide dust produced during a blast and the quantity of sulphur dioxide produced by the explosion. Laboratory studies have determined some of the relationships between the mineralogy of ores and their explosibilities. The amount of sulphur dioxide produced during the laboratory tests was measured by infrared spectrometry.

INTRODUCTION

Sulphide ore dust explosions are caused by the explosive used for breaking rock in base metal mines igniting a cloud of metal sulphide dust. Both pyrite (FeS_2) and pyrrhotite (FeS) can react to form hematite (Fe_2O_3) or magnetite (Fe_3O_4), by the following reactions:

$$4\ FeS_2 + 11\ O_2 = 2\ Fe_2O_3 + 8\ SO_2 \quad (1)$$

$$3\ FeS_2 + 8\ O_2 = Fe_3O_4 + 6\ SO_2 \quad (2)$$

$$4\ FeS + 7\ O_2 = 2\ Fe_2O_3 + 4\ SO_2 \quad (3)$$

$$3\ FeS + 5\ O_2 = Fe_3O_4 + 3\ SO_2 \quad (4)$$

Unlike explosions of coal dust, the major problem is not usually the actual blast damage, but rather the toxic sulphur dioxide gas generated in the explosion. The identification of the cause of the ignition of a coal dust explosion in a mine can often be difficult, because the coal dust cloud can be ignited by various means, such as a methane/air explosion, electric sparks, flames, frictional sparks or the thermite reaction. Sulphide dust explosions can always be linked to the primary blast. Despite this apparent simplification, a solution has not been found, partly because there has not been until recently a recognition of the seriousness of the problem by the mining industry. A fatality in 1985 at the Geco Mine in Manitouwadge, Ontario, by poisoning from SO_2 generated in a sulphide dust explosion, has stimulated the present work. The advent of newer techniques, such as vertical crater retreat, has increased the probability of sulphide dust explosions. Recent legislation in Canada has given workers the right for a workplace environment free from safety and health hazards, which will increase the pressure on the mining industry to eliminate possible exposure to SO_2.

The earliest published reports on sulphide dust explosions were by the U.S. Bureau of Mines (Gardner 1926, 1928). Seven miners were killed by either SO_2 or by "inhaling burning sulphide dust" in the U.S. in the period 1924-1926. In addition to SO_2, H_2S was also detected when blasting high-sulphur ores in wet environments.

The Horne mine in Noranda, Quebec was reported to have had 76 fires caused

by dust explosions between 1949 and 1963 (Cant 1963); a number of other Canadian mines have had problems with dust explosions (Job 1975). Canada carried out studies on sulphide dust explosions from 1960-1968. Laboratory work indicated that more than 18% combustible content constituted a possible explosion hazard (Montgomery 1962).

Enright (1988) has given a detailed account of sulphide dust explosions that have occurred in Sweden, Australia, South Africa and Canada. Day (1980) gave a detailed description of a major sulphide dust explosion in Canada, along with a proposed mechanism, and attributed the increased frequency of such explosions to the increasing size of blastholes. In addition, Japan has also had a sulphide dust explosion, which occurred in their Yanahara mine in 1964 (Takagi 1987) and the U.S.S.R. have indicated that they have had explosions in some high-pyrite mines (Polikarpov 1982).

COLLABORATIVE RESEARCH PROGRAM

The first conference ever devoted to sulphide dust explosions was held at Manitouwadge, Ontario, sponsored by Geco Div. of Noranda Minerals Ltd., in October 1986 (Dainty 1987). Among the conclusions were: "problems of sulphide dusts are more frequent than previously considered by individual companies"; "precautionary measures currently in use may be unsuitable or inadequate"; and "dust explosions can be associated with stope blasting, development or secondary blasting".

Subsequently, a Sulphide Dust Control Group, consisting of representatives from mining companies, explosive manufacturers, universities and governments, was organized to coordinate research in this area; four meetings have been held. In addition, a session of 8 papers was held at the 90th Annual General Meeting of the Canadian Institute of Mining and Metallurgy in Edmonton in May 1988. In order to develop a data base for explosions, a standard occurrence report has been distributed to the mines; 18 incidents have been reported up to January 1989. A list of control measures has been prepared for the mines, based on current knowledge:

1. avoid long delay blasting, because a dust cloud can be formed and then ignited;
2. wash down area to prevent the blast from forming a dust cloud generated from previous blasts;
3. use finely-divided water sprays at entryways;
4. stem all holes where possible using inert plugs (not drill cuttings);
5. evacuate personnel from area before blasting;
6. confirm that area is free from SO_2 before re-entering.

LABORATORY STUDIES

Minimum Ignition Temperature (MIT)

A standard Godbert-Greenwald furnace (Dorsett 1960) was used to measure the MIT of seven pure minerals (diameter <38 μm), shown in Table 1, and 20 different ores.

TABLE 1 - MIT OF PURE MINERALS

Mineral	MIT (°C)
pyrrhotite	380
chalcopyrite	480
pyrite	510
tetrahedrite	530
galena	550
sphalerite	560
argentite	780

Unlike other dusts, sulphide ore dusts did not produce a flame in the Godbert-Greenwald furnace; they produced only sparks. The quantity of sparks increased with temperature, but even as high as 800°C, pyrrhotite generated only a shower of sparks, not a flame. The occurrence of a single spark was used as the limit criterion. It is a moot point as to whether indeed sparks should be used to define a MIT. As will be seen

later, the relative MIT values do not correlate with the explosibility tests. For example, sphalerite and tetrahedrite are in the same range of MIT as pyrite, yet they do not explode. MIT values of sulphide dusts are probably more closely related to their propensity for spontaneous combustion than their explosibility.

Hartmann Chamber Tests

Our first set of explosions were carried out using the standard Hartmann chamber (Dorsett 1960). A CEC 1000 strain-gauge pressure transducer was mounted on the top of the chamber; its output signal was amplified by a modified Honeywell 1885A-SGC module, then stored in a Nicolet 4094 digital oscilloscope. A control box was built to allow sequencing of events to a millisecond. Since the sulphide dusts would not explode in air using an electric spark discharge, the tests were carried out in pure oxygen. For convenience, all samples were tested at a concentration of 1600 g/m^3, which should be close to the optimum.

The maximum pressures during the explosion, P_m, and the maximum rate of pressure rise, $(dP/dt)_m$, of the pure minerals are given in Table 2. All of them have a particle size <106 µm, except for pyrrhotite, which is somewhat finer (38-75 µm). The explosion pressures are clearly not related simply to the percentage of sulphur present, as has been postulated. Galena is unusual, not only because it is explosible with relatively little sulphur, but because it formed the sulphate, rather than the oxide, and no SO_2 was detected. Thermodynamically, the sulphate is favoured over the oxide for reaction of all the metal sulphides. However, ferric sulphate decomposes at the relatively low temperature of 480°C, thus accounting for the lack of observance of iron sulphate.

In these explosion tests, pyrite produced only black powder (magnetite); by contrast, pyrrhotite produced only red powder (hematite). Using a finer fraction of pyrrhotite (which generated a higher explosion pressure), a mixture of hematite and magnetite was produced. Although at room temperature the reaction of both pyrite and pyrrhotite are thermodynamically favoured to form hematite, at higher temperatures, such as occur in strong explosions, thermodynamics predicts the reverse.

TABLE 2 - EXPLOSION PARAMETERS OF PURE MINERALS

mineral	%S	P_m (kPa)	$(dP/dt)_m$ (MPa/s)
pyrite	53.0	460	10
pyrrhotite	36.4	420	20
chalcopyrite	34.9	300	2
galena	13.4	310	4
sphalerite	32.9	no reaction	
tetrahedrite	26.0	no reaction	
argentite	10.0	no reaction	

Analyses of the residues from the explosion tests were carried out by scanning electron microscopy. Pyrite produced a variety of virtually perfect spheres between 5 and 50 µm in diameter (Fig. 1). Chalcopyrite produced less spherical particles. Although microanalysis indicated that these particles did not contain any sulphur, the distorted spheres indicate that the temperature reached during their formation was not sufficient to fuse the particles completely. Galena also produced highly spherical particles; microanalysis showed that these particles contained lead and sulphur, thus confirmining the formation of sulphate.

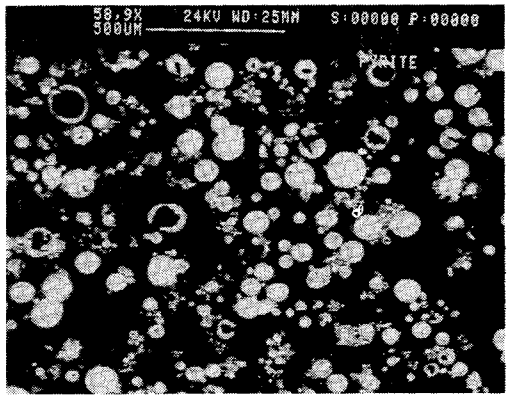


Figure 1

Table 2 shows that the explosibility is not simply a function of the sulphur content of the mineral. Twenty-seven ores of various composition were collected from 10 different mines and tested in the Hartmann chamber in the same way as was done for the pure minerals. For clarity, the explosibilities were normalized relative to pure pyrite, which should be the most explosive. Defining the relative explosibility (RE) of pyrite as 100, the relative explosibility of all the other samples are then:

$$RE = \frac{100\ P_m\ (\text{sample})}{P_m\ (\text{pyrite})} \qquad (5)$$

Figure 2 shows the relative explosibility as a function of sulphur content for all the ores (but not the pure minerals). In general, explosibility increases with sulphur content, and there appears to be a cut-off at 26%, below which there are no explosions. However, the scatter of data is rather high; in addition, there were no ores with high galena content, which, as noted above, would probably explode at much lower sulphur contents.

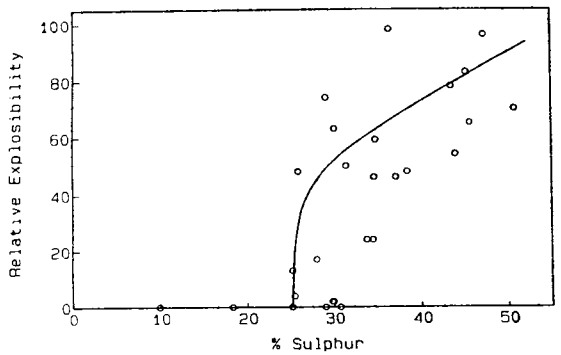


Figure 2

Equation 5 applied to the pure minerals yielded RE values of 90 for pyrrhotite, 65 for chalcopyrite, 70 for galena and 0 for sphalerite. Thus, using the mineralogical analysis, the RE of each ore sample can be predicted by multiplying the above values by the percentages of each component. A graph of the actual RE vs. the predicted RE for all the ore samples (but not the pure minerals) is shown in Fig. 3. The straight line shown is the theoretical line. It would appear that this line provides a prediction of the upper limit of the explosibility.

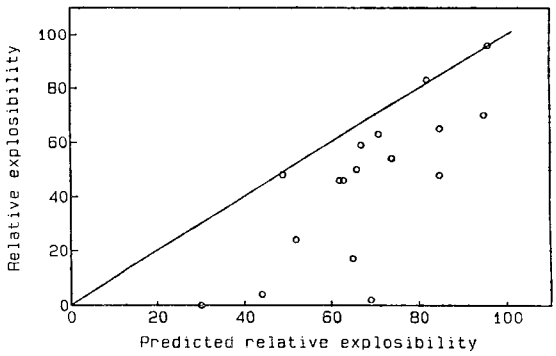


Figure 3

20-L Vessel Explosion Tests

A stainless-steel vessel similar to the one used by the U.S. Bureau of Mines (Cashdollar 1985) was used for these tests. The control and pressure-detection systems were the same as were used for the Hartmann apparatus. A paramagnetic oxygen analyzer was connected to the 20-L vessel to allow measurement of the oxygen remaining after the explosion test, as well as for making up mixtures for testing in oxygen-enriched or in oxygen-deficient atmospheres.

An infrared spectrophotometer (Miran-80 Computing Infrared Analyzer, with a 20-m gas cell) was also connected to the 20-L vessel for measuring the quantity of SO_2 produced in the explosion. The main absorption band of SO_2 at 7.3 μm was not used, because it is located in the midst of the absorption band due to water and thus is subject to interference. Instead, a weaker band at 9 μm, in a spectral region free from

interference by water and carbon dioxide, was used.

As mentioned above, the sulphide ore dusts cannot be ignited by an electric spark. Instead of testing in pure oxygen as was done for the Hartmann tests, a stronger ignitor (a 5-kJ Sobbe pyrotechnic) was used in a normal atmosphere. This system should provide a more realistic simulation of the actual occurrence. The Sobbe ignitor generates an explosion pressure of 50 kPa in our vessel under the operating conditions used. The maximum pressure in the explosion tests on a dust is approximately the sum of the explosion pressure due to that dust plus the pressure due to the ignitor. Therefore, the pressure trace of the ignitor carried out without dust was subtracted from the experimental trace to yield the "true" pressure trace. The justification for this procedure was that the maximum pressures thus obtained for lycopodium dust were the same as obtained for tests on lycopodium using electric sparks as the ignitor. On the other hand, $(dP/dt)_m$ was higher for lycopodium with dust than with electric sparks. This may be due to additional turbulence caused by the chemical ignitor, which affects that parameter much more than P_m.

Of critical importance in laboratory tests in dust explosions is the creation of a homogeneous dust cloud. The dispersion technique, the pressure of the dispersing air, the timing of the air injection and the ignition must be optimized. We have built a glass vessel of about the same size as the stainless-steel vessel and which uses the same dispersion system, so as to allow the dust dispersion process to be videotaped. Frame-by-frame viewing allows identification of appropriate parameters for testing.

Sulphide dusts, being much denser than coal and most other explosible dusts, require a shorter, more intense pressure pulse for adequate dispersion. The test conditions were selected such that ignition occurred at a pressure of one atmosphere. It is known that the explosion pressure is proportional to the initial pressure. In the Hartmann apparatus, the initial pressure is about 30 kPa, thus the experimental pressures are about a factor of 1.3 too high (neglecting the other deficiencies of the Hartmann apparatus).

The explosion pressures obtained for pyrite and two size fractions of pyrrhotite are shown in Fig. 4. The curves are all S-shaped, making it reasonable to define the minimum explosible concentration (MEC) as being the steepest point on the curve. The maximum explosion pressure appears to be about the same for both size fractions of pyrrhotite, but the MEC of the finer fraction is much lower. The pyrite, which is coarser, has a smaller MEC than the pyrrhotite, indicating that it is more explosible.

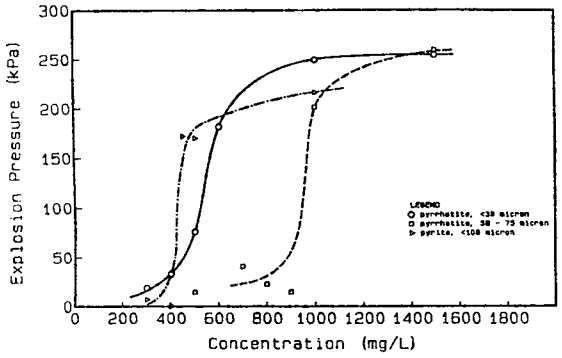


Figure 4

$(dP/dt)_m$ for the same three samples are shown in Fig. 5. The effect of particle size on this parameter is greater than it is on P_m. The separation of each curve into the regions of explosible and non-explosible is not as clear-cut, partly due to the intrinsically greater scatter of $(dP/dt)_m$ compared to P_m.

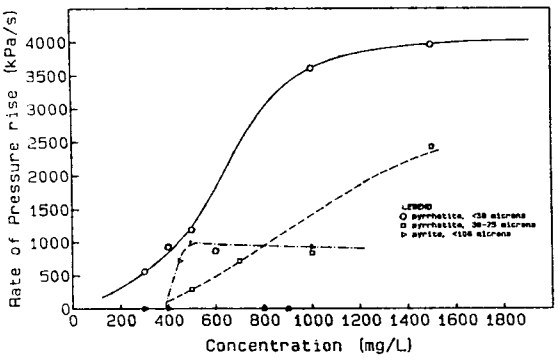


Figure 5

The quantity of SO_2 produced for the same three samples is shown in Fig. 6. The curves are quite similar to Fig. 4, because both P_m and $n(SO_2)$ are directly proportional to the amount of reaction that has occurred. The quantity of oxygen consumed (on a molar basis) was roughly two times the quantity of SO_2 generated. The theoretical values for reaction of pyrrhotite and pyrite are 1.75 and 1.38, assuming that they each react to hematite.

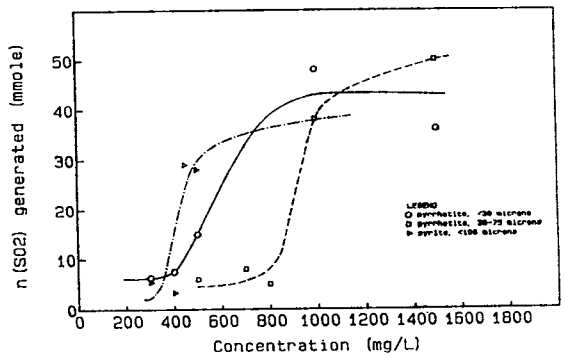


Figure 6

These tests produced mainly hematite, with some magnetite, for both pyrrhotite and pyrite. In the Hartmann apparatus, the same sample of pyrite produced magnetite; the difference being that the tests had been done in pure oxygen, which produced stronger explosions, as shown also by the higher explosion pressures.

Chalcopyrite had only limited reaction up to 1200 g/m^3. Limited testing of galena indicated that the MEC is between 1000 and 1300 g/m^3. Galena produced significant quantities of SO_2, in contrast to the Hartmann tests, in which no SO_2 was detected. The reason is that the latter were carried out in pure oxygen, which would tend to favour reaction (7) that uses more oxygen than reaction (6):

$$PbS + 1\text{-}1/2\ O_2 = PbO + SO_2 \qquad (6)$$

$$PbS + 2\ O_2 = PbSO_4 \qquad (7)$$

This hypothesis is supported by the ratio of SO_2 detected: O_2 consumed being 5, compared to a ratio of 1.5 predicted on the basis of (6) being the only reaction occurring.

Two explosion tests, at a concentration of 1000 g/m^3, carried out on different samples of sphalerite, both with particle size of <38 μm, yielded absolutely no evidence of explosion or reaction, thus confirming the tests carried out in the Hartmann chamber.

Figure 7 shows the explosion pressure for tests carried out on pyrrhotite, (38-75 μm fraction) at a concentration of 1500 g/m^3, as a function of oxygen concentration. The curve shows a sharp break between the region of explosibility and non-explosibility at 17.2% O_2, which can be defined as the minimum oxygen concentration for this sample.

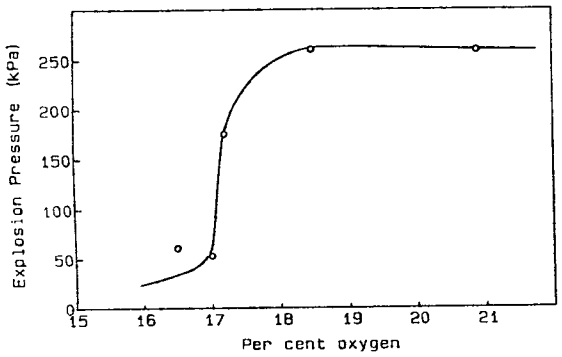


Figure 7

IN-MINE TESTS

Electrochemical SO_2 sensors were evaluated in a mine which has high pyrite ore and therefore has had many sulphide dust explosions. The sensors proved to be sufficiently reliable and stable that they are recommended for use by all mines having problems with sulphide explosions. The highest concentration of SO_2 measured was 910 ppm after a large blast.

Dust probes were constructed that would allow measurements of air-borne dust during four blasts, none of which had sulphide dust explosions. Most of the coarse dust settled within a second, but respirable dust took 10-15 minutes to decrease from 100 to 10 mg/m^3. Dust samples were also collected by pans placed on the floor at various distances from the blast; between 37 and 66% of this dust was below 38 µm. Laboratory tests reported above (Fig. 4-6) showed that the <38 µm size fraction was much more hazardous than larger fractions; hence, emphasis in the laboratory should be on the very fine fraction.

CONCLUSIONS

Laboratory tests have shown that pyrite, pyrrhotite, galena and chalcopyrite are all explosible when dispersed in air, and a sufficiently large ignitor is used. Sphalerite is non-explosible under those same conditions. The percentage of sulphur cannot be used as a reliable measure of the risk of explosion, except where the mineralogy of the ore is similar. The amount of SO_2 produced in the explosions of pyrite and pyrrhotite is proportional to the explosion pressure, indicating a quantitative conversion of the sulphur in the ore to SO_2. Explosions of galena produce sulphate in larger quantities than SO_2.

In the past three years, Canadian industry, industry and academia, with assistance from Australia and the U.S., have made significant progress in the understanding of sulphide ore dust explosions. It is hoped that application of this knowledge will lead to safer and more productive base metal mines.

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RECHERCHES SUR LES EXPLOSIONS DE POUSSIERE DE MINERAUX DE SULFURE AU CANADA

K.J. Mintz, E.D. Dainty

Ces dernières années, l'industrie minière canadienne des roches dures a pris conscience du besoin d'éliminer les explosions de poussières de minéraux de sulfure provoquées par les opérations de tirages. La cause la plus importante des dégâts ne provient pas des explosions par elles-mêmes, mais des gaz toxiques d'anhydride sulfureux produits. L'industrie minière, les industries connexes, les universités et les gouvernements ont lancé un programme en collaboration pour effectuer des recherches sur les causes de ces explosions de poussières de sulfure et pour établir des méthodes permettant si possible de les contrôler. A l'heure actuelle, des équipements sont développés pour mesurer la quantité de poussières de sulfure produite au cours d'un tirage et la quantité d'anhydride sulfureux produite par l'explosion. Des études de laboratoire ont établi plusieurs rapports entre la minéralogie des minerais et leurs capacités explosives. La quantité d'anhydride sulfureux produite au cours d'essais en laboratoire a été mesurée au moyen de la spectrométrie à infra-rouges.

FORSCHUNG AUF DEM GEBIET DER SCHWEFELERZSTAUB - EXPLOSIONEN IN KANADA.

K. J. Mintz, E. D. Dainty

In den letzten Jahren ist es der kanadischen "hard-rock mining" Industrie immer deutlicher bewußt geworden, daß es notwendig ist, die durch Spreng-Arbeiten verursachten Schwefelerzstaubexplosionen zu unterbinden. Die größte Unfallsgefahr liegt nicht bei der Explosion selbst, sondern viel mehr in der damit verbundenen Erzeugung von giftigen Schwefeldioxidgasen. Die Bergbau-Industrie, andere beteiligte Industriezweige, die Forschungsstellen und die Regierung haben ein gemeinsames Programm eingeleitet, um den Grund und die mögliche Kontrolle dieser Schwefelstaubexplosionen zu untersuchen.
Instrumente zur Messung des bei einer Sprengung erzeugten Sulfidstaubes und der bei einer Explosion erzeugten Schwefeldioxide steht in Entwicklung. Laborversuche haben einige der Zusammenhänge zwischen der Mineralogie des Erzes und seiner Explosivität bestimmt. Die Menge des im Labor erzeugen Schwefeldioxides wurde mit Infrarot-Spectrometern gemessen.

ИССЛЕДОВАНИЕ ВЗРЫВОВ СЕРНИСТОРУДНОЙ ПЫЛИ В КАНАДЕ

К.Дж. Минц и Э.Д. Дейнти

В последние годы Канадская горная промышленность уделяла большое внимание необходимости прекращения взрывов сернисторудной пыли, которые вызываются подрывными работами. Главный вред наносится не самими взрывами, а сопровождающими их токсическими сернокислыми газами. Горная и другие близкие индустрии, а также академические круги и местные органы власти начали совместную программу исследований, направленных на изучение причин и возможных методов контроля над взрывами сернистой пыли. Конструируется оборудование для измерения количества сернистой пыли и сернокислого газа, производимых в результате взрыва. Экспериментальные исследования показали, что существует корреляция между минералогией пород и их взрывоопасностью. Количество сернокислого газа, произведенного в процессе экспериментов, измерялось с помощью инфракрасной спектроскопии.

加拿大硫矿尘末爆炸的研究

K. J. 明兹， E. D. 丹退

近年来加拿大硬岩石矿冶工业发觉到必须杜绝由爆破作业引起的硫矿尘末的爆炸事件。此类事故中人员的受伤并非由爆炸直接所致，而是由于爆炸产生的二氧化硫气体造成的。矿业界，其他有关工业界，学术界和政府部门已经联合起来开展了一项协作的研究项目来寻找这些硫矿粉尘爆炸的起因及其可能的控制方法。我们正在研制用来测量硫矿粉尘爆炸时所产生硫矿粉尘和二氧化硫气体数量的仪器。实验室试验已经找到了矿石的矿石性与其爆炸性之间的一些关系。实验室里我们用红外线色谱仪来测量爆炸试验中所产生的二氧化硫量。

EXPLOSION TESTING AND OPERATIONAL EXPERIENCES WITH FLAMEPROOFED DIESEL EXHAUST SYSTEMS WITH HEAT EXCHANGE

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All diesel engines used in German hard coal mines normally are equipped with plate trap and water box at the intake and exhaust sides. Explosion tests with such installations have revealed that flameproofness was not always ensured despite the prescribed minimum safe gap (width and length) of the flame trap. Hence, to correctly assess flameproofness, explosion tests with original components became necessary. Test gas is a mixture of methane and hydrogen.

For some time now exhausts in the form of tube bundle heat exchangers have been tested, which additionally are equipped with plate traps. The re-test of such exhaust systems after several years underground service showed no effect on flameproofness. Today also ribbon flame traps and glass ball traps are in test exhaust systems after several years underground service showed no effect on flameproofness. Today also ribbon flame traps and glass ball traps are in test application which are quite satisfying.

INTRODUCTION

For decades diesel engines have been holding a firm place as driving unit in German coal mining. As diesel engines present a possible ignition source of methane-air-mixtures, they shall be flameproof (Lehmann 1939; requirements of the LOBA [mining regulatory authority], Northrhine Westphalia 1954, 1976, 1987). The intake and exhaust installations must be protected by flame arrestors in the form of plate traps. In plate traps having a plate width of 50 mm, a maximum gap of 0.8 mm was permitted, but examination in comprehensive explosion tests showed that although the maximum gap was observed, the flameproofness was not ensured in all cases. Obviously form and geometry of the installation had an influence on the effectivness of the flame traps (Wiemann 1982 and 1984; Scholl 1983). Therefore the flameproofness of each type of installation is proved in type tests by appropriate explosion tests. Here, the maximum permitted gap is established (von Schaubert 1986).

TEST METHOD OF FLAMEPROOFNESS

For the testing of flameproofness, i.e. the ability to withstand an internal explosion without the transmission of flame, intake and exhaust systems in final configuration including the installed plate trap are used, which means, all components between engine and plate trap. The engine itself is not subjected to the tests.

The test principle is to surround the installation to be tested with an explosive gas-air-mixture, inside and outside. The point of ignition is at the plate trap side, facing the engine, in the intake or exhaust gas manifold. It is registered, whether the plate trap arrests the approaching explosion or the explosion flame passes across the plate trap and thus ignites the surrounding explosive mixture. The test gas used is a mixture of methane and hydrogen, containing a specific quantity of air. The composition of the gas mixture is given by a numerical pair, where the

first number indicates the percentage b.v. of methane, the second of hydrogen. The higher the portion of hydrogen in the mixture, the easier ignition is transmitted. On the basis of investigations with single gaps, a safety factor is attributed to the mixture just not transmitting ignition (Table 1).

Table 1: Attribution of just not ignition transmitting gas mixture and safety factor for the assessment of flameproofness of intake and exhaust systems

just not ignition transmitting gasmixture		MESG[1]	safety factor	sufficient flameproofness
gas mixture composition CH_4/H_2	gas mixture concentration % b.v.	mm		
100/ 0	7 - 11	1.20	1.00	no
90/ 10	9 - 13	1.05	1.14	no
80/ 20	10 - 14	1.00	1.20	yes
70/ 30	11 - 15	0.90	1.33	yes
60/ 40	12 - 16	0.80	1.50	yes
50/ 50	13 - 17	0.70	1.71	yes
40/ 60	14 - 18	0.70	1.71	yes
30/ 70	15 - 19	0.60	2.00	yes
20/ 80	17 - 21	0.50	2.40	yes
10/ 90	19 - 23	0.45	2.67	yes
0/100	28 - 32	0.35	3.35	yes

[1] refers to investigations with single gap

This safety factor is the quotient of maximum experimental safe gap for methane and the one of the test gas mixture. For intake and exhaust installations the not ignition transmitting gas mixture, or, the safety factor in methane, has been determined. One can speak of an adequate flameproofness, when a safety factor of 1.2 has been established experimentally, not transmitting ignition with an 80/20 gas mixture.

$$S = \frac{\text{MESG (methane)}}{\text{MESG (hydrogen)}}$$

In connection with the testing, each gap of the flame trap is measured with a gauge, in steps of 0.05 mm. The greatest, just fitting gauge sensor width is defined as the maximum permitted gap for plate traps in a defined type of inlet or exhaust installation.

EXHAUST EQUIPMENTS ON THE BASIS OF HEAT EXCHANGE

The usual flameproof exhaust equipment, comprising water injection into the exhaust gas manifold with superposed water box, are very large and heavy. This is unfavourable, because the space for exhaust systems in trackless vehicles is very restricted. In addition, water is ejected with the exhaust gas, presenting an additional strain to the carriage road. Therefore, an alternative to the usual exhaust cooling was searched. The basic requirements that the exhaust system should fulfill are:

- compactness of the exhaust equipment
- indirect exhaust cooling
- closed cooling circulation to minimize consumption of cooling agent
- flameproofness

These requirements can largely be fulfilled by exhaust systems on the basis of heat exchanger. In the two

types, used at present, the exhaust gas is cooled in two steps: step one is a tube bundle heat exchanger with a cooling circulation which is independent from the diesel engine. First the exhaust gases are cooled down to approximately 130 °C. Then the gases are further cooled down in the second step by means of an exhaust gas dilution system. The gases entering the dilution system with very high velocity, take in ambient air, which then is mixed with the exhaust gases in the steel tube. By this, the temperature of the exhaust gases are lowered to 70 °C, a permitted value. Such exhaust installations are called "dry exhaust gases cooling systems". A plate flame trap is provided between heat exchanger and dilution system, the latter being installed already outside the explosive area.

EXPLOSION TESTS IN HEAT EXCHANGER EXHAUST SYSTEMS

Exhaust system type A

The heat exchanger of exhaust system type A is flanged to the diesel engine by a short water cooled pipe bend in such a way that the heat exchanger is always in a parallel position to the engine, just beside the manifold at the engine side (Figure 1). The exhaust gases coming from the heat exchanger are directed to the plate trap through a compensator and a pipe. All system components behind the heat exchanger are not water-cooled.

The heat exchanger in type A is a tube bundle heat exchanger with two pipes and two jacketed paths. One of the pipe bottoms is removibly connected to the jacket flange, the other one can freely move. The cap is double walled and consists of two chambers with openings for the exhaust gas inlet and outlet. The hot engine exhaust gases flow radially over the cap into the first chamber, from there through the first tube bundle, then they are deflected by 180° at the bottom of the tube, flow through the other tube bundle back into the second chamber and leave the heat exchanger via the cap in axial direction.

The result of the tests on flameproofness was that the exhaust equipment is safe in a 60/40 gas mixture - 60 % b.v. methane, 40 % b.v. hydrogen. The maximum permitted gap of the plate trap is 0.60 mm.

This test result was obtained with a brand-new exhaust system. It could not be excluded, however, that during operational service time the safety of the exhaust system decreases. Therefore, the exhaust system was re-examined after two years operation service on a trackless vehicle. The heat exchanger had not been cleaned in this period, only the plate trap had been serviced regularly. Therefore, the re-test on explosion behaviour was carried out with a sooted system. The exhaust equipment proved flameproof even after a 2 years service. However, it was noted at the beginning of the tests that the explosions in the exhaust system developed slowly and even did not run to the plate trap in the first instance. By the dispersion of soot deposits, the explosion flame was almost extinguished. This means that in practical operation soot deposits are more explosion inhibiting rather than intensifying.

Exhaust system type B

In type B the connection to the engine is a water cooled compensator (Figure 2), followed by two heat exchangers, which are connected in a parallel position. Behind the second heat exchanger the plate trap is arranged and, with an appropriate pipe, the exhaust gas dilution system. The heat exchanger applied is a tube bundle heat exchanger with one pipe path and two solid bases, the two caps being water cooled as well. The exhaust gases are directed axially into and out of the heat exchanger via the respective cap.

Explosion tests with this system were carried out in two different arrangements. The exhaust system was connected with the manifold either with a straight compensator (arrangement 1) or a compensator with a 90° pipe bend (arrangement 2).

Although in both arrangements the same type of plate trap with a maximum permitted gap of 0.65 mm was used, minor differences in ignition transmission were observed. In test arrangement 1 no ignition transmission with a 80/20 gas mixture was noted. In test arrangement 2 no ignition

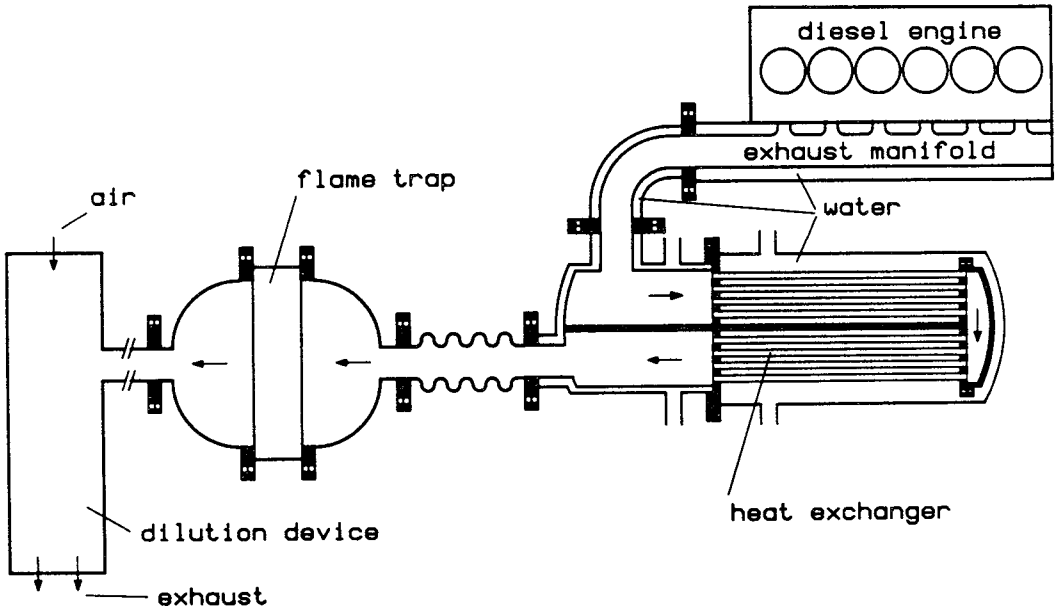


Figure 1: Exhaust system type A

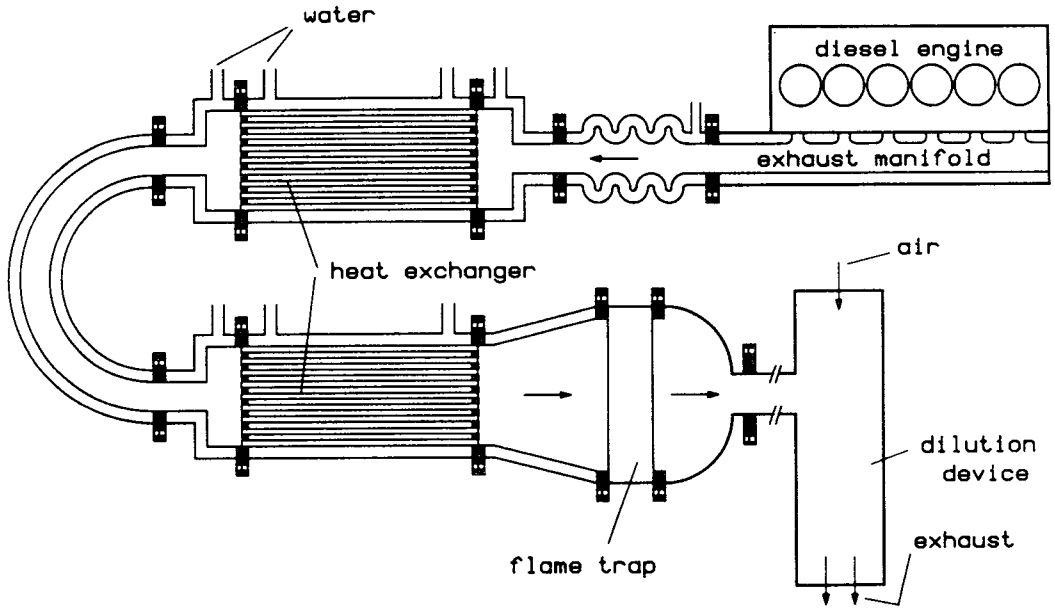


Figure 2: Exhaust system type B

transmission was obtained with a 70/30 mixture. So it could be established that the geometry of the installation has a great influence on ignition transmission.

BASIC EXPERIENCES WITH FLAMEPROOFED HEAT EXCHANGERS

The exhaust systems on the basis of heat exchangers have been in operation since 1983. At the end of 1988 the diesel engines of more than 20 trackless vehicles were equipped with heat exchangers at the exhaust side. Most of the vehicles are driven by a 6-cyl. diesel engine of the company MWM, the so-called RAG-Standard engine. Smaller vehicles are equipped with 2, 3, and 4-cyl. diesel engines. The dry exhaust gas cooling on the basis of heat exchanger had proved a good alternative to the conventional water box. The closed cooling circulation offered largely independence from the water supply underground. Safety restrictions with regard to flameproofness were not realized. This was confirmed by the re-testing of the exhaust system type A. Just like in the case of old exhaust installations with water box, the flameproofness of the dry exhaust gas cooling depends upon the correct state of the plate trap.

ALTERNATIVES TO THE PLATE TRAP

Although plate traps have proved their usability with regard to safety, they show some disadvantages. Plate traps are quite heavy and liable to contamination which necessitates relatively much service work. Incorrect service can lead to increased gaps. So, an alternative to the plate trap was searched. Crimped ribbon flame traps and ball traps were taken into consideration. Both types of flame trap already have been applied successfully in methane drainage installations (Scholl 1979, Bartknecht 1967).

CRIMPED RIBBON FLAME TRAP

This type of flame trap consists of flat and crimped stainless steel ribbons of 10 mm width each. A flat and a crimped ribbon are rolled up. So an almost round disk with a great number of triangle-shaped ducts is formed. The geometrical characteristics for flameproofness of ribbon flame traps are the clear hight of the triangles and the duct. The maximum triangle hight of the type considered here is 0.7 mm (Figure 3). The three ribbon disks of a complete flame trap are put in series in changed roll up direction. So the ducts in these traps run in zig-zag-form.

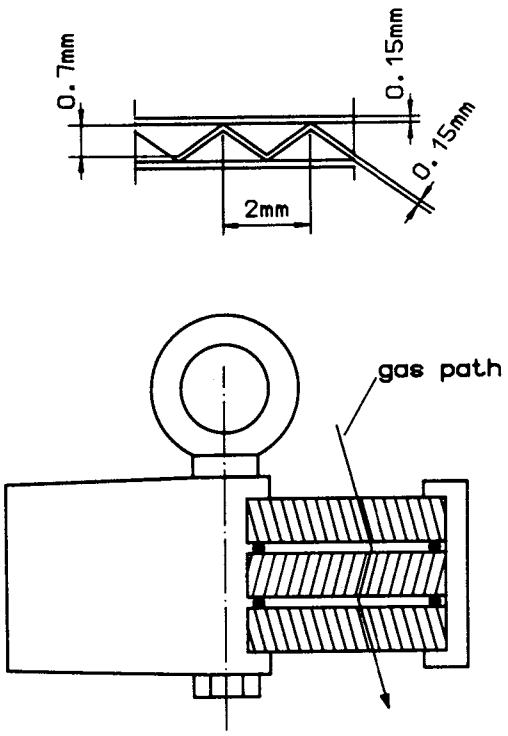


Figure 3: Crimped ribbon flame trap

This type of crimped ribbon flame trap has been tested in combination with the exhaust system type A in explosion tests. Because the exhaust system is designed for installation on diesel engines with two or three cylinders, the heat exchanger is correspondingly smaller and more compact. Besides the ribbon trap, a plate trap with maximum gap of 0.7 mm has been tested, with the result that the flameproofness of an exhaust system is not only be achieved by plate traps but by ribbon flame traps as well. In tests with ribbon flame traps no ignition transmission was obtained in 60/40 gas mixtures, while no more ignition transmission was obtained in 80/20 gas mixtures with plate traps.

Because of the positive test results this exhaust system with triple ribbon

trap has 1987 been installed underground with satisfactory result on a two-cylinder diesel engine.

Glas ball flame trap

For the exhaust system B a flame trap has been constructed filled with glass balls, 5 mm in diameter (Figure 4), between two parallel stainless steel wires mesh. The diameter of the wire is 1 mm with 3 mm width of mesh. The glass ball filling is 60 mm.

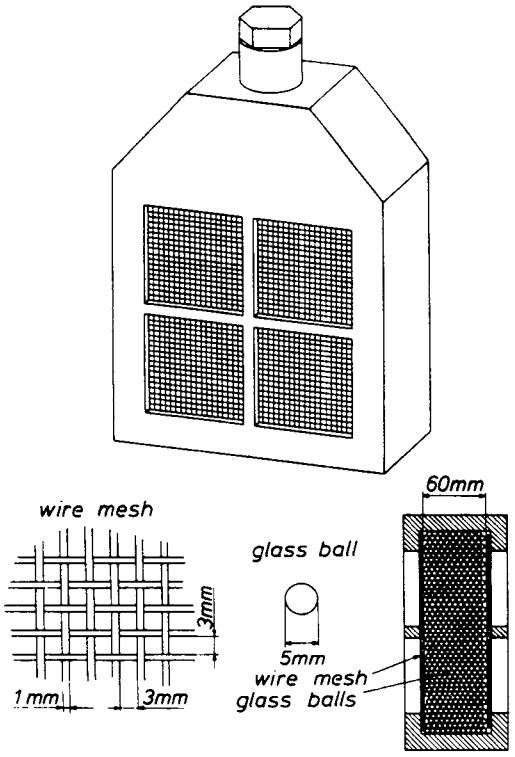


Figure 4: Glass ball trap

Explosion tests have demonstrated that the glass ball flame trap prevents ignition transmission in a 70/30 gas mixture, which thus flameproofs exhaust system B.

This positive test result 1986 led to the equipment of a trackless vehicle with 6-cylinder diesel engine with a ball flame trap at the exhaust side. At the end of 1988 six trackless vehicles had been equipped with ball flame traps. Up till now, the underground test installations deliver satisfactory results. The operating persons stress the favourable maintenance work. Generally the ball trap is cleaned by just blowing in air. In the case of heavy contamination the dirty balls are just simply replaced for new ones. The dirty balls can be cleaned with water. With the same passing through area the decline in pressure is a bit higher as in the case of plate traps (Figure 5). Ball traps therefore present a higher flow resistance, leading to a higher exhaust gas counter pressure in the manifold of the diesel engine. Negative effects to practical operation of the presently six so equipped vehicles could not be noted. For this reason ball traps were not enlarged up till now with a subsequent reduction of flow resistance.

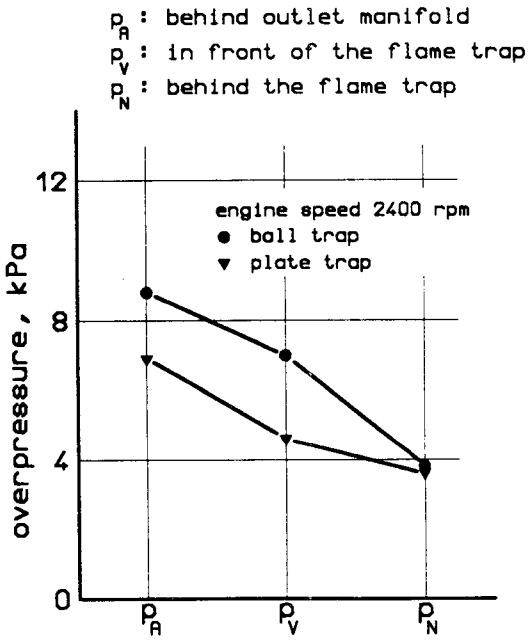


Figure 5: Exhaust gas pressure in the exhaust system type B

SUMMARY

Because of the increasing application of trackless vehicles in German underground coal mining, alternatives to the usual direct water cooling of the diesel engine exhaust gases have been searched. Explosion tests with heat exchanger exhaust systems show that they are flameproof in combination with plate traps.

Crimped ribbon flame traps and glass ball traps are possible alternatives. Appropriate explosion tests have

proved their effectiveness. Because of their simple construction and cleaning and longer service time ball traps are favoured.

LITERATURE

Anforderungen des Landesoberbergamtes Nordrhein-Westfalen an die Bauart von Zugkatzen mit Eigenantrieb durch Dieselmotoren (Dieselkatzen) für Einschienenhängebahnen im Untertagebetrieb von Steinkohlenbergwerken (Bauvorschriften für Dieselkatzen) vom 8. Juni 1976

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EXPLOSIONSTECHNISCHE PRÜFUNG UND BETRIEBSERFAHRUNGEN MIT SCHLAGWETTERSICHEREN ABGASSYSTEMEN FÜR DIESELMOTOREN AUF DER BASIS VON WÄRMEAUSTAUSCHERN

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Im deutschen Steinkohlenbergbau eingesetzte Dieselmotoren wurden bisher auf der Ansaug- und Abgasseite grundsätzlich mit Plattenschutzpaketen ausgerüstet. Ferner mußte auf der Abgasseite eine Wasservorlage vorhanden sein. Explosionsversuche mit derartigen Systemen haben jedoch gezeigt, daß trotz Einhaltung der vorgeschriebenen Grenzwerte für den Plattenschutz die Schlagwettersicherheit nicht immer gegeben war. Deshalb wurde neuerdings im Rahmen der Bauartzulassung eine Explosionsprüfung an Originalbauteilen eingeführt, die eine zuverlässige Beurteilung der Schlagwettersicherheit ermöglicht. Als Prüfgas dienen Mischgase aus Methan und Wasserstoff.

Seit einiger Zeit werden auch Abgassysteme in Form von Rohrbündelwärmeaustauschern eingesetzt, die ebenfalls mit Plattenschutzpaketen ausgerüstet sind. Die Explosionsprüfung an Neuanlagen sowie an einer untertage eingesetzten Anlage bestätigten die Schlagwettersicherheit dieser Abgassysteme. Heute werden anstelle des Plattenschutzes auch Flammensperren aus Bandsicherungen oder aus Glaskugelschüttungen erprobt. Diese Einsätze verliefen bisher zufriedenstellend.

EINLEITUNG

Der Dieselmotor hat im deutschen Steinkohlenbergbau unter Tage seit Jahrzehnten seinen festen Platz als Antriebsaggregat. Da Dieselmotoren als eine mögliche Zündquelle für Grubengas/Luft-Gemische anzusehen sind, müssen die Motoren schlagwettergeschützt ausgeführt werden (Lehmann 1939; Anforderungen des Landesoberbergamtes NW 1954, 1976 und 1987). So sind u.a. die Ansaug- und Abgasanlagen durch Flammensperren in Form von Plattenschutzpaketen gesichert. Für Plattenschutzpakete mit einer Spaltlänge von 50 mm waren Spaltweiten bis zu maximal 0,8 mm zulässig. Bei der Überprüfung solcher Systeme haben umfangreiche Explosionsversuche jedoch gezeigt, daß trotz Einhaltens dieser Spaltweite die Schlagwettersicherheit nicht immer gegeben war. Offensichtlich beeinflussen Form und Größe der Anlagen die flammensperrende Wirkung der Plattenschutzpakete (Wiemann 1982 und 1984; Scholl 1983). Deshalb muß die Schlagwettersicherheit derartiger Systeme im Rahmen einer Bauartprüfung durch Explosionsversuche nachgewiesen werden. Dabei wird für jedes System die maximal zulässige Spaltweite des zugehörigen Plattenschutzpaketes festgelegt (von Schaubert 1986).

PRÜFMETHODE ZUR BEURTEILUNG DER SCHLAGWETTERSICHERHEIT

Für die Prüfung des Zünddurchschlagverhaltens müssen Ansaug- und Abgassysteme in Originalausführung einschließlich der vorgesehenen Plattenschutzpakete verwendet werden, und zwar alle Anlagenteile, die zwischen dem Motor und dem jeweiligen Plattenschutz liegen. Der Motor selbst wird nicht mit in diese Prüfung einbezogen.

Das Prinzip der Prüfung besteht darin, daß das zu prüfende System in ein explosionsfähiges Gas/Luft-Gemisch gebracht wird. Dasselbe Gemisch befindet sich auch innerhalb des Systems, so daß zu beiden Seiten des Plattenschutzes explosionsfähige Atmosphäre vorhanden ist. Die Zündung des Gemisches erfolgt auf der dem Motor zugewandten Seite des Plattenschutzes, wobei der Zündort i. a. im Ansaug- bzw. Abgassammelrohr liegt. Dabei wird festgestellt, ob der Plattenschutz die auf ihn zulaufende Explosion stoppt oder ob ein Zünddurchschlag erfolgt, weil die Explosionsflamme durch den Plattenschutz hindurchläuft und das außen anstehende Gemisch ebenfalls zur Reaktion bringt.

Das verwendete Prüfgas ist ein Mischgas aus Methan und Wasserstoff, das in bestimmten Anteilen mit Luft gemischt wird. Die Zusammensetzung des Mischgases wird durch ein Zahlenpaar angegeben. Die erste Zahl gibt die Volumenanteile von Methan und die zweite die von Wasserstoff an. Je höher der Wasserstoffanteil in einem Mischgas ist, um so zünddurchschlagfreudiger ist es. Aufgrund von Überlegungen, die sich auf Untersuchungen an einem Einzelspalt beziehen, kann den gerade noch zünddurchschlagsicheren Mischgasen ein Sicherheitsfaktor zugeordnet werden (Tabelle 1). Dieser Sicherheitsfaktor ergibt sich als Quotient aus Grenzspaltweite für Methan und Grenzspaltweite des gerade zünddurchschlagsicheren Mischgases. Für die Ansaug- und Abgassysteme wird jeweils das zünddurchschlagsichere Mischgas ermittelt bzw. der Sicherheitsfaktor gegenüber Methan festgestellt. Von einer ausreichenden Schlagwettersicherheit kann immer dann ausgegangen werden, wenn die Prüfung mindestens einen Sicherheitsfaktor von 1,2 ergibt. D.h., daß bei dem Mischgas 80/20 keine Zünddurchschläge erfolgen dürfen.

Tabelle 1: Zuordnung zwischen gerade noch zünddurchschlagsicherem Mischgas und Sicherheitsfaktor zur Beurteilung der Schlagwettersicherheit von Ansaug- und Abgassystemen

gerade noch zünddurchschlagsicheres Mischgas		experimentell ermittelte Grenzspaltweite 1)	Sicherheitsfaktor	ausreichende Schlagwettersicherheit
Mischgaszusammensetzung CH_4/H_2	Mischgaskonzentration in Vol.-%	in mm		
100 / 0	7 - 11	1,20	1,00	Nein
90 / 10	9 - 13	1,05	1,14	Nein
80 / 20	10 - 14	1,00	1,20	Ja
70 / 30	11 - 15	0,90	1,33	Ja
60 / 40	12 - 16	0,80	1,50	Ja
50 / 50	13 - 17	0,70	1,71	Ja
40 / 60	14 - 18	0,70	1,71	Ja
30 / 70	15 - 19	0,60	2,00	Ja
20 / 80	17 - 21	0,50	2,40	Ja
10 / 90	19 - 23	0,45	2,67	Ja
0 / 100	28 - 32	0,35	3,43	Ja

1) bezieht sich auf Untersuchungen am Einzelspalt

$S = \frac{\text{Grenzspaltweite (Methan)}}{\text{Grenzspaltweite (Mischgas)}}$

Im Zusammenhang mit der Prüfung werden die Plattenschutzpakete Spalt für Spalt mit Hilfe einer Fühlerlehre in Abstufungen von 0,05 mm vermessen. Das größte, gerade noch passende Fühlerlehrenmaß wird als maximal zulässige Spaltweite für die in dieser Bauart von Ansaug- bzw. Abgassystem zu verwendenden Plattenschutzpakete festgelegt.

ABGASSYSTEME AUF DER BASIS VON WÄRMEAUSTAUSCHERN

Herkömmliche schlagwettersichere Abgasanlagen, die aus Wassereinspritzung in die Abgassammelleitung mit nachgeschalteter Wasservorlage bestehen, sind sehr voluminös und schwer. Dies ist von Nachteil, da der auf Gleislosfahrzeugen verfügbare Raum sehr begrenzt ist. Auch wird z.T. von den Abgasen mitgerissenes Wasser ausgestoßen, was zu einer zusätzlichen Belastung des Fahrweges führt. Deshalb wurde nach einer Alternative zur herkömmlichen Abgaskühlung gesucht. Die wesentlichen Anforderungen, die ein solches Abgassystem zu erfüllen hat, lassen sich wie folgt zusammenfassen:

- Das Abgassystem sollte möglichst kompakt sein.
- Die Abgaskühlung sollte indirekt erfolgen.
- Um den Kühlmittelverbrauch zu minimieren, sollten nur geschlossene Kühlkreisläufe Verwendung finden.
- Das Abgassystem muß schlagwettersicher sein.

Diese Forderungen können weitgehend durch Abgassysteme, die nach dem Wärmeaustauscherprinzip arbeiten, erfüllt werden. Bei den zwei z.Z. eingesetzten Systemen erfolgt die Abgaskühlung in zwei Stufen. Die Stufe 1 besteht aus einem Rohrbündelwärmeaustauscher mit einem eigenen Kühlwasserkreislauf, der von dem des Dieselmotors i.a. unabhängig ist. Die Abgase werden zunächst auf ca. 130 °C heruntergekühlt. Die weitere Abkühlung der Abgase erfolgt in einer zweiten Stufe mit Hilfe eines Abgasverdünners. Die in den Verdünner mit hoher Geschwindigkeit eintretenden Abgase saugen Umgebungsluft an, die sich mit den Abgasen im Strahlrohr vermischt. Hierdurch wird die Temperatur der Motorabgase unter die zulässige Austrittstemperatur von 70 °C gesenkt. Derartige Abgassysteme werden auch als "trockene Abgaskühlsysteme" bezeichnet. Zwischen Wärmeaustauscher und Abgasverdünner ist als Flammensperre ein Plattenschutz vorgesehen, so daß der Abgasverdünner bereits außerhalb des explosionsgefährdeten Bereichs angeordnet ist.

EXPLOSIONSVERSUCHE IN ABGASANLAGEN AUF WÄRMEAUSTAUSCHERBASIS

Abgasanlage System A

Der Wärmeaustauscher der Abgasanlage System A wird über einen kurzen wassergekühlten Rohrbogen derart an den Dieselmotor angeflanscht, daß der Wärmeaustauscher stets parallel zur Motorlängsseite unmittelbar neben dem motorseitigen Abgassammelrohr liegt (Bild 1). Die aus dem Wärmeaustauscher austretenden Abgase gelangen über einen Kompensator und ein Rohr zum Plattenschutz, dem ein weiteres, zum Abgasverdünner führendes Rohr nachgesetzt ist. Alle nach dem Wärmeaustauscher verlegten Anlagenteile sind nicht mehr wassergekühlt. Bei dem hier verwendeten Wärmeaustauscher handelt es sich um einen Rohrbündelwärmeaustauscher mit Schwimmkopf sowie zwei Rohrwegen und zwei Mantelwegen. Einer der Rohrböden ist lösbar mit dem Mantelflansch verbunden, der andere kann sich frei bewegen. Die Haube ist doppelwandig ausgeführt und besteht aus zwei Kammern mit entsprechenden Öffnungen für den Abgasein- und -austritt. Die heißen Motorabgase strömen radial über die Haube in die erste Kammer, von dort weiter durch das erste Rohrbündel, werden am Rohrboden um 180° umgelenkt, strömen durch das andere Rohrbündel in die zweite Kammer zurück und verlassen den Wärmeaustauscher in axialer Richtung über die Haube.

Die Untersuchung des Zünddurchschlagverhaltens ergab, daß die Abgasanlage beim Mischgas 60/40, bestehend aus 60 Vol.-% Methan und 40 Vol.-% Wasserstoff, zünddurchschlagsicher ist. Für den Plattenschutz liegt die maximal zulässige Spaltweite bei 0,60 mm.

Dieses Versuchsergebnis wurde an einer fabrikneuen Abgasanlage erreicht. Da aber nicht auszuschließen ist, daß sich im Verlauf des betrieblichen Einsatzes das Zünddurchschlagverhalten ändert, wurde die Abgasanlage nach einem zweijährigen Einsatz auf einem Gleislosfahrzeug erneut untersucht. Eine Reinigung des Wärmeaustauschers war nicht erfolgt; lediglich der Plattenschutz war routinemäßig gewartet worden. Die explosionstechnische Nachuntersuchung erfolgte somit im verrußten Zustand. Die Abgasanlage erwies sich auch nach dem zweijährigen Betriebseinsatz als schlagwettersicher. Allerdings fiel zu Beginn dieser Untersuchungen auf, daß die Explosionen in der Abgasanlage nur schwer anliefen und zunächst den Plattenschutz nicht einmal erreichten. Durch Aufwirbeln von Rußablagerungen wurden die Explosionsflammen praktisch erstickt. D.h., im Betrieb werden Rußablagerungen eher zu einer Behinderung als zu einer Verstärkung von Explosionen führen.

Abgasanlage System B

Bei der Abgasanlage System B erfolgt die Verbindung zum Motor über einen wassergekühlten Kompensator (Bild 2).

Dann folgen zwei Wärmeaustauscher, die über einen wassergekühlten Rohrbogen so miteinander verbunden sind, daß die Wärmeaustauscher parallel zueinander liegen. Hinter dem zweiten Wärmeaustauscher befindet sich der Platten-

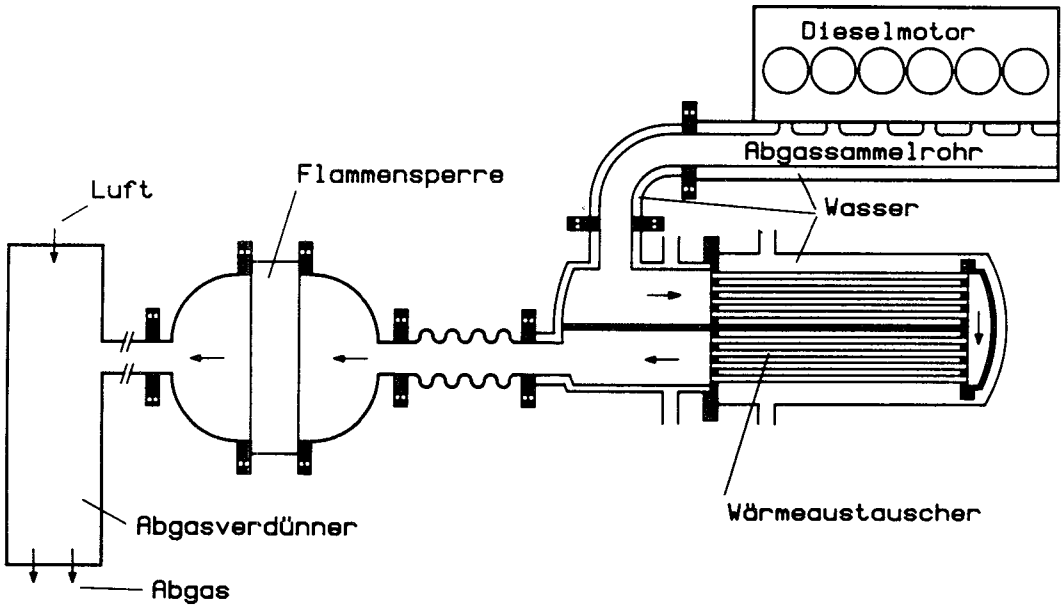


Bild 1: Abgasanlage System A

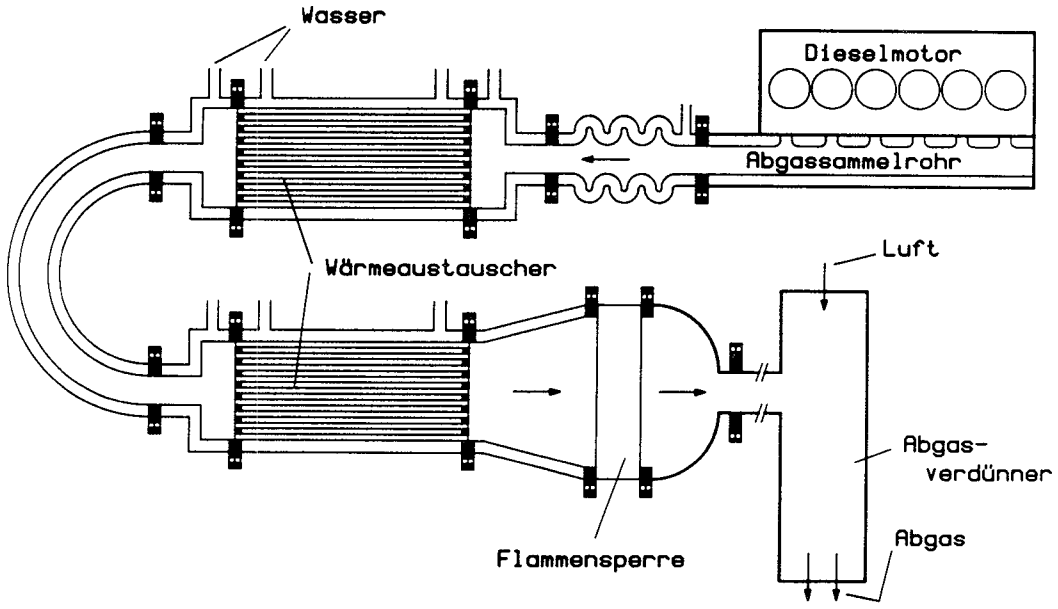


Bild 2: Abgasanlage System B

schutz, dem über eine entsprechende Rohrleitung ein Abgasverdünner nachgesetzt ist. Bei dem eingesetzten Wärmeaustauscher handelt es sich um einen Rohrbündelwärmeaustauscher mit zwei festen Böden, wobei die beiden Hauben ebenfalls wassergekühlt sind. Die Abgase treten in den Wärmeaustauscher jeweils axial über die entsprechende Haube ein bzw. aus ihm aus.

Die explosionstechnische Prüfung der Abgasanlage erfolgte in zwei verschiedenen Anordnungen. Dabei erfolgte die Anbindung der Abgasanlage an das Abgassammelrohr entweder über einen geraden Kompensator (Anordnung 1) oder über einen Kompensator mit 90°-Rohrbogen (Anordnung 2).

Obwohl bei beiden Anordnungen derselbe Plattenschutz mit einer maximalen Spaltweite von 0,65 mm verwendet wurde, ergaben sich geringfügige Unterschiede im Zünddurchschlagverhalten. Bei der Anordnung 1 wurden bei dem Mischgas 80/20 keine Zünddurchschläge mehr registriert. Bei der Anordnung 2 traten bereits bei dem Mischgas 70/30 keine Zünddurchschläge mehr auf. Damit bestätigte sich auch hier, daß die Anlagengeometrie von deutlichem Einfluß auf das Zünddurchschlagverhalten ist.

BETRIEBSERFAHRUNGEN MIT WÄRMEAUSTAUSCHERN IM HINBLICK AUF DEN SCHLAGWETTERSCHUTZ

Die beschriebenen Abgasanlagen auf Wärmeaustauscherbasis sind seit 1983 im Einsatz. Ende 1988 waren die Dieselmotoren von mehr als 20 Gleislosfahrzeugen auf der Abgasseite mit Wärmeaustauschern ausgerüstet. Dabei werden die meisten Fahrzeuge durch den 6-Zyl.-Dieselmotor der Firma MWM, dem sog. RAG-Einheitsmotor, angetrieben. Kleinere Fahrzeuge sind aber auch mit 2-, 3- oder 4-Zyl.-Dieselmotoren desselben Herstellers ausgerüstet. Die trockene Abgaskühlung auf der Grundlage des Wärmeaustauschers hat sich als eine sinnvolle Alternative zur bisher üblichen Wasservorlage herausgestellt. Durch den geschlossenen Kühlkreislauf konnte eine gewisse Unabhängigkeit von der Wasserversorgung im Grubengebäude erzielt werden. Sicherheitstechnische Nachteile insbesondere auf dem Gebiet des Schlagwetterschutzes konnten nicht beobachtet werden. Dies hat auch die explosionstechnische Nachuntersuchung der Abgasanlage System A bestätigt.

Wie bei den alten Abgasanlagen mit Wasservorlage hängt auch bei der trokkenen Abgaskühlung die Schlagwettersicherheit vom einwandfreien Zustand des Plattenschutzes ab.

ALTERNATIVE FLAMMENSPERRE ZUM PLATTENSCHUTZ

Obwohl der Plattenschutz sich vom sicherheitstechnischen Standpunkt aus bewährt hat, weist er doch einige Nachteile auf. Der Plattenschutz ist relativ schwer und verschmutzt leicht. Sein Wartungsbedarf ist relativ hoch. Unsachgemäße Wartung kann zu einer Vergrößerung der Spaltweiten führen. Deshalb wurde nach Ersatz für den Plattenschutz gesucht. Hier boten sich die Flammensperre aus Bandsicherungen und die Flammensperre aus kugelförmigem Schichtgut an. Beide Flammensperrensysteme haben sich bereits bei Grubengasabsaugeanlagen bewährt (Scholl 1979, Bartknecht 1967).

Bandsicherungen als Flammensperre

Eine Bandsicherung besteht aus einem glatten und einem schräg geriffelten Edelstahlband von je 10 mm Breite. Beide Bänder werden aufeinandergelegt und spiralförmig aufgewickelt. So entsteht eine runde Scheibe mit einer Vielzahl nahezu dreieckförmiger Kanäle. Die für das Zünddurchschlagverhalten charakteristischen Abmessungen einer solchen Bandsicherung sind die lichte Dreieckshöhe sowie die Kanallänge. Die lichte Dreieckshöhe beträgt bei der hier betrachteten Ausführung maximal 0,7 mm (Bild 3). Da die Kanäle leicht schräg verlaufen, ist ihre Länge geringfügig größer als die Breite der Sicherung von 10 mm. Da sich je nach Wicklungsrichtung der Bandsicherung auch die Richtung der schräg verlaufenden Kanäle ändert, werden die drei zu einer Flammensperre gehörenden Bandsicherungen stets in wechselnder Wicklungsrichtung hintereinander angeordnet. Dadurch entsteht innerhalb dieser Flammensperre ein im Zickzack verlaufender Kanal.

Eine derartige Bandsicherung wurde in Verbindung mit einer Abgasanlage nach dem System A explosionstechnisch untersucht. Da die Abgasanlage jedoch nur für Dieselmotoren mit zwei bzw. drei Zylindern vorgesehen ist, ist der Wärmeaustauscher entsprechend kleiner

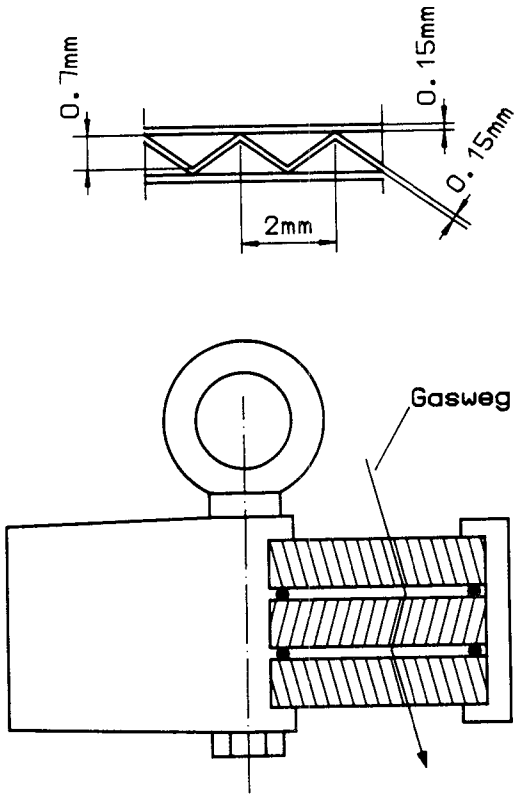


Bild 3: Flammensperre aus 3fach Bandsicherung

und kompakter gebaut. Neben der Band sicherung wurde auch ein Plattenschutz mit einer maximalen Spaltweite von 0,7 mm untersucht. Die Untersuchungen ergaben, daß die Schlagwettersicherheit der Abgasanlage nicht nur durch einen Plattenschutz, sondern auch durch Flammensperren aus 3fach-Bandsicherungen sichergestellt werden kann. Bei Verwendung der Bandsicherung ergaben sich bei dem Mischgas 60/40 keine Zünddurchschläge. Bei Einsatz des Plattenschutzes traten dagegen erst bei dem Mischgas 80/20 keine Zünddurchschläge mehr auf.

Aufgrund dieses positiven Versuchsergebnisses kam es zum Untertageeinsatz dieser Abgasanlage mit 3fach-Bandsicherung in Verbindung mit einem 2-Zyl.-Dieselmotor. Der seit 1987 laufende Einsatz ist aus sicherheitstechnischer Sicht bisher positiv zu bewerten.

Flammensperre aus kugelförmigem Schichtgut

Für die Abgasanlage System B wurde eine Flammensperre aus kugelförmigem Schichtgut gebaut (Bild 4). Als Schichtgut werden Glaskugeln mit einem Durchmesser von 5 mm verwendet. Die Kugelpackung befindet sich zwischen zwei parallel angeordneten Drahtgeweben aus rostfreiem Stahl. Der Drahtdurchmesser beträgt 1 mm und die Maschenweite 3 mm. Die Dicke der Kugelpackung beträgt 60 mm.

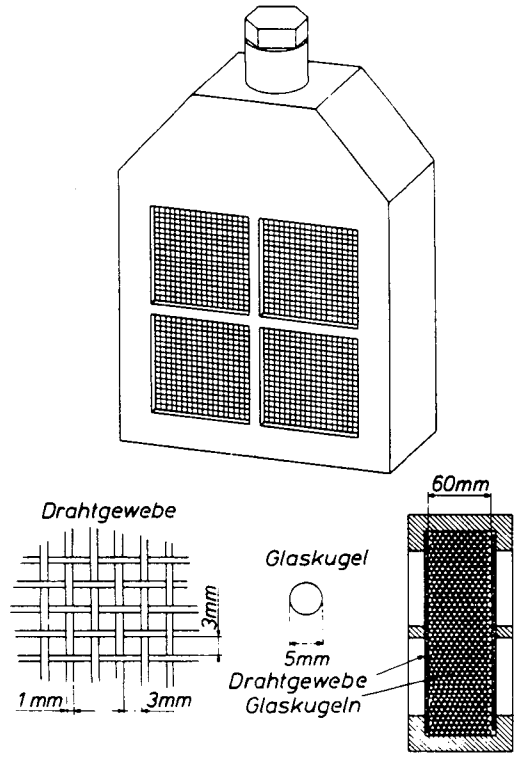


Bild 4: Flammensperre aus kugelförmigem Schichtgut

Die Explosionsversuche ergaben, daß die Glaskugelsperre den Zünddurchschlag von Explosionen des Mischgases 70/30 verhindert und damit die Abgasanlage System B schlagwettersicher ist.

Aufgrund dieses positiven Prüfergebnisses wurde 1986 ein Gleislostransporter mit 6-Zyl.-Dieselmotor auf der Abgasseite mit einer Flammensperre aus Glaskugeln ausgerüstet. Bis Ende 1988 ist die Zahl der mit einer Glaskugelsperre ausgerüsteten Gleislosfahrzeuge auf sechs erweitert worden. Die untertägige Erprobung kann bisher als zu-

friedenstellend bezeichnet werden. Vom Bedienungspersonal wird insbesondere die gute Handhabbarkeit der Kugelsperre bei Wartungsarbeiten hervorgehoben. Im allgemeinen genügt zum Reinigen das Ausblasen der Kugelsperre mit Luft. Bei starker Verschmutzung werden die Kugeln einfach ausgeschüttet und durch saubere Kugeln ersetzt. Die Reinigung der verschmutzten Kugeln kann dann durch Wasser unter Zusatz von Kaltreiniger erfolgen. Bei gleicher Durchtrittsfläche ist der Druckabfall an der Kugelsperre allerdings etwas höher als am Plattenschutz (Bild 5).

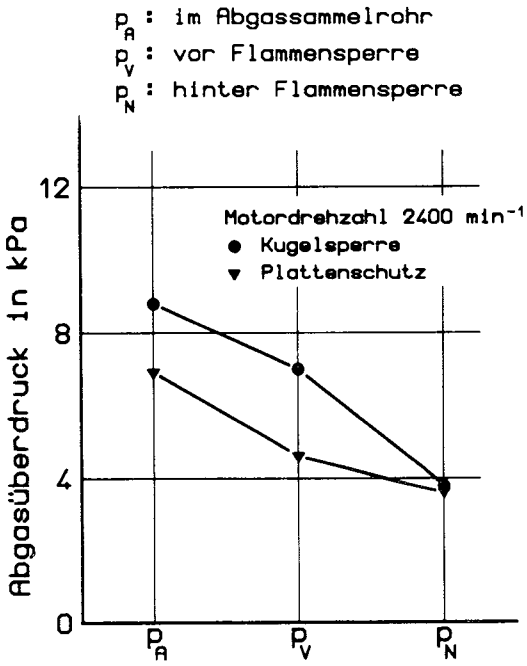


Bild 5: Druckverhältnisse in der Abgasanlage System B

Die Kugelsperre stellt somit einen höheren Strömungswiderstand dar, der auch zu einem höheren Abgasgegendruck am Abgassammelrohr des Dieselmotors führt. Negative Auswirkungen auf den Betrieb der sechs zur Zeit so ausgerüsteten Fahrzeuge konnten bisher noch nicht festgestellt werden. Deshalb wurde auch bisher auf eine Vergrößerung der Kugelsperre und damit eine mögliche Verringerung des Strömungswiderstandes verzichtet.

ZUSAMMENFASSUNG

Im Zuge des verstärkten Einsatzes von Gleislosfahrzeugen im deutschen Steinkohlenbergbau untertage wurden Alternativen zur herkömmlichen direkten Wasserkühlung der Dieselmotorabgase erprobt. Explosionsversuche mit Abgassystemen nach dem Wärmeaustauscherprinzip zeigen, daß diese in Verbindung mit Plattenschutzpaketen schlagwettersicher sind.

Für den bisher üblichen Plattenschutz können Flammensperren aus Bandsicherungen oder aus kugelförmigem Schüttgut (sog. Glaskugelsperren) eine Alternative sein. Entsprechende Explosionsversuche haben ihre Wirksamkeit bewiesen. Insbesondere werden für Glaskugelsperren aufgrund des einfachen Aufbaus, der guten Reinigungsmöglichkeiten und höherer Standzeiten Vorteile gesehen.

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ESSAIS D'EXPLOSIONS ET EXPERIENCES PRATIQUES SUR DES SYSTEMES D'ECHAPPEMENT IGNIFUGES DIESEL A ECHANGE DE CHALEUR

Werner Wiemann

Tous les moteurs diesel utilisés dans les mines de charbon allemandes sont normalement équipés de collecteurs à plaques et de récipients à eau aux extrémités d'admission et de sortie. Des essais d'explosions effectués sur ces équipements ont permis d'établir qu'ils n'étaient pas toujours antidéflagrants, malgré l'espace minimum de sécurité prescrit (largeur et longueur) du pare-flamme. Pour évaluer correctement l'antidéflagration, il a donc été nécessaire d'effectuer des essais d'explosions avec les éléments d'origine. Le gaz utilisé au cours de ces essais était un mélange de méthane à 80% par volume et d'hydrogène à 20% par volume, ce qui assurait une sécurité suffisante vis-à-vis du méthane.

Depuis un certain temps, des essais d'évacuation sont effectués sur des échangeurs de chaleur en faisceaux de tubes munis de collecteurs à plaques. Après plusieurs années de fonctionnement souterrain, une deuxième vérification de ces systèmes d'évacuation n'a révélé aucune modification de leurs propriétés antidéflagrantes. Aujourd'hui, des essais sont également effectués sur des pare-flammes en bandes et des collecteurs en boules de verre; ils se révèlent tout à fait satisfaisants.

ИСПЫТАНИЯ НА ВЗРЫВООПАСНОСТЬ И ОПЫТ ЭКСПЛУАТАЦИИ ОГНЕСТОЙКИХ ДИЗЕЛЬНЫХ ВЫХЛОПНЫХ СИСТЕМ С ТЕПЛООБМЕННИКАМИ

Вернер Виман

Все дизельные двигатели, используемые на каменноугольных шахтах Германии, обычно оборудованы плоскостными ловушками и водяными затворами на всасывании и выхлопе. Испытания на взрывобезопасность двигателей с таким оборудованием показали, что пожаробезопасность не всегда обеспечивается, несмотря на предусмотренный минимальный зазор безопасности (в ширину и длину) в огневой ловушке. Следовательно, чтобы правильно оценить пожаробезопасность, необходимо провести испытания на взрыв с использованием исходных компонентов. Испытательный газ представляет собой объемную смесь 80% метана и 20% водорода, что удовлетворяет условиям безопасности по сравнению с чистым метаном.

Выхлопные системы с многотрубчатыми теплообменниками и дополнительно оборудованными плоскостными ловушками испытываются уже в течение некоторого времени. Повторные испытания таких выхлопных систем после нескольких лет подземной эксплуатации показало отсутствие изменений в степени пожаробезопасности. Испытываемые в настоящее время ленточные и стеклянно-шаровые огневые ловушки показыают вполне удовлетворительные результаты.

有热交换的防火柴油抽风装置的爆炸试验和操作经验

宛尔纳维芒

德国硬煤矿里所有的柴油机都在进气出气口设有板闸门和水箱．从这种设备的爆炸试验结果来看，即使阻焰器符合最小安全间隙（宽度和长度），防火并不能得到保证．所以要正确地评定防火的效果，必须要做原件的爆炸试验．试验气体用百分之八十体积的甲烷和百分之二十体积的氢气，这样可以得到对甲烷的安全性．

管束热交换器的抽风装置已经试验过多年，这种装置并备有板闸门．这种抽风装置设在地下矿井几年后再经过检验，发现对防火并无影响．现今我们也用带状阻焰器和玻璃球阀门做过试验，情形非常良好．

LATEST DEVELOPMENTS WITH VENTILATION TECHNIQUES, WATER SPRAY SYSTEMS AND GAS EMISSION JUDGING METHODS IN MACHINE-DRIVEN HEADINGS

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1. Introduction

Over the period between 1975 and 1987 a total of 72 methane ignitions caused by coalwinning and road-driving machines was reported to the inspectorates in the Federal Republic of Germany. This means a frequency of 5,5 events per year with variations between 2 and 11 events per year. An evaluation of the ignitions assigned to the kind of machine causing them (Fig. 1a) shows that point-attack roadheaders and fullface tunneling machines account for 67 % of the ignitions and thus constitute a particular risk factor. Taking into consideration the number of machines used (Fig. 1b) the facts become even more evident. Thus, in 1987, a number of 129 point-attack roadheaders and fullface tunneling machines accounted for 34 % of a total of 377 coalwinning and road-driving machines, but for 87,5 % of the eight ignitions in 1987. Considering that never an ignition was caused by impact rippers, the above-mentioned disproportion gives even more reason for concern.

Generally, hot surfaces or sparks were found to be the ignition source and, in turn, to be caused by cutting rocks with worn or damaged picks. In particular, rocks with a quartz content of more than 30 % or with pyrite inclusions turned out to be especially prone to ignitions. In the majority of cases the ignited methane accumulation had been formed by a locally insufficient dilution effect of ventilation. In some cases, gas blowers from the floor strata or from faulted zones had been cut into.

Even though the events were only gas deflagrations, i. e. a somewhat moderate form of methane ignition, we have to consider each event of that kind as a potential fire damp explosion whose prevention is and remains a task of highest priority. Actually, also gas deflagrations can be a fatal hazard as shown by one of the events which caused injuries to seven miners including three fatalities.

2. Ventilation

The face area of machine-driven headings is, from the viewpoint of ventilation, a problematic zone since on the one hand dust is generated and, on the other hand, gas and heat emission ta-

ke place. The problem is that for rapid and homogeneous dilution of the methane, high ventilation air speeds, above all directly at the heading face near the cutting tools, are required while efficient dust control necessitates reduced ventilation air speeds, except if the air draught is generated by the deduster system itself. Eventually, air cooling requires additional fans in the face zone because of the relatively high pressure requirements of the air coolers.

Ventilation systems

For the above-described optimization problem three solutions have been developed for headings with point-attack roadheaders (Fig. 2): one solution for low gas emission and without air cooling (system A) and two solutions for higher gas emission and with air cooling (systems B and C).

All three systems need to meet the following common specifications:

- The ventilation air stream coming from the main ventilation air duct line needs to be sufficient for assuring the minimum ventilation air speed in the roadway (0,5 m s^{-1} for inseam roadways; 0,2 m s^{-1} for roadways in the rock).

- During cutting, the duct line end nearest to the heading face is closed by the swivel damper in order not to destroy the dust wall, and the air streams discharged tangentially from the Coanda-type ducts assure sufficient ventilation air speed in the overlapping zone between the main/secondary ventilation air duct line and the deduster duct line.

- When the roadheader is not in operation the heading face is ventilated by the free-jet from the main/secondary ventilation air duct line by turning the swivel damper. In that case the exhaust end of the duct line in question must not be by more than $5\sqrt{A}$ (A = roadway cross-section) outby from the heading face.

The above-mentioned specifications result e. g. in the air flow rates shown on Fig. 2 for A = 18 m^2 (point-attack roadheader).

Ventilation by airmover

For assuring sufficient ventilation air speeds at the heading face - in particular also in the sumping zones behind the cutter head - when the swivel damper is closed compressed-air supplied airmovers are used. Ventilation of the face area by the dedusting system only is not permitted. Since the standard airmovers with flat-jet exhaust are comparably heavy and noisy (94 dB(A)) the twin-jet airmover (Fig. 3) was developed, which, for identical ventilation performance, can be run at lower compressed-air pressure. The weight could be reduced to 11 kg and the noise level down to 75 dB(A).

Ventilation by free-jets

An improvement of the ventilation of the zone between the exhaust end of the main/secondary air duct line and the machine body is possible by using the jet manipulator (Fig. 4).

This device is a two-element swivel damper with openings exhibiting the shape of a sector of circle. In "Maltese cross" position, four free-jets are

generated and propagated almost without mutual influence. Due to the larger total jet surface, an intensified exchange of masses and impulses takes place so that the penetration depth of the total jet is reduced to approximately $3\sqrt{A}$; the jet's range therefore does not go inby the machine body and thus does not destroy the dust wall. In retracted position the same conditions are obtained as with a normally opened swivel damper, i. e. the free-jet ventilates the heading face.

As an alternative to the Coanda-type ducts the secondary-exhaust was developed. This device comprises two steel sheet ducts of different diameters connected in such a way that a sickle-shaped opening remains between the two ducts (Fig. 5).

From this opening a free-jet with a range of about $5\sqrt{A}$ is discharged in axial direction. If the secondary-exhaust is arranged in a distance of about $2\sqrt{A}$ from the exhaust end of the main/secondary ventilation air duct line a good ventilation of the zone up to the machine body can be assured without troubling the dust wall. The discharge opening can be turned in such a way that certain parts of the roadway are ventilated more intensively. Furthermore, the size of the discharge opening and thus the size of the free-jet can be modified by means of a shiftable steel sheet for optimized adaptation to operation conditions. Figure 6 shows an example for the application of a secondary-exhaust in the secondary ventilation air duct line arranged in parallel. Also in the main ventilation air duct line and in the deduster duct line secondary-exhausts are used.

Ventilation with heavy-duty deduster systems

One may speak of a heavy-duty deduster system if it masters an intake air flow rate of at least 0,5 $m^3 s^{-1}$ per m^2 of roadway cross-section. Such a system assures perfect dedusting even if a free-jet actually reaches the heading face, i. e. even with an opened swivel damper or with a secondary-exhaust jet reaching the heading face, cutting may take place. A prerequisite therefore is, however, that the ratio of the deduster intake air flow rate to the air flow rate exhausted from the ventilation duct line equals approximately 3/1. Since for a sufficient ventilation of the heading face the exhausted ventilation air flow rate needs to be at least 0,15 $m^3 s^{-1}$ per m^2 of roadway cross-section, this requirement results in the above-mentioned intake air flow rate of at least 0,5 $m^3 s^{-1}$ per m^2 or, with A = 18 m^2, of 9,0 $m^3 s^{-1}$. Up to present, intake air flow rates up to 13 $m^3 s^{-1}$ have been attained.

Change-over from the sucking to the blowing mode of the cutting zone ventilation.

In headings with fullface tunneling machines it was found useful to change over from the sucking mode of the cutting zone ventilation induced by the deduster system to a blowing mode with simultaneous stop of the cutter head as soon as gas emission problems occur. This is done by a duct switch and corresponding swivel dampers (Fig. 7)

A further development of this system looks out for the swivel dampers being interlocked with the deduster in such a way that as soon as the deduster is stopped, automatic change-over to

blowing ventilation takes place. In that case it is important that the dust shield is well ventilated by a free-jet because methane-enriched air is forced along the dust shield into the free roadway cross-section.

Monitoring and shut-down

Fig. 2 also shows the layout of the sensor heads for air flow and methane concentration measurements and the switchzones S1 (all not-intrinsically safe electrical equipment in the heading), S2 (the not-intrinsically safe electrical equipment between the heading face and the exhaust end of the main ventilation air duct line) and S3 (the not-intrinsically safe electrical equipment between the heading face and the interruption point which exists while the main ventilation air duct line is being extended) which are controlled by the measuring equipment. The shut-down is carried out without delay as soon as a pre-selected threshold value (V_{min} or c_{max}) is attained. V_{min} equals the minimum ventilation air flow rate required for assuring the minimum ventilation air speed over the whole free roadway cross-section. c_{max} equals 1 % CH_4. Shut-down can take place either directly, i. e. by a conventional coupling control or by a fail-safe programmable control.

3. Water spray systems

The following conditions must be fullfilled for ignitions to take place: an explosive methane/air mix, an ignition source with temperatures above the ignition point of the methane/air mix and with sufficient energy, and sufficient time for the reaction of the methane/air mix with the ignition source. In machine-driven headings the actual ignition hazard is constituted by hot surfaces in the cutting track. Those hot surfaces are created by oxidation of the mineral pulverized and pre-heated by the cutting operation. This oxidation requires a certain reaction time.

For the suppression of the ignition hazard special water spray systems have been developed. Their most important technological characteristics are put together in table 1.

Cutting track water spray systems

The technical solution for reducing the ignition hazard which, up to present, is the most frequently practiced one is aimed totally or predominantly at cooling down the ignition source in the cutting track by a water spray. The effectiveness of such cutting track spray systems is assured if, in case of round-shank picks, a sufficient water flow rate at a sufficiently high pressure is delivered from a fullspray or a conical spray nozzle with a suitable diameter into the cutting track, 20 to 30 mm behind the tip of the pick.

The various pressures and nozzle diameters, as well as water consumption for the existing cutting track water spray systems are given in table 1.

The basic configuration of a cutting track water spray system comprises the cutting head with pick holders, water spray nozzles and supply pipes, the water adduction trough the cutting head gear to the cutting head, filters (10 to 25 µm) for the spray water, a high pressure pump for the spray water, and hydraulic and electric monito-

ring and control equipment for water pressure and water flow as well as for the shut-down of the cutting head as soon as given threshold values are attained.

The most important aim of development at the moment is to lower water consumption without injuring the ignition suppressing effect. Methods to realize this are the single-pick control, the sector control and the intermittend control.

The single-pick controlled cutting track water spray system is the most widely proved one. It works in such a way that each pick coming into contact with the face opens via axial movement and by the axially induced cutting force, a valve in the pick holder thus releasing the water jet directed into the cutting track (Fig. 8).

Since, during normal cutting operations, actually only few picks are in contact with the face - in hard rock 4 to 6, i. e. considerably less than the 8 to 12 picks when soft rock is cut - and since during idling no water is discharged at all, water consumption of about 20 to 25 l min^{-1} in hard rock and of 30 to 35 l min^{-1} in medium-strong rock (at a pressure of for example 200 bar and with 0,8 mm diameter nozzles) is remarkably low and definitely in conformity with requirements.

At present, a function check system for cutting track water spray systems on the basis of electronically gathered and filtered oscillation measurement data is developed. This system records the pressure impacts during opening and closing of the valves. Deviations of these pressures from specified threshold values can be identified, assigned to the plugged valve, and used for controlling shut-down of the cutting head.

Cutting track air/ water spray system

Cutting track air/water spray system constitute a generally different approach (table 1). By spraying an air/water mix of 0,25 m^3 min^{-1} of air at approx. 2,5 bar and of 0,5 l min^{-1} of water at approx. 2,5 bar as well via a nozzle with a diameter of 5 mm into each cutting track ignitions are reliably suppressed (Fig. 9).

The effect of this spray system is based on shortening the time of reaction between the hot surfaces of the ignition source and the methane/air mix by higher flow velocities, on cooling of the cutting track by a fine distribution of water droplets, and on displacement of the methane from the cutting track. Due to the lower pressures the design of the swivel joint for air and water does not cause any problem. Each cutting head is to be adjusted for a specified pressure by the manufacturer in order to assure the correct air to water ratio.

The cutting track air/water spray system is available at present for longitudinal cutting heads of point-attack roadheaders up to the 100 t class, and meanwhile well proven. Water consumption was stabilized to approx. 20 l min^{-1} and air consumption to 10 m^3 min^{-1}. This spray system can be monitored in the following way: if air or water adduction is interrupted or if water pressure exceeds a given threshold value, the cutting head is shut down automatically.

The drawbacks on the system are: reduced visibility and working comfort at the heading face because of the formation of fog, more frequent maintenance and larger dimensions of the dedust-er systems.

Horizontally and vertically arranged fan-shaped pressure spray system

The working principal of the horizontally and vertically arranged fan-shaped pressure spray system (table 1) is to limit the accumulation of inflammable methane/air mixes to the smallest possible volume in order to prevent ignitions.

This system comprises two components, viz. the horizontally arranged fan-shaped nozzle arrays and the vertically arranged fan-shaped nozzle arrays (Fig. 10).

The horizontal component comprises a single or multiple nozzle carrier with a total of 26 to 38 nozzles arranged behind the cutting head. The two parts of the vertical component with its 18 nozzles are arranged on the gear head between the two cutting head halves. The total of 44 to 56 nozzles delivers a water flow rate of 51 to 65 l min^{-1} at a pressure in the nozzle carriers of 150 bar.

From the juxtaposed and superposed nozzles the water is discharged at high velocity towards the heading face. The fan-shaped jets filled with water droplets pass over the upper part of the cutting head and, in addition, - directed from the center of the cutting head to both sides - touch the picks and the heading face, and effect the cooling of the hot surfaces there. The pulverized water jets entrain an air flow with a multiple volume of that of the water.

In order to assure permanent effectiveness of the system it is planned to monitor water consumption and operation pressure in such a way that attaining threshold values is reliably detected so that the cutting head can be shut down automatically.

The ventilation air speeds assured anywhere at the heading face alone by the ventilation equipment operational during cutting, i. e. without the spray system of the cutting head being in action, should not fall below 0,7 m s^{-1}. Furthermore, the ventilation conditions should correspond to those obtained when running a heavy-duty deduster system.

4. Gas emission judgement

In seams prone to gas outbursts machine-driven headings are not allowed. This means that with desorbable gas contents > 9 m^3 t^{-1} a method for the early detection of gas outburst hazard is required which is comparable to the determination of the desorption characteristic V_{30} practized in drilling and shot-firing headings. It was looked therefore at the possibility of determining a so called "pseudo-V_{30} value" by the use of existing measuring devices in headings with point-attack roadheaders.

It was found that the methane concentration in a measuring plane 100 m outby the heading face exhibited a behavior which is typical for the various phases of the cutting cycle. Thus, cutting in the coal resulted regularly in concentration peaks (Fig. 11), which, as to their shape, were quite similiar to

the known shot-firing peaks. Then followed higher levels caused by loading. In between, lower values were recorded which were conditioned by the duration of cutting rock. If the minimum values of the concentration curve are connected to form a lower envelope curve, two sections are generated reflecting the gas emission from the recently cut coal and from the coalfaces. By integration, the methane inflows belonging to these sections can be determined. The distribution of the methane inflow (28 % from the recently cut coal and 72 % from the coalfaces) found in the example shown on Fig. 11 was confirmed by gas content determinations on the heading face, the roadside, and the coal produced.

For 10 cutting cycles the methane inflow from the recently cut coal was related to the quantity of the coal cut. This pseudo-V_{30} value ranged between 1,2 and 3,9 $m^3\ t^{-1}$ at an average of 1,9 $m^3\ t^{-1}$. This value was compared to "genuine" V_{30} values determined in a drilling and shot-firing heading in the same seam. The latter values ranged between 0,8 and 3,9 $m^3\ t^{-1}$ at an average of 1,8 $m^3\ t^{-1}$. The congruence can be said to be satisfactory. The evaluation described was carried out by means of a process computer.

While in headings with point-attack roadheaders drilling advance boreholes is just a matter of organization, headings with fullface tunneling machines lacked up to present the possibility to carry out such drillings. For these headings, too, however, it may become necessary to take exploration or stress relief measures when approaching faults or prior to opening seams. Fullface tunneling machines of recent design therefore are equipped with a stationary hydraulic drilling machine on the support setting platform. This drilling machine allows drilling of boreholes in three positions in parallel to the roadway axis. Furthermore, boreholes of up to 120 mm of diameter can be drilled through the slot between dust shield and support at an angle of 13 gon relative to the axis of the roadway.

5. Summary

Machine-driven headings exhibit a particularly high risk with respect to methane ignition. Therefore, high priority is assigned to further development of ventilation and water spray systems. In particular, the following subjects are dealt with: ventilation by airmover, ventilation by free-jets, ventilation with heavy-duty deduster systems, change-over from the sucking to the blowing mode of the cutting zone ventilation, monitoring of ventilation and shut-down of electrical equipment, cutting track water spray systems, cutting track air/water spray system, and horizontally and vertically arranged fan-shaped pressure spray system. Eventually, the state of the art of gas-emission related assessment of machine-driven headings by means of a so called "pseudo-V_{30} value", is discussed as well as recent designs with respect to the integration of advance drilling technology into the concept of fullface tunneling machines.

Table 1: Water spray systems

Spray systems		Medium	Cutting head	Pick holder	Nozzle-Ø in mm/ type of nozzle [3]	Flow rate per nozzle and holder in l min^{-1} ± manufacturer's tolerance in %	Water pressure at holder or nozzle carrier resp. air pressure in bar	Monitoring: Pressure	Monitoring: Flow rate
Cutting track water sprays	Single-pick control	water	Transverse Longitudinal	System Voest-Alpine	0,8 V/or 0,8/10-12°K	3,2 ± 10	120	yes	1)
	Sector control ≧ 180°	water	Transverse	System Krummenauer R. 15.01	0,7/20° K	1,7 ± 5	120	yes	yes
	Intermittend control ≧ 180°	water	Transverse	System Voest-Alpine	0,7/10 - 12°K	At continous operation: 3,2 ± 10. At a pulsation frequency of 36-63 Hz: 1,2 ± 10	120	yes	yes
Cutting track air/water sprays	Full spray	air and water	Longitudinal	System Paurat (Boart)	5 (air) 0,8 (water)	250 ± 5 0,5 ± 5	app. 2,5 app. 2,5 Adduction machine entrance	yes yes	yes yes
HVFPS [4]	Nozzles at the cutting bracket	water	Transverse	Free	0,4-0,6 /V 0,4-0,6 /V	Number of nozzles: 26-38 2x9 [2]; Total flow rate l min^{-1}: 51-65 [2]	150	yes	yes

[1]) Technically not jet realized

[2]) Number and total flow rate according to cutting head width (1600 - 2100 mm)

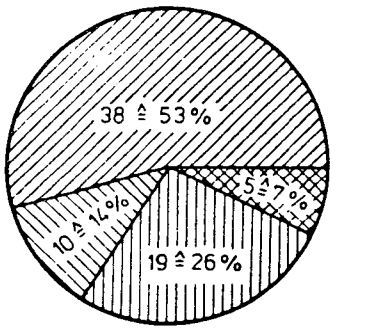
[3]) V = Fullspray nozzle, K = Conical spray nozzle

[4]) HVFPS = Horizontally and vertically arranged fan-shaped pressure sprays

Fig.1 Break-down of methane ignitions by the kind of machine causing the ignition

Federal Republic of Germany

a) Period 1975 – 1987



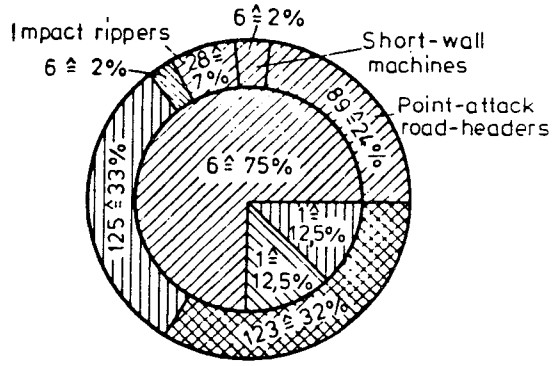
Point-attack roadheaders

Fullface tuneling machines

Shearing winning machines

Stripping winning machines

b) In 1987



(Inner circle) kind of machine causing the ignition

(Outer ring) break-down of winning and roadheading machines run in 1987

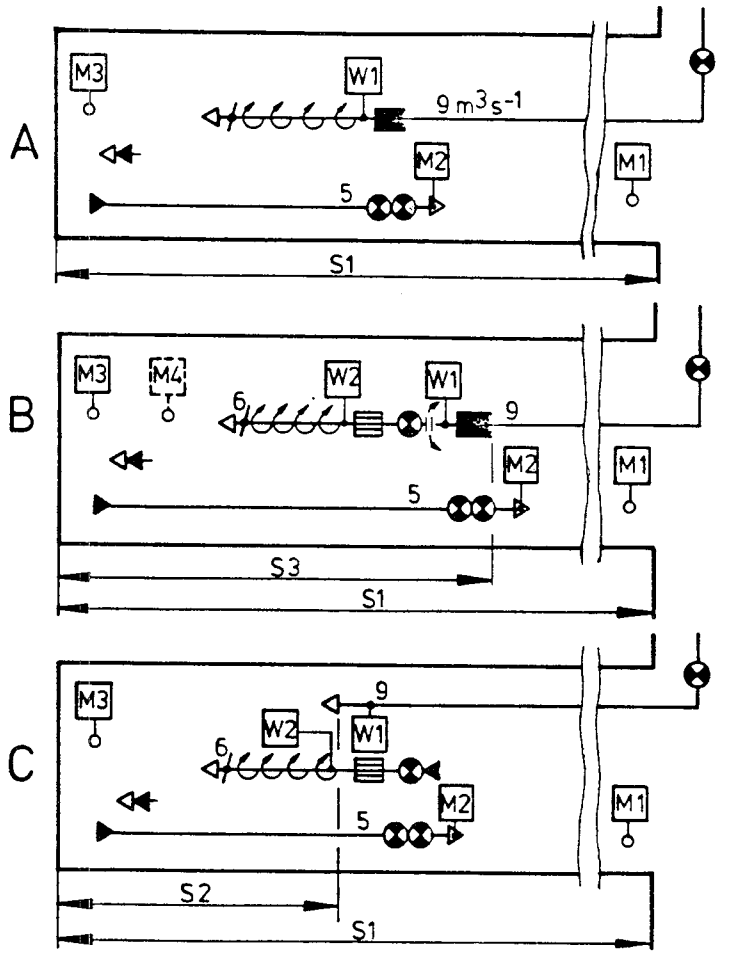


Fig.2 Ventilation systems

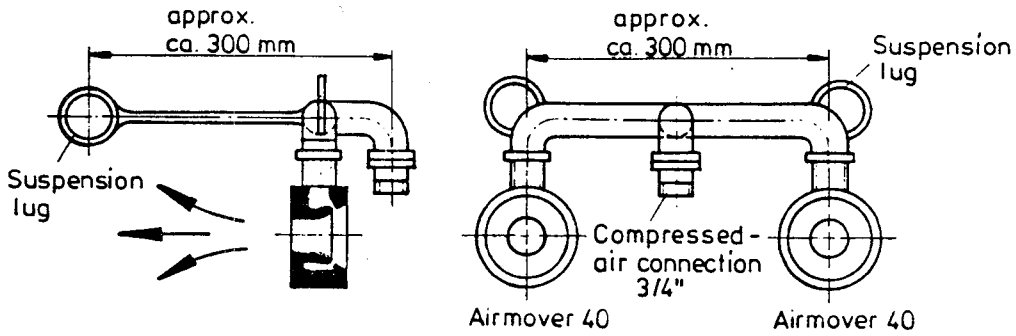


Fig.3 Twin-jet airmover

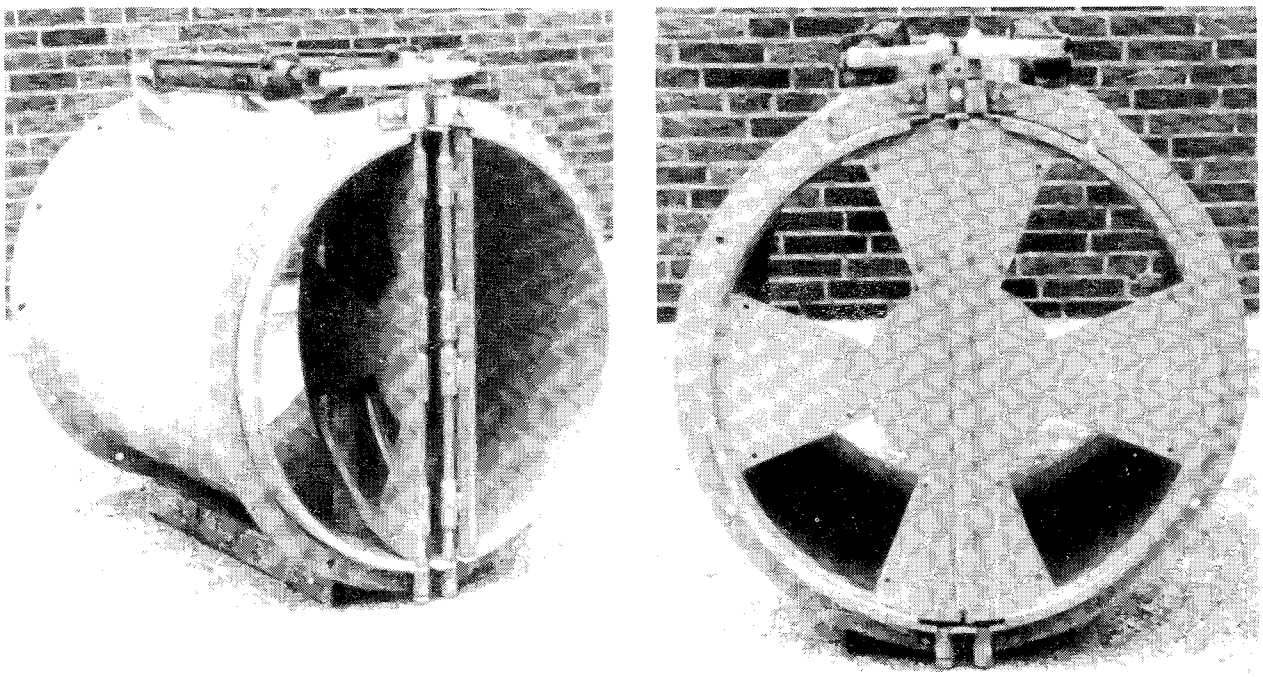


Fig. 4 Jet manipulator

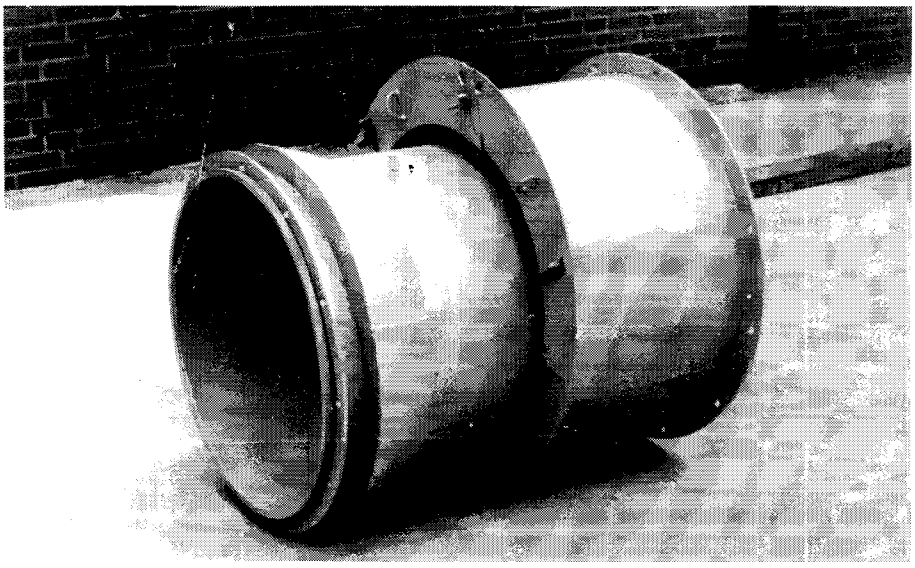


Fig. 5 Secondary-exhaust

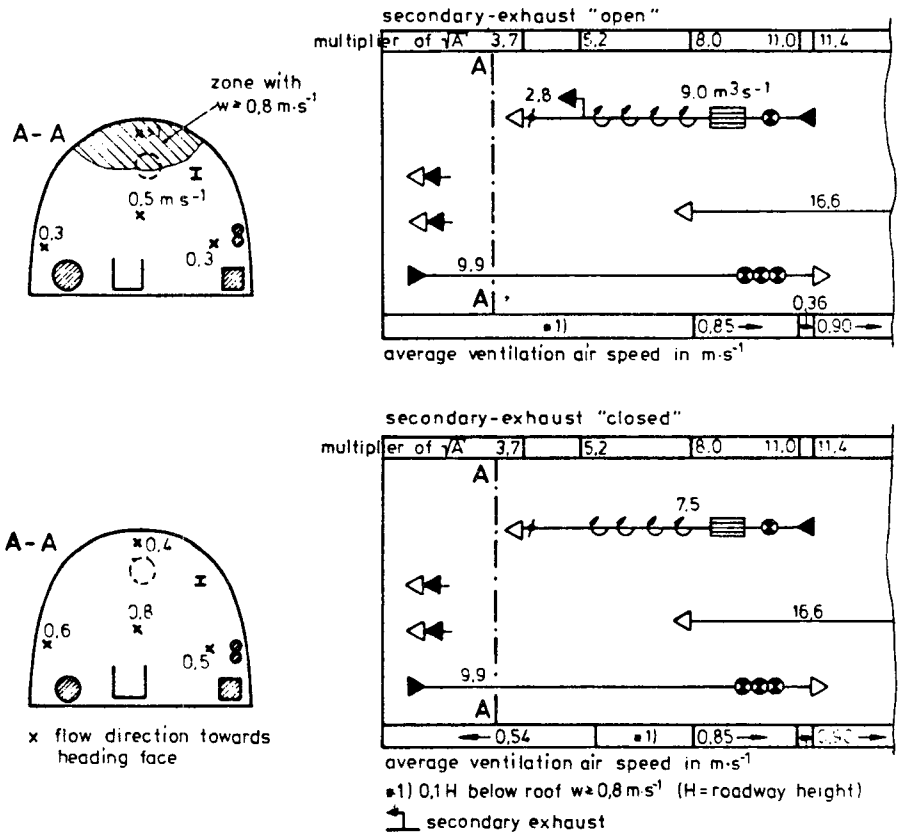


Fig.6 Secondary-exhaust in secondary ventilation air duct line arranged in parallel

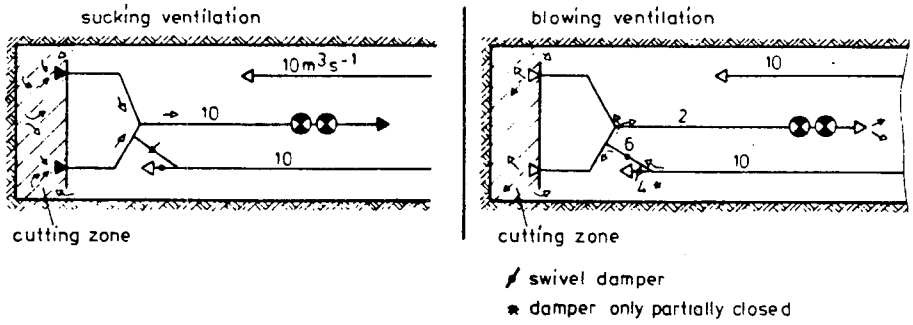


Fig.7 Cutting zone ventilation

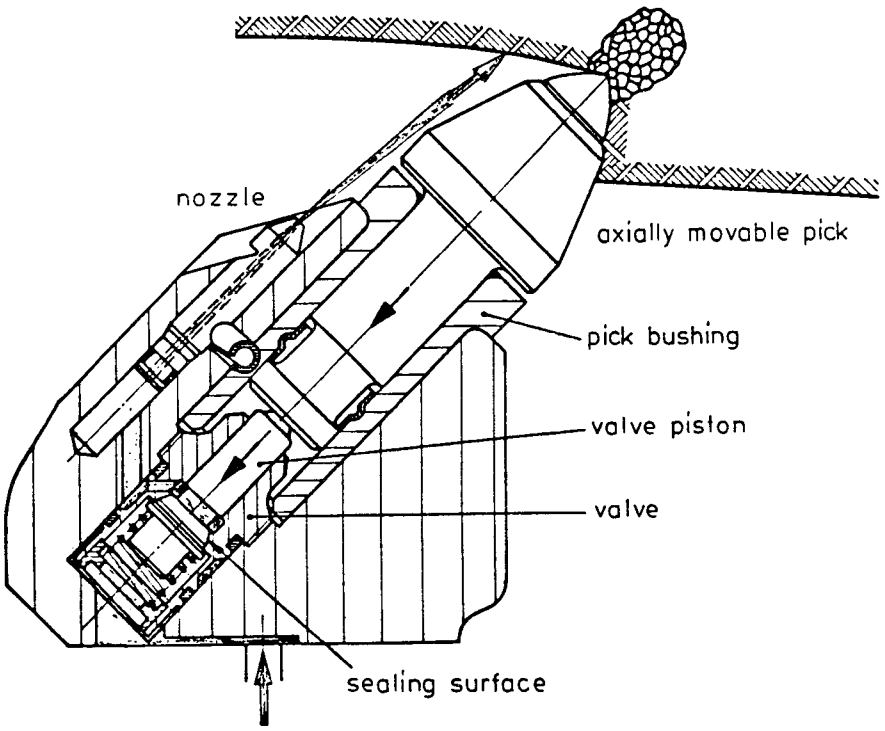


Fig 8 Principle of the single-pick controlled cutting track water spray system of the Voest-Alpine AG

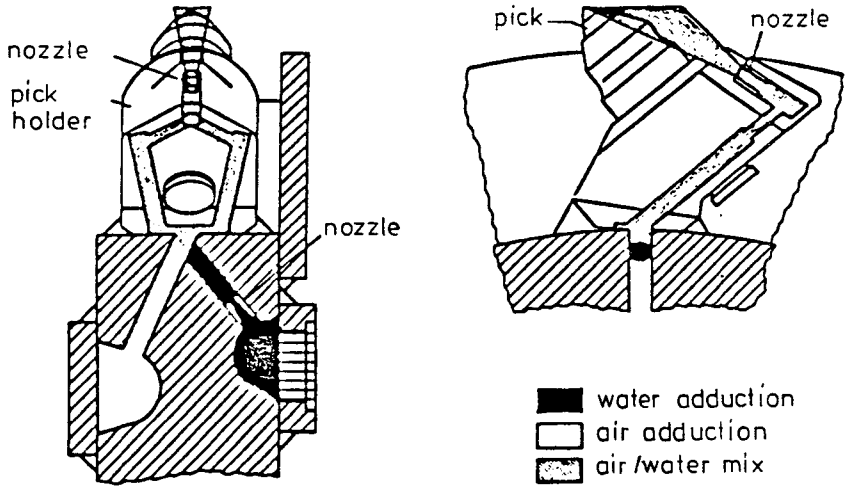


Fig 9 Nozzle for the cutting track air/water spray system of the Paurat GmbH

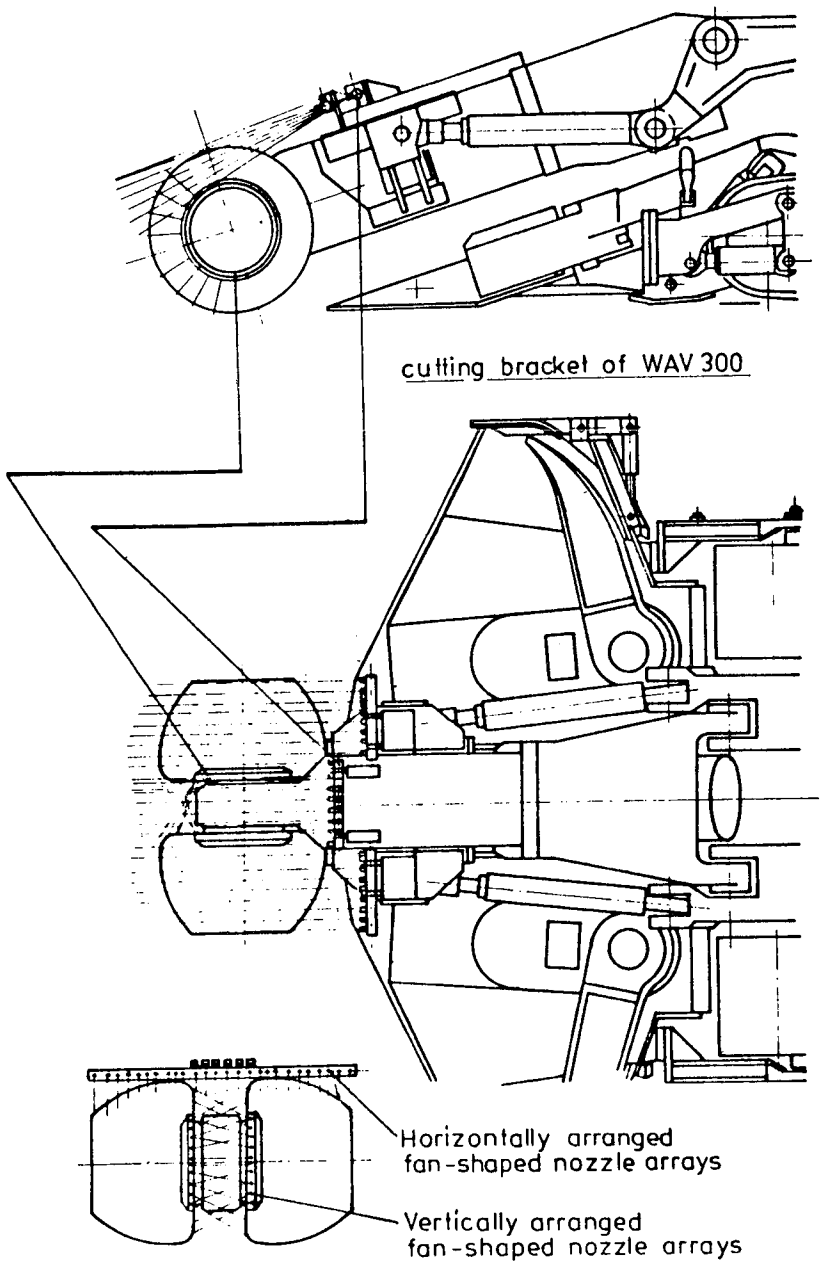
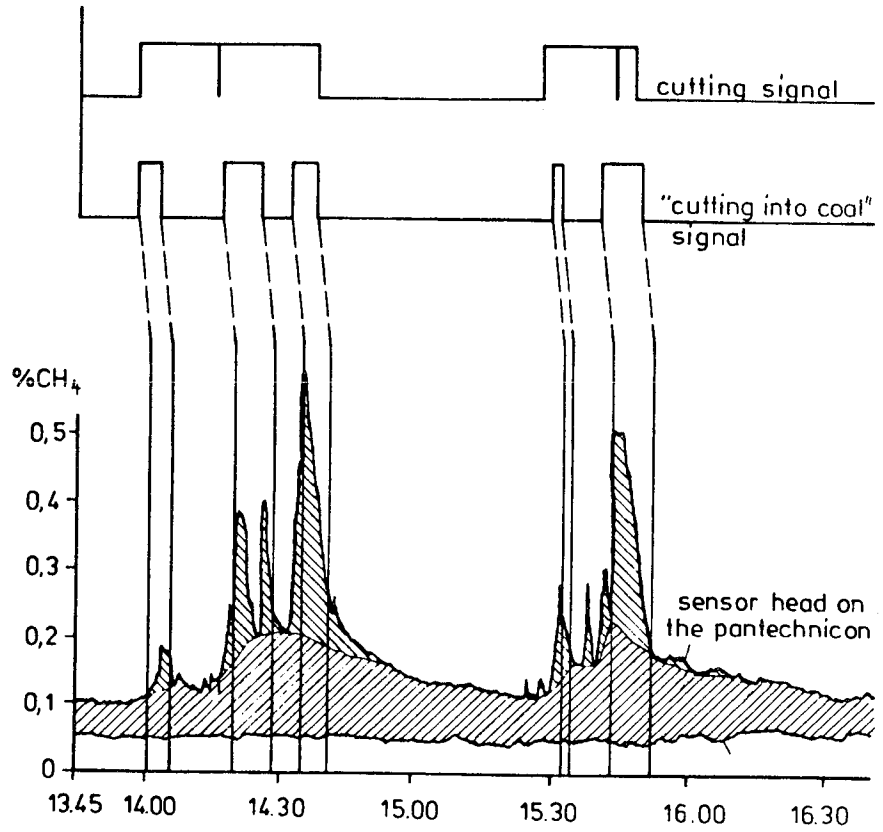


Fig 10 Horizontally and vertically arranged fan-shaped pressure spray system



	methane volume	methane inflow
coal total	64.51 m^3	0.379 $m^3 min^{-1}$ (28 %)
faces total	165.33 m^3	0.973 $m^3 min^{-1}$ (72 %)
grand total	229.84 m^3	1.352 $m^3 min^{-1}$ (100 %)

Fig 11 Evaluation of concentration curves during the cutting cycle of a point-attack roadheader

NEUERE ENTWICKLUNGEN BEI DER BEWETTERUNG, DER BEDÜSUNG UND DER AUSGASUNGSTECHNISCHEN BEURTEILUNG VON MASCHINELLEN STRECKENVORTRIEBEN

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1. Einleitung

Im Zeitraum 1975 bis 1987 sind den Bergbehörden der Bundesrepublik Deutschland 72 Entzündungen von Grubengas gemeldet worden, die durch Gewinnungs- und Vortriebsmaschinen verursacht worden sind. Daraus ergibt sich eine Ereignishäufigkeit von 5,5 pro Jahr, wobei die Spanne von 2 bis zu 11 Ereignissen pro Jahr reicht. Eine Analyse der Zündungen nach der Art der verursachenden Maschine (Bild 1a) zeigt, daß Teil- und Vollschnittmaschinen mit 67 % einen Risikoschwerpunkt darstellen. Dies wird noch deutlicher, wenn man eine Wichtung an der Anzahl der eingesetzten Maschinen vornimmt (Bild 1b). So entfielen im Jahr 1987 von insgesamt 377 Gewinnungs- und Vortriebsmaschinen nur 129 oder 34 % auf den Bereich der Teil- und Vollschnittmaschinen, während sich die acht Zündungen dieses Jahres zu 87,5 % auf diesen Bereich konzentrierten. Berücksichtigt man, daß von Schlagkopfmaschinen noch nie eine Zündung ausgegangen ist, dann wird das Mißverhältnis noch krasser.

Als Zündursache wurden in der Regel heiße Flächen oder Funken ermittelt, die vornehmlich beim Schneiden im Gestein an verschlissenen oder beschädigten Meißeln entstanden waren. Hierbei haben sich Gesteinspartien mit einem Quarzanteil von mehr als 30 % oder mit Pyriteinlagerungen als besonders zündfreudig erwiesen. Die entzündete Grubengasansammlung hatte sich in den meisten Fällen durch eine örtlich unzureichende Spülwirkung der Bewetterung gebildet; gelegentlich sind aber auch Gasbläser aus dem Liegenden oder aus Störungsklüften angeschnitten worden.

Obwohl es sich immer nur um Gasbrände, also milde Formen der Entzündung von Grubengas gehandelt hat, ist davon auszugehen, daß jedes derartige Ereignis eine potentielle Schlagwetterexplosion darstellt, deren Verhütung nach wie vor oberste Priorität genießt. Im übrigen sind auch Gasbrände lebensbedrohend, wie ein Vorfall mit sieben Betroffenen, darunter drei tödlich Verletzten, gezeigt hat.

2. Bewetterung

Der Ortsbereich maschineller Streckenvortriebe stellt eine wettertechnische Problemzone dar, da einerseits Staub entsteht und andererseits Grubengas und Wärme zutreten. Die Problematik besteht darin, daß es zur schnellen und gleichmäßigen Ver-

mischung des Grubengases hoher Wettergeschwindigkeiten, vor allem an der Ortsbrust im Bereich der Schneidwerkzeuge, bedarf, für eine wirkungsvolle Entstaubung die Wettergeschwindigkeiten dagegen möglichst klein sein sollten, es sei denn, sie würden von der Entstaubungs-Anlage selbst erzeugt. Schließlich macht die Wetterkühlung wegen des relativ hohen Druckbedarfs von Wetterkühlern den Einsatz zusätzlicher Ventilatoren im Ortsbereich erforderlich.

Bewetterungssysteme

Für das so umschriebene wettertechnische Optimierungsproblem haben sich in Streckenvortrieben mit Teilschnittmaschine im wesentlichen drei Lösungen herausgebildet (Bild 2), und zwar eine bei geringerer Ausgasung und ohne Kühlung (System A) und zwei bei stärkerer Ausgasung und mit Kühlung (Systeme B und C).

Wesentliche gemeinsame Forderungen an diese Systeme sind:

- Der in der Hauptluttenleitung herangeführte Frischwetterstrom muß ausreichen, um die Mindestwettergeschwindigkeit in der Strecke (0,5 m/s für Flözstrecken, 0,2 m/s für Gesteinsstrecken) sicherzustellen.

- Die Absperrklappe ist beim Schneiden geschlossen, um die Staubwand nicht zu zerstören, und die tangential aus den Coanda-Lutten austretenden Wetter sorgen für ausreichende Wettergeschwindigkeiten im Überlappungsabschnitt zwischen Haupt- bzw. Nebenluttenleitung und Entstauberleitung.

- Die Ortsbrust wird beim Nichtschneiden nach Öffnung der Absperrklappe durch den aus der Haupt- oder der Nebenluttenleitung austretenden Freistrahl bewettert. Dazu darf das Ausblasende der jeweiligen Luttenleitung nicht weiter als $5\sqrt{A}$ (A = Streckenquerschnitt) von der Ortsbrust entfernt sein.

Aus den genannten Forderungen ergeben sich z. B. die in Bild 2 eingetragenen Wetterströme, und zwar für A = 18 m^2 (Teilschnittmaschine).

Strahldüsenbewetterung

Um auch bei geschlossener Absperrklappe ausreichende Wettergeschwindigkeiten an der Ortsbrust - insbesondere auch in den durch Vorschneiden in der Kohle entstehenden Einbruchschlitzen - zu gewährleisten, werden druckluftbetriebene Strahldüsen eingesetzt; eine Bewetterung des Ortsbereiches allein durch die Entstaubungsanlage ist nicht zulässig. Da die üblichen Strahldüsen mit Flachstrahlvorsatz verhältnismäßig schwer (27 kg) und laut (94 db(A)) sind, wurde die Doppelstrahldüse (Bild 3) entwickelt, die bei gleicher Wetterleistung mit einem geringeren Druckluftdruck betrieben werden kann. Das Gewicht konnte auf 11 kg und der Schalldruckpegel auf 75 dB(A) gesenkt werden.

Freistrahlbewetterung

Eine Verbesserung der Bewetterung des Raumes zwischen dem Ausblasende der Haupt- bzw. Nebenluttenleitung und der Vortriebsmaschine läßt sich mit dem Strahlmanipulator erreichen (Bild 4).

Hierbei handelt es sich um eine zweigeteilte, mit sektorförmigen Aussparungen versehene Absperrklappe. Im ausgeklappten Zustand werden vier Freistrahlen er-

zeugt, die sich weitgehend ohne gegenseitige Beeinflussung ausbreiten. Durch die insgesamt größere Strahloberfläche findet ein intensiverer Massen- und Impulsaustausch statt, so daß sich die Eindringtiefe des Gesamtstrahles auf etwa $3\sqrt{A}$ verringert; der Strahl reicht also nur noch bis zur Maschine und zerstört die Staubwand nicht. Im eingeklappten Zustand stellen sich dieselben Verhältnisse wie bei einer gewöhnlichen Absperrklappe ein, d. h. der Freistrahl bespült die Ortsbrust.

Als Alternative zu Coanda-Wirbellutten wurde der Sekundäraustrag entwickelt. Dies ist eine Einrichtung, bei der zwei Blechlutten verschiedenen Durchmessers so miteinander verbunden sind, daß zwischen ihnen eine sichelförmige Öffnung freibleibt (Bild 5).

Aus dieser tritt ein Freistrahl in axialer Richtung mit einer Eindringtiefe von etwa $5\sqrt{A}$ aus. Baut man den Sekundäraustrag in einer Entfernung von etwa $2\sqrt{A}$ vom Ausblasende der Haupt- bzw. Nebenluttenleitung ein, dann ist auch auf diese Weise eine gute Bewetterung des Bereiches bis zur Vortriebsmaschine ohne Zerstörung der Staubwand möglich. Die Ausblasöffnung kann so gedreht werden, daß bestimmte Teile der Strecke besonders intensiv bespült werden. Außerdem kann man mit Hilfe eines verschiebbaren Bleches die Größe der Ausblasöffnung und damit des Freistrahls verändern und dem Betrieb optimal anpassen. Bild 6 zeigt ein Beispiel für den Einsatz eines Sekundäraustrages in der parallelen Nebenluttenleitung. Auch in der Hauptlutten- und in der Entstauberleitung werden Sekundärausträge eingesetzt.

Bewetterung mit Hochleistungs-Entstaubungsanlage

Von einer Hochleistungs-Entstaubungsanlage spricht man, wenn ein Rohluftstrom von mindestens 0,5 $m^3 s^{-1}$ je m^2 Streckenquerschnitt abgesaugt wird. Mit ihr ist eine einwandfreie Entstaubung selbst dann gewährleistet, wenn ein Freistrahl die Ortsbrust erreicht; d. h. es kann bei geöffneter Absperrklappe oder mit einem bis zur Ortsbrust wirkenden Sekundäraustrag geschnitten werden. Voraussetzung dafür ist jedoch, daß das Verhältnis des Rohluftstromes zum ausgeblasenen Wetterstrom etwa 3 zu 1 beträgt. Da für eine ausreichende Bespülung der Ortsbrust der als Freistrahl austretende Wetterstrom mindestens 0,15 $m^3 s^{-1}$ je m^2 Streckenquerschnitt betragen muß, führt diese Forderung zu dem genannten Rohluftstrom von mindestens 0,5 $m^3 s^{-1}$ je m^2 oder bei A = 18 m^2 von 9,0 $m^3 s^{-1}$. Bisher sind Rohluftströme bis 13 $m^3 s^{-1}$ erzielt worden.

Umschalten von saugender auf blasende Schneidraumbewetterung

In Streckenvortrieben mit Vollschnittmaschine hat es sich bewährt, die von der Staubabsaugung bewirkte saugende Schneidraumbewetterung, sobald Ausgasungsschwierigkeiten auftreten, bei gleichzeitigem Stillsetzen des Schneidkopfes auf blasend umzustellen. Dies geschieht mit einer Luttenweiche und entsprechenden Absperrklappen (Bild 7).

Eine Weiterentwicklung dieses Systems sieht vor, die Absperrklappen so mit dem Entstauber zu verriegeln, daß immer dann selbsttätig auf Blasen umgestellt wird, wenn der Entstauber außer Betrieb geht. Wichtig ist

dabei, daß der Staubschild durch einen Freistrahl gut bespült wird, da mit Methan angereicherte Wetter am Staubschild vorbei in den freien Streckenquerschnitt gedrückt werden können.

Überwachung und Abschaltung

Bild 2 zeigt auch die Anordnung der wettertechnischen Meßköpfe für Wetterstrom und Methangehalt sowie die von den Meßgeräten angesteuerten Schaltbereiche S1 (sämtliche nichteigensicheren elektrischen Betriebsmittel im Vortrieb) oder S2 (die nichteigensicheren elektrischen Betriebsmittel zwischen Ortsbrust und Ausblasende der Hauptluttenleitung) bzw. S3 (die nichteigensicheren elektrischen Betriebsmittel zwischen Ortsbrust und der bei der Verlängerung der Hauptluttenleitung entstehenden Trennstelle). Die Abschaltung erfolgt unverzögert, sobald ein vorgewählter Grenzwert $\dot{V}_{min}$ bzw. c_{max} verletzt wird. Hierbei entspricht $\dot{V}_{min}$ dem Mindestwetterstrom, der benötigt wird, um auch im größten freien Querschnitt der Strecke die Mindestwettergeschwindigkeit einzuhalten; c_{max} ist gleich 1 % CH_4. Die Abschaltung kann direkt, d. h. mit Hilfe der Koppelgliedtechnik, oder unter Einsatz einer sicherheitsgerichteten speicherprogrammierbaren Steuerung erfolgen.

3. Bedüsung

Bei Zündvorgängen sind folgende Voraussetzungen erforderlich: ein zündfähiges Methan-Luft-Gemisch, eine Zündquelle mit Temperaturen über dem Zündpunkt des Methan-Luft-Gemisches und mit genügend großer Energie sowie eine ausreichende Reaktionszeit zwischen Methan-Luft-Gemisch und Zündquelle.

Im maschinellen Streckenvortrieb kommt die eigentliche Zündgefahr von den heißen Flächen in der Schneidspur. Diese entstehen durch Oxidation des beim Zerspanen pulverisierten und vorerhitzten Gesteins. Die Oxidation benötigt eine gewisse Reaktionszeit.

Zur Unterdrückung der Zündgefahr sind spezielle Bedüsungseinrichtungen entwickelt worden. Die wichtigsten technischen Merkmale dieser Einrichtungen sind in Tabelle 1 zusammengestellt.

Wasser-Schneidspurbedüsung

Die bisher am häufigsten verfolgte technische Lösung für die Verminderung der Zündgefahr zielt ganz oder überwiegend auf eine Kühlung der Zündquelle in der Schneidspur durch Bedüsung mit Wasser ab. Die Wirksamkeit dieser Schneidspurbedüsung ist gegeben, wenn beim Rundschaftmeißel ein ausreichender Wasserstrom bei einem genügend hohen Druck aus einer Voll- oder Kegelstrahldüse mit einem geeigneten Durchmesser in die Schneidspur gespritzt wird, und zwar 20 bis 30 mm hinter der Meißelspitze.

Die unterschiedlichen Drücke und Düsendurchmesser und damit der Wasserbedarf der verschiedenen Einrichtungen zur Wasser-Schneidspurbedüsung ergeben sich aus Tabelle 1.

Der grundsätzliche Aufbau einer Wasser-Schneidspurbedüsung umfaßt den Schneidkopf mit Meißelhaltern, Wasserdüsen und -zuleitungen zu denselben, die Wasserzuleitung zum Schneidkopf mit Drehdurchführung durch das Schneidkopfgetriebe, Filter für das Bedüsungswasser mit einer Filterfeinheit von 10 bis 25 µm, eine Hochdruckpumpe für das Be-

düsungswasser und hydraulische und elektrische Überwachungs- und Regeleinrichtungen für den Wasserdruck und den Wasserstrom sowie für das Abschalten des Schneidmotors bei Unterschreiten der Soll-Werte.

Das wichtigste Entwicklungsziel ist zur Zeit die Senkung des Wasserverbrauchs ohne Beeinträchtigung der zündunterdrückenden Wirkung. Wege dazu sind die Einzelmeißelsteuerung, die Sektorsteuerung und die intermittierende Steuerung.

Die umfangreichsten Erfahrungen liegen über die einzelmeißelgesteuerte Wasser-Schneidspurbedüsung vor. Diese arbeitet so, daß jeder längsbewegliche Meißel, der beim Schneidvorgang in Eingriff kommt, durch die auf ihn axial übertragene Schneidkraft ein am Boden des Meißels im Meißelhalter befindliches Ventil öffnet und einen Wasserstrahl zur Bedüsung der Schneidspur freigibt (Bild 8).

Weil beim normalen Schneidvorgang tatsächlich nur wenige Meißel im Eingriff sind, und zwar im Hartgestein mit 4 bis 6 deutlich weniger als in weichem Gestein mit 8 bis 12 Meißeln, und weil im Leerlauf überhaupt kein Wasser austritt, ist der Wasserbedarf mit rd. 20 bis 25 l min^{-1} im Hartgestein und 30 bis 35 l min^{-1} in mittelfestem Gestein bei einem Druck von beispielsweise 200 bar und Düsen mit einem Durchmesser von 0,8 mm erfreulich gering und entspricht vollständig den Anforderungen.

Zur Zeit wird ein Funktionskontrollsystem für Wasser-Schneidspurbedüsungen auf der Basis von elektronisch erfaßten und gefilterten Schwingungsmessungen entwickelt, mit dem die Druckstöße beim Öffnen und Schließen der Ventile ermittelt und aufgezeichnet werden. Abweichungen des Druckes von festgelegten Grenzwerten können erkannt, dem verstopften Ventil zugeordnet und zum Abschalten des Schneidkopfes genutzt werden.

Luft-Wasser-Schneidspurbedüsung

Einen grundsätzlich anderen Ansatz stellt die Luft-Wasser-Schneidspurbedüsung (Tabelle 1) dar. Durch Versprühen eines Luft-Wasser-Gemisches von 0,25 m^3 min^{-1} Luft mit rd. 2,5 bar und 0,5 l min^{-1} Wasser mit rd. 2,5 bar über Düsen mit einem Durchmesser von 5 mm in jede Schneidspur wird eine sichere Zündunterdrückung erreicht (Bild 9).

Die Wirkung dieser Bedüsung beruht auf einer Verkürzung der Reaktionszeit zwischen den heißen Flächen der Zündquelle und dem Methan-Luft-Gemisch aufgrund hoher Strömungsgeschwindigkeiten, einer Kühlung der Schneidspur durch feinverteilte Wassertröpfchen und einer Verdrängung des Methans aus der Schneidspur. Aufgrund des geringen Druckes gestaltet sich die Ausführung der Drehdurchführung für Luft und Wasser problemlos. Jeder Schneidkopf ist für einen bestimmten Druck vom Hersteller zu justieren, um das richtige Verhältnis von Luft und Wasser zu gewährleisten.

Die Luft-Wasser-Schneidspurbedüsung ist zur Zeit für Längsschneidköpfe von Teilschnittmaschinen bis zur 100-t-Klasse verfügbar und erprobt; es stellen sich ein Wasserverbrauch von rd. 20 l min^{-1} und ein Luftverbrauch von 10 m^3 min^{-1} ein. Diese Bedüsung kann wie folgt überwacht werden: Wird die Zufuhr von Luft oder Wasser unterbro-

chen oder übersteigt der Wasserdruck einen Grenzwert, wird der Motor des Schneidkopfes automatisch abgeschaltet.

Die Nachteile sind: Verschlechterung der Sichtverhältnisse und des Arbeitskomforts an der Ortsbrust wegen Nebelbildung, häufigere Wartung und größere Dimensionierung der Entstaubungsanlage.

Fächerdruck- und Schwertbedüsung

Die Fächerdruck- und Schwertbedüsung (Tabelle 1) hat die Aufgabe, die Ansammlung zündfähigen Methan-Luft-Gemisches auf kleinsten Raum zu begrenzen und auf diese Weise eine Zündung zu verhindern.

Das Bedüsungssystem setzt sich aus zwei Komponenten, der Fächerdruck- und der Schwertbedüsung, zusammen (Bild 10).

Die Fächerdruckbedüsung besteht aus einem ein- oder mehrteiligen Düsenträger mit insgesamt 26 bis 38 Düsen, welche vor dem Schneidkopf angeordnet sind; die Schwertbedüsung mit 18 Düsen befindet sich an dem Getriebehals zwischen den beiden Schneidkopfhälften. Die insgesamt 44 bis 56 Düsen versprühen bei einem Druck im Düsenträger von 150 bar einen Wasserstrom von 51 bis 65 l min^{-1}.

Aus den neben- bzw. übereinander angeordneten Düsen tritt das Wasser mit hoher Geschwindigkeit auf die Ortsbrust gerichtet aus. Die mit Wassertropfen erfüllten, fächerförmigen Strahlen tangieren oben den Schneidkopf, treffen zusätzlich, von der Schneidkopfmitte aus nach beiden Seiten gerichtet, auf Meißel und Ortsbrust und bewirken dort die Kühlung heißer Flächen. Die in feine Tropfen zerstäubenden und einen Luftstrom bewegenden Wasserstrahlen reißen ein Vielfaches an Luft aus der Umgebung mit.

Damit die Wirksamkeit der Bedüsung ständig erhalten bleibt, ist eine Überwachung des Wasserverbrauchs sowie des Betriebsdruckes in der Weise vorgesehen, daß ein Unterschreiten von Grenzwerten sicher erkannt und der Schneidkopf in einem solchen Fall automatisch stillgesetzt wird.

Die ohne Betrieb der Bedüsung des Schneidkopfes allein durch die beim Schneiden betriebenen Bewetterungseinrichtungen überall an der Ortsbrust erzeugten Wettergeschwindigkeiten sollten 0,7 m s^{-1} nicht unterschreiten. Im übrigen sollten die Bewetterungsverhältnisse denjenigen bei Einsatz einer Hochleistungs-Entstaubungsanlage entsprechen.

4. Ausgasungstechnische Beurteilung

In gasausbruchsgefährlichen Flözen ist maschineller Streckenvortrieb verboten. Das bedeutet, daß bei desorbierbaren Gasinhalten >9 m^3 t^{-1} ein Verfahren für die Früherkennung von Gasausbruchsgefahr benötigt wird, das der in Sprengvortrieben angewandten Ermittlung des Desorptionskennwertes V_{30} vergleichbar ist. Es wurde daher untersucht, ob sich in Streckenvortrieben mit Teilschnittmaschine unter Nutzung der vorhandenen Meßtechnik ein sogenannter Pseudo-V_{30}-Wert bestimmen läßt.

Man erkannte, daß die Methankonzentration in einer der Ortsbrust im Abstand von 100 m nachfolgenden Meßebene für die unterschiedlichen Arbeitsvorgänge typische Verläufe aufwies. So führte das Schneiden in der Koh-

le regelmäßig zu Konzentrationsspitzen (Bild 11), die in ihrem Aussehen den bekannten "Sprengkurven" ähnelten. Es folgten dann Überhöhungen, die durch die Ladearbeit verursacht wurden. Dazwischen lagen Rückgänge, die von der Dauer des Schneidens im Gestein bestimmt waren. Verbindet man die Minima der Konzentrationsverläufe zu einer unteren Hüllkurve, so entstehen zwei Flächen, welche die Ausgasung aus der frisch geschnittenen Kohle bzw. aus den Streckenstößen repräsentieren. Durch Integration können die zugehörigen Methanzuströme bestimmt werden. Die am Beispiel des Bildes 11 gefundene Verteilung des Methanzustroms (28 % aus der frisch geschnittenen Kohle und 72 % aus den Streckenstößen) wurde durch Gasinhaltsbestimmungen an der Ortsbrust, den Seitenstößen und der Förderkohle bestätigt.

Für 10 Schneidvorgänge wurde der Methanzustrom aus der Kohle auf die jeweils geschnittene Kohlenmenge bezogen; dieser Pseudo-V_{30}-Wert schwankte zwischen 1,2 und 3,0 $m^3\ t^{-1}$ bei einem Mittelwert von 1,9 $m^3\ t^{-1}$. Er wurde echten in einem Sprengvortrieb im gleichen Flöz ermittelten V_{30}-Werten zum Vergleich gegenübergestellt; diese schwankten zwischen 0,8 und 3,9 $m^3\ t^{-1}$ bei einem Mittelwert von 1,8 $m^3\ t^{-1}$. Die Übereinstimmung ist als zufriedenstellend zu bezeichnen. Die beschriebene Auswertung erfolgt mit einem Prozeßrechner.

Während bei Streckenvortrieben mit Teilschnittmaschine die Herstellung von Erkundungsbohrungen lediglich eine Organisationsfrage ist, boten Streckenvortriebe mit Vollschnittmaschine bisher keine Möglichkeiten für gezielte Erkundungsbohrungen. Auch hier kann es jedoch erforderlich sein, bei Annäherung an Störungen oder vor dem Anschneiden von Flözen Erkundungs- oder Entspannungsmaßnahmen durchzuführen. Vollschnittmaschinen neuester Bauart sind deshalb auf den Ausbauplattformen mit einer stationären hydraulischen Lafettenbohrmaschine ausgerüstet. Mit ihr können im Auffahrprofil drei Bohrlochpositionen parallel zur Streckenachse gebohrt werden. Außerdem können zwischen Staubschild und Ausbau Bohrlöcher bis zu 120 mm Durchmesser unter einem Winkel von 13 gon zur Streckenachse hergestellt werden.

5. Zusammenfassung

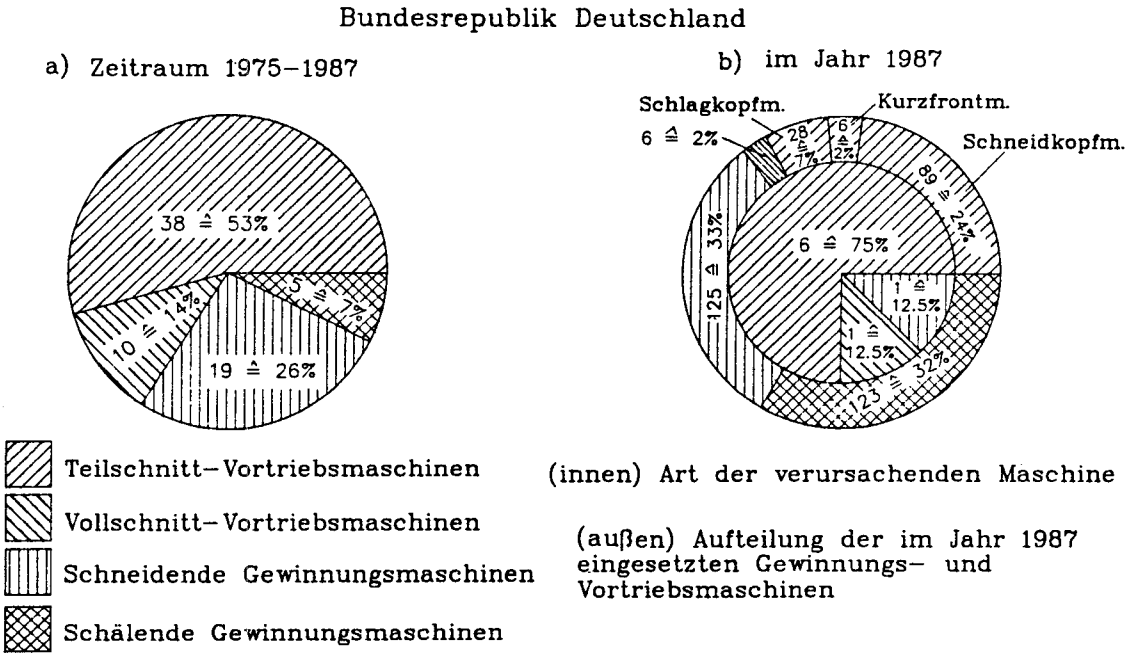
Im Hinblick auf die Gefahr der Entzündung von Grubengas stellt der maschinelle Streckenvortrieb einen Risikoschwerpunkt dar; daher kommt der Weiterentwicklung der Bewetterungs- und der Bedüsungssysteme hohe Priorität zu. Im einzelnen werden behandelt: die Strahldüsenbewetterung, die Freistrahlbewetterung, die Bewetterung mit Hochleistungs-Entstaubungsanlage, das Umschalten von saugender auf blasende Schneidraumbewetterung, die Überwachung der Bewetterung und die Abschaltung der elektrischen Betriebsmittel, die Wasser-Schneidspurbedüsung, die Luft-Wasser-Schneidspurbedüsung und die Fächerdruck- und Schwertbedüsung. Schließlich wird auf den neuesten Stand bei der ausgasungstechnischen Beurteilung maschineller Flözstreckenvortriebe mit Hilfe des sogenannten Pseudo-V_{30}-Wertes und bei der Integration der Vorbohrtechnik in das Vollschnittmaschinenkonzept eingegangen.

Tabelle 1: Bedüsungseinrichtungen

	Einrichtungen zur Bedüsung	Medium	Schneidkopf	Meißelhalter	Düsen-Ø in mm Düsenform 3)	Volumenstrom je Düse und Halter in l/min Herstellertoleranz in %	Wasserdruck am Halter bzw. Düsenträger ggf. Luftdruck in bar	Überwachung Druck	Überwachung Volumenstrom
Wasser-Schneidspurbedüsung	Einzelmeißel-steuerung	Wasser	Quer-Längs-	System Voest-Alpine	0,8 V oder 0,8/10-12° K	3,2 ±10	120	ja	1)
	Sektorsteuerung ≧180°	Wasser	Quer-	System Krummenauer R. 15.01	0,7/20 ° K	1,7 ± 5	120	ja	ja
	intermittierende Steuerung ≧180°	Wasser	Quer-	System Voest-Alpine	0,7/10-12 ° K	Bei Dauerbetrieb: 3,2 ±10. Bei einer Pulsfrequenz von 36 - 63 Hz: 1,2 ±10	120	ja	ja
Luft-Wasser-Schneidspur-bedüsung	Voll-Bedüsung	Luft und Wasser	Längs-	System Paurat (Boart)	5 (Luft) 0,8 (Wasser)	250 ±5 0,5 ±5	rd. 2,5 rd. 2,5 Zuleitung Maschinen-eingang	ja ja	ja ja
Fächerdruck und Schwert-bedüsung	Düsen am Schneid-arm	Wasser	Quer-	frei	0,4 - 0,6 V 0,4 - 0,6 V	Düsenzahl: 26 - 38; 2 x 9 2) — Volumenstrom ges. l/min: 51 - 65; 2)	150	ja	ja

1) z. Z. technisch noch nicht möglich. 2) Anzahl und gesamter Volumenstrom je nach Schneidkopfbreite (1600 - 2100 mm)
3) V = Vollstrahldüse, K = Kegelstrahldüse

Bild 1 Aufteilung der Grubengaszündungen nach der Art der verursachenden Maschine



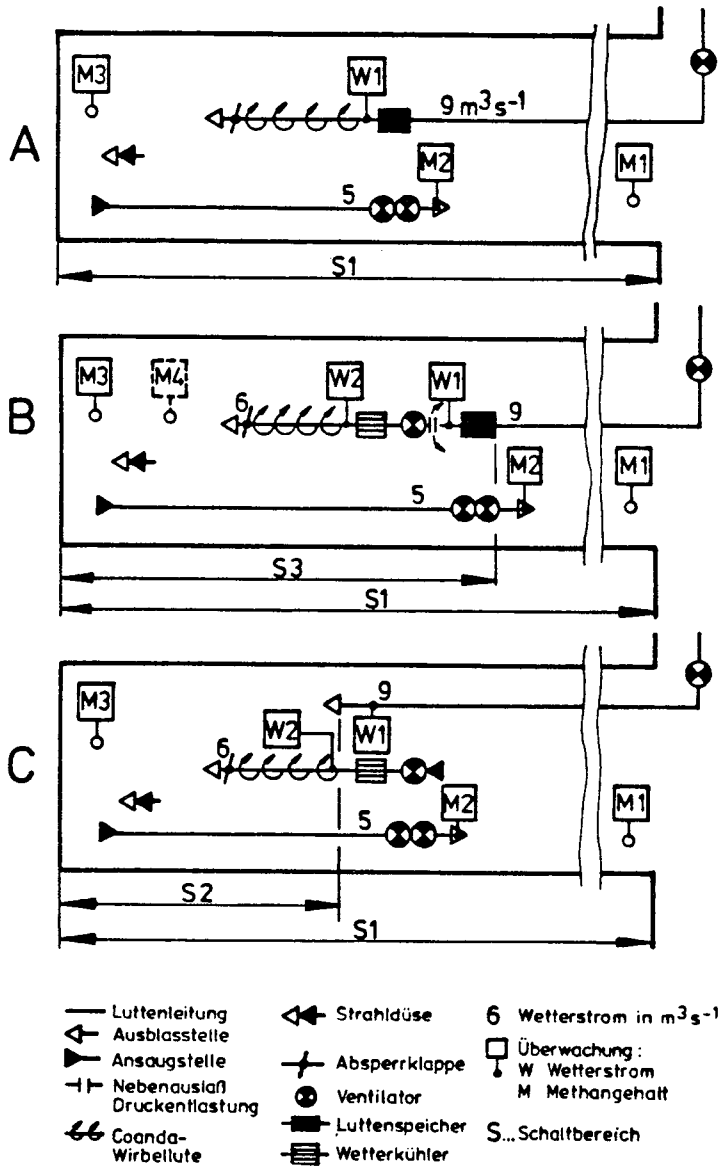


Bild 2 Bewetterungssysteme

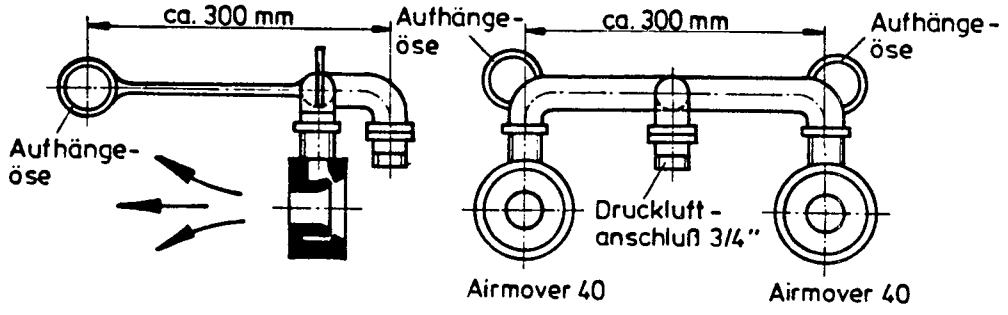


Bild 3 Doppelstrahldüse

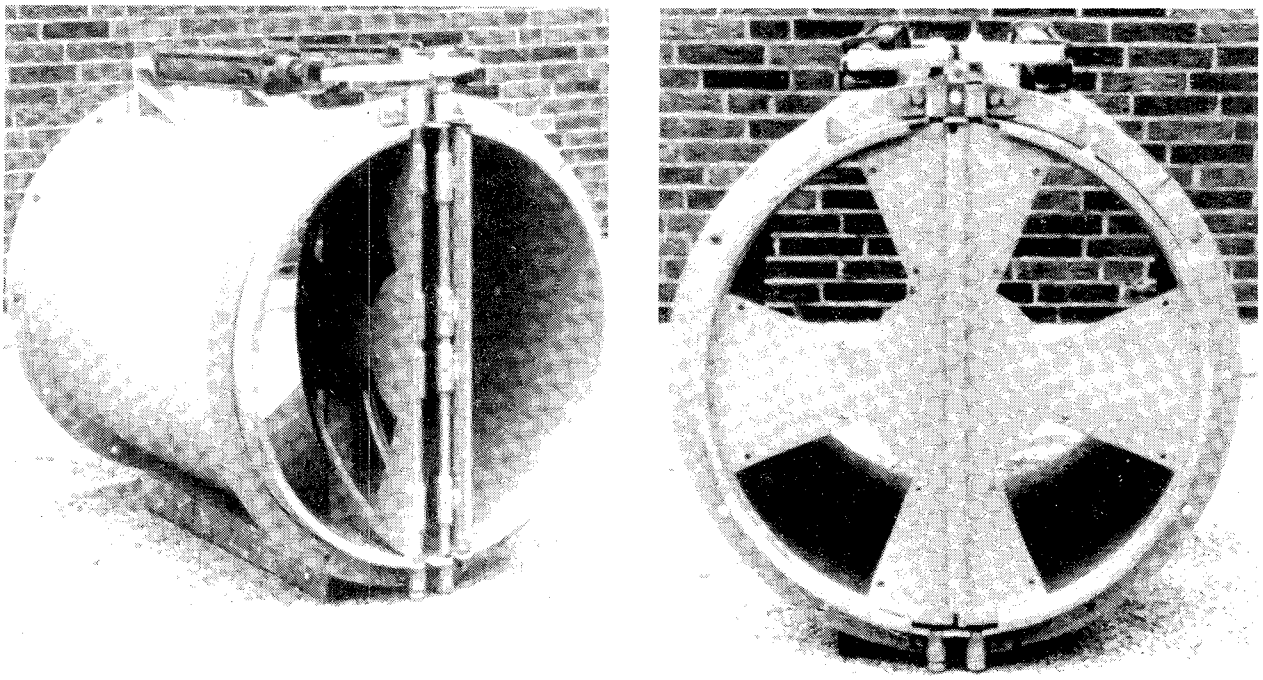


Bild 4 Strahlmanipulator

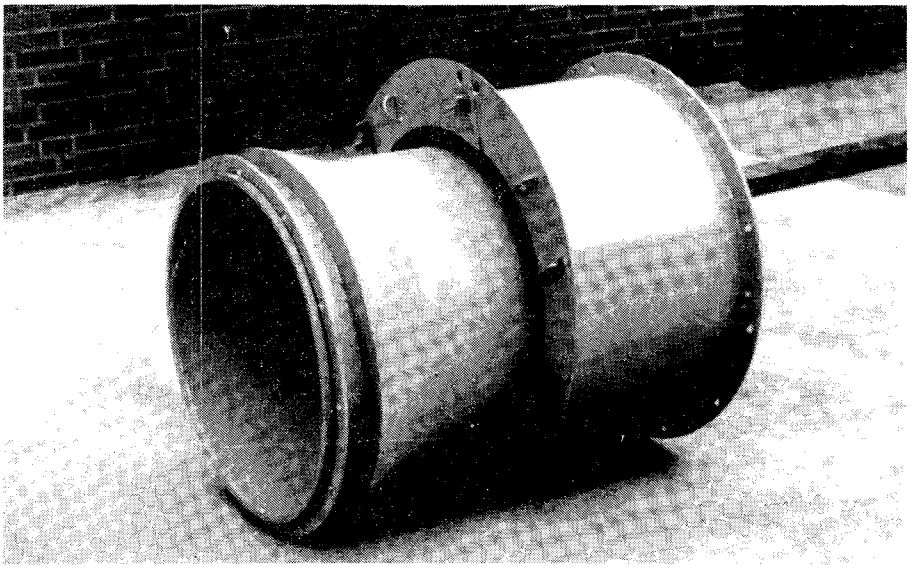


Bild 5 Sekundäraustrag

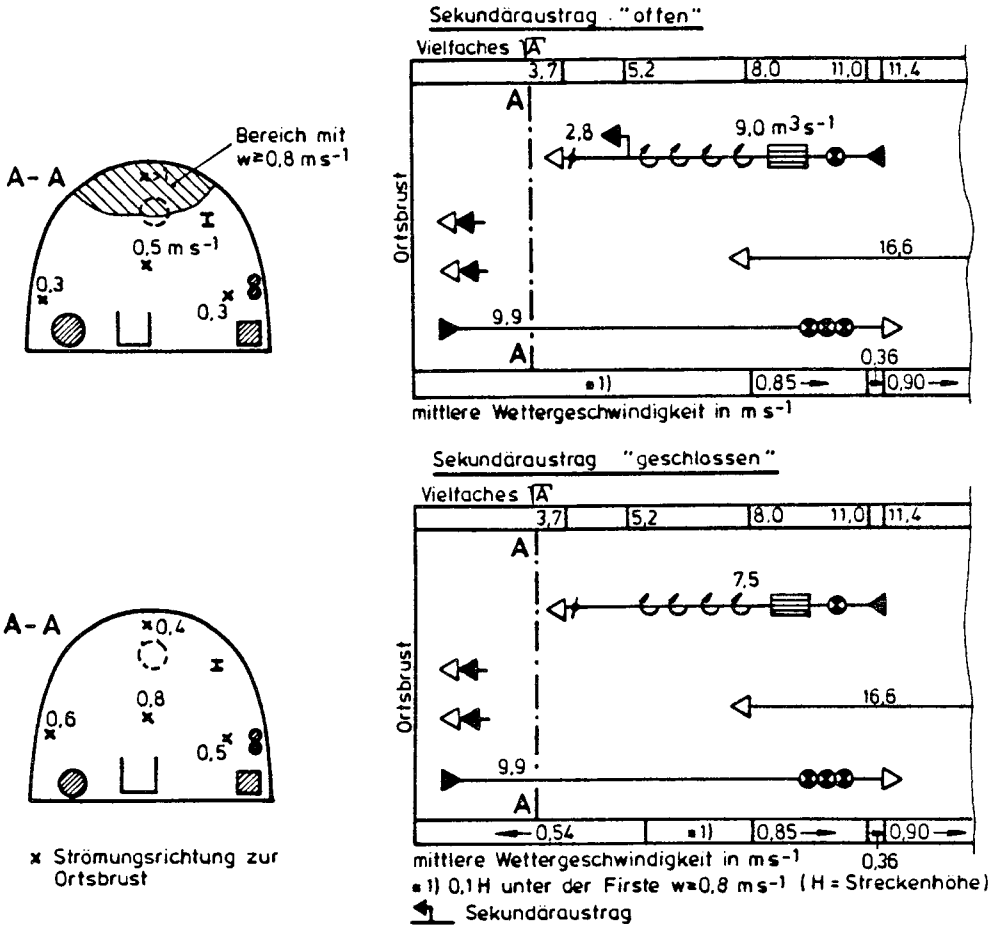


Bild 6 Sekundäraustrag in paralleler Nebenluttenleitung

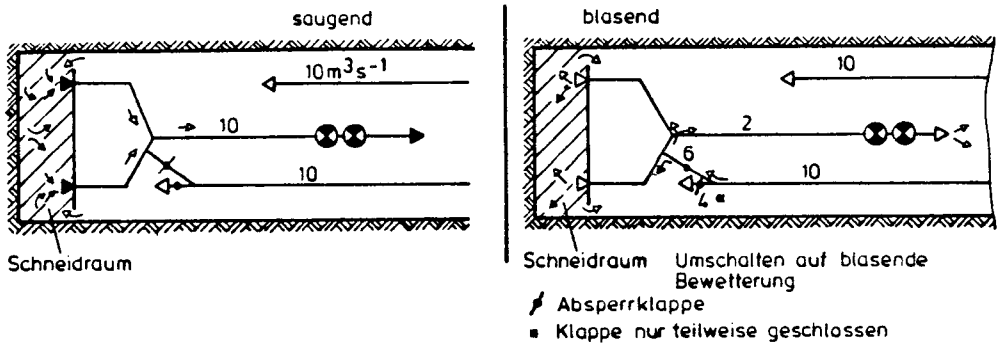


Bild 7 Schneidraumbewetterung

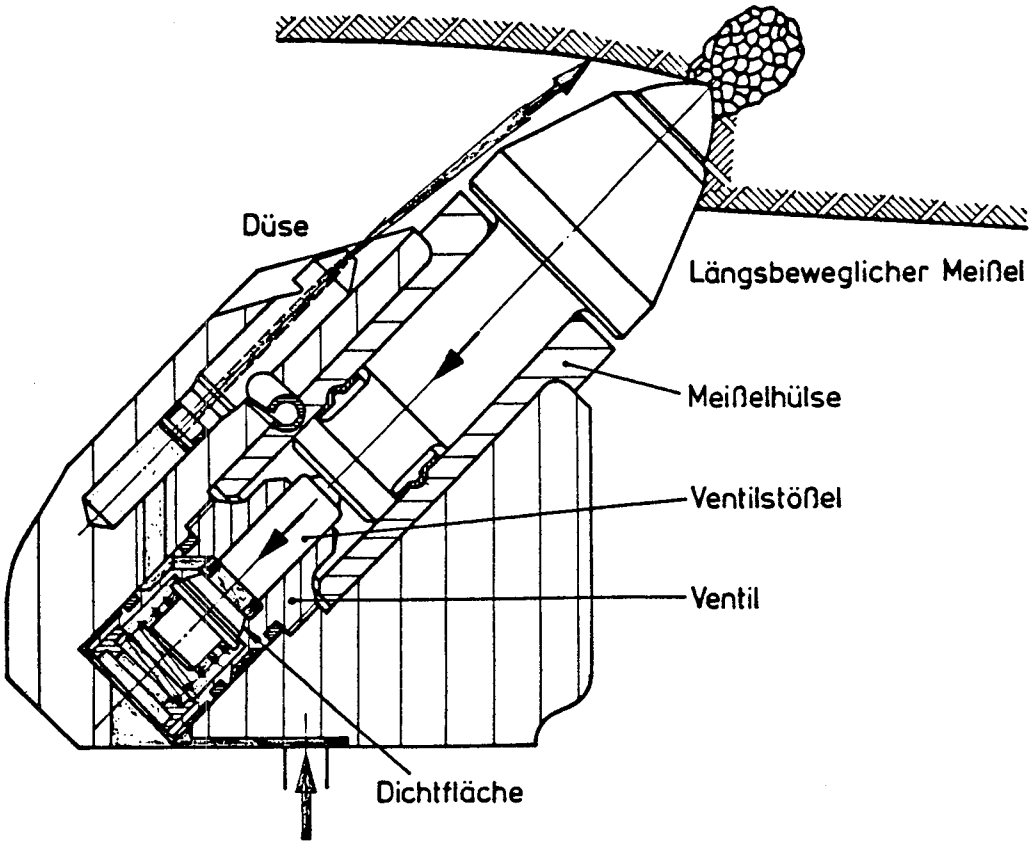


Bild 8 Prinzip der einzelmeißelgesteuerten Schneidspurbedüsung der Voest-Alpine AG

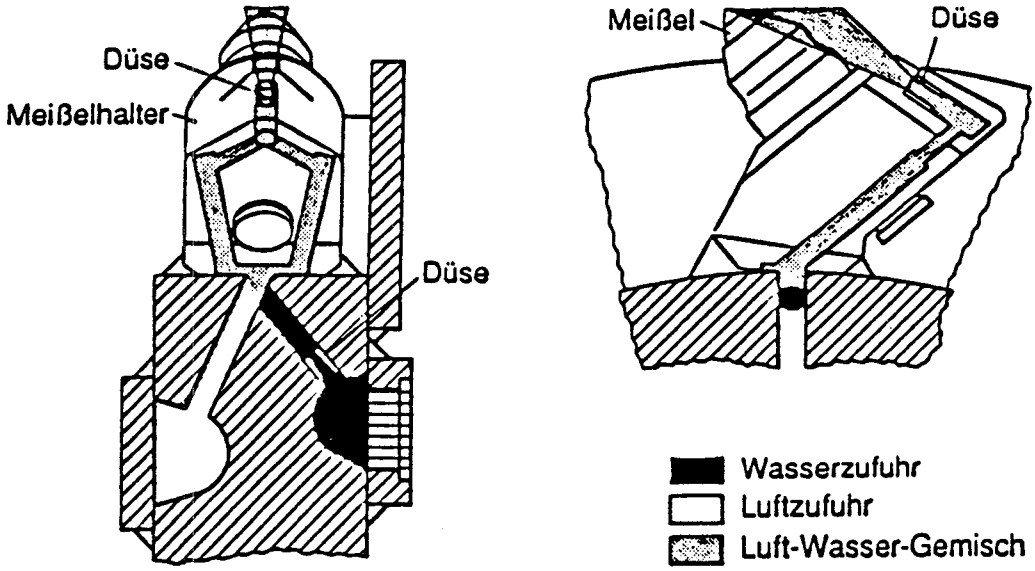


Bild 9 Düse für die Luft-Wasser-Schneidspurbedüsung der Paurat GmbH

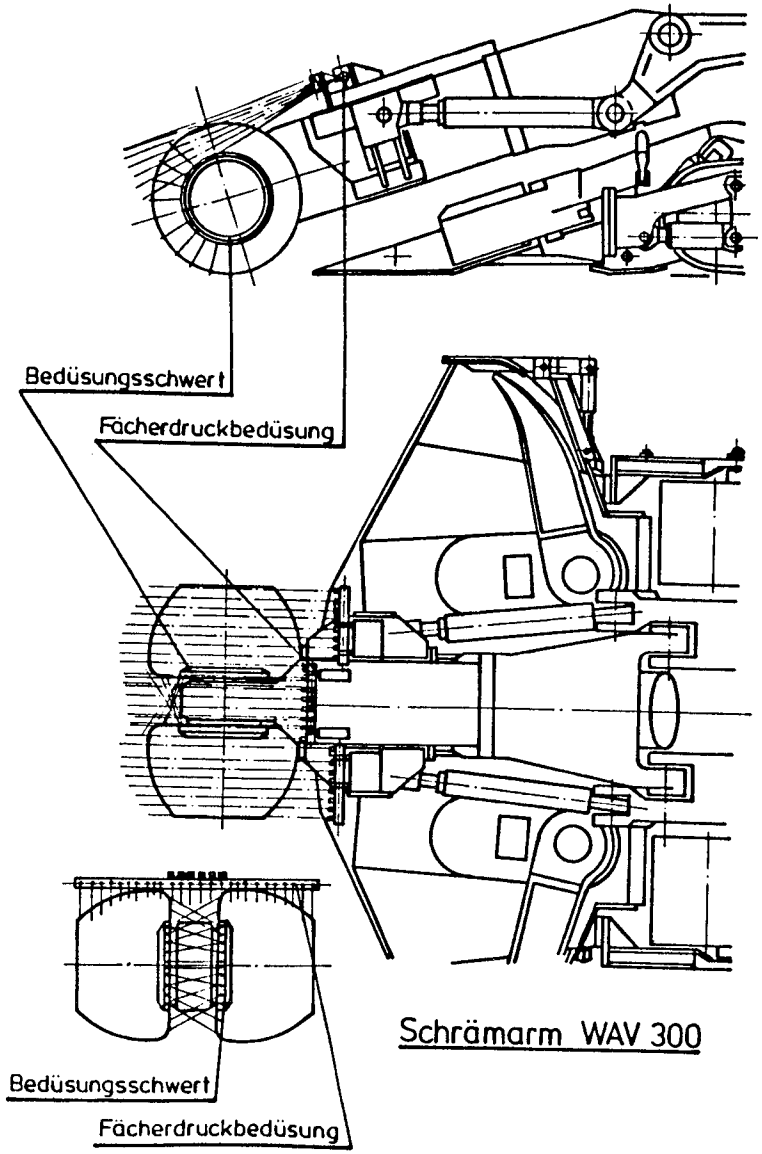
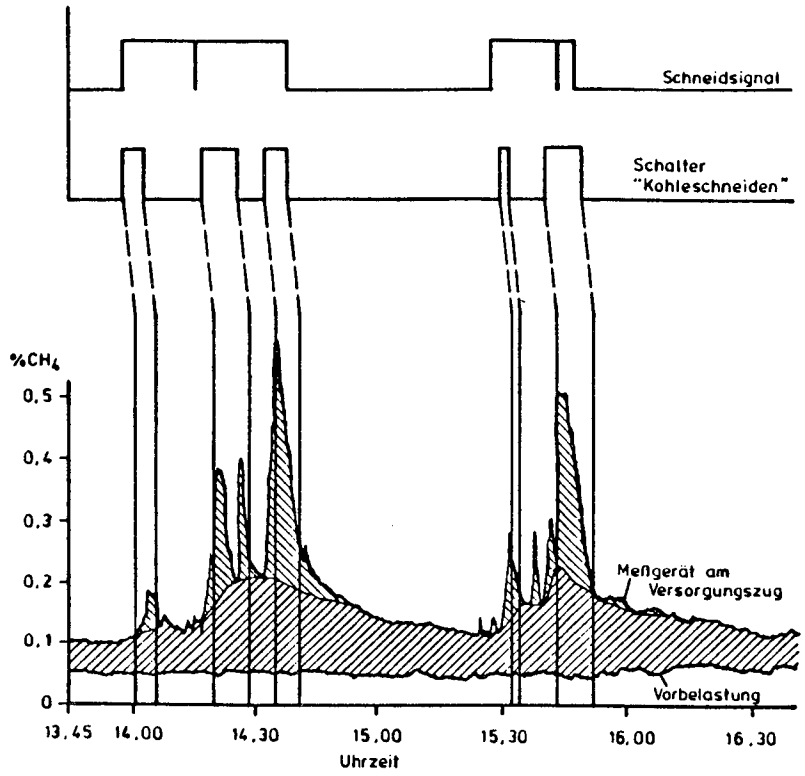


Bild 10 Fächerdruck- und Schwertbedüsung



	Methanmenge:	Methanzustrom:	
Summe Kohle	64,51 m^3	0,379 $m^3 min^{-1}$	(28 %)
Summe Stoße	165,33 m^3	0,973 $m^3 min^{-1}$	(72 %)
Gesamt:	229,84 m^3	1,352 $m^3 min^{-1}$	(100 %)

Bild 11 Auswertung von Konzentratrationsverläufen während des Schneidvorganges der Teilschnittmaschine

DERNIERS DEVELOPPEMENTS DES METHODES D'EVALUATION DES TECHNIQUES DE VENTILATION, DES SYSTEMES DE VAPORISATION D'EAU ET DES EMISSIONS GAZEUSES DANS LES AVANCEES

K. Noack, G. Hinderfeld, W.E. Marx W. Marth

Les inflammations de méthane sont causées principalement par les machines d'avancement de voies et de percement de tunnels. L'optimisation des systèmes connus de ventilation et d'arrosage d'eau est donc tout aussi importante que l'amélioration des techniques de surveillance et d'ouverture de circuits.

Même les méthodes de ventilation et d'arrosage les plus sophistiquées peuvent échouer si une émission soudaine de méthane se produit. Pour n'interdire l'utilisation de machines d'avancement de voies que dans les couches comportant des risques réels de dégagements soudains, on est en train de rechercher un coefficient de jugement approprié, sur la base de dispositifs de mesure déjà installés et l'on prévoit d'intégrer des techniques de forage de pointe dans les machines de percement des tunnels.

НОВЕЙШИЕ ДОСТИЖЕНИЯ В РАЗРАБОТКЕ ВЕНТИЛЯЦИОННОЙ ТЕХНИКИ, ВОДОРАСПЫЛИТЕЛЬНЫХ СИСТЕМ И МЕТОДОВ ОЦЕНКИ ВЫДЕЛЕНИЯ ГАЗА ПРИ МЕХАНИЗИРОВАННОЙ ПРОХОДКЕ

К. Ноак, Г. Хиндерфельд, В.-Е. Маркс и В. Марф

Воспламенение метана связано в основном с работой проходческих машин. Максимальное использование известных вентиляционных и водораспылительных систем так же важно, как и усовершенствование методов наблюдения и быстрой остановки работ. В случаях внезапной эмиссии метана даже самые современные вентиляционные и водораспылительные системы могут отказать. Для того, чтобы оценить, насколько целесообразен запрет на использование проходческой техники в повышенно взрывоопасных пластах, разрабатывается подходящий коэффициент, который будет использовать данные с уже существующих измерительных устройств. Одновременно планируется применение более прогрессивной проходческой технологии.

机器采矿井中通风技术，喷水系统和气体溢出鉴定方法的最新发展

R。诺阿克， G。兴德尔菲特， W。E。马克斯 及 W。马克斯

甲烷引燃常因开路及掘矿时发生。故优化通风及喷水系统与改进监测和开关技术是同等重要的。如果甲烷突然溢出，则最先进的通风和喷水系统也势将失效。为了禁止掘矿机用在有突出危害的煤层处，我们根据已安装好的测量设备制定了一个适当的鉴定系数，并计划将挖掘机与开路机并成一体。

COMPARATIVE INVESTIGATION INTO EXPLOSIBILITY OF BROWN COAL AND BITUMINOUS COAL DUST IN SURFACE AND UNDERGROUND TEST INSTALLATIONS

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ABSTRACT

The explosion behaviour of brown and bituminous coal (coking coal) are compared. WBK-Bergbau-Versuchsstrecke (BVS) has determined in a pressure-tight vessel the explosion characteristics with varying water content as well as the quantity of stone dust preventing explosion self-propagation. At Tremonia Experimental Mine the build-up behaviour of lignite dust explosions was investigated in a 8-m² gallery and compared with the results obtained with bituminous coal dust.

Taking into consideration the inert matter content and particle size distribution, lignite proved more explosive than coking coal. The high water content of lignite with approximately 45 % hides the higher explosibility which is revealed by drying of dust deposits or adding fine particles. When stone dusting as explosion protection means is applied, 80 % by weight is the minimum limiting value.

INTRODUCTION

The aim of the investigation was, whether in underground lignite mines the same explosion protection measures as in hard coal mines can successfully be applied. For this purpose the explosion behaviour of brown lignite was determined and some preliminary tests with regard to the preventive measures 'stone dusting and dust consolidation' were carried out.

For the determination of explosion characteristics the BVS 40-l-pressure-tight vessel was used, comparative tests with lignite and hard coal dust were carried out at Tremonia Experimental Mine in galleries with large cross sections.

SMALL-SCALE TESTS AT BVS

To be able to make general statements the samples to be studied should be as representative as possible. This proved difficult in the case of lignite. It was known from petrographic profile survey that the overall thickness of the seam to be investigated, as well as the thickness of characteristic seam strata varied considerably.

Preliminary tests not only established that specific seam strata differed from others with regard to explosion behaviour, but also that, for example, high-volatile, bituminous-rich matter are enriched in the underground dust. Because settled dust had to be investigated this had to be taken into consideration.

The natural dust quantities, collectable with justifiable effort, generally are not sufficient for large-scale tests. So, less representative average samples, taken from a raw-coal bunker of an underground mine, were used in the large-scale tests. Part of the samples was ground in a pin mill at BVS, another part in a rod mill at Tremonia Experimental Mine. The different analysis results

of the two sub-samples on ash i.wf and volatile matter i.waf (BVS, sample N° 123, Table 1, and VG, Table 5) confirmed the inhomogeneous structure of the seam.

The investigation of natural dust samples (unprocessed) taken in an underground lignite mine lead to two essential conclusions:

1. The deposited dust is, in most cases, much dryer than the unmined coal, which contains more than 40 % by weight water.

2. In many cases there is a vertical relationship in the gallery cross-section, such that dust in the upper part of the gallery contains water between 10 and 20 % by weight, whereas dust in the floor area has water contents between 30 and 40 % by weight.

To correctly assess the most easily reacting dust in the event of an explosion, samples should be taken from the roof area.

One dust quantity, sufficient for vessel tests (800 g), was taken in a gate road by clearing away dust from support caps and has subsequently been mixed. This kind of dust, which had been moved and at the time dried by the ventilation flow was rather fine and dry (sample N° 291, Table 1).
The third type of dust (bituminous) listed in Table 1 is a typical coal dust, which equals the Dortmund standard coking coal dust in explosion behaviour, cf. Table 5. The slightly higher inert matter content is compensated by the higher volatile matter content.

Table 1:

sample N°	median value	ash i.wf	water	volatile matter i. waf
	µm	%	%	%
123	26	22.1	29.6	65.5
291	16	23.6	17.3	69.8
bitum.	18	5.5	0.5	26.0

TESTS IN THE 40-L-PRESSURE-TIGHT VESSEL

The almost spherical 40-l-pressure-tight vessel has fully been described in literature (Reeh 1979). At the beginning of the test a defined quantity of dust is filled into a small dust reservoir and pressurized with 0.9 MPa compressed air. After opening of a magnetic valve the dust is blown into the vessel via a ball shaped nozzle and is ignited by a chemical igniter 170 ms after distribution.

The pressure curve of the combustion reaction is recorded. The record contains the maximum explosion pressure (p_{max}), the maximum rate of pressure rise (dt/dp), and the reaction time (Δt) from the moment of ignition to maximum pressure. From these three values an explosion characteristic (kex) is formed:

$$kex = \sqrt{\frac{p_{max}}{\Delta t} \cdot \left(\frac{dp}{dt}\right)_{max}} \quad MPa/s \qquad (1)$$

It is a measure of the specific energy of the combustion energy and allows the comparison of the explosion behaviour of different dusts.

RESULTS

Figure 1 shows the explosion characteristics of different dust-air concentrations determined in three tests each. All three samples are explosive over a wide concentration range. The lower explosion limit of the natural lignite sample N° 291 and the bituminous coal dust is about 70 g/m³.

The lowest characteristics were obtained with ground lignite. The coarse granulation and 52 % by weight of inert matter did not allow for higher values.

Values obtained with the natural lignite sample N° 291 demonstrate that lignite is able to react more violently. This dust still contained more than 40 % by weight inert matter of which only 17.3 % water by weight (cf. Table 1).

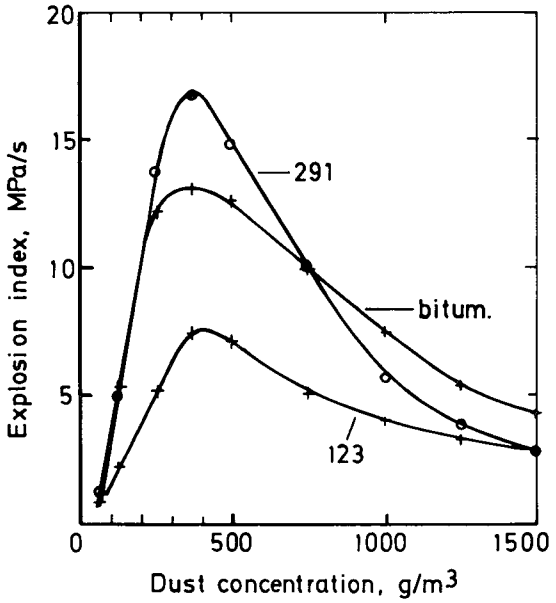


Figure 1.

EFFECT OF WATER CONTENT

Comparison of the two lignite dusts shows that the water content seems to have a great influence on explosion behaviour. Hence this parameter will be regarded closer.

Three sub-parts of a sample (N° 301) which had a water content of only 8.1 % after grinding, have been wetted again with water. So four samples were obtained different only in one parameter. With different dust concentrations, as shown in Figure 1, their explosion characteristics were determined as well as the maximum values in the curves obtained. These maximum values are listed in Table 2, related to the respective water contents.

Table 2:

sub-sample N°	ash i.wf %	water %	kex_{max} MPa/s
301/1	21.3	8.1	17.9
301/2	21.3	24.5	9.0
301/3	21.3	29.0	5.4
301/4	21.3	43.0	0.8

As it was expected, reactivity decreases as the water content increases. Figure 2 is a good representation of this relationship. The equalising straight line intersects the abscissa at 42.8 %, which means that "dust" with 42.8 % water content, corresponding to 64 % total inert matter, is victually no longer explosive.

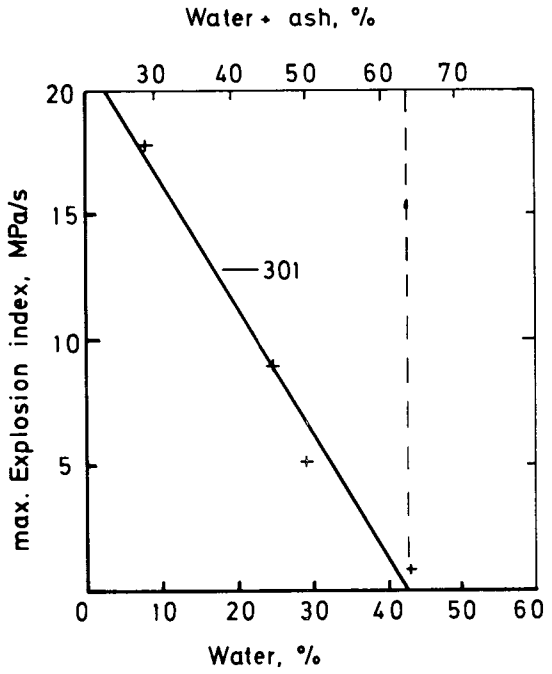


Figure 2.

The determined explosion characteristic of 0.8 MPa/s with 43 % water content (sub-sample N° 301/4)is partly to be attributed to the strong primer compound usually used in the 40-l-vessel. This chemical igniter releases an energy of 5 kJ.

Figure 3 shows the values obtained with a method similar to the one described above of a number of samples, prepared in drying and grinding processes and also the values of five mine-moist natural samples. Their inert matter content is listed in Table 3 (sample N° 291 has been added to the table).

Table 3:

sample N°	ash i.wf %	water %	median value µm
291	23.6	17.3	16
117	23.9	28.6	23
304	25.4	26.2	25
305	27.2	33.9	43
306	27.2	36.5	39

Figure 3 shows that the natural, unprocessed, samples reacted very differently in the explosion vessel, ranging from very dangerous to not dangerous. The drawn limiting curve is slightly steeper than the equalising line in Figure 2. It intersects the abscissa at approximately 65 % inert matter.

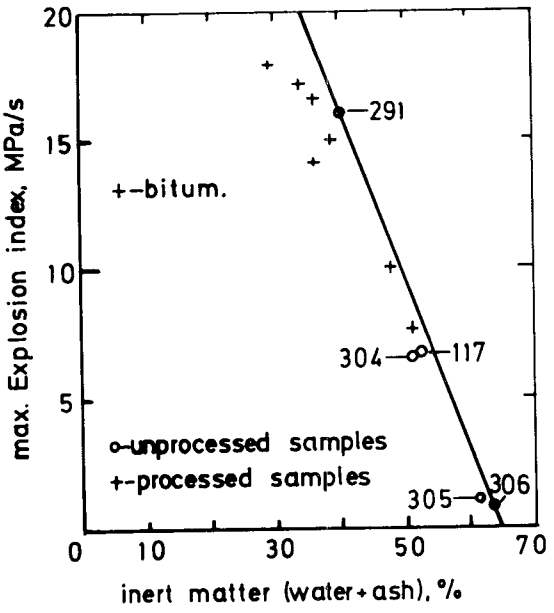


Figure 3.

The water contents of the two not dangerous samples were 33.9 % and 36.5 %. This is not extremely high, compared to sample 123 in Figure 1, which contained almost 30 % water but nevertheless yielded clearly higher explosion characteristics. But two natural samples yielded lower explosion values, also because of their coarse granulation together with ash contents of about 27 %.

The high values of sample 291 must be attributed to the fineness and - as already mentioned - to the low water content of 17.3 % which is not sufficient to essentially reduce blowing up and dispersability.

This means in practical operation underground that high explosion risk exist, when fine dust with median values below 20 µm and water contents below 20 % is present in sufficient quantity. Fineness and water content are influenced by dust binding in only a limited way. In cutting coal winning and conveying operations fine dust is produced which not only is dried and moved by the ventilation flow, but which also is deposited in the upper part of the roadway cross-sectional area, where it concentrates, other than the coarse dust. This also occurs during belt conveying.

PREVENTIVE EXPLOSION PROTECTION MEASURES

To reduce explosion risk, coal dust must either be neutralized or removed, the latter just being possible for a limited time and in a limited area. In general operation dust is neutralised by binding it with hygroscopic salt or by stone dusting.

Tremonia Experimental Mine has investigated the applicability of dust binding measures to brown lignite dust, according to the "Test regulations for dust binding substances and admixtures". In the tests $CaCl_2$ solution, scales and powder were used. Laboratory tests with different relative humidity established however that with none of the dust bindung means applied in underground coal mining satisfactory results were obtained.

This means that stone dusting is the only appropriate preventive explosion protection measure for use in the brown lignite dust.

EFFECT OF STONE DUST ADMIXTURE

It has been investigated whether 80 % by weight inert matter in deposited dust, required in German hard coal mining, is valid also in brown lignite mining or whether this value had not to be reached.

For worst case consideration a lignite dust with water and ash content as low as 9.8 % and 11 % was tested (sample 244, median value 23 µm). Sub-samples were produced by admixing 40, 60, 70 and 80 % stone dust to the original dust. So, the total inert matter contents in the mixtures were calculated by

$$I_{ges} = (1-G/100) \times (A+G)/G \qquad (2)$$

or, equation transformed, by

$$G = (I_{ges}-A-W) \; / \; (1 - \frac{A+W}{100}) \qquad (3)$$

where

Iges- total inert matter % by weight
G - stone dust admixture % by weight
A - ash content of coal % by weight
W - water content of coal % by weight

The values calculated for sample 244 are listed in Table 4 (for comparison the respective values of sample 123, Table 1 are added in Table 4).

The explosion characteristics, obtained with the five sub-samples N° 244 are given in Figure 4. Figure 5 gives the maximum values of the curves, also a linearly decreasing relationship at total inert matter contents between 20 and approximately 80 % and a stronger decrease only above this value. With 85 % total inert matter (equal to 81.1 % total stone dust admixture - according to equation 3) the right curve intersected the abscissa axis.

To also determine the limit stone dust admixture to ash and water rich lignite dust, sample 123 has been processed and analysed like sample

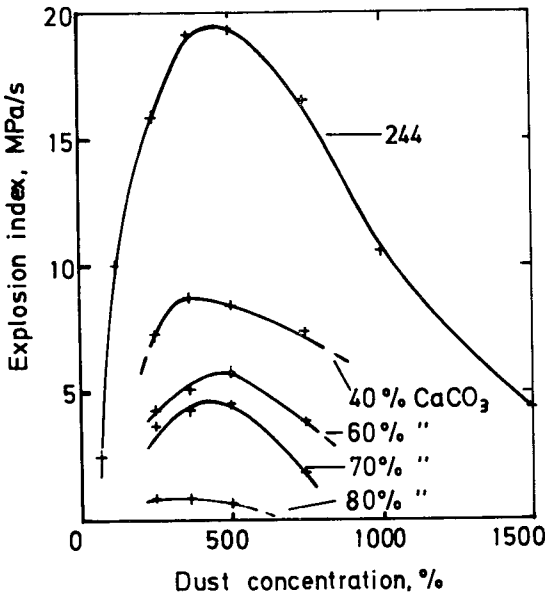


Figure 4.

244. The left curve in Figure 5 gives the results. Despite the high inert matter content of 51.7 % in the original sample, 60 % stone dust were still not sufficient to prevent explosibility. Only with a total inert

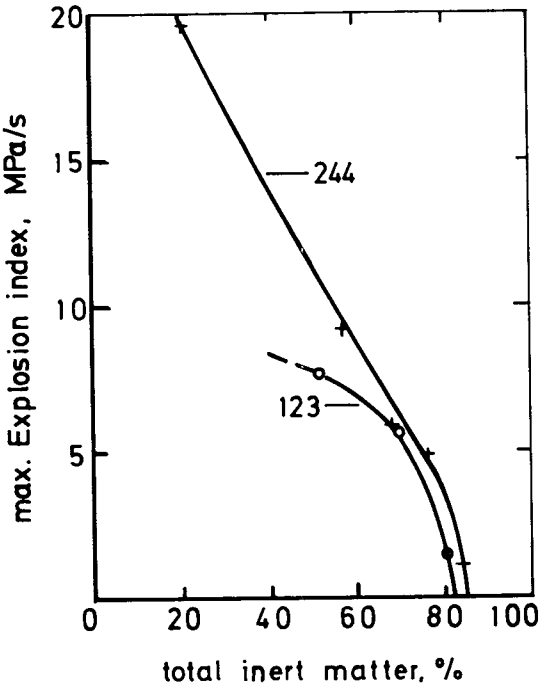


Figure 5.

matter content of 81 %, corresponding to 62.7 % stone dust admixture, the abscissa axis is intersected.

Table 4:

stone dust admixture % b.w.	total inert matter in sample N° 244 %	N° 123 %
0	20.8	51.7
40	57.0	1.0
60	68.3	0.7
70	76.2	-
80	84.2	-

The results of these vessel tests establish that in the case of low ash and dry lignite dust more than 80 % stone dust may be neccessary to prevent explosibility. On the other hand, natural lignite samples generally contain more than 35 % by weight inert matter (this orienting value has been defined as a limit value in the course of the investigations; this does not exclude, however, that it can be lower in individual cases).

To render such a natural dust with 35 % inert matter non explosive, a stone dust portion of according to equation 3) 77 % by weight is neccessary.

TESTS IN A 200 M SURFACE GALLERY

BVS disposes of a 200 m, 2.5 m² surface gallery to verify the vessel results. Some preliminary tests with lignite dust N° 123 (Table 1) have been carried out with varied initiating strength and length of dust zone (40 to 90 m) with a dust concentration of 200 g/m³.

As was expected from the vessel tests, in all cases the explosion was self propagating. The flame length was more than twice the dust zone.

Compared to similar coking coal dust explosions (dusts corresponding to Table 1), lignite dust yielded slightly higher values. The mean flame velocities were up to 30 % higher, whereas the maximum explosion pressure were approximately identical.

TESTS AT TREMONIA EXPERIMENTAL MINE

In surface and underground tests with coal dust, Tremonia Experimental Mine uses the so-called "Dortmunder Normalstaub" (Dortmund standard dust), a bituminous coal dust (coking coal). The most important characteristics of this dust are listed in Table 5. In numerous explosion tests data were gathered about the build-up of bituminous coal dust explosions (Michelis 1986). Additional tests with brown lignite in a view of preventive and constructive explosion protection measures were carried out in 1988 at Tremonia Experimental Mine. The characteristics of the lignite tested are also given in Table 5.

UNDERGROUND EXPLOSION TESTS

Tremonia Experimental Mine disposes of underground explosion galleries with a total length of 1700 m and cross-sections, ranging between 8 and 20 m². The tests described here were carried out in a 700 m gallery with 8 m² cross-section (Figure 6). This cross-section corresponds to the one gallery encountered in West German underground lignite mines.

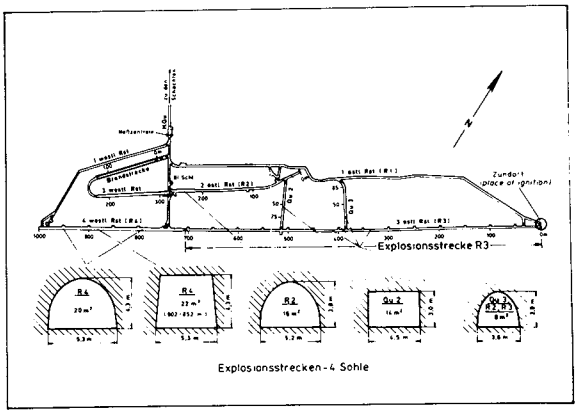


Figure 6.

Table 5:

Composition (mean values)		bituminous coal (Dortmund standard dust)	lignite coal (brown coal from underground)
water content	%	0.5	8.7
ash content	%	3.5	20.5
volatile matter (wf)	%	23.0	47.2
volatile matter (waf)	%	24.0	59.5
median value	µm	18 - 22	18 - 22
particle size distribution ± 50 µm	approx. %	95	90

INFLUENCE OF THE INITIATOR

It was assumed that in the tests the used ignition sources were strong enough to ignite coal dust and that the ignition process was reproducible. To initiate explosions methane as well as explosives were used.

Coal dust explosions can be ignited by explosives either rock explosives or black powder. However, the mere detonation of explosives does not necessarily lead to a coal dust explosion, the presence of a specific quantity of pre-dispersed coal dust is necessary. When this coal dust - air mixture is ignited by the explosive initiator, a self-propagating coal dust explosion starts.

The standard initiator used by Tremonia Experimental Mine for starting coal dust explosions, consists of 10 kg pre-dispersed coal dust (Dortmunder Normalstaub) and 2.5 kg black powder (Table 6 N° 7). In comparative tests also a gelatinous and a powder-form rock explosive were used.

The most frequent initiator of coal dust explosions in coal mines are methane accumulations (Michelis 1986). The effects of different initiators with regard to maximum static pressure and dynamic pressure in the 8 m² explosion gallery are listed in Table 6. The values of static pressure were generally recorded at the roadhead and dynamic pressures 50 m from there. The arrangement of the blasting mortars and the HRD-bottles, from which the coal dust is pre-dispersed with a pressure of 6 MPa, are shown in Figure 7.

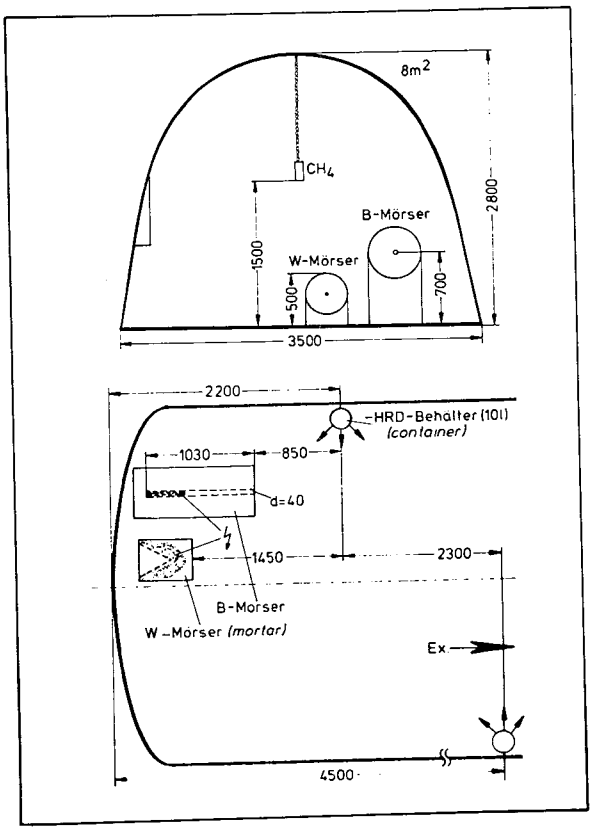


Figure 7.

Table 6:

N°	type of initiator	max. pressure stat. kPa	max. pressure dyn. kPa	released theoretically energy MJ
	A. explosive initiator			
	black powder			
1.	1250 g	3.9	-	0.345
2.	2500 g	6.0	-	0.685
	gelatinous explosive			
3.	1 cartridge = 250 g	32.3	2.9	0.275
	powderform explosive			
4.	1 cartridge = 240 g	29.6	1.9	0.240
5.	2 cartridges = 480 g	42.3	3.5	0.480
	B. combined initiator:			
6.	1250 g black powder + 10 kg hard coal dust	28.6	6.2	300
7.	2500 g black powder + 10 kg hard coal dust	37.5	8.5	300
8.	240 g powderform explosive + 5 kg hard coal dust	24.3	1.8	150
9.	240 g powderform explosive + 10 kg hard coal dust	28.6	5.8	300
10.	240 g powderform explosive + 5 kg lignite dust	35.0	2.3	60
11.	240 g powderform explosive + 10 kg lignite dust	30.0	5.3	120
12.	4.5 m^3 methane (10 % by vol.)	49.0	14.5	162
13.	3.5 m^3 methane (8 % by vol.)	26.5	4.0	126

In the 8 m^2 explosion gallery a dynamic pressure of 8 kPa, generated by the initiator, is required to start a reproducible propagating coal dust explosion. Therefore, the initiators N° 7 and N° 12 in Table 6 are normally used. Although smaller quantities of methane and explosive in combination with pre-dispersed coal dust are able to ignite coal

Table 7:

Test arrangement 1. coal dust quantity of the initiator (kg) 2. fuel zone (m) 3. specific fuel quantity (g/m³)	lignite° max. pres. stat. kPa	dyn. kPa	flame lenght m	max. flame velocity m/s	bituminous coal max. pres. stat. kPa	dyn. kPa	flame length m	max. flame velocity m/s
1. 2. 3.								
5 / 20 / 125	29.4	4.1	56	56	-	-	-	-
5 / 40 / 125	31.4	6.8	81	86	92.5	52	140	215
					(pre-dispersed coal dust)			
10 / 20 / 125	44.8	12.8	56	110	-	-	-	-
10 / 20 / 300	73.7	57.2	94	175	33.0*	6.0	58	65
					56.0+	22.0	75	19
10 / 40 / 300	91.7	41.4	119	216	81.5*	22.0	125	146
					84.0+	37.5	120	209

o Initiator in all tests with lignite = 1 cartridge (240 g) powderform explosive

* Initiator with 1.25 kg black powder
\+ Initiator with 2.50 kg black powder

dust explosions, their build-up and course differs considerably, and they are hardly reproducible.

For the tests with lignite dust an initiator combination was searched that yielded comparable results of pressure build-up as bituminous coal dust. The results obtained with Numbers 9 and 11 in Table 6 demonstrate that with a combination of 240 g (one cartridge) of powderform explosive and 10 kg of pre-dispersed hard coal or lignite dust almost identical values were found. Consequently, in explosion tests with lignite dust, exclusively powderform explosive was used as initiator (cf. Table 6 N° 11).

OTHER FACTORS INFLUENCE THE BUILD-UP OF COAL DUST EXPLOSIONS

Besides the type of initiator, other important factors that influence the build-up of coal dust explosions are:

- length of fuel zone
- specific fuel concentration
- type of fuel offer
- composition of chemo-physical structure of the fuel.

In the comparative tests, the length of fuel zone was 20 and 40 m and the fuel concentration 125 g and 300 g/m³ volume of the gallery. Coal dust was dispersed pneumatically in the whole cross-section. It had completely been settled at the moment of ignition so it had to be raised by the dynamic pressure. Bituminous coal dust and lignite dust were used as fuel, shown in Table 5.

RESULTS

In Table 7 the test results, regarding maximum explosion pressure (static and dynamic), flame length, and flame velocity, obtained in the

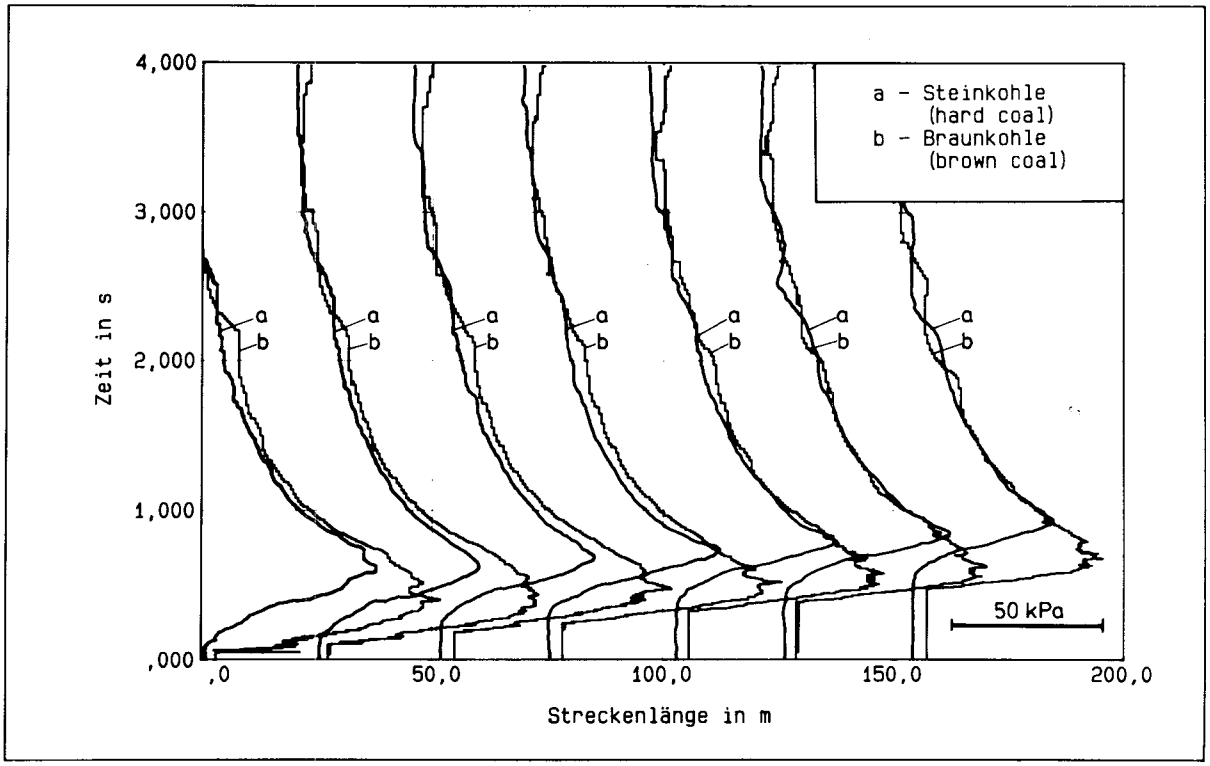


Figure 8.

respective test arrangements, are listed.

In all tests bituminous coal dust had a weaker pressure development than lignite. This can also be seen from Figures 8 and 9. Figure 8 is a representation of the pressure-time-curve of static pressure obtained with a 20 m long fuel zone and Figure 9 with a 40 m fuel zone. Figures 8 and 9 show the spontaneous ignition behaviour and the higher values of pressure rise $\Delta p/\Delta t$ of lignite. The mean rate of pressure rise was related to the measuring results obtained at 100 m as shown in Figures 8 and 9, about 176 kPa/s for bituminous coal dust and about 340 kPa/s for lignite dust.

RECOMMENDATIONS FOR CONSTRUCTIVE EXPLOSION PROTECTION

Because the lignite dust tested is similar in explosion behaviour to bituminous coal dust, the same protective measures as in bituminous coal mines against already build-up explosions apply. Constructive explosion protection in the form of water-trough explosion barriers can either be installed in concentrated or distributed form. The latter should be prefered, because in the case of ignition the effects of explosion are more limited (Figure 10).

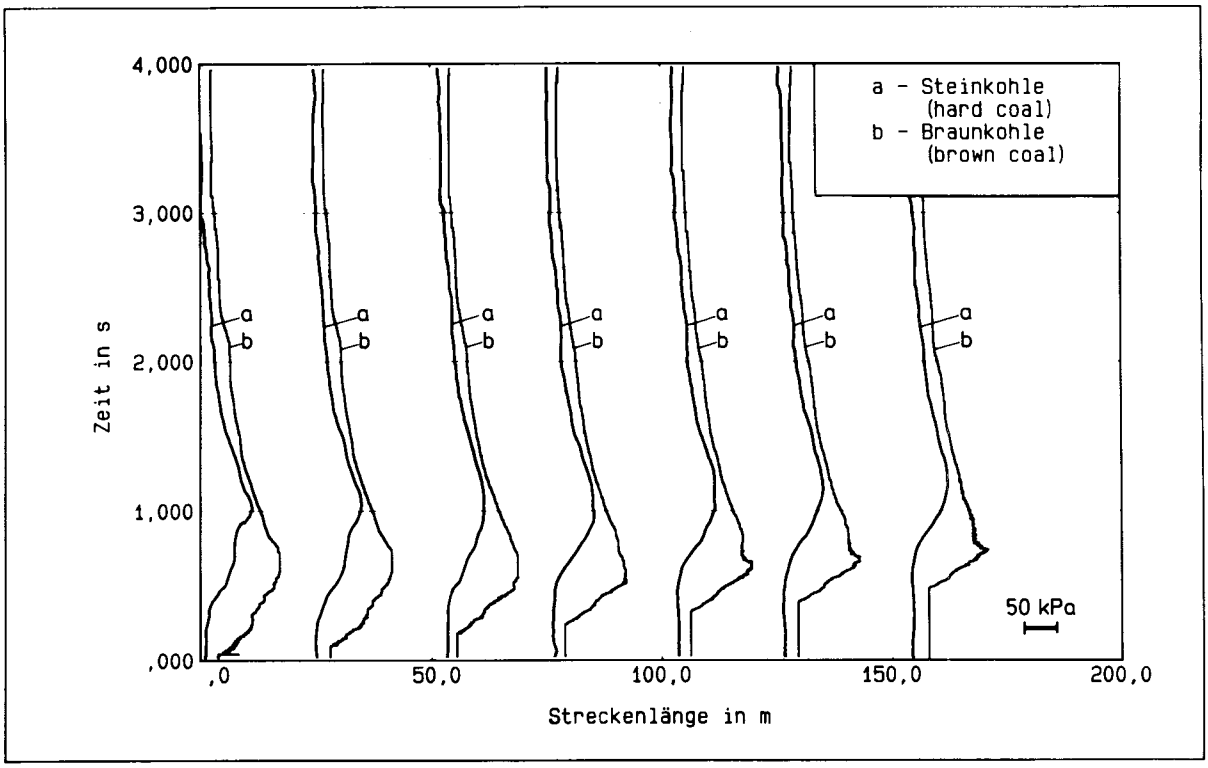


Figure 9.

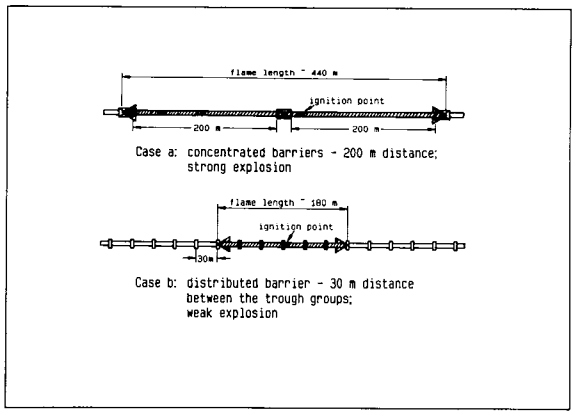


Figure 10.

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VERGLEICHENDE UNTERSUCHUNGEN VON EXPLOSIONSFÄHIGEM BRAUN UND STEINKOHLENSTAUB IN ÜBER UND UNTERTÄGIGEN VERSUCHEN

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ZUSAMMENFASSUNG

Weichbraunkohle und Steinkohle (Fettkohle) wurden explosionstechnisch verglichen. Von der WBK-Bergbau-Versuchsstrecke (BVS) wurden in einem Druckgefäß bei Variation des Wassergehaltes Explosionskennziffern sowie die Gesteinstaubmenge ermittelt, die bei Zugabe zur Braunkohle eine selbständige Explosionsausbreitung verhindert. Mit Untertageversuchen bei der Versuchsgrube Tremonia (VG) wurden in einer 8-m²-Strecke das Anlaufverhalten von Explosionen mit Braunkohlenstaub ermittelt und den Ergebnissen mit Steinkohlenstaub gegenübergestellt.

Bei Berücksichtigung des Inertgehalts und Kornaufbaus erwies sich die Braunkohle als explosionsgefährlicher als die Fettkohle. Der hohe natürliche Wassergehalt der Braunkohle mit rd. 45 % verdeckt diese größere Gefährlichkeit; nach Austrocknen von Staubablagerungen oder nach Anreicherung feinkörniger Anteile wird sie explosionsgefährlich. Für den präventiven Explosionsschutz dürfen bei Anwendung des Gesteinstaubverfahrens 80 Gew.-% im abgelagerten Staub nicht unterschritten werden.

EINLEITUNG

Um zu prüfen, ob in Braunkohlen-Tiefbauen dieselben Explosionsschutzmaßnahmen wie in Steinkohlengruben sinnvoll und anwendbar sind, sollten neben Versuchen zur Feststellung des allgemeinen Explosionsverhaltens von Weichbraunkohle einige orientierende Versuche, insbesondere im Hinblick auf die präventiven Maßnahmen "Gesteinstaubstreuung und Staubbindung" durchgeführt werden.

Zur Kenngrößenermittlung wurde dazu bei der WBK-Bergbau-Versuchsstrecke ein 40-l-Druckgefäß verwendet, während vergleichende Explosionsversuche mit Braun- und Steinkohlenstaub in Strecken mit großen Querschnitten auf der Versuchsgrube Tremonia stattfanden.

KLEINVERSUCHE BEI DER BVS

Um allgemeingültige Aussagen machen zu können, müssen möglichst repräsentative Proben untersucht werden. Das erwies sich im Falle der Braunkohle als schwierig. Aus der petrographischen Profilaufnahme war bekannt, daß sowohl die Gesamtmächtigkeit des zu untersuchenden Flözes als auch die Mächtigkeit markanter Flözstreifen in weiten Grenzen schwanken.

Vorversuche zeigten dann aber, daß sich bestimmte Flözstreifen nicht nur explosionstechnisch von anderen unterscheiden, sondern daß sich z.B. höher-flüchtige bitumenreiche Bestandteile auch im untertägigen Staub anreichern. Da aber letztlich der abgelagerte Staub zu beurteilen ist, mußte er entsprechend berücksichtigt werden.

Allerdings ist die mit vertretbarem Aufwand gewinnbare Naturstaubmenge für Großversuche i.a. zu gering. Daher konnte für die geplanten Großversuche auf eine (weniger repräsenta-

tive) große Durchschnittsprobe nicht verzichtet werden. Sie stammt aus dem Rohkohlenbunker einer Tiefbaugrube. Ein Teil dieser Probe wurde bei der WBK-Bergbau-Versuchsstrecke in einer Stiftmühle gemahlen, ein Teil bei der Versuchsgrube Tremonia in einer Stabschwingmühle.

Die unterschiedlichen Analysenergebnisse für Asche und Flüchtige Bestandteile i.waf bei den beiden Teilproben (BVS, Probe Nr. 123, Tabelle 1, und VG, Tabelle 5) bestätigten die Aussagen über den inhomogenen Flözaufbau.

Die in Strecken von Braunkohlen-Tiefbaugruben entnommenen natürlichen Staubproben (Naturproben) führten zu zwei wichtigen Erkenntnissen:

1. Der abgelagerte Staub ist meist deutlich trockener als die unverritzte Kohle, die über 40 Gew.-% Wasser enthält.
2. Im Streckenquerschnitt besteht häufig eine vertikale Abhängigkeit derart, daß Staub aus der oberen Streckenhälfte nur Wassergehalte zwischen 10 und 20 % aufweist, gegenüber dem Gut aus dem Sohlenbereich mit 30 bis 40 Gew.-%.

Um den Staub, der im Falle einer Explosion am leichtesten mitreagieren würde, richtig beurteilen zu können, müßte also Probegut aus dem Firstbereich von Strecken untersucht werden.

Eine für Gefäßversuche ausreichende Probe (800 g) fand sich in einer Förderstrecke, in der Staub von mehreren Ausbaukappen abgefegt und später gemischt wurde. Dieser Staub, der durch den Wetterstrom transportiert und dabei getrocknet worden war, erwies sich als relativ fein und trocken (Probe Nr. 291, Tabelle 1).

Beim dritten in Tabelle 1 aufgeführten Staub (bitum.) handelt es sich um einen typischen Fettkohlenstaub, der dem in Tabelle 5 aufgeführten Dortmunder Normalstaub explosions-technisch gleicht. Der etwas höhere Aschegehalt der Probe wird durch den höheren Gehalt an Flüchtigen Bestandteilen kompensiert.

Tabelle 1:

Probe Nr.	Median-wert µm	Asche i.wf %	Wasser %	Fl.Best. i.waf %
123	26	22,1	29,6	65,5
291	16	23,6	17,3	69,8
bitum.	18	5,5	0,5	26,0

VERSUCHE IM 40-L-DRUCKGEFÄSS

Das der Kugelform weitgehend angenäherte 40-l-Druckgefäß ist ausführlich in der Literatur beschrieben (Reeh, 1979). Zur Durchführung eines Explosionsversuchs in diesem Gefäß wird eine abgewogene Staubmenge in einen kleinen separaten Vorratsbehälter gegeben und mit Druckluft von 0,9 MPa beaufschlagt. Nach Öffnen eines Magnetventils wird dieser Staub durch eine Kugeldüse im Explosionsraum verteilt und etwa 170 ms nach Beginn der Aufwirbelung mit einem chemischen Zündsatz gezündet.

Der zeitliche Druckverlauf der Verbrennungsreaktion wird registriert, und der Aufzeichnung werden der maximale Explosionsdruck (p_{max}), der zeitliche maximale Druckanstieg $(dp/dt)_{max}$ und die Reaktionszeit (Δt) vom Beginn der Zündung bis zum Erreichen des Druckmaximums entnommen. Aus diesen drei Werten wird eine Explosionskennziffer (kex) gebildet:

$$kex = \sqrt{\frac{p_{max}}{\Delta t} \cdot \left(\frac{dp}{dt}\right)_{max}} \quad MPa/s \qquad (1)$$

Sie ist ein Maß für die spezifische Leistung der Verbrennungsreaktion und gestattet den Vergleich des Explosionsverhaltens verschiedener Stäube.

ERGEBNISSE

Die bei verschiedenen Staub/Luft-Konzentrationen in jeweils drei Versuchen ermittelten Explosionskennziffern sind in Bild 1 wiedergegeben. Alle drei Proben sind in einem weiten Konzentrationsbereich explosionsfähig. Die untere Explosionsgrenze

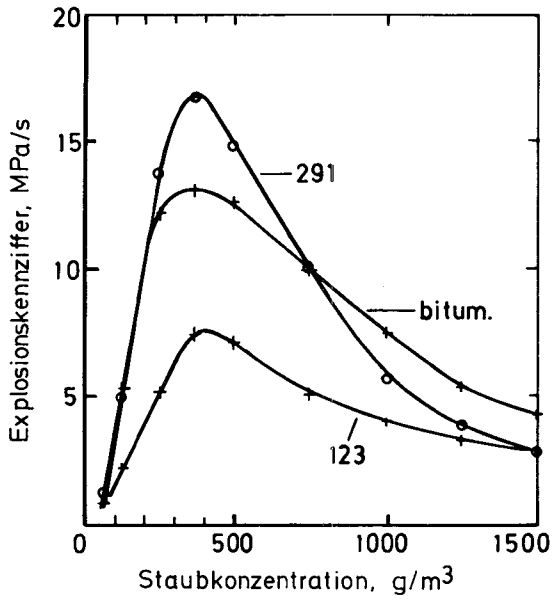


Bild 1.

der Naturprobe 291 sowie der Fettkohle liegt im Bereich von 70 g/m³.

Die gemahlene Braunkohle ergab die niedrigsten Kennziffern. Die grobe Körnung und fast 52 % unbrennbare Bestandteile verhinderten höhere Werte.

Daß Braunkohle aber auch wesentlich heftiger reagieren kann, zeigen die mit der Naturprobe 291 ermittelten Werte. Dieser Staub enthielt zwar auch noch über 40 % unbrennbare Bestandteile, davon aber nur 17,3 % Wasser (s. Tabelle 1).

EINFLUSS DES WASSERGEHALTES

Vergleicht man die Explosionskennziffern der beiden Braunkohlenstäube, scheint der Wassergehalt das Explosionsverhalten deutlich zu beeinflussen. Deshalb wird dieser Parameter im folgenden eingehender untersucht.

Von einer ausgesuchten Probe (Nr. 301), die nach der Mahlung einen Wassergehalt von nur 8,1 % aufwies, sind drei Teilmengen durch Wasserzugabe wieder befeuchtet worden. Dadurch entstanden vier Proben, die sich nur in dem einen zu untersuchenden Parameter unterschieden. Von ihnen wurden, wiederum bei verschiedenen Staubkonzentrationen, entsprechend Bild 1, die Explosionskennziffern ermittelt und aus den erhaltenen Kurven die Maximalwerte abgegriffen. Diese Maximalwerte sind in Tabelle 2 den Wassergehalten gegenübergestellt.

Tabelle 2:

Teil-probe	Asche i.wf %	Wasser %	kex_{max} MPa/s
301/1	21,3	8,1	17,9
301/2	21,3	24,5	9,0
301/3	21,3	29,0	5,4
301/4	21,3	43,0	0,8

Erwartungsgemäß nimmt das Reaktionsvermögen mit steigendem Wassergehalt stark ab. Bild 2 gibt die Verhältnisse deutlicher wieder. Die Ausgleichsgerade schneidet die Abszisse bei 42,8 %. D.h., ein "Staub" mit 42,8 % Wasser, entsprechend rund 64 % Gesamt-Inertanteil, ist praktisch nicht mehr explosionsfähig.

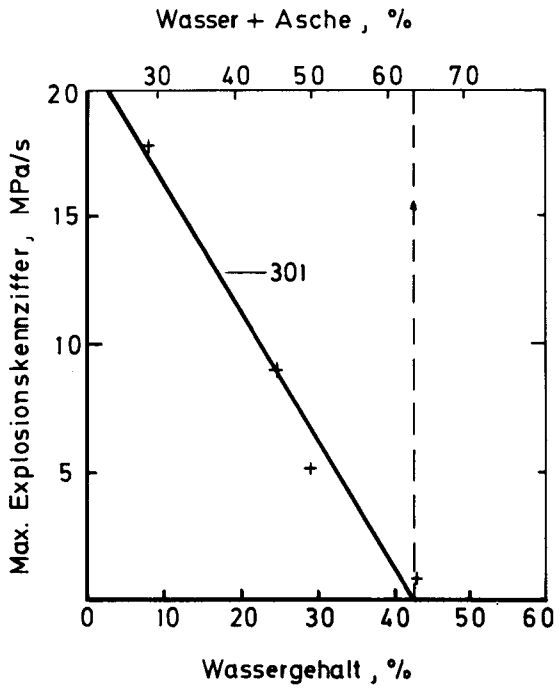


Bild 2.

Der für die Teilprobe 301/4 mit 43 % Wassergehalt ermittelte Explosionskennwert von 0,8 MPa/s ist zum Teil

durch den starken Zündsatz verursacht, der bei den Versuchen im 40-l-Gefäß üblicherweise verwendet wird; dieser chemische Zünder setzt eine Wärmeenergie von 5 kJ frei.

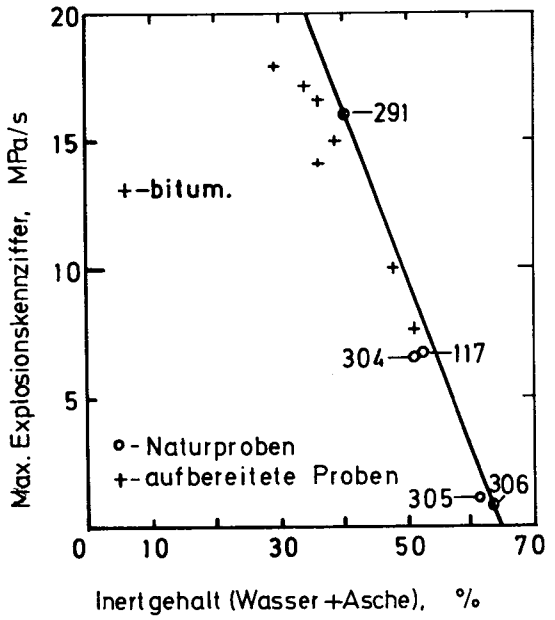


Bild 3.

In ähnlicher Weise wie in der vorhergehenden Darstellung sind in Bild 3 die Explosionskennziffern für eine Reihe weiterer Proben, die durch Aufbereitung (Trocknung und/oder Mahlung) hergestellt worden waren, sowie für fünf grubenfeuchte natürliche Staubproben eingetragen. Ihre Inertanteile gibt Tabelle 3 wieder (die Probe 291 ist erneut aufgeführt).

Tabelle 3:

Probe	Asche i.wf %	Wasser %	Medianwert µm
291	23,6	17,3	16
117	23,9	28,6	23
304	25,4	26,2	25
305	27,2	33,9	43
306	27,2	36,5	39

Wie Bild 3 ausweist, reagierten die Naturproben im Explosionsgefäß sehr unterschiedlich. Die Beurteilung reicht von sehr gefährlich bis ungefährlich. Die eingezeichnete Grenzkurve ist etwas steiler als die Ausgleichsgerade in Bild 2. Sie schneidet die Abszisse bei rund 65 % Inertanteil.

Die Wassergehalte der beiden ungefährlichen Proben waren mit rund 34 bzw. 36,5 % nicht einmal extrem hoch, wenn man sie mit der Probe 123 in Bild 1 vergleicht, die fast 30 % Wasser enthielt und dennoch deutlich höhere Explosionskennziffern ergab. Bei den beiden Naturproben 117 und 304 führten zusätzlich die grobe Körnung in Verbindung mit Aschegehalten um 27 % zu den niedrigen Explosionskennwerten.

Die hohen Werte bei der Probe 291 liegen in der Feinheit und - wie bereits erwähnt - im niedrigen Wassergehalt begründet. 17,3 % Wasser konnten weder die Aufwirbelbarkeit noch die Dispersionsfähigkeit entscheidend verringern.

Das bedeutet für die Praxis, daß immer dann, wenn unter Tage Feinstaub mit Medianwerten unter 20 µm und Wassergehalten unter 20 % in nicht nur verschwindend geringer Menge vorhanden ist, ein hohes Explosionsrisiko besteht. Feinheit und Wassergehalt können nämlich durch Staubbekämpfungsmaßnahmen nur in sehr begrenztem Maß beeinflußt werden. Sowohl bei der schneidenden Kohlengewinnung als auch bei der Förderung entsteht Feinstaub, der durch den Wetterstrom nicht nur zwangsläufig getrocknet und transportiert, sondern auch im oberen Teil des Streckenquerschnitts - relativ zum Grobgut - angereichert wird. Das ist bei Gleislosförderung ebenso unvermeidbar wie bei Gurtförderung.

PRÄVENTIVE SCHUTZMASSNAHMEN

Zur Verringerung der Explosionsgefahr muß man den Kohlenstaub entweder beseitigen oder ihn auf andere Weise unschädlich machen. Eine Beseitigung ist aber höchstens in Ausnahmefällen, d.h. nur örtlich und zeitlich begrenzt möglich. Für den Grubenbetrieb bleibt nur die Möglichkeit, den Staub mit hygroskopischen Salzen zu binden

oder ihn durch Gesteinstaubzugabe explosionsungefährlich zu machen.

Bei der Versuchsgrube Tremonia wurde untersucht, inwieweit das Staubbindeverfahren bei Weichbraunkohlenstaub anwendbar ist. Die Untersuchungen erfolgten nach den "Prüfbestimmungen für Staubbindemittel und Zusatzmittel". Erprobt wurden $CaCl_2$-Lösungen, -Schuppen und -Pulver. Die Laboruntersuchungen bei unterschiedlichen relativen Luftfeuchten ergaben jedoch, daß mit keinem der im Steinkohlenbergbau mit Erfolg eingesetzten Staubbindemittel befriedigende Resultate erzielt werden konnten.

Das bedeutet, daß für den vorbeugenden Explosionsschutz des zu beurteilenden Weichbraunkohlenstaubes nur das Gesteinstaubstreuverfahren übrig bleibt.

EINFLUSS VON GESTEINSTAUBZUSATZ

Zu klären war somit, ob die im Steinkohlenbergbau der Bundesrepublik Deutschland geforderten 80 Gew. % unbrennbaren Bestandteile im abgelagerten Staub auch bei der Braunkohle zur Explosionsverhütung ausreichen oder vielleicht sogar niedriger angesetzt werden können.

Um den gefährlichsten Fall zu berücksichtigen, ist für Versuche zunächst ein Braunkohlenstaub mit nur 9,8 % Wasser und 11 % Asche (Probe 244, Medianwert 23 µm) verwendet worden. Teilproben wurden in der Weise gewonnen, daß der Urprobe Gesteinstaub in Gewichtsanteilen von 40, 60, 70 und 80 % zugemischt wurde. Auf diese Weise ergaben sich Gesamt-Inertanteile in den Mischungen, die sich wie folgt errechnen:

$$I_{ges} = (1-G/100) \times (A+G)/G \qquad (2)$$

oder umgeformt

$$G = (I_{ges}-A-W) \;/\; (1 - \frac{A+W}{100}) \qquad (3)$$

hierin bedeuten

I_{ges}	- Gesamt-Inertgehalt	Gew.-%
G	- Gesteinstaubzumischung	Gew.-%
A	- Aschegehalt der Kohle	Gew.-%
W	- Wassergehalt der Kohle	Gew.-%

Die für die Probe 244 errechneten Werte sind in Tabelle 4 aufgelistet (zum Vergleich sind entsprechende Angaben für die Probe 123, Tabelle 1, mitaufgeführt).

Tabelle 4:

Gesteinstaubzumischung in Gew.-%	Gesamt - Inertgehalt bei Probe Nr. 244 %	bei Probe Nr. 123 %
0	20,8	51,7
40	57,0	1,0
60	68,3	0,7
70	76,2	-
80	84,2	-

Die mit den fünf Teilproben 244 ermittelten Explosionskennziffern sind in Bild 4 dargestellt.

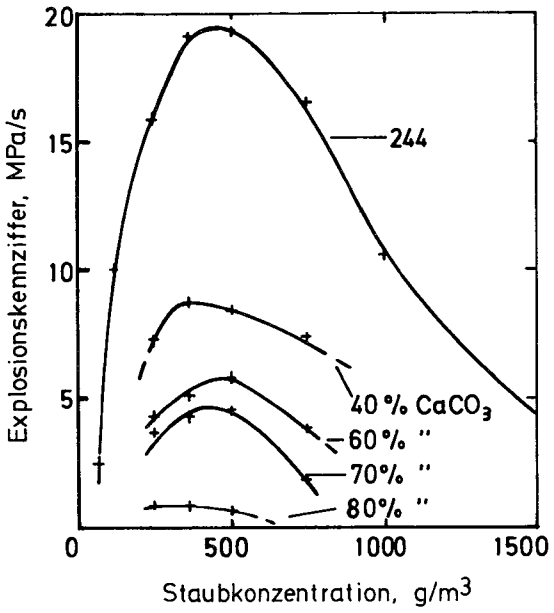


Bild 4.

Bild 5 gibt die Maximalwerte dieser Kurven wieder. Aus Bild 5 entnimmt man, daß bei Gesamt-Inertgehalten zwischen 20 und fast 80 % eine linear fallende Abhängigkeit besteht und erst oberhalb dieses Wertes ein stärkerer Abfall der Explosionskennziffer eintritt. Bei 85 % Gesamt-Inertanteil (entsprechend 81,1 % Gesteinstaub-

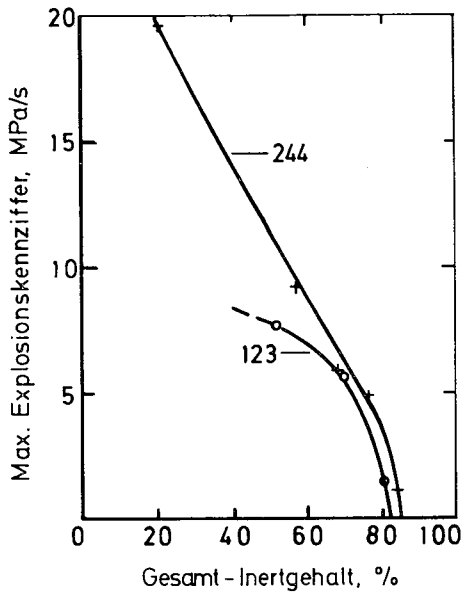


Bild 5.

zumischung gemäß Gleichung 3) schneidet die rechte Kurve die Abszisse.

Um auch für einen asche- und wasserreicheren Braunkohlenstaub die Gesteinstaub-Grenzkonzentration zu ermitteln, ist die Probe 123 in ähnlicher Weise wie die Probe 244 aufbereitet und untersucht worden. Die Ergebnisse sind ebenfalls in Bild 5 enthalten (linke Kurve). Trotz des hohen Inertgehalts von 51,7 % in der Urprobe reichten hier 60 Gew.-% Gesteinstaub noch nicht aus, die Explosionsfähigkeit ganz zu unterbinden. Erst bei 81 % Gesamt-Inertgehalt, das entspricht einer Gesteinstaubzumischung von 62,7 %, wird die Abszisse geschnitten.

Die Ergebnisse dieser Gefäßversuche zeigen, daß bei aschearmen und trockenen Weichbraunkohlenstäuben etwas mehr als 80 Gew.-% Gesteinstaub zur Explosionsverhütung erforderlich sein können. Andererseits werden Braunkohlenstaub-Naturproben i.a. mehr als 35 Gew.-% unbrennbare Bestandteile enthalten. (Dieser Richtwert hat sich im Verlauf der bisherigen Untersuchungen als unterer Grenzwert ergeben, was aber nicht ausschließt, daß er im Einzelfall auch niedriger sein kann.)

Um einen solchen Naturstaub mit 35 % unbrennbaren Bestandteilen explosionsungefährlich zu machen, ist nach Gleichung 3 ein Gesteinstaubanteil von ca. 77 Gew.-% erforderlich.

VERSUCHE IN DER 200-M-STRECKE ÜBER TAGE

Zur Überprüfung der bei den Gefäßversuchen gefundenen Ergebnisse ist bei der WBK-Bergbau-Versuchsstrecke eine 200 m lange Rohrstrecke von 2,5 m² Querschnitt vorhanden. In dieser Strecke sind einige orientierende Versuche mit dem Braunkohlenstaub Nr. 123 (Tabelle 1) durchgeführt worden. Variiert wurden die Initialstärke und die Streuzonenlänge (40 und 90 m) bei einer Staubkonzentration von 200 g/m³.

Wie aufgrund der Gefäßversuche zu erwarten war, kam es in der Strecke in allen Fällen zu einer selbständigen Explosionsausbreitung. Die Flamme war stets mehr als doppelt so lang wie die Streuzone.

Im Vergleich zu ähnlich angelegten Fettkohlenstaubexplosionen (mit Staub nach Tabelle 1) ergab die Braunkohle etwas höhere Werte. Die gemittelten Flammengeschwindigkeiten lagen bis zu 30 % höher, während die maximalen Explosionsdrücke fast gleich waren.

VERSUCHE BEI DER VERSUCHSGRUBE TREMONIA

Für Explosionsversuche mit Kohlenstaub, sowohl unter wie über Tage, verwendet die Versuchsgrube Tremonia den sogenannten "Dortmunder Normalstaub". Es handelt sich hierbei um einen Steinkohlenstaub (Fettkohle). Die wichtigsten Kenndaten dieses Staubes sind in der Tabelle 5 aufgeführt. Umfangreiche Explosionsversuche unter Tage lieferten bisher hinreichendes Datenmaterial über das Anlaufverhalten von Explosionen mit Steinkohlenstaub (Michelis 1986). Ergänzende Versuche hierzu wurden mit Weichbraunkohlenstaub im Hinblick auf anzuwendende präventive und konstruktive Schutzmaßnahmen im Jahre 1988 bei der Versuchsgrube Tremonia durchgeführt. Die Kenndaten des dabei verwendeten Braunkohlenstaubes sind ebenfalls in der Tabelle 5 enthalten.

Tabelle 5:

Zusammensetzung (Mittelwerte)		Steinkohle (Dortmunder Normalstaub)	Braunkohle (Hessische-Unter-Tage-Braunkohle)
Wassergehalt	Gew.-%	0,5	8,7
Aschegehalt	Gew.-%	3,5	20,5
Flüchtige Bestandteile (i.wf)	Gew.-%	23,0	47,2
Flüchtige Bestandteile (i.waf)	Gew.-%	24,0	59,5
Medianwert	µm	18 - 22	18 - 22
Körnungsanteil $\leq$ 50 µm	%	rd. 95	rd. 90

EXPLOSIONSVERSUCHE UNTER TAGE

Die Versuchsgrube Tremonia verfügt über ein untertägiges Explosionsstreckennetz von rund 1700 m Länge mit Streckenquerschnitten zwischen 8 bis 20 m². Die hier diskutierten Versuche wurden in einer 8 m² großen Strecke, die rund 700 m lang ist, durchgeführt (Bild 6). Der vorgegebene Querschnitt entspricht in etwa dem Streckenquerschnitt, der in westdeutschen Braunkohlenbergwerken unter Tage Verwendung findet.

EINFLUSS DES ZÜNDINITIALS

Bei den durchgeführten Versuchen wurde davon ausgegangen, daß die verwendeten Zündquellen in der Lage sind, Kohlenstaub zu zünden. Wichtig dabei war, daß der Zündvorgang reproduzierbar wiederholt werden konnte. Zum Einleiten der Explosion können sowohl Sprengstoffinitiale als auch Methan herangezogen werden.

Das Einleiten einer Kohlenstaubexplosion durch Sprengstoff ist mit Gesteinsprengstoffen oder mit Schwarzpulver möglich. Da allerdings das alleinige Zünden des Sprengstoffes noch zu keiner Kohlenstaubexplosion führen muß, ist es notwendig, daß eine bestimmte Menge an voraufgewirbeltem Kohlenstaub vorhanden ist. Dieses Kohlenstaub-Luftgemisch wird dann von dem Sprengstoffinitial gezündet und leitet eine Kohlenstaub-Laufexplosion ein.

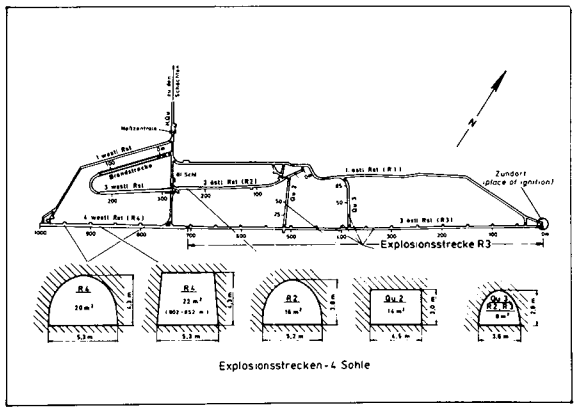


Bild 6.

Das Standardzündinitial der Versuchsgrube Tremonia zum Einleiten von Kohlenstaubexplosionen besteht aus 10 kg voraufgewirbeltem Kohlenstaub (Dortmunder Normalstaub) und 2,5 kg Schwarzpulver (Tabelle 6, Nr. 7). Im Rahmen der hier durchgeführten Vergleichsversuche kamen neben Schwarzpulver weiterhin auch ein gelatinöser und ein pulverförmiger Gesteinsprengstoff zum Einsatz.

Tabelle 6:

Lfd. Nr.	Art des Initials	max. Druck stat. kPa	dyn. kPa	theoretisch freigesetzte Energie MJ
	A. Sprengstoffinitial:			
	Schwarzpulver			
1.	1250 g	3,9	-	0,345
2.	2500 g	6,0	-	0,685
	gelatinöser Sprengstoff			
3.	1 Patrone = 250 g	32,3	2,9	0,275
	pulverförmiger Sprengstoff			
4.	1 Patrone = 240 g	29,6	1,9	0,240
5.	2 Patronen = 480 g	42,3	3,5	0,480
	B. Kombiniertes Initial:			
6.	1250 g Schwarzpulver + 10 kg Steinkohlenstaub	28,6	6,2	300
7.	2500 g Schwarzpulver + 10 kg Steinkohlenstaub	37,5	8,5	300
8.	240 g pulverförmiger Sprengstoff + 5 kg Steinkohlenstaub	24,3	1,8	150
9.	240 g pulverförmiger Sprengstoff + 10 kg Steinkohlenstaub	28,6	5,8	300
10.	240 g pulverförmiger Sprengstoff + 5 kg Braunkohlenstaub	35,0	2,3	60
11.	240 g pulverförmiger Sprengstoff + 10 kg Braunkohlenstaub	30,0	5,3	120
12.	4,5 m³ Methan (10 Vol.-%)	49,0	14,5	162
13.	3,5 m³ Methan (8 Vol.-%)	26,5	4,0	126

Das häufigste Zündinitial für die Einleitung von Kohlenstaubexplosionen stellt allerdings im Steinkohlenbergbau das Zünden von Methanansammlungen dar (Michelis 1986).

Die Auswirkungen der einzelnen Zündinitiale im Hinblick auf maximalen allseitigen (statischen) Druck und erzeugten Winddruck (dynamischen) Druck) in der 8 m² großen Explosions-

strecke sind in der Tabelle 6 wiedergegeben. Hierbei ergaben sich die Werte für den statischen Druck in der Regel im Streckenort und für den dynamischen Druck beim 50. Streckenmeter. Die Anordnung der Sprengmörser und der HRD-Behälter, aus denen der Kohlenstaub mit einem Druck von 6 MPa voraufgewirbelt wurde, ist aus Bild 7 ersichtlich.

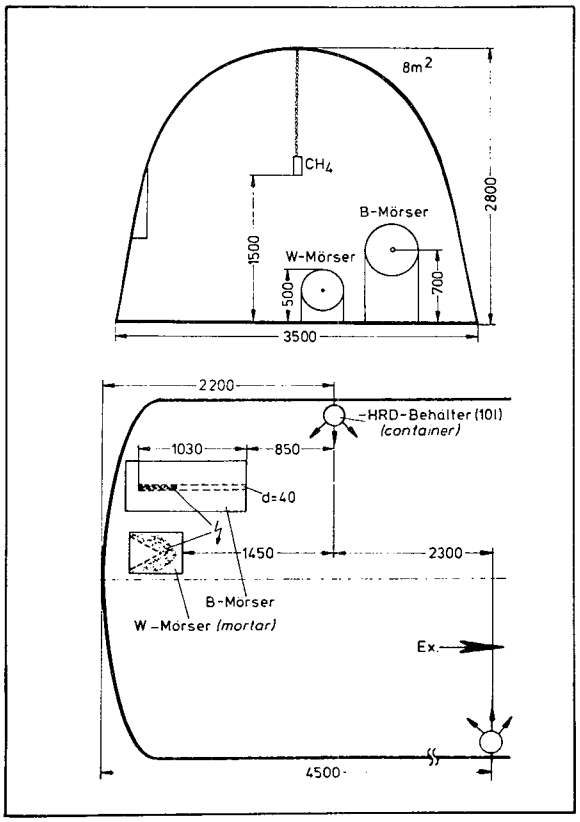


Bild 7.

In der 8 m² großen Explosionsstrecke wird zur Einleitung einer reproduzierbaren Kohlenstaub-Laufexplosion ein durch das Initial erzeugter Winddruck von etwa 8 kPa und mehr benötigt. Daher werden normalerweise bei den Versuchen als Standardinitiale die Initiale mit der Nummer 7 und 12 aus der Tabelle 6 verwendet. Zwar führen auch weniger Methan und geringere Mengen an Sprengstoff in Kombination mit voraufgewirbeltem Kohlenstaub durchaus zur Einleitung von Kohlenstaubexplosionen. Nur ist deren Entwicklung und Verlauf sehr unterschiedlich und wenig reproduzierbar.

Im Hinblick auf die Versuche mit Braunkohlenstaub wurde eine Initialkombination gesucht, die vergleichbare Ergebnisse bei der Druckentwicklung lieferte, wie bei Verwendung von Steinkohlenstaub. Wie die Ergebnise der Nr. 9 und 11 in der Tabelle 6 zeigen, konnten bei der Kombination von 240 g (1 Patrone) pulverförmigem Sprengstoff mit 10 kg voraufgewirbeltem Stein- oder Braunkohlenstaub nahezu gleiche Werte ermittelt werden. Bei den Explosionsversuchen mit Braunkohlenstaub wurde daher als Sprengstoffinitial ausschließlich 240 g pulverförmiger Sprengstoff eingesetzt (s. Tabelle 7).

WEITERE EINFLUSSGRÖSSEN AUF DEN ANLAUF VON KOHLENSTAUBEXPLOSIONEN

Abgesehen von der Art des Initials, das den Anlauf einer Kohlenstaubexplosion in der Anfangsphase zum Teil wesentlich beeinflußt, sind weitere Einflußfaktoren zu nennen wie:

- Länge der Brennstoffzone
- spezifische Brennstoffkonzentration
- Art der Brennstoffbereitstellung
- Zusammensetzung und chemisch-physikalischer Aufbau des Brennstoffes.

Bei den hier vorgestellten Vergleichsversuchen betrug die Länge der Brennstoffzone 20 m oder 40 m und die angebotene Brennstoffkonzentration 125 g oder 300 g/m³ Streckenvolumen. Der Kohlenstaub wurde bei diesen Versuchen pneumatisch über den gesamten Streckenquerschnitt verteilt und war zum Zeitpunkt der Zündung vollkommen sedimentiert, so daß er durch den Winddruck des Initials wieder aufgewirbelt werden mußte. Als Brennstoff wurde der in Tabelle 5 vorgestellte Steinkohlen- bwz. Braunkohlenstaub eingesetzt.

ERGEBNISSE

In der Tabelle 7 sind die Versuchsergebnisse im Hinblick auf maximalen Explosionsdruck (statisch und dynamisch), erreichte Flammenlänge und maximale Flammengeschwindigkeit in Abhängigkeit von der jeweiligen Versuchsanordnung wiedergegeben.

Versuchsanordnung 1. Kohlenstaubmenge des Initials (kg) 2. Brennstoffzone (m) 3. Spez. Brennstoffmenge (g/m³)	Braunkohle° max.Druck stat. kPa	dyn. kPa	Flammenlänge m	max. Flammengeschw. m/s	Steinkohle max.Druck stat. kPa	dyn. kPa	Flammenlänge m	max. Flammengeschw. m/s
1. 2. 3.								
5 / 20 / 125	29,4	4,1	56	56	-	-	-	-
5 / 40 / 125	31,4	6,8	81	86	92,5	52	140	215
					(Kohlenstaub voraufgewirbelt)			
10 / 20 / 125	44,8	12,8	56	110	-	-	-	-
10 / 20 / 300	73,7	57,2	94	175	33,0*	6,0	58	65
					56,0+	22,0	75	19
10 / 40 / 300	91,7	41,4	119	216	81,5*	22,0	125	146
					84,0+	37,5	120	209

o Initial bei allen Braunkohlenversuchen = 1 Patrone (240 g) pulverförmiger Sprengstoff

* Initial mit 1,25 kg Schwarzpulver

\+ Initial mit 2,50 kg Schwarzpulver

Die Versuche mit Steinkohlenstaub erwiesen sich bei allen Kombinationen im Hinblick auf ihren Druckverlauf als schwächer gegenüber dem Braunkohlenstaub. Dies wird auch aus den Bildern 8 und 9 ersichtlich. Die Darstellung in Bild 8 zeigt den Druck-Zeit-Verlauf des statischen Druckes bei einer 20 m langen und in Bild 9 bei einer 40 m langen Brennstoffzone. Die Bilder 8 und 9 zeigen das spontanere Zündverhalten sowie die höheren Werte für den zeitlichen Druckanstieg $\Delta p/\Delta t$ des Braunkohlenstaubes. Der gemittelte zeitliche Druckanstieg betrug, bezogen auf die Meßergebnisse beim 100. Streckenmeter in den Bildern 8 und 9, für die Steinkohle rund 176 kPa/s und für die Braunkohle rund 340 kPa/s.

EMPFEHLUNGEN ZUM KONSTRUKTIVEN EXPLOSIONSSCHUTZ

Da sich der hier untersuchte Braunkohlenstaub in seinem Explosionsverhalten als vergleichbar mit Steinkohlenstaub zeigte, sind auch die Schutzmaßnahmen des Steinkohlenbergbaus zur Bekämpfung einer bereits angelaufenen Explosion durchaus anwendbar. Der Einsatz von konstruktivem Explosionsschutz in Form von Wassertrog-Explosionssperren kann dabei entweder mit konzentrierten oder aufgeteilten Explosionssperren erfolgen, wobei der aufgeteilten Wassertrogsperre der Vorrang gegeben werden sollte, da im Falle eines Zündereignisses die Auswirkungen wesentlich kleiner gehalten werden können als beim Einsatz von konzentrierten Sperren (Bild 10).

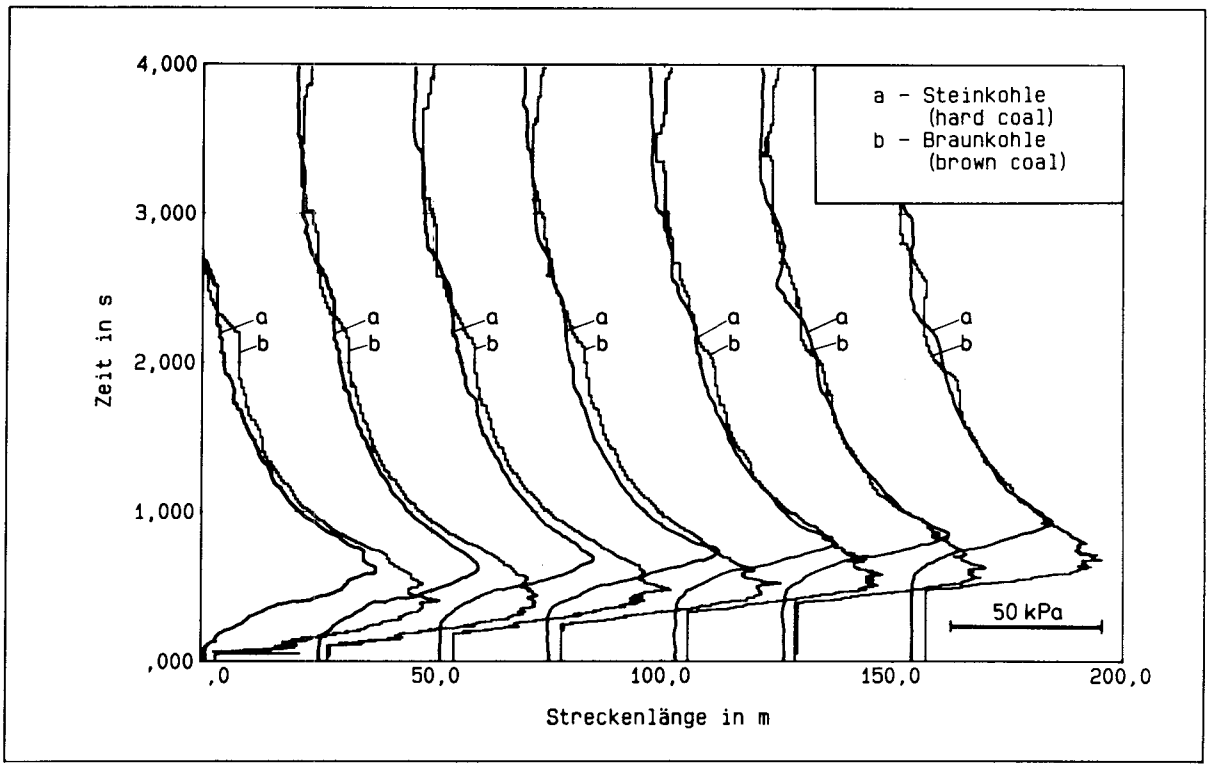

4,000
3,000
2,000
1,000
,000
Zeit in s
a - Steinkohle
(hard coal)
b - Braunkohle
(brown coal)
a
b
50 kPa
,0
50,0
100,0
150,0
200,0
Streckenlänge in m

Bild 8.

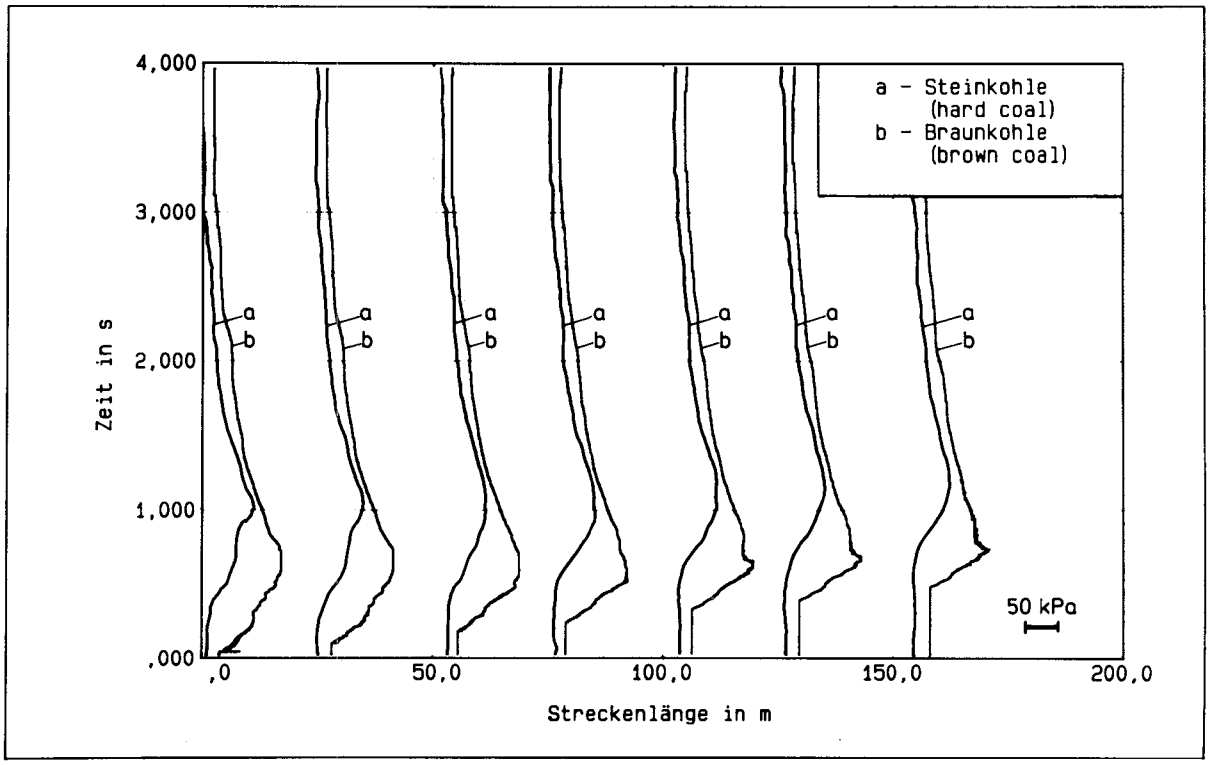

4,000
3,000
2,000
1,000
,000
Zeit in s
a - Steinkohle
(hard coal)
b - Braunkohle
(brown coal)
a
b
50 kPa
,0
50,0
100,0
150,0
200,0
Streckenlänge in m

Bild 9.

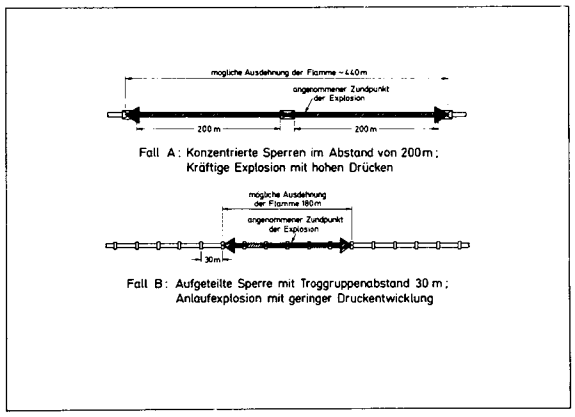


Bild 10.

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ETUDES COMPARATIVES DE L'EXPLOSIBILITE DES POUSSIÈRES DE CHARBON BRUN ET DE CHARBON BITUMINEUX DANS DES INSTALLATIONS D'ESSAIS DE SURFACE ET SOUTERRAINES

D. Reeh, J. Michelis

Une explosion qui s'est produite dans une mine profonde de charbon brun a donné lieu à une étude des conditions spéciales qui rendent le charbon brun Eocène explosif. L'épaisseur de la couche varie; sa forme est différente également, verticalement et horizontalement, sa teneur moyenne en eau est de 45%, sa teneur en cendres de 30% et sa teneur en matière volatil de 65% environ).

Des essais effectués par BVS dans une chambre de pression à 40-1 ont permis d'établir les caractéristiques d'explosion les plus importantes, avec teneur en eau variable, ainsi que la quantité de poussières minérales nécessaire pour empêcher la propagation d'une explosion. Les résultats d'essais effectués dans une galerie d'explosions souterraine de 8 m^2 de la Mine Expérimentale de Tremonia ont été comparés aux résultats d'essais en laboratoires et aux résultats obtenus avec du charbon bitumineux.

Comparées aux poussières standards de charbon bitumineux habituellement utilisées, qui contiennent 27% de matière volatile, les poussières de charbon brun ayant la même teneur en matière inerte et la même distribution de taille de particules se sont révélées beaucoup plus explosives. Ce fait est normalement dissimulé par la haute teneur en eau du charbon brun, sauf si les couches superficielles des dépôts de poussières se dessèchent, ou si de fines particules bitumineuses se rassemblent à certains endroits.

СРАВНИТЕЛЬНЫЕ ИССЛЕДОВАНИЯ ВЗРЫВООПАСНОСТИ БУРОГО УГЛЯ И БИТУМИНОЗНОЙ УГОЛЬНОЙ ПЫЛИ, ПРОВЕДЕННЫЕ НА ПОВЕРХНОСТИ И В ПОДЗЕМНЫХ ИСПЫТАТЕЛЬНЫХ УСТАНОВКАХ

Д. Рее и Дж. Михелис

Взрыв пыли в глубокой шахте, добывающей бурый уголь, заставил исследователей изучить •специальные условия, при которых мягкие эоценовые бурые угли становятся взрывоопасными. Мощность угольных пластов и их конфигурация меняются как в вертикальном, так и в горизонтальном измерениях. В них содержится 45% воды, 30% золы и около 65% летучих веществ. При испытаниях в камере под давлением 40-1 были установлены наиболее важные характеристики взрывов при различном содержании воды в образцах, а также количество каменной пыли, необходимое для предотвращения распространения взрыва. Результаты подземных испытаний, проведенных в штреке для взрывов на экспериментальной шахте Тремония, сравнивались с лабораторными испытаниями и с результатами, полученными в испытаниях с битуминозным углем. По сравнению с обычно встречающейся битуминозной угольной пылью, содержащей 27% летучих веществ, пыль бурых углей с тем же содержанием инертного материала, имеющая то же распределение мельчайших частиц, является более взрывоопасной. Этот факт обычно маскируется высоким содержанием воды в бурых углях, кроме тех поверхностных участков, где пыль высыхает или битуминозные частицы скапливаются в определенных местах.

褐煤和烟煤尘在地面和地下试验设备爆炸性的比较研究

D．雷埃，及 J．密希力斯

开始研究始新世软褐煤有爆炸性的情形是源于一个深井褐煤矿煤尘爆炸的事件．煤层厚度在水平垂直方向均有变化，形状亦很不同，含水量百分之 45，含灰量百分之 30，挥发物质约为百分之 65．

在矿业研究机构（BVS）的 40 升压力容器内做的试验中，用不同含水量和岩粉量测定了最重要的爆炸特性，以便了解防止爆炸传播的办法．在屈裂尼亚实验煤矿 8 平方米的地下爆炸坑道里做的试验结果用来与实验室的结果和烟煤所得的结果也作了比较．

与含有百分之 27 挥发物的平常用标准烟煤尘比较起来，含有同等的惰性物质和具有同样颗粒大小比例的褐煤尘，它的爆炸性高得很多．这个事实常常被褐煤的高含水量所遮盖，除非尘埃积沉的表面层变得十分干燥，或很细的烟煤粉聚积在某些地方．

LABORATORY STUDY OF ROCK DUST INERTING REQUIREMENTS: EFFECTS OF COAL VOLATILITY, PARTICLE SIZE, AND METHANE ADDITION

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ABSTRACT

The Bureau of Mines has conducted a systematic study of the rock dust inerting requirements for various coals, using a 20-L laboratory test chamber. Particle sizes of the coals ranged from very fine pulverized dusts to coarse (minus 20 mesh) dusts. The data show that, with respect to the rock dust inerting requirement, the particle size of the coals has as much of an effect as the standard volatility. In testing various $CaCO_3$ rock dusts against a bituminous coal dust, the finer sized rock dusts were more effective. The laboratory rock dust inerting data also showed reasonable agreement with full-scale mine data. Laboratory data on the effect of added methane gas also showed good agreement with mine data.

INTRODUCTION

There have been over 500 major dust and/or gas explosions in United States coal mines during the past 100 years (Nagy 1981). The frequency and magnitude of these explosion disasters have decreased as a result of regulations that require the addition of rock dust to the coal dust and that require the dilution of methane by ventilation air. However, the explosion hazard has not been completely eliminated, and major coal mine explosion disasters still occur.

The Bureau of Mines has been studying dust explosions in both laboratory and full-scale experimental mine tests since 1910. In the United States, the mining regulations have always been based on the results of the full-scale mine tests. Laboratory chambers have been used for preliminary screening tests before full-scale mine tests were conducted. Because the laboratory tests can be conducted in a much shorter time than the mine tests, a wider range of coal types and sizes can be studied in the laboratory chambers. During the past dozen years, the Bureau has designed an 8-L laboratory chamber (Hertzberg and Cashdollar 1977 and Cashdollar et al 1983) and a 20-L laboratory chamber (Cashdollar and Hertzberg 1985); data from these chambers agreed reasonably well with those from mine tests.

The 20-L chamber has become the standard laboratory test chamber at the Bureau for testing the explosibility and inerting of fuel dusts and gases. Some data on the explosibility of coals and other carbonaceous dusts as measured in the Bureau 20-L chamber have been reported previously. Those previous papers have discussed the inerting of a high volatile bituminous coal by various inhibitors (Hertzberg et al 1985), the ignition energy requirements for accurate measurements of flammability limits of dusts (Hertzberg et al 1988a), a volatility model for coal dust flame propagation (Hertzberg et al 1988b), a comparison of laboratory and mine research on lean explosibility limits and rock dust inerting of various coals (Cashdollar et al 1987), and the effect of volatility on the explosibility limits of coals and other carbonaceous dusts (Cashdollar et al 1989).

The present paper compares 20-L data on the limestone rock dust inerting of three bituminous coals with data from full-scale mine tests. There

is also new 20-L data on the rock dust inerting of various sizes of bituminous coals. In this paper, the terms "flammability" and "explosibility" are used interchangeably to refer to the ability of an airborne dust cloud to propagate a flame after it has been initiated by a sufficiently strong ignition source. The terms refer to a rapid deflagration and not a detonation.

EXPERIMENTAL APPARATUS AND DUST ANALYSES

The dust explosibility data in this paper were obtained in the 20-L laboratory chamber (Cashdollar and Hertzberg 1985) shown in figure 1. The near-spherical chamber is made of stainless steel and has a pressure rating of 21 bar. Optical dust probes (Cashdollar et al 1981) are used to measure the uniformity of the dust dispersion. The strain gauge pressure transducer measures the explosion pressure and a differentiator amplifier allows the direct measure of the rate of pressure rise. The dust to be tested can be placed either in the dust reservoir or on top of the dispersion nozzle at the bottom of the chamber. The standard test procedure was to partially evacuate the chamber to 0.1 bar absolute so that the blast of air (which dispersed the dust) raised the chamber pressure to about 1 bar absolute at the time of ignition. The standard ignition source used for the rock dust inerting measurements was a 5,000 J electrically activated pyrotechnic ignitor.

The proximate analyses of the four bituminous coals that were tested most extensively are listed in table 1. The coals were desiccated before testing, as shown by the low moisture contents in the table. The standard ASTM volatilities are listed for reference only. Other measurements have shown that the true volatilities are higher at the high heating rates found in dust flame fronts (Hertzberg and Ng 1987, Cashdollar et al 1989).

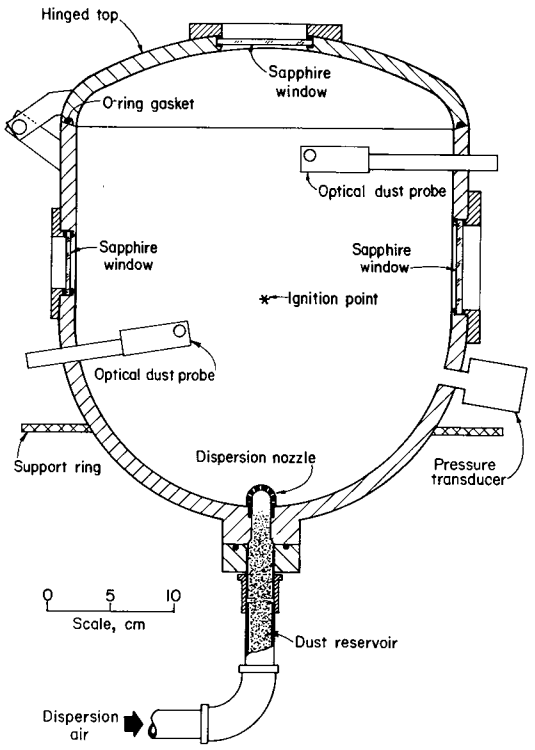


Figure 1. - Vertical cross section of the 20-L laboratory test chamber.

The particle size analyses for the various pulverized batches of these four bituminous coals are listed in table 2. The size distributions were determined from a combination of sonic sieving data and Coulter counter data. The first column lists the coal seam and identifies the various batches of each seam by number. The Pittsburgh seam is abbreviated as Pgh and the Pocahontas as Poc. The second and third columns list the weight percent of each dust that is less than 20 or 75 μm, respectively. In the fourth and fifth columns, D_s is the arithmetic surface mean diameter and D_w is the arithmetic volume or mass mean diameter. Since particle size distributions generally approximate a log-normal distribution, the geometric size parameters are also listed in the table in the sixth and seventh columns. In the sixth column, D_g is the geometric mass mean diameter. If the size distribution were exactly log-normal, D_g would also be

Table 1: Proximate analyses of selected coals

Coal dust, seam and state	Moisture, %	Ash, %	Volatility, %	Fixed carbon, %
Sunnyside, UT	0.8	6.6	38.5	54.1
Pittsburgh, PA	0.9	6.0	36.5	56.6
Freeport, PA	0.9	7.6	22.5	69.0
Pocahontas, VA	0.6	6.0	17.1	76.3

Table 2: Size analyses of the pulverized coals

Coal dust	<20 μm, %	<75 μm, %	D_s, μm	D_w, μm	D_g, μm	σ_g
Sunnyside-1	12	83	36	52	44	1.8
-2	25	73	24	62	40	2.7
-3	41	86	17	40	26	2.6
-4	18	100	28	38	33	1.7
Pgh-1	16	81	31	51	40	2.0
-2	24	100	25	35	30	1.8
-3	47	97	18	28	23	1.9
-4	100	100	8	9	8	1.5
-5	100	100	5	6	6	1.5
Freeport-1	27	73	23	55	37	2.6
-2	47	97	16	29	22	2.2
Poc-1	10	78	38	55	46	1.8
-2	27	65	19	63	39	3.0
-3	38	75	18	60	32	3.2
-4	43	86	16	40	25	2.7
-5	67	94	11	23	15	2.4
-6	67	100	13	17	16	1.6
-7	100	100	5	6	5	1.5

identical to the mass median particle diameter. For these pulverized coals, the size distributions are close enough to a log-normal distribution so that D_g usually differs by only a few percent from the mass median diameter. In the seventh column, σ_g is the geometric standard deviation based on the volume or mass distribution. For a log-normal curve, the characteristic width of the distribution is from D_g/σ_g to $D_g\sigma_g$. For example, the Pgh-1 coal has D_g = 40 μm and σ_g = 2.0. Therefore, one sigma smaller than the mean would be D_g/σ_g = 20 μm and one sigma larger than the mean would be $D_g\sigma_g$ = 80 μm. The larger σ_g is, the broader is the size distribution. For a perfectly monodisperse size distribution, σ_g would be 1.0. Most of the particle size distributions shown in table 2 are for pulverized coals from various pulverizers. The smallest sizes of the Pittsburgh (Pgh 3,4, and 5) and Pocahontas (Poc 6,7) seam coals were centrifugally classified.

Table 3 lists similar size data for the three coarse bituminous coals that were studied. The second through fifth columns list the percent passing through various sieves or, equivalently, the percent less than a certain particle size. For the coarse coals, the size distributions are skewed and therefore the mass median diameters are about 35 to 40% larger than the listed geometric mean diameters.

Table 3: Size analyses of the coarse coals

Coal dust	-200 mesh, <75 μm, %	-100 mesh, <150 μm, %	-40 mesh, <425 μm, %	-20 mesh, <850 μm, %	D_s, μm	D_w, μm	D_g, μm	σ_g
Sunnyside	20	35	69	100	92	311	199	3.0
Pittsburgh	21	35	67	99	89	325	202	3.2
Pocahontas	20	34	67	99	77	323	199	3.3

Table 4 lists particle size data for the rock dusts used in this study. The first is the standard limestone rock dust used at the Bureau for mine explosion tests. The second is this same limestone rock dust with 0.5% magnesium stearate added as a fluidizing agent to make the dust disperse more easily. The third is a sample of the stone dust used in tests at the Experimental Mine Tremonia in Dortmund, Germany. The fourth is a rock dust made from pulverized marble; this rock dust is used in some coal mines in the southern United States. The first three rock dusts have very similar size distributions. The marble rock dust has a higher percentage minus 200 mesh (less than 75 μm), but it also has less fines below 20 μm than the other rock dusts. The fifth rock dust in table 4 is the minus 325 mesh size fraction sieved from the regular limestone rock dust. The sixth is a very fine, centrifugally classified limestone dust.

ROCK DUST INERTING DATA

Figures 2, 3, and 4 are examples of the rock dust inerting data for Pgh-1 coal in the 20-L chamber. To test whether a particular percentage of rock dust will inert a coal, tests are conducted over a wide range of coal dust concentrations. Figure 2 shows the explosibility data for mixtures of 40% fluidized limestone rock dust and 60% coal dust. In the top part of the figure, the maximum rate of pressure rise for each test is plotted versus the coal dust concentration. For the 20-L data, the rate of pressure rise data are size normalized by multiplying by the cube root of the chamber volume. In the bottom part of figure 2, the pressure ratio is the maximum explosion pressure (corrected for the pressure rise due to the ignitor by itself) divided by the pressure at ignition (about 1 bar). Therefore, the pressure ratio (PR) is approximately the maximum absolute explosion pressure in bar or atm. For comparison, the dashed lines in figure 2 show the data for the coal dust without any added rock dust. Based on previous data (Hertzberg et al 1988a), the criteria for significant flame propagation are PR > 2 and $(dP/dt)V^{1/3} > 1.5$. Using these criteria the 40% rock dust tests produced explosions at all coal concentrations above ~80 g/m^3. At higher concentrations, the explosion pressures for the coal-rock mixtures were slightly lower than those for the coal by itself. However, the rates of pressure rise were only about half of those for the coal alone.

Figure 3 shows the data for mixtures of 70% rock dust and 30% coal dust. For these mixtures, both the maximum explosion pressures and rates of pressure rise are significantly lower than those for the coal alone or for the 40% rock dust mixture. However, almost all of the tests at high concentrations produced explosions, and therefore 70% rock dust is still insufficient to inert the mixture. In figure 4, the data show that only a few tests at 75% rock dust produced explosions. However, more than half of the tests at coal concentrations of 200 and 700 g/m^3 did produce explosions.

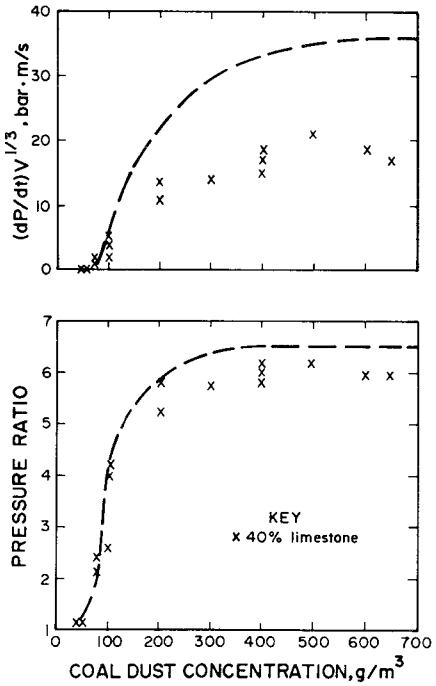


Figure 2. -Explosibility data for mixtures of 60% coal and 40% limestone compared to the dashed line data for coal alone.

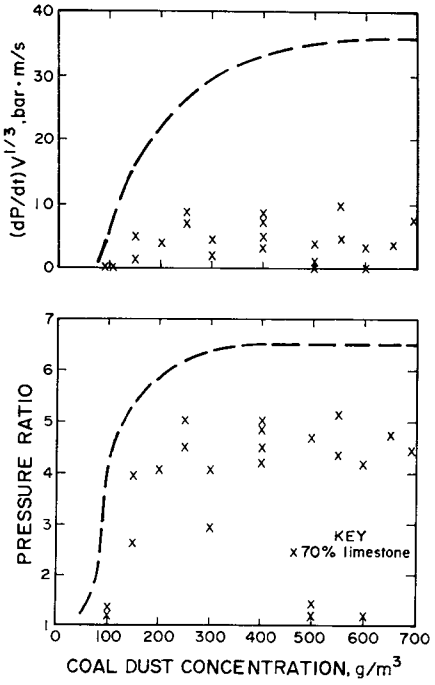


Figure 3. - Explosibility data for mixtures of 30% coal and 70% limestone compared to the dashed lines for coal alone.

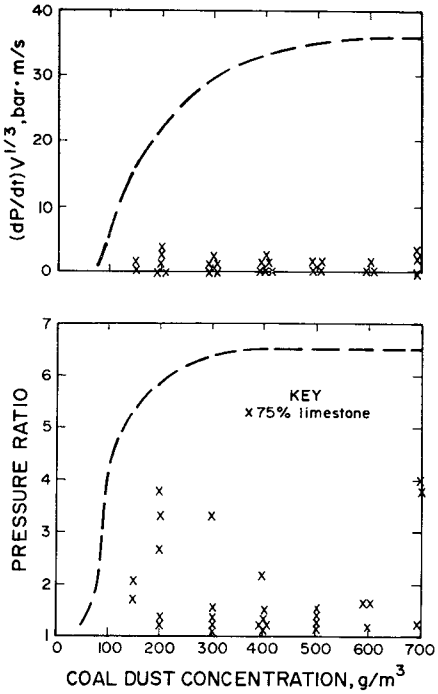


Figure 4. - Explosibility data for mixtures of 25% coal and 75% limestone compared to the dashed lines for coal alone.

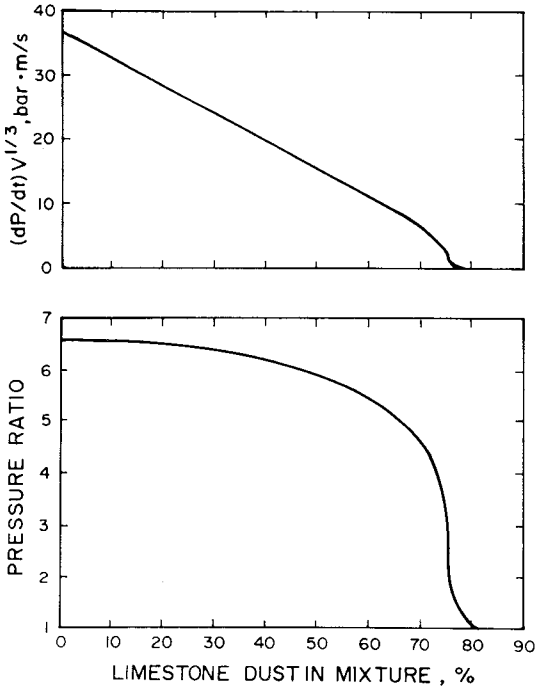


Figure 5. - Maximum explosion pressure and rate of pressure rise data for coal-rock dust mixtures.

Table 4: Size analyses of the $CaCO_3$ rock dusts

Rock dust	<20 μm, %	<75 μm, %	D_s, μm	D_w, μm	D_g, μm	σ_g
Limestone, regular	53	77	12	42	22	3.2
Limestone, fluidized	46	73	14	49	26	3.2
Limestone, German	51	79	15	41	24	2.9
Marble	34	94	18	35	26	2.2
Limestone, -325 mesh	75	100	8	14	11	2.1
Limestone, very fine	100	100	3	4	4	1.4

Therefore, the amount of fluidized limestone rock dust necessary to completely inert the Pgh-1 coal dust would be slightly more than 75%. That conclusion was confirmed by the fact that tests of an 80% rock dust mixture produced no explosions at any concentration.

A summary of all such 20-L data for mixtures of Pgh-1 coal dust and fluidized limestone rock dust is shown in figure 5. For each mixture tested, the average maximum pressure and rate of pressure rise values (measured at high coal dust concentrations) are plotted versus the amount of rock dust in the mixture. The pressure ratio declines only slightly with increased inhibitor content until a high percent of rock dust is reached. Near the inerting level, the pressure ratio decreases rapidly from a value of 4 at 73% rock dust to a value of about 1.1 at 80% rock dust. Based on the flammability criteria and the data in figure 5, the amount of fluidized limestone rock dust necessary to inert this Pittsburgh seam coal is about 76%. This is the inerting level as measured in the 20-L chamber using 5000 J ignitors. Because of the possible dependence on apparatus and the known variation with ignition energy, this inerting level should be considered not as an absolute value, but rather as a relative one. This will be discussed in more detail later in the paper when the data are compared to mine test data.

Data similar to those in figures 2 through 5 were obtained for all the coals listed in tables 2 and 3.

Table 5 lists the fluidized limestone rock dust inerting data obtained in the 20-L chamber for all the pulverized coals from table 2. The repeatability of these data is about ±2%. The amount of rock dust necessary to inert the Sunnyside, Utah coal ranged from 76 to 83% depending on the coal particle size distribution. In general, the finer pulverized coals required more rock dust to inert them. However, the inerting data do not correlate precisely with any one measure of the coal particle size in table 2 because the exact size distribution cannot be described by only one parameter. For these relatively broad size distributions, either an increase in finer particles (<20 μm) or a decrease in the larger particles (>75 μm) results in an increase in the amount of rock dust required to inert the coal-rock mixture. The same general observations can be made for the various sizes of Pittsburgh seam coal. The two finest sizes (Pgh-4 and Pgh-5) were centrifugally classified to remove all the large particles, and these two sizes required the greatest amount of rock dust to inert the coal-rock mixtures.

The two Freeport coals show the same size effects, with the finer size requiring more rock dust. The Pocahontas seam coal data in table 5 show the same correlation with particle size as the other coals. However, this lower volatility coal requires a somewhat lower amount of rock dust to inert, if similar sizes of the coals are compared. For example, Pgh-1 and Sunnyside-1 are

Table 5: Amount of fluidized limestone rock dust necessary to inert various fine size, pulverized coals.

Coal dust	Amount to inert, %
Sunnyside-1	76
-2	78
-3	83
-4	79
Pgh-1	76
-2	79
-3	79
-4	87
-5	89
Freeport-1	73
-2	79
Poc-1	63
-2	67
-3	74
-4	78
-5	83
-6	78
-7	84

similar in size to Poc-1, but they require 76% rock dust to inert compared to 63% for the Pocahontas. Also, Pgh-5 is comparable in size to Poc-7 but requires 89% to inert compared to 84% for the Pocahontas coal. Note that the difference in inerting amount required for the Pittsburgh versus Pocahontas coals is smaller for the finer sizes of the coal dusts. This may be due to increased amount and rate of volatiles generation from the finer dusts (Hertzberg and Ng 1987), especially for the lower volatility coal.

A comparison of the inerting effectiveness of the regular and fluidized limestone rock dusts in the 20-L laboratory chamber is listed in tables 6 and 7. The dashes in the tables indicate that no tests were made. For both the Pittsburgh and Pocahontas seam pulverized coals, somewhat more of the fluidized rock dust was required to inert the mixture. The reason for this may be that although the fluidized rock dust disperses better (which should make it more effective), it may also allow the coal dust to disperse better. Also listed in tables 6 and 7 are the Bureau data from the Bruceton Experimental Mine (BEM) and Lake Lynn Experimental Mine (LLEM). The BEM data are based on previous tests with a wide range of coals (Rice and Greenwald 1929 and Richmond et al 1975). Those early data have also been presented in more recent papers (Nagy 1981, Cashdollar et al 1987, and Weiss et al 1989). The coarse Pocahontas coal was not actually tested in the BEM, and the data listed in table 7 are for a coarse bituminous coal of similar volatility. The LLEM data are from the paper at this conference by Weiss et al (1989). In those other papers describing the mine results, the amount to inert was usually presented as the total incombustible content rather than just the limestone content as listed in tables 6 and 7. (The total incombustible includes the moisture and ash in the coal in addition to the limestone.) The data listed in tables 6 and 7 show that larger amounts of rock dust are required in the larger cross section entry of the LLEM compared to those required at the BEM, particularly for the lower volatility Pocahontas and for the coarse size coals. The 20-L laboratory data show reasonably good agreement with the mine data, especially for the pulverized coals. It is unrealistic to expect that a laboratory chamber would always give exact agreement with large-scale mine data over a wide range of coal volatility and particle size. However, the relative data from the 20-L chamber can be used for preliminary screening tests before full-scale mine tests. This provides a cost effective method of reducing the number of mine tests needed to document the inerting requirements for coals.

Table 8 lists inerting data for the various $CaCO_3$ rock dusts listed in table 4. All were tested against the Pgh-1 pulverized coal dust in

Table 6: Amount of limestone rock dust (%) necessary to inert pulverized bituminous coals

	20-L chamber		BEM	LLEM
	fluidized	regular	regular	regular
Sunnyside-1	76	-	77	75-79
Pgh-1	76	72	76	78-80
Poc-1	63	58	50-55	>68

Table 7: Amount of limestone rock dust (%) necessary to inert coarse bituminous coals

	20-L chamber		BEM	LLEM
	fluidized	regular	regular	regular
Sunnyside	51	-	59	68-72
Pittsburgh	50	47	58	>65
Pocahontas	42	37	30-35	52-56

Table 8: Amount of various rock dusts (%) necessary to inert Pittsburgh pulverized coal dust.

Rock dust	Amount to inert, %
Limestone, regular	72
Limestone, fluidized	76
Limestone, German	76
Marble	76
Limestone, -325 mesh	68
Limestone, very fine	63

the 20-L chamber. The German stone or rock dust has a size distribution similar to those of regular and fluidized limestone rock dusts from the United States. Its effectiveness is comparable to that of the fluidized rather than regular rock dust even though the optical dust probes show that its dispersion characteristics are comparable to those of the regular rather than fluidized limestone. The marble rock dust is also less effective than the regular limestone and comparable to the fluidized limestone rock dust. In this case also, the optical dust probes show that the dispersion characteristics of the marble are comparable to those of the regular limestone. From table 4, the marble dust has a higher percentage of mass below 75 μm but less below 20 μm than the other rock dusts. The smaller amount of very fine particles may reduce its effectiveness as a heat sink. The minus 325 mesh limestone was finer in size than the other rock dusts and required less to inert the coal. The very fine sized limestone was the most effective of these rock dusts. The explanation is that the finer dusts are probably more effective heat sinks.

The effect of small amounts of added methane gas on the inerting effectiveness of rock dust is shown in table 9. More rock dust is required to inert the mixture when the atmosphere contains added methane. Based on previous tests in the Bruceton Experimental Mine (Rice and Greenwald 1929), a formula was developed for the additional amount of rock dust, A, necessary for each percent of methane present. The formula is $A = (100 - I)/5$, where A is in percent and I is the total incombustible percent required in the absence of methane. The data in table 9 from the 20-L laboratory

Table 9: Amount of fluidized limestone rock dust (%) necessary to inert pulverized coal with added methane gas.

Coal dust	0% CH_4	1% CH_4	2.5% CH_4
Pgh-1	76	80	87
Poc-1	63	71	-

chamber also follow this empirical formula. Thus there is good agreement between the data from the 20-L laboratory tests and those from full-scale mine tests for the inerting requirements of mixtures of coal dust and methane.

A summary of rock dust inerting requirements measured in the 20-L chamber for a very wide range of carbonaceous dusts is shown in figure 6. Some of these data were presented previously (Cashdollar et al 1989), but the figure now contains new data for the pulverized coals in table 2. Data for the very fine, centrifugally classified coal dusts were not included. The graph shows the amount of fluidized limestone rock dust necessary to inert the various fuel dusts as tested in the 20-L chamber using 5000 J ignitors. The amount to inert is shown in two ways as a function of the standard ASTM volatility (on an "as received" basis). The first (open circles) is the total incombustible, which includes both the limestone and the ash and moisture in the coal. The coals tested include low volatile anthracites and various bituminous coals. The data have been extended to higher volatility by also testing a polyethylene-graphite mixture (50% volatility), gilsonite (a mined, asphaltic material with 85% volatility), and polyethylene (100% volatility). Proximate analyses of these dusts are listed either in table 1 or in the previous paper (Cashdollar et al 1989). The data curve for total incombustible required versus volatility agrees rather well with previous data for pulverized coals tested in the Bruceton experimental mine (Rice and Greenwald 1929 and Richmond et al 1975).

In the second way of presenting the data in figure 6, the triangles are the inerting data points for which the fixed carbon char is also considered to be part of the inert material. These data show that all of the fuel dusts are inerted at a

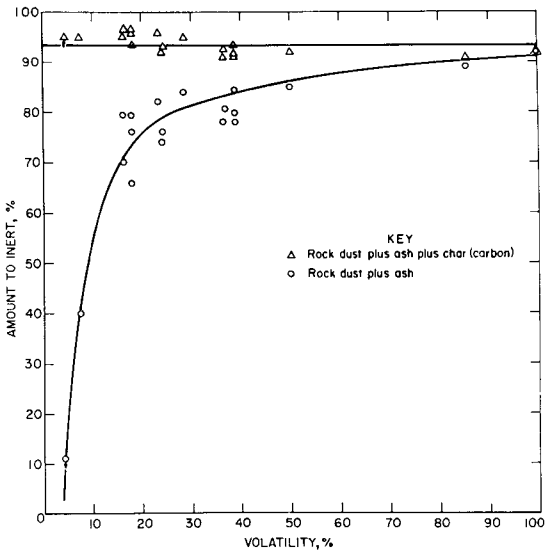


Figure 6. - Rock dust inerting requirements for carbonaceous dusts over a wide range of volatility.

volatility, when the fixed carbon is considered part of the inert. This shows that the added limestone rock dust, the ash and moisture in the coals, and the fixed carbon are all of similar effectiveness in inerting the dusts during the short time scale of flame propagation. As another test, graphite powder was used to inert the polyethylene dust, and 94% graphite was sufficient to inert the polyethylene-graphite mixture. This was only slightly more than the 92% limestone required to inert the polyethylene-limestone mixture.

DISCUSSION AND CONCLUSION

The data show that the particle size distribution of coals can have as much of an effect on the rock dust inerting requirements as the volatility of the coals. In general, the finer the size of the coal, the more rock dust is required to inert. However, any single measure of particle size (such as the percent minus 200 mesh) is insufficient to characterize the particle size distribution and rock dust inerting requirement. The data also show that finer sized rock dust

is more effective in inerting coal dust.

The 20-L laboratory data presented here are consistent with laboratory rock dust inerting data of other researchers (Banhegyi and Egyedi 1983, Vukanic and Vukanic 1983, Foniok 1985, and Krzystolik 1987). However, an exact comparison is difficult because of insufficient details of the experimental apparatus, test procedures, and propagation criteria used by the other researchers.

There is sufficiently good agreement between the 20-L laboratory chamber data and the experimental mine data to justify the use of the laboratory chamber for preliminary screening tests, thus improving the cost effectiveness of the full-scale mine testing program.

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ETUDE APPROFONDIE EN LABORATOIRE DES CONDITIONS D'INERTIE DES POUSSIERES DE ROCHES : EFFETS DE LA VOLATILITE DU CHARBON, DE LA TAILLE DES PARTICULES ET DE L'APPORT DE METHANE

Kenneth L. Cashdollar, Martin Hertzberg

Une étude systématique des conditions d'inertie des poussières de calcaire a été effectuée dans une chambre de laboratoire de 20-L. Les données d'inertie ont été comparées aux données des limites maigres d'explosibilité pour les mêmes charbons. La taille des particules de charbon s'échelonnait des poussières pulvérisées de 100 pct passant à la maille de moins de 200 aux poussières épaisses de 20 pct passant à la maille de moins de 200. Les poussières examinées comprenaient des poussières d'anthracites et de charbons bitumineux dont la volatilité allait de faible à élevée.

En ce qui concerne les conditions d'inertie des poussières de roches, les données révèlent que la taille des particules de charbon a autant d'influence que leur volatilité. Elles révèlent également que le composant fixe de carbone ou de charbon animal des charbons est essentiellement inerte au cours d'explosions de courte durée.

Les données d'inertie des poussières de roches obtenues en laboratoire rejoignent celles obtenues par des essais miniers à grande échelle.

UMFASSENDE LABORSTUDIE DER ANFORDERUNGEN AN DAS EINSTAUBEN MIT STEINSTAUB: EFFEKTE DER FLÜCHTIGEN BESTANDTEILE DER KOHLE, KORNGRÖßE UND METHANGASZUSATZ.

Kenneth L. Cashdollar, Martin Hertzberg

Eine systematische Studie über die Anforderungen an das Einstauben mit Kalksteinstaub wurden in einer 20 l fassenden Laborkammer durchgeführt. Die Daten wurden mit den Daten, die sich aus den Experimenten mit den Explosivitätsgrenzen für dieselben Kohlen ergaben, verglichen. Die Korngröße der Kohlen reichte von 100% minus 200 Maschenweite bis zu groben Stäuben mit 20% minus 200 Maschenweite. Die Studie umfaßte Anthrazitkohle, sowie niedrig und hoch flüchtige Steinkohle.

In Bezug auf die Wirksamkeit des Einstaubens mit Steinstaub zeigen die Daten, daß die Korngröße des Staubes einen ebenso großen Einfluß hat wie die Flüchtigkeit. Außerdem zeigt sich, daß feste Kohlenstoffanteile oder Schwelkohleanteile in der Kohle während der kurzen Zeit der Explosion inert bleiben.

Die Ergebnisse dieser Studie werden mit denen aus Großversuchen in Gruben verglichen, und ihre Übereinstimmung wird aufgezeigt.

ВСЕСТОРОННЕЕ ЭКСПЕРИМЕНТАЛЬНОЕ ИССЛЕДОВАНИЕ УСЛОВИЙ ИНЕРТНОСТИ ПОРОДНОЙ ПЫЛИ: ВОЗДЕЙСТВИЕ ЛЕТУЧЕСТИ УГЛЯ, РАЗМЕРА ПЫЛЕВЫХ ЧАСТИЦ И ДОБАВЛЕНИЯ МЕТАНА.

Кеннеф Л. Кашдоллар и Мартин Херцберг

Систематическое изучение условий, необходимых для увеличения инертности известняковой пыли для различных видов угля, было проведено в лабораторной камере #20-Л. Инерционные данные сравнивались с данными по пределам отощенной взрывоспособности для тех же видов угля. Размер угольных частиц варьировал от пульверизованной пыли, состоящей на 100% из фракции минус 200 меш до крупной пыли состоящей на 20% из фракции минус 200 меш.

Результаты показывают, что в рассмотрении условий, необходимых для увеличения инертности породной пыли, размер угольных частиц имеет столь же большое значение, как и летучесть. Результаты также подтверждают, что связанный углерод или древесная составляющая угля, в основном, инертны в течение того короткого промежутка времени, когда происходит взрыв. Лабораторные данные были сопоставлены с результатами полномасштабных полевых испытаний в шахте: было показано хорошее совпадение результатов.

岩粉惰性化要求的综合实验室研究：
煤的挥发性，颗粒大小，及甲烷掺合的影响

肯尼斯 L．凯希达勒 及 马丁 赫兹柏格

我们在 20 升实验室容器里用各种煤炭作了石灰岩粉惰性化要求的有系统的研究．这惰性化的数据结果用来与同样煤炭的爆炸性下限值作了比较．煤粒的大小从细粉百分之百下筛 200 孔，到粗粉百分之 20 下筛 200 孔．研究的粉末包括无烟煤和低挥发至高挥发的烟煤．

就岩粉惰性化要求而论，数据指出煤的颗粒大小与挥发性有相等的影响．数据并指出煤里的固定炭和炭成份在短促的爆炸过程里并不起任何作用．

实验室岩粉惰性化数据与实尺度矿里试验结果相比，十分接近．

Session XII

NEW TECHNOLOGIES, I

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ELECTRICAL PROTECTION BY MEANS OF MICROELECTRONIC INTEGRATED CIRCUIT FOR SPECIAL APPLICATION

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ABSTRACT

After many years of research, all the requirements for electronic protection of electrical equipment in open cast mines and pit mines for unearthed (IT) and earthed networks including:

- overload, and short circuit protection,
- protection from too high contact voltage for direct (L.V.) and indirect contact (H.V.),
- insulation control for IT networks,
- checking the integrity of earthing of load and earthers, together with the control are combined in one microelectronic ASIC for general use, and in two ASIC for methane pit mines. A high reliability of protection for all voltage levels, universal application and easy replacement of electronic module, make such protection very economical particularly in Exd enclosures, and by applying of screened cables reduce the danger from electricity to the very margin of the scale of hazards in pit mines.

COMMON PARAMETERS OF LOGIC PROCESSING OF ELECTRICAL PROTECTION

The basis of this new system of electrical protective connection /by means of microelectronic I.C./ consists in designing the protection in such a way that it does not solve the problem only partially, which is the existing practice, but the protections are treated in a complex way with interactive connection. That means that the protection follows one general algorithm covering all possible combinations of interdependence either simultaneously or sequentially. This means that the processing can be logic, in combination of digital and analog values, with an adjustable reaction threshold.

Such electrical protective connection can be applied to all electric power systems and to different voltages, provided that the logic of the micrologic unit is unique, and by its construction equal. It is only used as desired or as required, which is defined by the project of the installation. It is essential that the input network data such as current, voltage and resistance are prepared for logic processing, which means that they do not need power, but logic zero or unity, or an adjustable voltage threshold for the logic input, where the new data are added, such as characteristic resistance for some correct state, while some deviation + or - is considered a fault. Such electric protective connections are shown on two examples of power supply system marked according to the international classification IEC 364 as follows:

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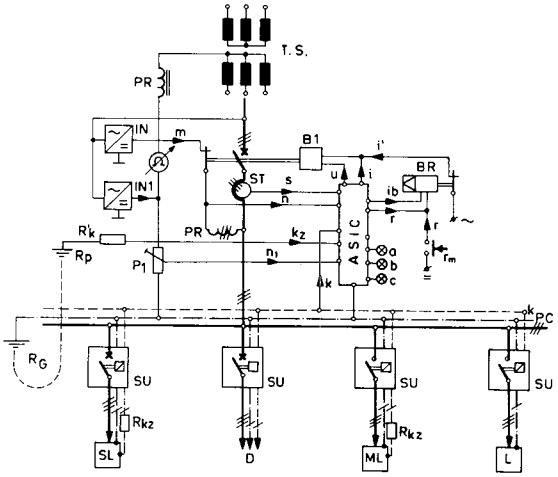


Figure 1. IT system of non earthed network, with permanent monitoring of insulation resistance against earth.

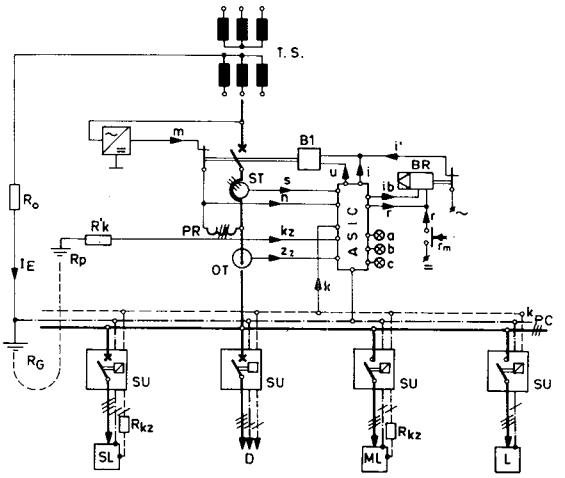


Figure 2. IT system of earthed network through resistance Ro/or impedance/ for earthfault current limitation.

With direct earthing of the neutral point of power supply source, the use is suitable for TT and TN networks too. For all cases for electrical protection, a universal switching unit according to Figure 3 is used, for which the following input data are required:

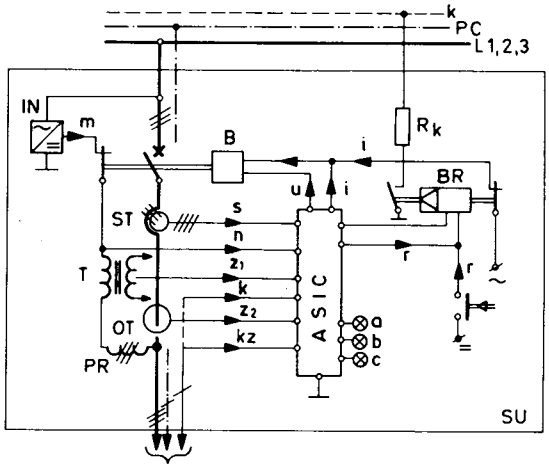


Figure 3. Switching unit for all type of fields.

s - Network current taken as voltage from the secondary of an "unloaded" current transformer. The voltage obtained is linearly dependent on the current, so that it gives the burden data /0-1/. In, and the short circuit data /2-10/. In, which is adjusted as necessary. These data can be obtained through a standard current transformer with secondary current 5A or 1A, and one intermediate transformer 5A or 1A to 5-10 V, or to current e.g. 1 mA which is, via resistance 10 kOhm, transmitted as a signal of 10 V for the rated current value. The same datum can be obtained also by a current/voltage transformer with the induced voltage in the toroid winding, so that it can be wound around the insulated conductor on plastic core.

On such transformer, the induced voltage "u" is a function of current I through the conductor, and the activation of protection is adjusted with voltage "u" in relation to the actual current for overload, or as its multiple for short circuit.

z1 - Alternating voltage of power supply zero point displacement

in case of fault in the network, which means that it includes the power supply neutral point asymmetry. The power supply neutral point displacement can be recorded in star connection, or if neutral point is formed by means of 3 coils, or 3 capacitors. For networks of up to 1,200 V a.c. rated voltage, it is most convenient to carry out by means of three coils and one transformer according to Figure 4, and for networks over 1,200 V rated voltage, by means of capacitances obtained by bandaging the phase conductors with adhesive conducting tape according to Figure 5.

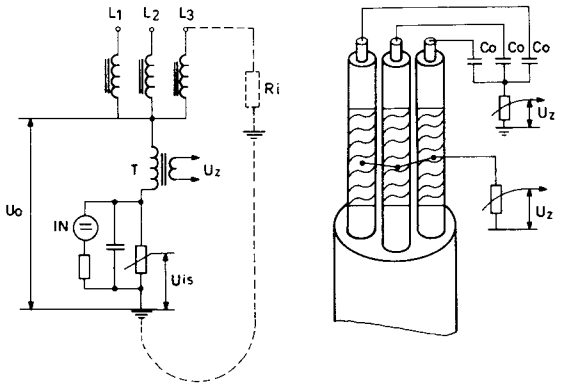


Figure 4. Indication of fault voltage in low voltage networks (1,200 V a.c.).

Figure 5. Indication of fault voltage in high voltage network.

In case of earthfault, a.c. voltage Uo will appear on the neutral point, and it will be transformed to the favourable value Uz, as one of the data for the activation of earthfault protection. Similarly, for high voltage networks the most favourable way for fault voltage indication is arrangement according Figure 5.

By means of conductive self-adhesive tape insulated on one side, whose conductive parts are interconnected together, the capacitive neutral point with three capacities Co is obtained, and the appearance of voltage Uz indicates earthfault.

z2 - The a.c. voltage depends on the leakage current in the network which means, it includes the current unbalance in the branch due to earth leakage. The fault current can be indicated through the summing current, core balance transformer, which sums up all currents of individual phases, or the state when

$$i_1 + i_2 + i_3 \neq 0$$

That means the leakage current is given in form of voltage for digital processing, by the core balance transformer OT. Data z_1 and z_2 are compared to each other in the phase of the same half-cycle and as such are memorized, but only as logic zero or logic unity which is further processed only logically.

n1 - d.c. voltage indicating the value of insulation resistance of the energized network regardless the part of the network in operation /only for unearther IT networks, Figure 1/.

n - The datum as under n1 but only of the branch up to the voltage switching to the branch, which means the measurement of the insulation resistance of the part of network or the load at no-voltage condition before closing the switching device. This insulation measurement can be seen in Figures 1, 2 and 3, in each field. From a d.c. power supply source marked with "m", through the artificial neutral point via reactors PR and transformer T, the insulation of the branch is measured before voltage switching on. If the insulation is correct, the sufficient voltage for digital processing gives permission for

the branch voltage switching on. In case of an earthfault, the voltage /n/ will not be sufficient and the closing of the switching device will be interlocked.

k - The connected characteristic resistance against earth Rk as an indication of correctness of the branch condition. If at the same time the information about fault is received through "s" - signal, and an indication of correct state of pilot circuit anded with resistance Rk, means that fault occurred in the next branch. This k datum is obtained from the electronic unit ASIC which gives the digital "1" if the characteristic resistance according to Figure 6 is connected to its input.

As it can be seen in the figure, if resistance Rk, within tolerances + R, is connected to the R unit input, the R unit will give a logic "1".

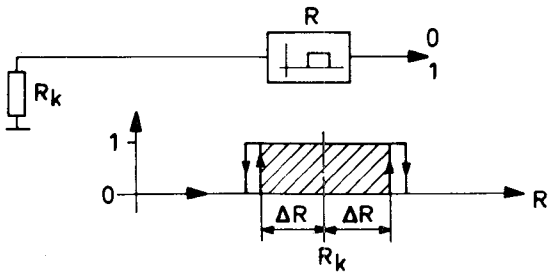


Figure 6. Electronic R - unit sensitive to characteristic resistance Rk.

In the opposite case, with a short circuit on the input, or interruption, or deviations bigger than + R, the electronic unit will give logic "0" for further logic data processing.

Electrical protection covers the following 6 phenomena:

(1) - overload
(2) - earthfault (indirect contact)
(3) - dangerous contact voltage (direct contact)
(4) - short circuit
(5) - checking of earthing integrity

(6) - permanent monitoring of insulation /only for unearthed IT network system/, as well as their signalling in a corresponding way e.g. by means of LED. The following examples are given in figures:

a - green: steady light good condition, while 0.5 Hz blinking means a reduced insulation value for IT /Figure 1/,
b - red blinking 2 Hz for an earthfault and 8 Hz for a short circuit,
c - red blinking 8 Hz for insufficient earthing or damage of earthing integrity, or increased earther resistance or faulty pilot circuit for monitoring of earthing integrity.

In order to explain more in detail the design of the protective link, we shall describe the protection of two characteristic network systems according to Figures 1 and 2. In all cases the following is shown:

- the main switching panel Gp near the transformer and the first distribution of the installation,
- switching panel for the rest of the network "D" /e.g. feeding the next busbars/,
- switching panel for loads moved when energized ML,
- switching panel for special load "SL" to be protected from direct contact voltage which should not be disconnected in case of earthfault then only in case of short circuit,
- typical switching panel for other loads "L".

In Figures 1, 2 and 3 are shown, with standard electrotechnical symbols, the examples of network with busbars for three phases /L1, L2, L3/, the protective conductor /PC/.

On the additional busbar "k" appears the signal "k" with characteristic resistance Rk against earth in case of fault on one distribution branch in case of: - short circuit /alternatively overload/ with automatic "reset" for an unearthed IT network (Figure 1).

By means of "k" conductors connected via "k" busbars, the proceeding switching panel in the direction of energy source is automatically reset /in order not to disturb the supply of the parallel branch of the network and the faultless loads.

The other markings mean:

IN1 - d.c. power supply source for permanent monitoring of the power source insulation, and after switching on, of the complete network under voltage, for an unearthed IT system /Figure 1/,

IN - d.c. power supply source for the control of the branch before the network switching on, in other panels, marked with "m",

B1 - the main circuit breaker in the network /after the power transformer TS/,

B - circuit breakers in the network /on the distribution busbars/, and for loads a contactor can be provided,

BR - bistable relay /memorizes the last state also without voltage/,

PR - reactors for forming artificial neutral point of the branch, which are in high-voltage networks replaced by capacitors Co wound on phase conductors /Figures 4 and 5/.

T - transformer, whose primary is carried out as reactor PR, indicating the displacement of neutral point of the network in case of an earthfault; in high-voltage networks it is replaced by resistance connected to the neutral point, carried out with capacitors Co, and it gives datum Z1 /Figures 4 and 5/.

ST - current voltage transformers with open circuit secondary which give datum "s" in voltage form,

OT - core balance transformers for indication of fault current /earthfault/, which gives datum Z2 in form of "u" voltage as a function of earth leakage current.

ASIC - Application Specially Integrated Circuit with the exactly determined functions described on the basis of the protection algorithm. For methane pits, the input data from the network are conducted through an additional hybrid unit which ensures intrinsic safety of the input signals,

Rz - the main earther of the network (protective earthing),

Rp - auxiliary earther for the monitoring of Rz earther resistance.

The possibility of not switching off of the special load "SL" in case of an earthfault is particularly evident in IT networks, where the protection from indirect contact is provided, while the protection against direct contact is ensured by other measures, e.g. mechanical obstacles. This is particularly important for e.g. fans, electronic processing, telemetry or communication devices, if they have no stand-by power supply source, and whose disconnection would cause additional danger and indirect damages.

The evident usefulness of tripping the first earthfault in case of other loads reduces considerably the probability of short circuit, and consequently, fire, and explosion risk. This can be very efficiently achieved with IT unearthed network system with permanent monitoring of insulation resistance against earth. Short circuit can be practically nearly completely eliminated, if the installation is carried out with special cables with semi-conductive sheathing around the phase conductors with incorporated

uninsulated protective conductor in the semi-conductive layer, or with screening of the phase conductors with earthed metallic strand. In such case, every damage of the phase conductor insulation results in an earthfault through the semi-conductive layer or metallic stranding. In this way the occurrence of short circuit and its consequences, e.g. ignition of methane or coal dust, are prevented. To eliminate the possibility of short circuit occurrence is particularly important in hazardous areas endangered by fire and explosion e.g. in methane pits, storages of explosives or spaces endangered by fire hazard. Besides, if a short circuit occurs before an earthfault, in some flameproof enclosure, the earthfault will occur in case of establishing a stable electric arc, which will surely activate the earthfault protection regardless the reliability of actuation of short circuit protection.

CHECKING AND SWITCHING ON

In unearthed IT three-phase network system, e.g. up to 1,200 V, neutral point is "earthed" only through the d.c. measuring voltage supply source /of high resistance/. The permanent monitoring of insulation resistance against earth is performed with d.c. voltage, e.g. 50 V for low voltage network /and 100-[illegible] V for higher voltages/ which constantly controls the switching of the network. Before switching-on, the insulation resistance of the branch is checked via reactors PR up to the first distribution busbars /L1, L2, L3/. Only with the favourable voltage n1 and voltage n the main circuit breaker B1 can be switched-on provided the following logic conditions with:

- that the previous tripping was not effected by the signal "i" through the bistable relay BR,
- that, by signal kz, the correct protective earthing resistance Rz has been ascertained, which form, together with Rp and Rk, the characteristic resistance.

If the previous tripping occurred due to a short circuit with signal "i" by activation of BR relay, reclosing will be possible only if, the resistance Rk is connected to earth on any distribution branch, which will through the input signal k, reset the BR relay via "r", and enabling closing via "u" main circuit breaker B1. If the short circuit occurred on the busbars of the first distribution or on the connection to the busbar this can be done only manually after a warning that short circuit is eliminated.

By closing the circuit breaker B1, the control voltage of power supply IN is switched off, and the busbar insulation is controlled by d.c. voltage of power supply source IN1. In this way all the protection devices are supplied and activated, and the checking of the insulation resistance of each branch after the busbar without network voltage is started by d.c. voltage from "m" power supply sources. The insulation resistance is measured via the primary of transformer T and reactors PR. The circuit breaker or contactor /SU" units/, can be switched-on under the following conditions:

- that the previous tripping was not caused by short circuit via input signal "s" and by activation of bistable relay BR i.e. "i" signal,
- that the branch insulation is sufficient, which is shown by the sufficient voltage of n signal,
- that signals z1 and z2 are not of the same phase in the moment of closing,
- if the previous tripping was caused by short circuit by activation of BR relay, reclosing will be possible only if k-signal of characteristic resistance Rk in some panel of the following distribution appears at the input, which signal will reset BR relay via "r". For the load panel, resetting is possible only by button "r".
- for load panel "ML", if is confirmed the correct earthing of load's enclosure by signal "kz"

which can be applied to all loads, but must be applied to the loads which are moved during operation e.g. continuous miner.

In case of a special load "SL" which is not tripped by earthfault, reclosing is possible if:

- the insulation of "SL" is correct, which is confirmed by signal "n",
- the earthing of "SL" is correct, which is confirmed by signal "kz" and,
- if the previous tripping was not due to short circuit by activation of BR relay by signal "s" and tripping signal "i".

PROTECTION ACTUATION IN CASE OF FAULTS

/1/ Overload

Overload is shown by signal "s" with voltage up to the adjusted threshold, when tripped the corresponding switching device. It can be also "memorized" and reset manually or, after the disappearance of the overload, automatically, as desired or as designed. By steploss adjusting of the current for activation threshold, and the slope of the curve, undisturbed starting is enabled, and overload protection is ensured.

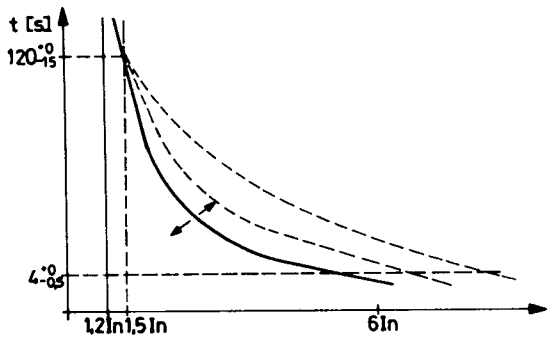


Figure 7. Adjustable characteristic of overload protection

/2/ Earthfault /Indirect Contact/

For unearthed IT networks /Figure 1/ an earthfault is manifested by disappearance of the d.c. input signal "n_1", which is only signalled in the main panel and the panel of "SL". With disappearance of input signal "n" will occur also on all other panels in the network, but tripping will occur only in case if a.c. signal of the same phase appears on z1 and z2. This means that an earthfault on that branch has occurred.

In case of an earthfault, in the "SL" panel no tripping will occur, if signal "kz" indicates correct earthing. The signalling will point to the fault to be eliminated on "SL" load, as soon as possible. First earthfault means potential danger of short circuit through another earthfault in another phase, for which the protective conductor must be dimensioned. For earthed IT networks /Figure 2/ there is no d.c. signal "n" except before closing, and it disappears by switching-on. An earthfault in a network is manifested by a.c. signals of the same phase z1 and z2. In the main field the earthfault will be only signalled, and tripping will occur only if it assumes short circuit values, and, as such, the fault will be examined under "short circuit". In "SL" panel, the fault of signals z1 and z2 will be only signalled, and tripping will occur only if the input signal "kz" indicates to high earthing resistance, or if the earthfault current is at the same time a short circuit. We must point out that according to Figure 2, tripping due to earthfault and short circuit is signalled by "i" signal through the bistable relay BR, and that only in cases of short circuit, also tripping of the main panel can occur. However, if the earthfault protection /or short-circuit protection/ is activated, and the switching device of one of fields: D or L, or ML panels is switched off, then one of Rk resistances is connected through BR relay to earth, and the bus "k" transmits this datum to ASIC logic of the main field which closes automatically immediately energizing the busbars. If an earthfault occurs on branch L or ML, its closing can be done only by manual reset by means of a

pushbutton or by mechanical reset of the bistable relay BR. If an earthfault occurs on branch D which supplies the following distribution in the network, reclosing can occur only if through "k" signal the information is obtained that in the following distribution some of the characteristic resistances Rk is connected to earth, which means that an earthfault or short circuit occurred on the following branch or load of the following distribution in the direction of energy flow.

/3/ Earthfault /Direct Contact/

It is considered that this earthfault occurs with the main resistance of order 1 - 10 kOhms, which means that it is not a direct connection of phase and earth. Such contact will cause signal z1, and it is enough that datum z2 records fault current of 30 mA, which means that for voltage of e.g. 500 V, it will react to resistance of 10 kOhm and lower. Unnecessary trippings due to disturbance are not expected because the displacement of neutral point on transformer T due to e.g. external influence /atmospheric discharge, static electricity etc./ will not occur simultaneously.

Acting of protection is identical as described under /2/ earthfault - indirect contact. Special load "SL" which does not trip in case of earth-fault, does not assume the possibility of direct contact, which must be secured by special measures of closing and interlocking. For unearthed IT network systems, this protection is particularly suitable because it does not react only to the level of d.c. signal "n" but also to its derivation on time, so that it will act quickly or momentarily to the quick drop of network insulation resistance against earth independently on RC network constant which retards this decrease of d.c. signal. Namely, if a man touches a phase, the insulation resistance of the network against earth drops momentarily, and the derivation of signal or dR/dt will be reliable datum for quick tripping together with z1 and z2, which will be for this branch of the same direction. Consequently, with tripping time smaller than 100 ms, it can be considered a very reliable protection against direct contact /IEC 479/. After voltage failure, reclosing is possible only if the contact with phase has disappeared and the insulation is correct. Reclosing is conditioned by checking the correct insulation resistance before closing the switching device. Automatic reclosing is conditioned by small time delay e.g. 2 s in order to discharge the network voltage according to the network RC constant.

/4/ Short Circuit

Acting of protection in case of short circuit is identical for all network systems and the tripping remains selective regardless the difference of values of short circuit currents of different busbars. It also enables the use of a contactor as a switching device in the network, on condition that at least one main circuit breaker exists in the network, which will reliably break the short circuit, and whose breaking current is higher than the maximum possible short circuit current in the network. In other words from distribution D, according to the figures, the network can contain contactors instead of circuit breakers, because, if the contactor does not break the short circuit current within a few ms, an electric arc will appear for a short time on the open contacts of the contactor, and it will be interrupted by the main circuit breaker. After the activation of protection in all panels through which the short circuit current flows, all bistable relays BR will be switched over through signal "s" and "i" connecting the characteristic resistor Rk to earth, which means that the short circuit current has passed through the panel and the short circuit has not occurred up to that point. Voltage reclosing will occur when k bus obtains "k" signal that one of the characteristic

resistances "Rk" to earth is connected. If the distribution with busbars is connected via D panel, and a short circuit occurred on one of its branches, signal "k" in the field will momentarily automatically reset the BR relay and switch on the branch. In the same way, the main panel will be switched on, if it has been tripped due to short circuit on any branch. It should be noted that the conducting of signal "k" for the first distribution is not a problem because the main panel and the first distribution are always in the immediate vicinity, but the conducting along the whole network of distributions on various locations requires an additional insulated pilot conductor in the cable for "k" signal. However, nowadays, special cables for mines have an insulated conductor for pilot circuit, which can be used for this purpose.

/5/ Earthing Integrity

Since by pilot circuit of characteristic resistance Rk, the correctness of earthing of the protective conductor "PC" and of the earthing of the load casing to the protective conductor can be controlled the special load "SL" is not necessary to be tripped in case of an earthfault only, because the touch voltage of indirect contact is harmless, e.g. 50 V. This checking is effected in the main panel in such a way the earthing of the protective conductor "PC" and the resistance of the main earther "Rz" are checked through one auxiliary earther by signal "kz". For unearthed IT networks the resistances "Rk' + Rp + Rz" constitute the characteristic resistance "Rk" adjusted in such a way that it can slightly deviate downwards, but not more than e.g. 50-100 Ohm upwards. A case of an interruption, or increased resistance, or direct earthing of "kz" conductor indicates the desintegrity of earthing or the lack of control of earthing integrity, which is of special importance for "SL" loads remaining in operation also during the earthfault on them. In such case the touch voltage can become dangerous, as well as for ML consumers at which the interruption of earthing is more probable than in case of other loads which are not movable under voltage.

/6/ Insulation Against Earth

It is based on the principle that for the unearthed IT networks in the main panel the d.c. voltage from power supply source IN1 is connected via reactors PR to the transformer neutral point T.S., and a d.c. voltage parallelly connected with the insulation resistances of phases against earth is conducted into ASIC logic, as datum n1. Before closing the circuit breaker B1 of the main panel this will be the checking of the insulation of the power supply source /transformer secondary/.

After closing, of main breaker B1, it will monitor the whole supply network under voltage. This means that when the whole network is energized, the ohmmeter will indicate the leakage current of the whole network and, consequently, also the insulation resistance. At the same time this monitoring d.c. voltage will appear, on the neutral point of reactors PR of all the branches depending on the total insulation resistance of the network, and for this reason, the input n must be high-ohmic the same as the internal resistance of d.c. voltage supply source. Dropping of the insulation resistance anywhere in the network will be manifested by the reduction of all voltages "n_1" and "n" in the whole network, which will be one of the data for tripping, or only signalling if the insulation resistance drop does not endanger the safety by too high touch voltage.

MICROLOGIC CONTROL-SIGNALING UNIT "ASIC"

The operation of micrologic unit ASIC is described, according to its function for universal use for protection against all the mentioned kinds of faults. Each input function is treated in accordance with

the protection requirement and the output function can be used for tripping, closing, resetting, but it is always signalled.

The micrologic unit meets all the described functions, and its role depends on the project or the design of corresponding switching field. The design of the micrologic unit is hybrid-type with the use of standard I.C. logic circuits, which is, special for this purpose, designed with a fixed processing programme. All the described functions are performed by one hybrid micrologic unit of dimensions 45 x 55 x 12.5 mm, and for methane pits the intrinsic safety of the input data is secured by the interface in the additional hybrid circuit of equal dimensions, while all together is placed in a box with connectors of dimensions 115 x 90 x 50 mm, shown in Figure 8, where the hybrid micrologic circuit can be also seen. Together with the power supply sources, the electronic protection block has dimensions: 115 x 90 x 100 mm.

CONCLUSION

IT system unearthed network is used in Yugoslavian mines for more than 30 years, and quick-acting earthfault protection cca 15 years and experience has shown excellent results in relation to direct and indirect contact protection by earthfault and particularly exposed result in disappearance of short circuit. However introducing quick-acting earthfault protection increased disturbances of sometime unnecessary disenergizing of power supply but with interactive processing by microelectronic of several input data for switching off in case of fault all unnecessary disenergizing disappeared and selection of fault is obtained.

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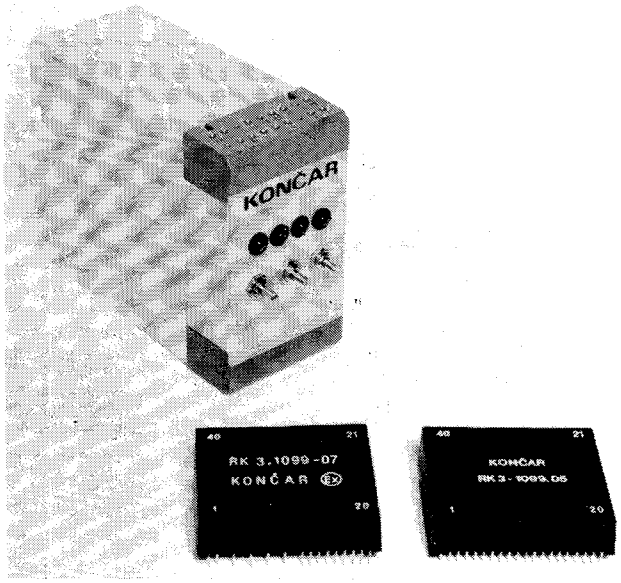


Figure 8. Hybrid microelectric ASIX-s and control unite for electric protection.

PROTECTION ELECTRIQUE PAR CIRCUITS INTEGRES MICROELECTRONIQUES POUR DES APPLICATIONS SPECIALES

Nenad Marinovic

Après de nombreuses années de recherches, toutes les conditions de protection électronique des équipements électriques des mines à ciel ouvert et des mines de fond, ainsi que le contrôle, ont été combinés dans un ASIC microélectronique à utilisation généralisée et dans deux ASIC destinés aux mines de fond à méthane, pour des réseaux hors de terre (IT) et mis à la terre (TN), y compris:

- protection contre les surcharges, les dérangements dûs aux mises à la terre accidentelles et les courts-circuits,
- protection contre les tensions de contact trop élevées pour les contacts directs (L.V.) et indirects (H.V.),
- contrôle de l'isolation pour les réseaux IT,
- vérification de l'intégrité des mises à la terre des charges et des conducteurs à la terre.

Leur excellente protection pour tous les niveaux de tension, leur application universelle et le remplacement rapide du module électronique leur permettent d'être très économiques, particulièrement dans les boîtiers SXD. En outre, grâce à leurs câbles blindés, ils minimisent le risque électrique dans les mines de fond.

ELEKTRISCHE SICHERHEIT DURCH ANWENDUNG VON MIKROELEKTRONISCHEN INTEGRIERTEN SCHALTKREISEN FÜR SPEZIELLE ANWENDUNGEN.

Nenad Marinovic

Nach langjähriger Forschung können alle Anforderungen für die elektronische Sicherung von elektrischen Ausrüstungen in Tagebauen und Gruben für ungeerdete (IT) und geerdete (TN) Netze in einem mikroelektronischen ASIC für allgemeine Anwendung und in zwei ASIC für gashaltige Gruben zusammengefaßt werden. Darin enthalten sind die folgenden:

- Überlast-, Erdschluß- und Kurzschlußüberwachung
- Schutz vor zu hoher Kontaktspannung bei direktem (L. V.) und indirektem Kontakt (H. V.),
- Isolationsüberwachung bei IT Netzen
- Test der vollständigen Erdung von Lasten und Erdungen.

Hohe Zuverlässigkeit des Schutzes bei allen Spannungsgrößen, universelle Anwendbarkeit und leichte Austauschbarkeit der elektronischen Module machen diese Art des Schutzes wirtschaftlich, besonders in explosionsgeschützten Gehäusen. Die Verwendung von abgeschirmten Kabeln reduziert die durch Elektrizität entstehenden Gefahren auf ein Minimum.

ИСПОЛЬЗОВАНИЕ МИКРОЭЛЕКТРОННОЙ ИНТЕГРАЛЬНОЙ СИСТЕМЫ ДЛЯ ЗАЩИТЫ ЭЛЕКТРООБОРУДОВАНИЯ

Ненад Маринович

В процессе многолетних исследований были разработаны спецификации для электронной защиты электрооборудования в карьерах и открытых месторождениях, которые включают заземленную и незаземленную системы. Эти системы состоят из следующих функциональных элементов:
- зашита от перегрузок и коротких замыканий;
- защита от слишком высокого контактного напряжения в случае прямого (низкое напряжение) и непрямого (высокое напряжение) контакта;
- изоляция для незаземленной сети;
- проверка качества работы по заземлению.

Эти элементы объединены в одну микроэлектронную схему общего пользования и в две микроэлектронные схемы для метаноопасных шахт. Высокая надежность защиты при любых уровнях напряжения, возможность применения в любых ситуациях и легкость замены электрических элементов делают предлагаемую систему защиты экономически эффективной. Применение защищенных кабелей сводит к минимуму опасность от электрического тока в карьерах.

用微电子集成电路的电力保护

能纳特　马林诺维支

经过多年的研究，所有露天矿和井矿里具有接地（IT）和不接地（TN）网路的电力设备用电子保护的要求和控制一并在一个微电子 ASIC（一般用）或两个 ASIC（甲烷井矿用）中实施．这些要求包括：
- **过载，接地故障，和短路的保障**
- **直接接触（L.V.）和间接接触（H.V.）接触电压过高的保障**
- **IT 网路绝缘控制**
- **载荷和地线接地完整性的保障**

由于在不同高低电压值有很大的可靠性，电子组件更换的方便和应用范围的广大，使得这些保障非常经济，特别用在 EXd 护栏上．此外籍屏蔽电缆的应用可使井矿中由于电所产生的危害大为减少．

ПРОБЛЕМЫ ЭЛЕКТРОБЕЗОПАСНОСТИ В РУДНИЧНЫХ СЕТЯХ С ВЫСШИМИ ГАРМОНИКАМИ В НАПРЯЖЕНИИ

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РЕЗЮМЕ

Высшие гармоники могут в значительной степени усложнить условия безопасной эксплоатации электрических установок в шахтах и рудниках. В докладе рассматриваются проблемы безопасности в рудничных электрических установок с напрежением до 1000 V в системах с изолированной нейтралью. Анализированы электробезопасные качества однофазных и трехфазных сетей. На базе полученых уравнений и экспериментально определенных данных о частотном и амплитудном спектре высших гармоников в реальных сетях доказывается повышенная опасность от электрического тока как источник эксплозий и пожаров и для человека.

Дискутируется эффективность защит от электрического тока при наличий высших гармоников в напряжении, методы измерения и контроля изоляции и проблемы эффективной статической и динамической компенсации емкостной составляющей в токах замыкания на землю.

ВВЕДЕНИЕ

Высшие гармоники, возникающие в сетях с низким напряжением или проникающие в них, являются результатом расширенного применения все более мощных нелинейных электрических нагрузок-нерегулируемые и регулируемые преобразовательные установки, газоразрядные источники света, дуговые печи и др. Мощность преобразовательных установок в металлургии достигла GVA. На рудниках и шахтах применяется тиристорный привод с единичной мощностью до несколько MVA. В наиболее распространенных трехфазных мостовых схемах возникают высшие гармоники с частотой, кратной в 5, 7, 11, 13, 17, 19 раз основной частоте и амплитудами, достигающие десятки %, а в гармониках более высокого порядка (кратность к= = 30 - 40) максимальные значения находятся в границах до 2 % амплитуд основной гармоники. В осветительных установках с ртутными, люминесцентными и галогенными лампами удельные значения высших гармоник следующие: для третьей гармоники-до 21 %, для пятой - 9.5 %... для 23 и 25 - до 10 %... для 31 - 8.6 %. Экспериментально исследованы амплитудный и частотный спектры высших гармоник в электрической установке рудничного подъема с приводом по системе генератор-двигатель с мощностью 315 кW. На рис. 1 показаны форма напряжения /а/ и спектр гармоник /б/ в сети с переменным напряжением, а на рис. 2 - форма /а/ и спектр гармоник /б/ в сети с постоянным током. Высшие гармоники создают неприятные проблемы: значительный рост потерь в электрических машинах и трансформаторах, уменьшение точности при учете электрической энергии и релейной защиты,

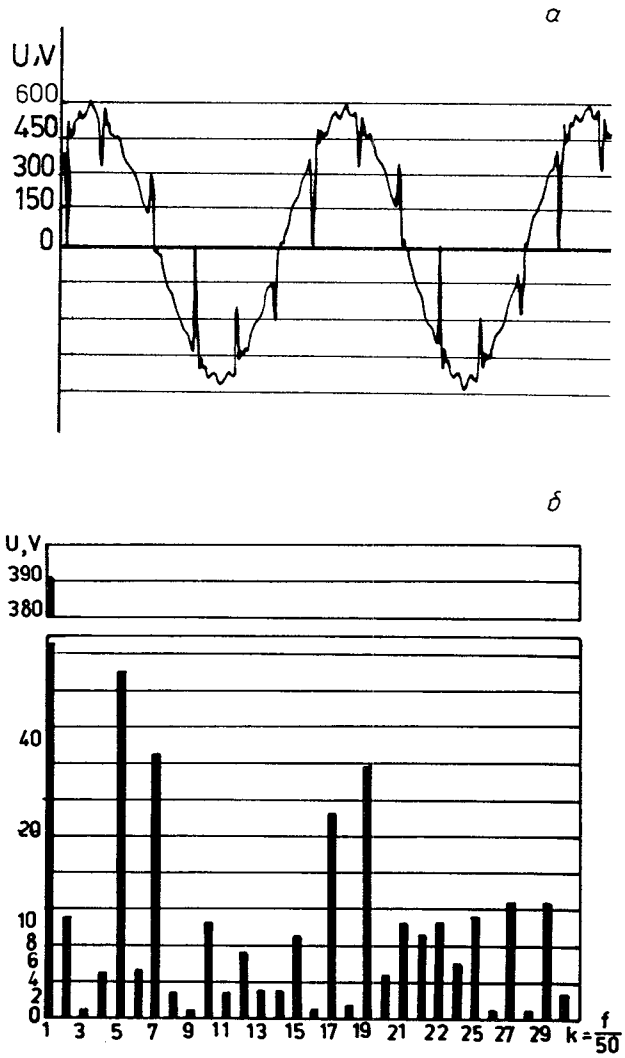


Рис. 1

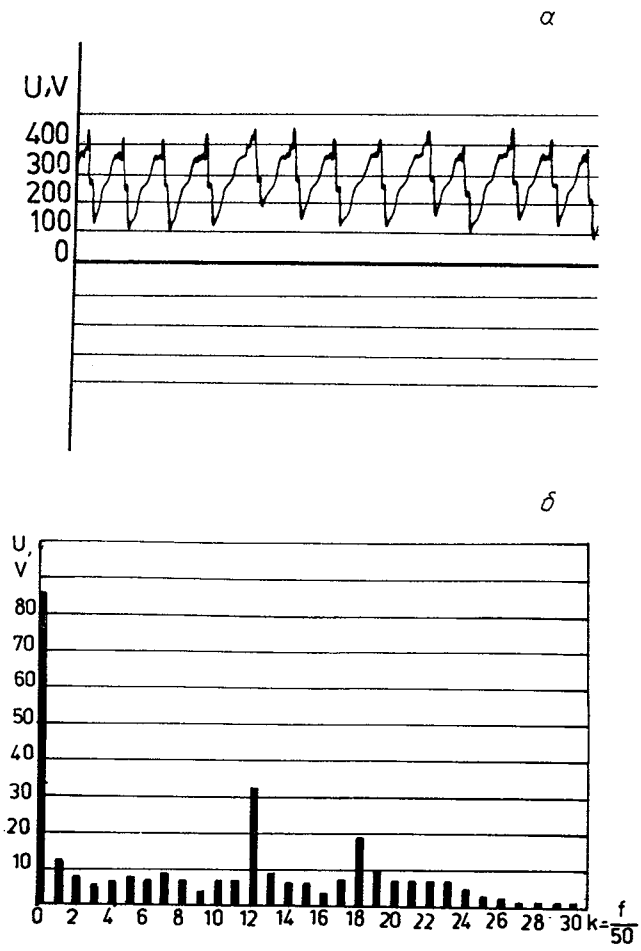


Рис. 2

усложнение компенсации реактивной нагрузки и режима работы конденсаторных батарей и т.д. К сожалению, высшие гармоники ставят серьезные проблемы и в области электробезопасности. Они выражены особено сильно в системах с изолированной нейтралью, которые единствено допускаются в рудниках много европейских стран, включительно и НР Болгарии.

ЭЛЕКТРОБЕЗОПАСНОСТЬ РУДНИЧНЫХ СЕТЕЙ

Электробезопасные качества сетей с изолированным по отношению к земле источником определяются в большой степени их изолационной проводимостью по отношению к земле. При высоких значениях изоляционного сопротивления, ток человека при однополюсном касании и ток утечки могут быть существено ограничены до безопасных значений. Эта возможность легко доказывается теоретически, хотя и осуществляется на практике в различной степени. Оказывается, что высшие гармоники создают дополнительные трудности при осуществлении этого принципа. Эти трудности определяются нарастанием тока

утечки и тока человека при однополюсном касании. Этому эффекту больше всего способствует емкость сети по отношению к земли, чья проводимость сильно растет для высших гармоник.

В трехфазных системах /рис. 3/ с несинусоидальным периодическим напряжением ток утечки I_y и ток человека $I_ч$ зависят от дискретного спектра гармоник и от соотношения активных и емкостных составляющих проводимостей фаз по отношению к земли Y_A , Y_B , Y_C . Ток человека и ток утечки обуславливаются только напряжениями с прямой и обратной последовательностью, которые генерированы в результате несинусоидальной формы фаз напряжений.

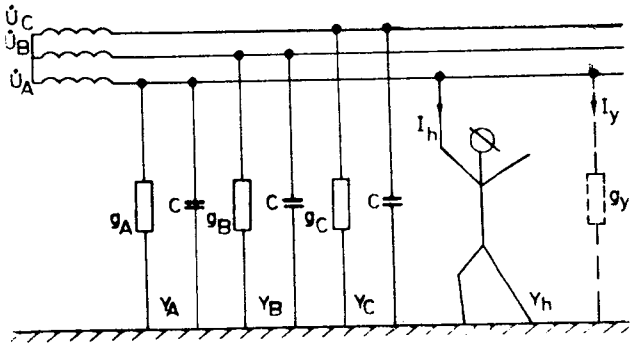


Рис. 3

Известно, что напряжения с прямой последовательностью имеют частоту, которая кратна к = 3n + 1, а напряжения с обратной последовательностью - частоту, которая кратна к = = 3n + 2. Их амплитуды дефинированы коэффициентом кратности $K_K = \frac{U_{km}}{U_m}$, который является отношением максимальных значений несинусоидального напряжения и напряжения соответствующей гармоники.

Например, при симметрии изоляционных проводимостей, ток утечки определяется зависимостью:

$$I_y = 3g_y \frac{U_m}{\sqrt{2}} \sqrt{\sum_{\substack{n=0\\k=1}}^{\substack{k=3m+1\\n=m}} K_k^2 \left| \frac{Y_k}{3Y_k + g_k} \right|^2 + \sum_{\substack{n=0\\k=2}}^{\substack{k=3m+2\\n=m}} K_k^2 \left| \frac{Y_k}{3Y_k + g_y} \right|^2}$$

а ток человека:

$$I_h = 3 \frac{U_m}{\sqrt{2}} \sqrt{\sum_{\substack{n=0\\k=1}}^{\substack{k=3m+1\\n=m}} K_k^2 \left| \frac{Y_k Y_h}{3Y_k + Y_h} \right|^2 + \sum_{\substack{n=0\\k=2}}^{\substack{k=3m+2\\n=m}} K_k^2 \left| \frac{Y_k Y_h}{3Y_k + Y_h} \right|^2}$$

Количественное представление об ошибке, допускаемой при пренебрежении высших гармоник - если ток утечки и ток человека определяются только по напряжению с основной частотой /к= 1/, получается из кривых на рис. 4 и рис. 5. Эти зависимости относятся к экспериментально определенным напряжениям с дискретным спектром на рис. 1 б. В зависимости от соотношения активной и емкостной проводимости, действительный ток утечки может быть до 2,5 разов выше, а ток человека до 2 раз выше, чем вычисленный без учета высших гармоник ток.

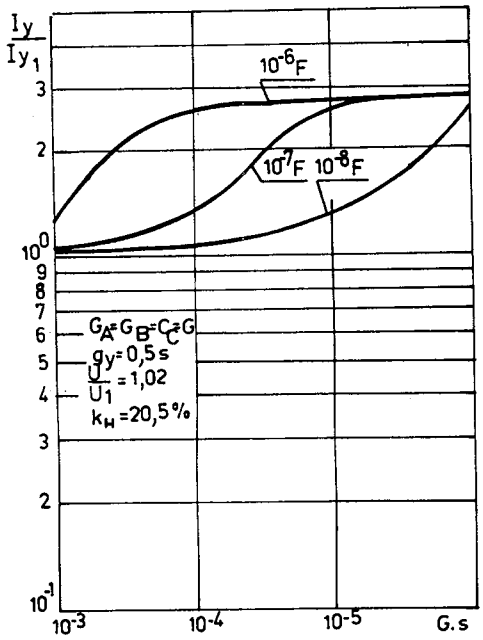


Рис. 4

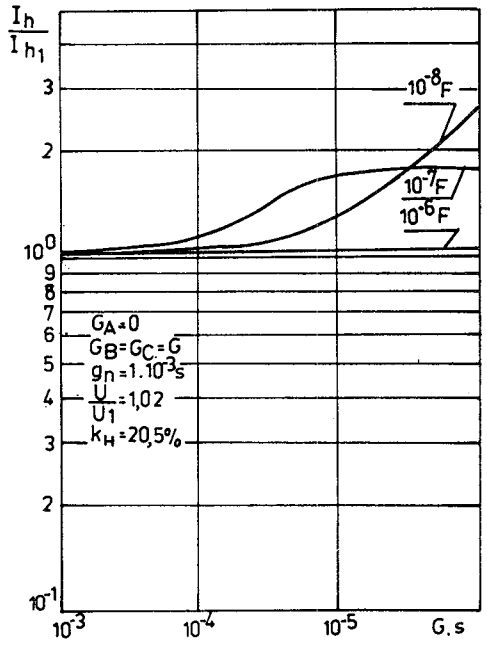


Рис. 5

Это обстоятельство, естественно, сказывается и при определении безопасных значений изоляционного сопротивления.

Критическое значение активной изоляционной проводимости, которое обеспечило бы безопасность человека при однополюсном касании, определяется условием $I_h \leqslant I_\delta$, т.е. ток человека не должен превышать принятое безопасное значение I_δ при наиболее неблагоприятном сочетании изоляционных проводимостей. Для ситуации на рис.3:
при $g_A=0$, $g_B=g_C=g_{кр}$ $Y_h=g_h$
доказывается зависимость:

$$I_\delta \geqslant I_h = \frac{U_m}{\sqrt{2}} 3g_h \sqrt{\sum_{k=1}^{m} K_k^2 \frac{g_{кр}^2+k^2\omega^2c^2}{(2g_{кр}+g_h)^2+9k^2\omega^2c^2}}$$

где к может получать значения от 1 до целых чисел, которые соответствуют номеру гармоники, с исключением чисел, кратных 3, т.е. исключаются гармоники напряжения, образующие систему с нулевой последовательностью.

Критическое значение изоляционного сопротивления можно определить и по отношению к воспламеняющей способности тока утечки, если принять что существуют минимальные значения тока, мощности и энергии, которые являются необходимыми /хотя и недостаточными/ условиями возбуждения пожара в рудничной атмосфере.

Наиболее неблагоприятный случай возникновения взрыва или пожара, вызванных током утечки, получается при соприкосновении фазового провода и оболочки заземленного сооружения. Ток утечки ограничится проводимостью остальных двух фаз по отношению к земли и сопротивлением заземления. Последнее регламентировано, следовательно существует такое критическое значение изоляции, которое ограничит ток утечки под вероятным воспламеняющим значением.

Максимальное значение тока утечки получается в условиях $g_y=g_з$ и $g_в=g_c$: $g_A=0$.

Функциональная связь между критическим значением проводимости /сопротивления/, параметрами тока утечки и дискретным спектром составных гармоник напряжения следующая:

$$I_y \leqslant I_{в\delta} = \frac{U_m}{\sqrt{2}} g_з \sqrt{\sum_{k=1}^{m} \frac{K_k^2(g_{кр}^2+k^2\omega^2c^2)}{(2g_{кр}+g_з)^2+9k^2\omega^2c^2}}$$

При двухпроводных сетях /рис. 6/ несинусоидальное напряжение может содержать и постоянную составную часть /нулевая гармоника - к = 0/. Таким является случай в сетях постоянного тока. Воспринятая оценка электробезопасных качеств подобной сети только по постоянной составной части напряжения неточна, особенно когда используются преобразователь-

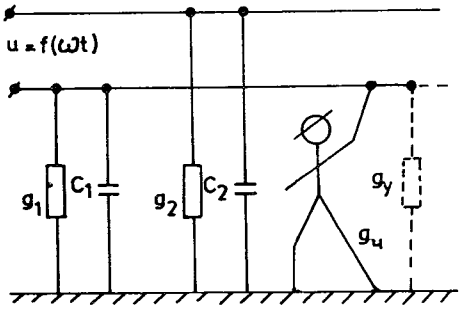


Рис 6

ные установки. Ток человека и ток утечки должны определяться при учете высших гармоник. Так например, при симметрии изоляционной проводимости по отношению к земле, они определяются уравнениями:

$$I_h = U_m g_h \sqrt{\left(\frac{k_0 g}{2g+g_h}\right)^2 + \frac{1}{2}\sum_{k=1}^{m} \frac{k_k^2(g^2+k^2\omega^2c^2)}{(2g+g_h)^2+4k^2\omega^2c^2}}$$

$$I_y = U_m g_y \sqrt{\left(\frac{k_0 g}{2g+g_y}\right)^2 + \frac{1}{2}\sum_{k=1}^{m} \frac{k_k^2(g^2+k^2\omega^2c^2)}{(2g+g_y)^2+4k^2\omega^2c^2}}$$

При сети постоянного тока, полученного тиристорным выпрямителем и с формой и дискретным спектром напряжения, которые показаны на рис. 2 а и б, ошибка, вызванная пренебрежением высших гармоник, может достичь сотни %. По отношению к току человека это иллюстрируется рис. 7, а по отношению к току утечки - рис. 8. Они показывают кратность соответствующих токов по отношению к значениям, вычисленных только по постоянной составной части. Для параметров конкретной исследованной сети действительные токи с учетом высших гармоников, могут оказаться от 1,4 до 4 раз выше, чем токи, определенные только по постоянной составляющей /в зависимости от величины активного сопротивления/.

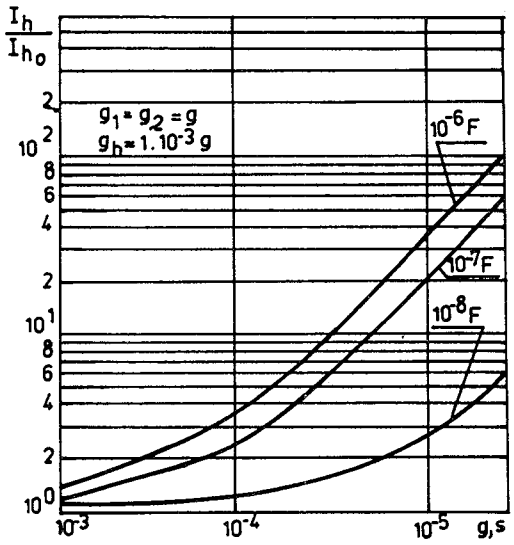


Рис. 7

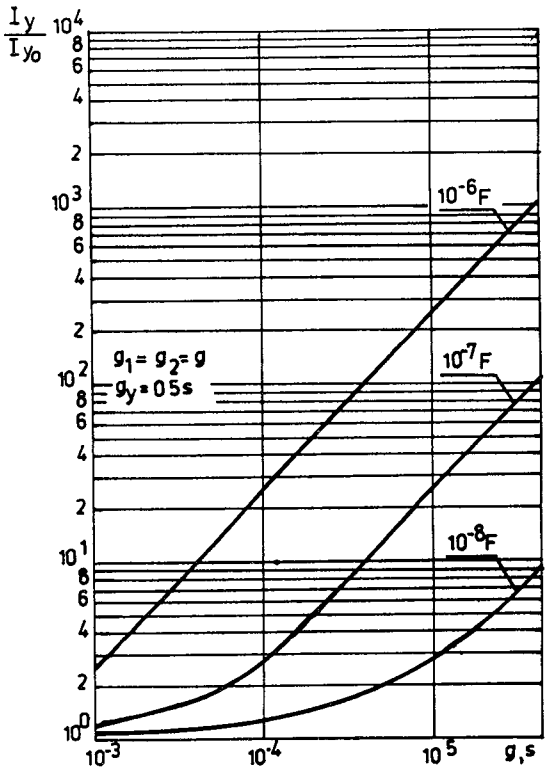


Рис. 8

Критические изоляционные сопротивления также зависят от дискретного спектра гармоник: по отношению к безопасному току человека при $g_1 = 0$; $g_2 = g_{кр}$; $C_1 = C_2 = C_3$:

$$I_\delta \geq I_h = U_m g_h \sqrt{\left(\frac{k_0 g_{кр}}{g_{кр}+g_h}\right)^2 + \frac{1}{2}\sum_{k=1}^{m} k_k^2 \frac{(g_{кр}^2 + k^2\omega^2 c^2)}{(g_{кр}+g_h)^2 + 4k^2\omega^2 c^2}}$$

и по отношению тока, который может вызвать пожар или взрыв рудничной атмосферы при $g_1 = 0$; $g_y = g_3$; $g_2 = g_{кр}$; $c_1 = c_2 = c$:

$$I_{\delta\delta} \geq I_y = U_m g_3 \sqrt{\left(\frac{k_0 g_{кр}}{g_{кр}+g_3}\right)^2 + \frac{1}{2}\sum_{k=1}^{m} \frac{k_k^2 (g_{кр}^2 + k^2\omega^2 c^2)}{(g_3+g_{кр})^2 + 4k^2\omega^2 c^2}}$$

Из сказанного можно сделать заключение, что электробезопасные качества рудничных сетей значительно ухудшаются под влиянием высших гармоник в напряжении. Степень этого усложнения зависит от дискретного спектра /амплитудного и частотного/ гармоник, от соотношения между активной и емкостной составными частями изоляционной проводимости, от сопротивления сети заземления и от неблагоприятности ситуации, определенной проводимостями отдельных фаз и места соприкосновения или утечки. Пренебрежение влияния высших гармоник допустимо только после количественной оценки любого конкретного случая.

ИЗМЕРЕНИЕ И КОНТРОЛЬ ИЗОЛЯЦИОННЫХ ПАРАМЕТРОВ СЕТЕЙ

Известные методы измерения и контроля изоляционных параметров сетей со синусоидальным напряжением или с постоянным напряжением, в которых носителем информации является утечка, в общем случае неприменимы при напряжениях, которые содержат гармонические составные части.

Высшие гармоники в напряжении влияют существенно на емкостную проводимость и в меньшей степени - на активную проводимость сети по отношению к земли. Это определяет и целесообразность введения параметров "спектральная проводимость" и "спектральное сопротивление" изоляции в сети с высшими гармониками в напряжении. Они дефинируются как отношение эффективных значений несинусоидальных токов и напряжений:

$$y^s = \frac{I}{U} \qquad Z^s = \frac{U}{I}$$

После введения коэффициентов кратности и выражения эффективных значений несинусоидального тока и напряжения при помощи эффективных значений их составных гармоник, упомянутые выражения получают следующий вид:

$$y^s = \sqrt{\frac{\sum_{k=0}^{m} k_k^2 y_y^2}{\sum_{k=0}^{m} k_k^2}} = \sqrt{\frac{\sum_{k=0}^{m} [k_k^2 (g_k^2 + k^2\omega^2 c^2)]}{\sum_{k=0}^{m} k_k^2}}$$

$$Z^s = \sqrt{\frac{\sum_{k=}^{m} k_k^2}{\sum_{k=0}^{m} \left(\frac{k_k}{Z_k}\right)^2}} = \sqrt{\frac{\sum_{k=0}^{m} k_k^2}{\sum_{k=0}^{m} \frac{k_k^2}{r_k^2 + \left(\frac{1}{k\omega c}\right)^2}}}$$

Дефинированные таким образом спектральные проводимости и импеданс характеризуются только модулем и ими можно выражать только эффективные значения несинусоидальных токов и напряжений.

Спектральные активная проводимость и сопротивление определяются уравнениями:

$$g^s = \sqrt{\frac{\sum_{k=0}^{m} k_k^2 g_k^2}{\sum_{k=0}^{m} k_k^2}} \qquad r^s = \sqrt{\frac{\sum_{k=0}^{m} k_k^2}{\sum_{k=0}^{m} \frac{k_k^2}{r_k^2}}}$$

а спектральные емкостная проводимость и емкостное сопро-

тивление определяются уравнениями:

$$B^S = \omega C \sqrt{\frac{\sum_{k=0}^{m} K_k^2 K^2}{\sum_{k=0}^{m} K_k^2}} \qquad X^S = \frac{1}{\omega C} \sqrt{\frac{\sum_{k=0}^{m} K_k^2}{\sum_{k=0}^{m} K_k^2 K^2}}$$

Когда измеряется или контролируется состояние изоляции при неизменном дискретном спектре напряжения /в зарядных устройствах с нерегулируемым напряжением, осветительных установках и во всех нерегулируемых выпрямителях с неизменной нагрузкой/ достаточно работать со спектральными изоляционными параметрами и их необходимо прямо определять.

При оценке изоляционного состояния сетей с изменяющимся дискретным спектром гармоник в напряжении возможны два подхода. Первый - определить изоляционное состояние спектральными проводимостями для наиболее неблагоприятного по отношению к безопасности спектра и второй - определить экспериментальным путем активное изоляционное сопротивление /или проводимость/ и емкость любой из гармоник, например основной, и при помощи вышеуказанных зависимостей определить соответственные спектральные проводимости для возможных дискретных спектров.

Следовательно, практический интерес вызывают как методы измерения и контроля спектральной проводимости, так и методы, с помощью которых можно измерять и определять сопротивление и емкость сети для целесообразно подобранной гармоники.

Для измерения проводимости составной гармоники применяется метод ампер-вольтметра, причем в схему включается узкополосный фильтр с импедансом для частоты измерения $Z_{фк}$ - 0.

Для измерения спектральной проводимости изоляции применяется та же схема измерения, но с обязательным условием, что приборы должны измерять эффективные значения тока и напряжения при несинусоидальной форме.

Наибольшие возможности измерения изоляционных параметров дает схема на рис. 9. Напряжение $U_{кхх}$ измеряется в режиме холостого хода, а ток $(U_{ккз})$ - в режиме однофазового короткого замыкания для любой гармоники

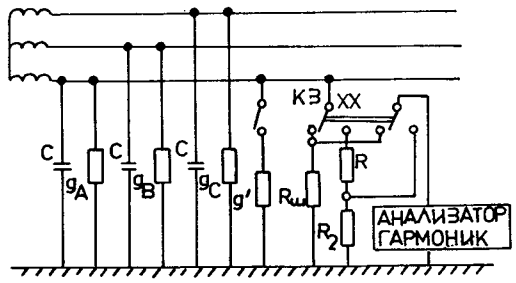


Рис. 9

в отдельности. Спектральная проводимость вычисляется по формуле:

$$y_\Sigma^S = \frac{R_2}{(R_1 + R_2) R_ш} \sqrt{\frac{\sum_{k=0}^{m} U_{ккз}}{\sum_{k=0}^{m} U_{кхх}}}$$

а после включения дополнительной проводимости:

$$y_\Sigma^S = \frac{R_2}{(R_1 + R_2) R_ш} \sqrt{\frac{\sum_{k\ 0}^{m} U'_{ккз}}{\sum_{k\ 0}^{m} U'_{кхх}}}$$

Активная и реактивная спектральная проводимость определяются из зависимостей:

$$g_{\Sigma}^{s} = g_{\Sigma} = \frac{y^{s'^2} - y^{s^2} - g'^2}{2g'}$$

$$b_{\Sigma}^{s} = \sqrt{y_{\Sigma}^{s^2} - g_{\Sigma}^{2}}$$

Можно определить и емкость сети:

$$c = \frac{b_{\Sigma}^{s}}{3\omega} \sqrt{\frac{\sum_{k=0}^{m} U_{kxx}^{2}}{\sum_{k=0}^{m} (k U_{kxx})^{2}}}$$

Контроль состояния изоляции в сетях с несинусоидальным напряжением характеризуется некоторыми специфическими особенностями.

Контроль изоляции с вольтметрами, включенные между фазами и землей более неэффективен, так как емкость, которая симметрична, определяет преимущественно общую проводимость изоляции, и чувствительность контроля по отношению к активной проводимости будет низкой. Использование вольтметра, включенного между нейтралью и землей вообще недопустимо, так как последний будет реагировать на все гармоники, которые кратны 3 и образуют систему с нулевой последовательностью и вольтметр будет индикировать асимметрию без ее реального существования.

Асимметры, которые имеют ограниченное применение для контроля изоляции в сетях с синусоидальным напряжением, являются значительно более неэффективным средством контроля в сетях, содержащих высшие гармоники - они обладают более низкой чувствительностью при асимметрии в активной проводимости; необходимым условием пропорциональности напряжения измерительного элемента от тока и напряжения с нулевой последовательностью является наличие чистого активного сопротивления как у фильтра, так и у измерительного элемента.

При контроле изоляции внешним источником с переменным напряжением наиболее эффективным является использование кратной 3 частоты по отношению к частоте основной гармоники /промышленная частота/.

ЗАЩИТНЫЕ УСТРОЙСТВА ОТ ТОКА УТЕЧКИ

Высшие гармоники в напряжении влияют существенно и на эффективность защитных устройств от тока утечки. Теоретические и экспериментальные исследования позволяют систематизировать следующие соображения.

Необходимым условием точной оценки активной изоляционной проводимости устройствами предварительного контроля, является обязательная фильтрация оперативного напряжения. Таким образом устраняется как влияние составных гармоник в напряжении, так и влияние емкости на контролированную часть сети.

Эффективность вентильных схем непрерывного контроля изоляционного состояния всей сети значительно уменьшается. Ее качества можно сохранить частично, если в измерительной схеме преобладают элементы с активным сопротивлением. Вентильные схемы неприменимы в сетях с изменяющимся дискретным спектром гармоник.

Одним из наиболее перспективных является принцип непрерывного контроля с помощью внешнего источника напряжения. Пред-

посылки его применения в сетях с переменным током следующие: постоянное оперативное напряжение не должно содержить гармоник, т.е. оно должно быть качественно фильтрировано; необходимо использовать присоединительные фильтры с активным сопротивлением; необходимо применять небольшую индуктивность для дискриминации высших гармоник тока с нулевой последовательностью.

Сужены и возможности защит, построенных на принципе быстродействующего предохранительного шунтирования. Это вызывается трудностями при идентификации фазы с утечкой при использованных в настоящее время методах. Эффективность этого вида защиты сохраняется только при прямых однофазных земных соединениях. Во всех осталяных случаях ее чувствительность уменьшается. Частично это компенсируется комбинацией предохранительного шунтирования с защитой, которая контролирует изоляцию с помощью источника постоянного напряжения.

Серьезные проблемы возникают и при защитах, содержащих устройства для статической и еще в большей степени динамической компенсации емкостной составной части тока утечки. Компенсация только основной гармоники не решает полностью задачу ограничения полного тока утечки. В определенной ситуации может оказаться значительно эффективнее компенсировать некоторую из высших гармоник. Это требует нового подхода при решении оптимизационной задачи минимизации тока утечки.

В заключении надо еще раз подчеркнуть, что сети, которые содержат высшие гармоники в напряжении, действительно создают много проблем при их безопасной эксплуатации. Механическое применение предохранительных мер и средств, которые используются в сетях без гармоник, может компрометировать усилия обеспечить электротехническую безопасность.

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PROBLEMS OF ELECTRIC SAFETY IN MINE NETS WITH HIGHER VOLTAGE HARMONICS

Mento Menteshev

The increasing use of non-linear devices such as thyristor drive controls, gas discharge lighting, and solid state rectifiers is impressing higher current and voltage harmonics on mine electrical power systems. The problems arising from the presence of these harmonics has not been adequately investigated. This paper presents results of investigations into harmonic problems on both grounded and ungrounded single- and three-phase mine electrical power systems up to 1,000 V AC. Currents containing higher harmonics and their effects on the human physiological response are discussed. Increased danger of methane explosions arising from the presence of high harmonics is also explored. The effect of harmonics on the response of electrical protective devices and compensation schemes to mitigate these effects are also presented.

PROBLEMES CONCERNANT LA SECURITE ELECTRIQUE DANS LES GRILLAGES DE MINES AYANT DES HARMONIQUES PLUS IMPORTANTES

M. Menteshev

L'utilisation accrue de dispositifs non linéaires tels que les commandes à entraînement par thyristor, l'éclairage par décharge de gaz et les redresseurs transistorisés entraîne des harmoniques plus importantes des courants et des tensions des systèmes électriques miniers. Les problèmes que pose la présence de ces harmoniques n'ont pas été examinés de manière adéquate.

Notre rapport présente les résultats de recherches concernant les problèmes d'harmoniques; ces recherches ont été effectuées sur les systèmes électriques à la fois mis à la terre et non mis à la terre, monophasés et triphasés, allant jusqu'à 1 000 V CA. Nous y examinons les courants comportant des harmoniques plus élevées et leurs effets sur la physiologie du corps humain. Nous étudions l'accroissement du danger d'explosions de méthane provoqué par la présence d'harmoniques élevées. Nous y présentons également l'influence des harmoniques sur les dispositifs de protection électriques, ainsi que les dispositifs de compensation visant à atténuer ces effets.

PROBLEME MIT DER SICHERHEIT IN UNTERTAGE-NETZEN MIT OBERWELLENSPANNUNG

M. Menteshev

Der gestiegene Einsatz von nicht linearen Einrichtungen, wie z. B. Thyristor-Steuerungen für Antriebe, Gasentladungsröhren für Beleuchtungszwecke und Halbleitergleichrichter, belastet das elektrische Netz der Mine mit mehr Oberwellenspannung. Die Probleme, die mit dem Vorhandensein dieser Oberwellenspannung auftreten, wurden noch nicht in ausreichendem Maße untersucht.

Der Vortrag beschreibt die Endergebnisse der Untersuchung des Oberwellenproblems in beiden, den geerdeten und den nicht geerdeten, Einphasen- und Dreiphasen-Stromsystemen einer Grube bis zu 1,000 VAC. Ströme, die mehr Oberwellenspannung enthalten, und ihre Wirkung auf die menschliche Physiologie werden diskutiert. Es wurde auch die gesteigerte Gefahr einer Methanexplosion durch das Vorhandensein von Strömen mit viel Oberwellenspannung erforscht. Die Auswirkung der Oberwellenspannung auf die Reaktion von elektrischen Schutzeinrichtungen und Kompensationseinrichtungen zur Milderung der Auswirkungen werden vorgestellt.

煤矿电网中高压谐波的安全问题

M。孟特敬夫

最近阿流晶体管控制器，排气照明，和固态整流器应用很多。这些对矿井电力动力系统都会产生很高的电流和电压谐波。关于这谐波的问题尚未充分讨沦过。本文报道一些对高达交流１０００伏特的接地及不接地的单相，三相电力系统的谐波问题的研究结果。这里将讨论含有这谐波的电流和它对人类生理反应的影响。文中亦探讨了由于这谐波而增加的甲烷爆炸的可能性。这里同时讨论到了谐波对电力保护设备的影响和如何减轻其影响的方法。

ENVIRONMENT MONITORING AT CZECHOSLOVAK UNDERGROUND COLLIERIES

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ABSTRACT

Monitoring of the mine atmosphere and relaying to the surface are described. An intrinsically safe methanometric system is usually an autonomous continuous system, independent of the colliery mains. Separate systems monitor air velocity, CO and CO_2, static pressure, pressure differences, as well as methane in the drainage system. Special emphasis is placed on the CO concentration as an indication of spontaneous (endogenic) combustion. Actual operation in an active mine is described. Need for international cooperation is expressed.

The measurement of mine atmosphere parameters at Czechoslovak's deep mines has had a long tradition. The largest Czechoslovakian hard coal coalfield with underground mining methods is situated in the Ostrava-Karvina Coal Basin where most collieries are characterized by a high gas emission rate. Methane has been an everlasting enemy of mine workers, and therefore great attention has been focused on it.

Besides the traditional manual measurement of methane content, methane data-logging systems with peripheral sensors located in individual underground workings and a central signalling system at the surface control room have been used since the beginning of the seventies. At first, data-logging systems of the West German Drager company were used. However, since the early eighties, Czechoslovak systems with home-made sensors and evaluation microcomputers have been produced. Similar to Drager sensors, the Czechoslovak sensors are based on thermocatalytic methane combustion on platinum. In their development, some parameters have been achieved by which these sensors differ from sensors of other manufacturers. Due to refined design of the combustion element, outstanding features have been achieved which practically eliminate the ambiguity of data measured with a pronounced inversion at higher methane contents (Figure 2). The measuring procedure is discontinuous with measuring intervals of about two minutes, in general.

The sensors are intrinsically safe, enabling the measurement of methane content development in the monitored workings even at high gas content, i.e., methane concentration exceeding the limit at which other electrical equipment (except for intrinsically safe appliances) is automatically switched off from the electric mains. For this reason, the sensors have no local power supply, but are supplied by a line from the surface.

Measuring characteristics of the sensor ensures an easy and precise evaluation by microcomputer (Figure 1). The output of the sensor corresponds to the electric frequency, i.e., the frequency interval from 600 to 1,500 Hz corresponds to the content range of 0 to 3% of CH_4. During the inactive period between the individual measuring cycles, the sensor transmits a 600 Hz frequency. By checking this frequency, the

integrity of the line to the sensor is also checked. To start the measuring cycle, a reverse supply voltage impulse is given by a program. The sensors are then activated. A gradual typical tuning of the frequency of activated sensors occurs at first at lower frequencies; later the frequency rises and after about three seconds is stabilized on the value corresponding to the CH_4 content at the measuring point. After about 4.5 seconds since the beginning of the measuring cycle, the frequency drops sharply again to 600 Hz. The processor during these 4-second intervals is measuring the period time (frequency) of characteristic points in all activated channels by a multiplex method while performing five measurements in total on each sensor in the intervals of T = 0,1s; T = 0,5s; T = 3,5s; T + 3,85s and T + 4s. The first two measurements characterize the condition of the sensor (sensor failure, faulty connection to the line), while the other three values determine a specific frequency; their arithmetic average is calculated and converted to a percent value of the methane content; this final value is stored into the microcomputer memory. After twenty seconds, measured values of all 120 possible sensors integrated into the system are updated. A data evaluation phase follows (for instance, comparison of limits, etc.) together with output report of sensor matrix on the CRT display. If anomalous conditions are detected, they are listed by the printer, together with a commentary featuring the actual condition. The last phase of data processing is represented by a storage of all measured values together with real time and date on magnetic media. In the pause preceding the beginning of another measuring cycle, the time and date are updated by the system and eventual commands from the keyboard for representation of additional data on the screen are expected.

On the communication terminal, the data on actual conditions at individual measuring points are updated continuously. These data are schematically displayed in the matrix order in which the left matrix pertains to methane detectors and the right matrix pertains to automatic cut-out of the electric power in the monitored working sites, (Figure 3).

The following conditions are distinguished by the matrix symbols.

Measuring matrix

+ sensor connected, integrated into the system, is O.K.

A presignalling limit reached, warnings

H alarm level reached

R predetermined measuring range exceeded, indicating within the interval of 3.1 - 12%

I inversion, beyond about 12%

P failure, measurement curve does not comply with the typical course

F frequency loss, frequency lower than 450 Hz, connection interrupted

C calibration of sensors, maintenance

Tripping matrix

+ switching off point defined in the system

V electric power switched off (automatic disconnection)

Z reswitching allowed

On this display, the lower third is reserved for conversation with the system. Detailed data on each sensor can be displayed: the warning and alarm limits include the data measured in the past 20 minutes; it is possible to display data from all sensors in a selected working.

Due to high methane emission rates of the exploited seams, the methanometric system is generally an

autonomous continuously operating system with sensors independent of colliery mains, enabling automatic switching off of the electric power in gassy areas. The power can be switched on again remotely only by the dispatcher (if reswitching is allowed by the computer), according to Czechoslovak regulations.

For monitoring of other safety information, a separate data concentrator is used. The concentrator evaluates data on air velocity, carbon monoxide and carbon dioxide contents, static pressures and pressure differences, as well as the methane concentration in the drainage system (Figure 4). The concentrator also controls the operation of the booster fans and the opening of dams, etc. These autonomous systems can be covered by control computers on the pit level, or eventually on the enterprise level.

Together with penetration into greater depths and more adverse geological conditions, the risk of endogenic fires (spontaneous combustion) has increased sharply. For an early detection of endogenic or exogenic fire risk, a mere monitoring of limit values is not sufficient, it is also necessary on the basis of time development of carbon monoxide content to provide for:

- elimination of misleading influences, i.e., shotfiring, passing of mine locomotives, eventually larger deviations of analyzer indications (four minute trends)

- calculation of short time trends of carbon monoxide content (always the last 180-minute interval values measured must be available).

- calculation of long-term trends (for a single day trend, 24 average hourly values are to be available, for a single week trend 7 average daily values must be available, etc.).

- comparison of calculated trends with predetermined limit values depending on the type of analyzers and their location in the ventilation network.

For implementation of sophisticated algorithms of regression analysis it is necessary to apply efficient 16-bit microcomputers in a multiprocessor mode. After long-term laboratory tests, we started with a verification at a colliery, using thirty carbon monoxide sensors, twenty anemometers and ten combined sensors for measuring carbon monoxide and products of combustion supplied by the Sieger Company.

Similar autonomous microcomputer subsystems have been developed in the past years for other mine technologies, such as production, transport, energy supply and preparation processes. The unification of microcomputers and their peripherals, unification of control room design, their equipping by unified minicomputers (Czechoslovak equivalent of PDP) and particularly an extensive modular software for real time control, enabled design and realization of a comprehensive information and control system at the May Day (1. Maj) Colliery.

This colliery with a daily output of 17,500 tons includes 4 pits and a coal preparation plant. A single pit, No. 2 Darkov, operates six working faces and eleven headings. There is a central enterprise control room, four pit control rooms, an underground control room for rail transport control, and in the preparation plant there are a central control room plus three control rooms designed for dirt disposal, washing and shipping.

The control rooms feature remarkable conversation possibilities, variability in hardware and software configurations, graphic colour displays of the processes, with possibilities of complex consulting algorithms, autonomous operation in case of breakdown of superior

control levels, and ability to maintain the data obtained.

Input data acquisition is performed automatically by means of data concentrators. Manual inputs are introduced by means of terminals. Logic processing is carried out by SMEP minicomputers in a standby mode. For human communication, graphic colour display of alarms, technology mnemonics, schemes and histogramms, are available, as is detailed data presentation displaying the occupied underground workings or technological, eventually transport, junctions. The operation of the system is also facilitated by an acoustic signalling output, synthetic voice output and automatic feedback into the process by binary commands (switching off the electric power when alarm limits of methane content are exceeded, control of pilot motors in compressed air network, etc.), including transmission of selected data to the upper management level.

The pit control room is designed to control the mine operation in real time and is equipped with four colour semigraphic monitors (Zoom 1, Zoom 2, Overview and Alarms), with standby monitors SM 7202 connected with the environmental concentrator (MBK 100) and data concentrator MMK-120 for measuring methane. The monitors are put into operation if a failure of the SMEP computer occurs and environmental information from colour semigraphic monitors is available. The control room is also equipped with a printer, and a display of remote terminal network with a keyboard connected to the upper level (introduction of manual inputs), with a passive mine map, and finally with an operator's console with conversation facility, a technological keyboard for communicating with the system, and a control and signalling part. In the console there are outlets of optical and acoustic signalling transmitting selected information on environment, ventilation and methane drainage which are useful in situations when the whole computing system eventually fails.

For the building set of control rooms, an application software DISAM has been developed. It is an information system for the control room operator which also enables the connection of the local terminal network. The purpose of the whole operation is to collect information from the data concentrators and to form an image of the real condition of the monitored processes in the memory. For each sensor, an information block is maintained in the storage which enables further processing. The real time is given by an interval of the information acquisition by the data concentrator.

Based on such a data base, so-called technologic modules can operate, processing the operating data according to requirements defined by the process control system. The information system can be complemented during operation of technologic modules, or they can be cancelled eventually.

The software provides:

- data acquisition from the mining process and updating of the data base
- recording and handling of special conditions of the process, so-called alerts
- calculations of technologic values on basis of the predetermined algorithms (technologic modules) and their storage into disk files
- representation of all system information by means of graphic colour displays
- operation control of those displays by means of a technological keyboard (the operator's console)
- updating of sensors of "technological modules" of data concentrators during the system operation

- restarting of the system in case of outage

- data transfer to a superior level

- information extending by voice synthesizer (MHK-100 data concentrator) through telephone exchange with individual extensions (telephone, tannoy, etc.)

- control of terminal network operation

The main feature of the information system is the general character of solutions which enables during the system operation to vary both the data processing method and their representation on screens of the control room, with minimum requirements on the professional level of programming by the operator.

The above-mentioned monitoring and control systems have proved fully successful in practice and the initial distrust of mine employees for computing techniques was overcome. The best proof that mine employees have learned to utilize the advantages of computer control is the fact that they themselves require further applications, modifications and innovations of the already manufactured systems.

Although it is true that on the one hand that we can set up high quality automation systems, on the other hand we still face the problem of insufficient range of present sensors and need sensors based on principles (electrochemical cells, semi-conductor sensors, optoelectronics, etc.) which objectify the input data, eliminate the human factor, improve and extend the quality of knowledge of the process controlled and provide a computing technique with higher communicating ability.

This area in a number of cases is directly connected to basic research and is very time-consuming and financially demanding. Therefore it appears useful to develop international contacts and cooperation, particularly in this area, and to coordinate the utilization of creative capacities in a desired direction.

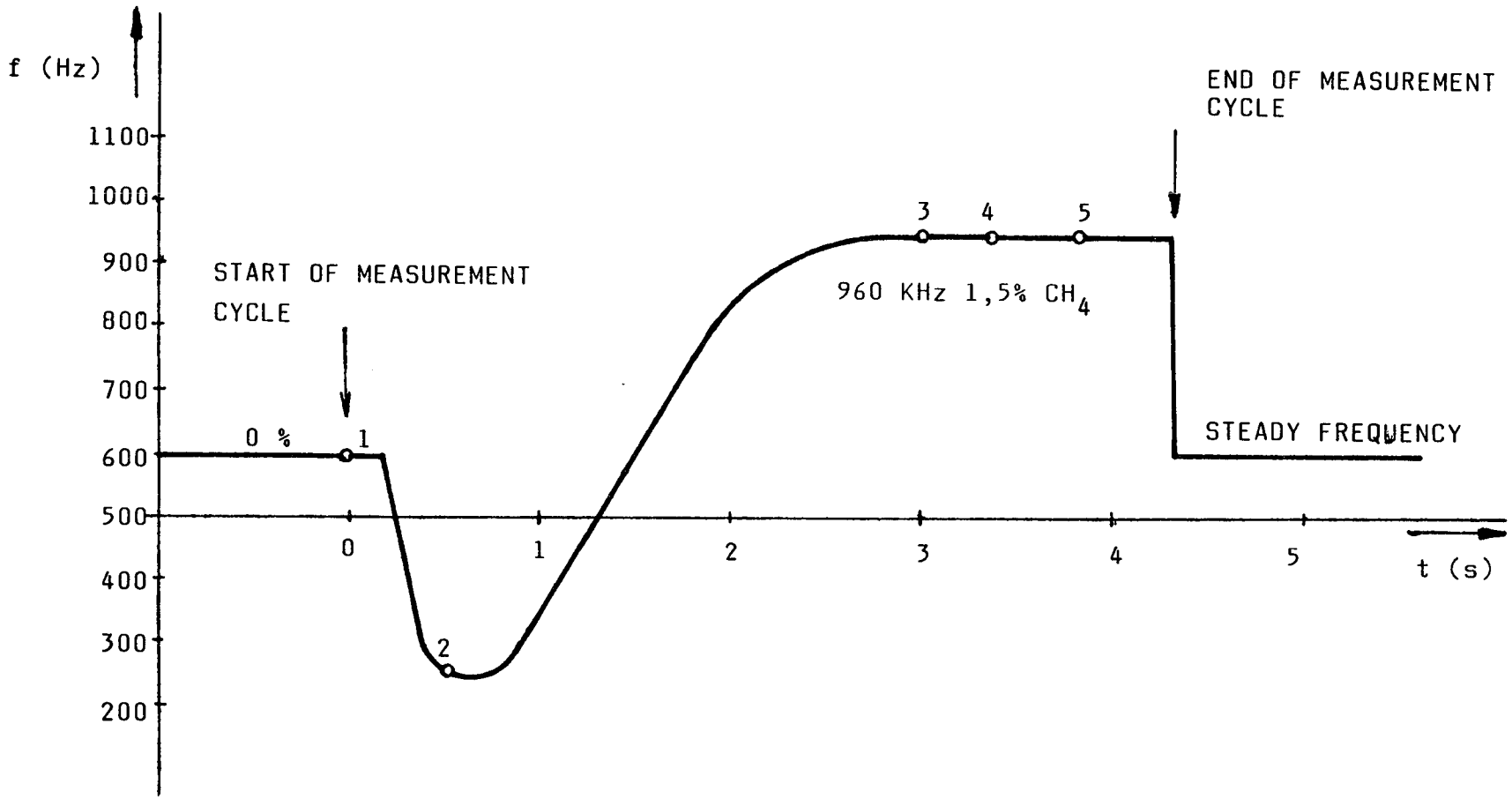


FIGURE 1: TRANSDUCER OUTPUT SIGNAL ACROSS TIME CURVE

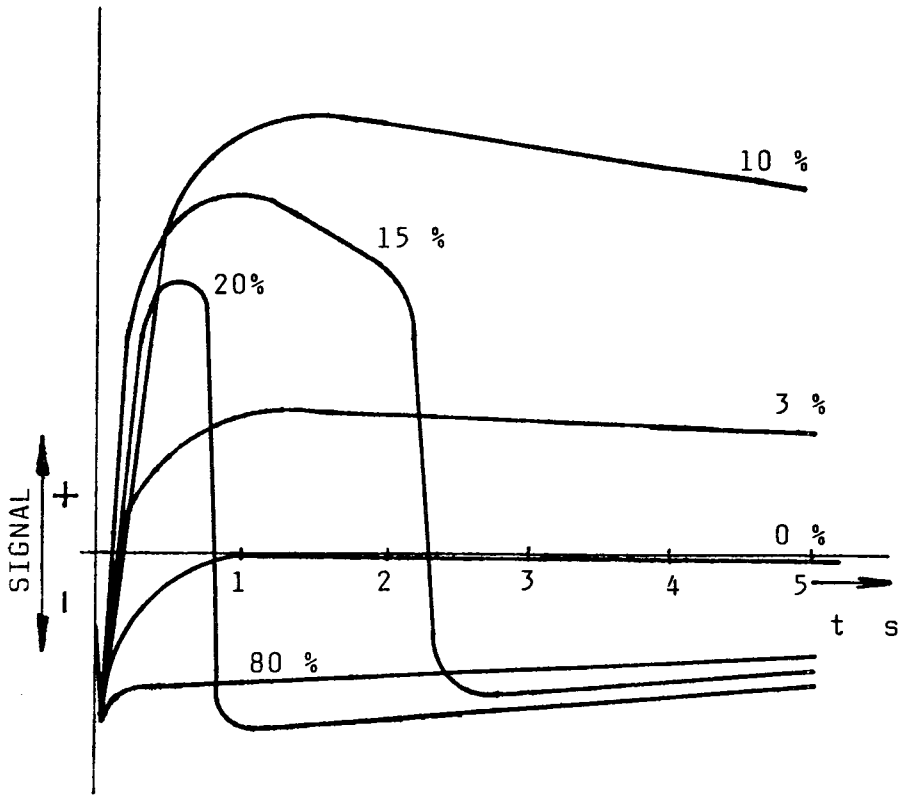
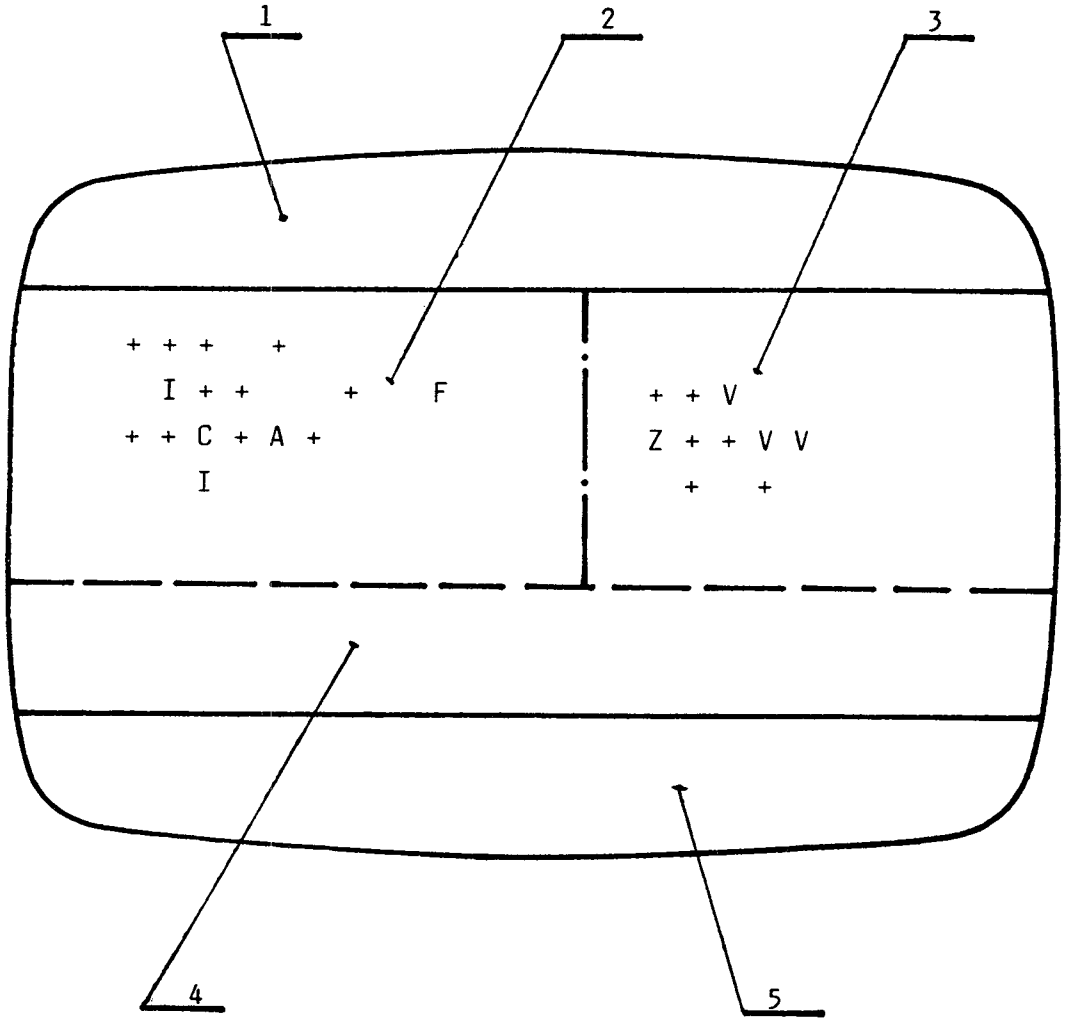


FIGURE 2: DYNAMIC CHARAKTERISTICS OF SENZOR WITHOUT FILTR AND ELECTRONICS



1 - GENERAL INFORMATION (DATE, TIME ETC.)
2 - MATRIX OF SENSORS
3 - MATRIX OF POWER SUPPLY BREAKING
4 - SUPPLEMENTARY INFORMATION
5 - COMMAND ECHO

FIGURE 3: INFORMATION DISPLAY

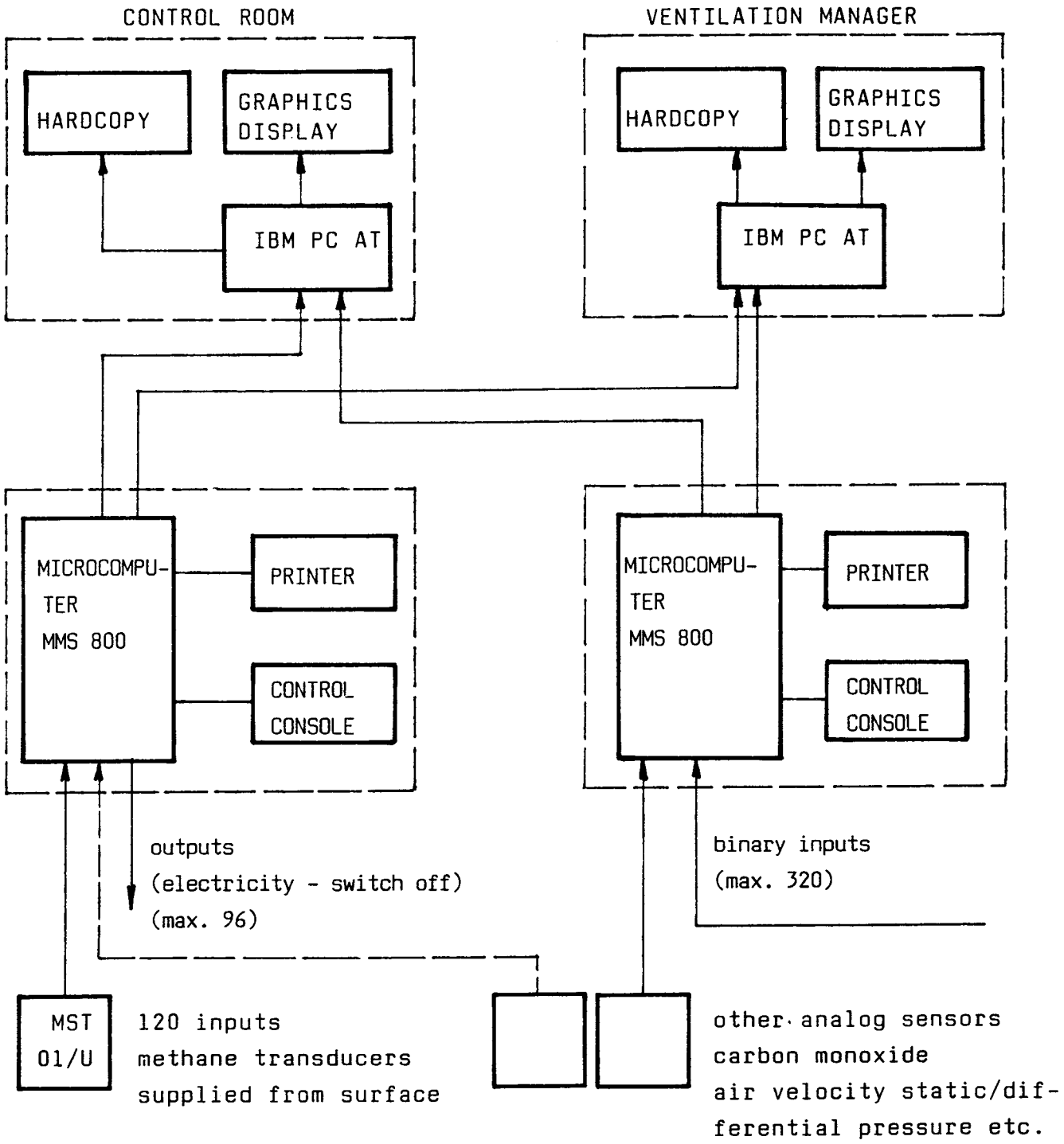


Figure 4: Environmental Management

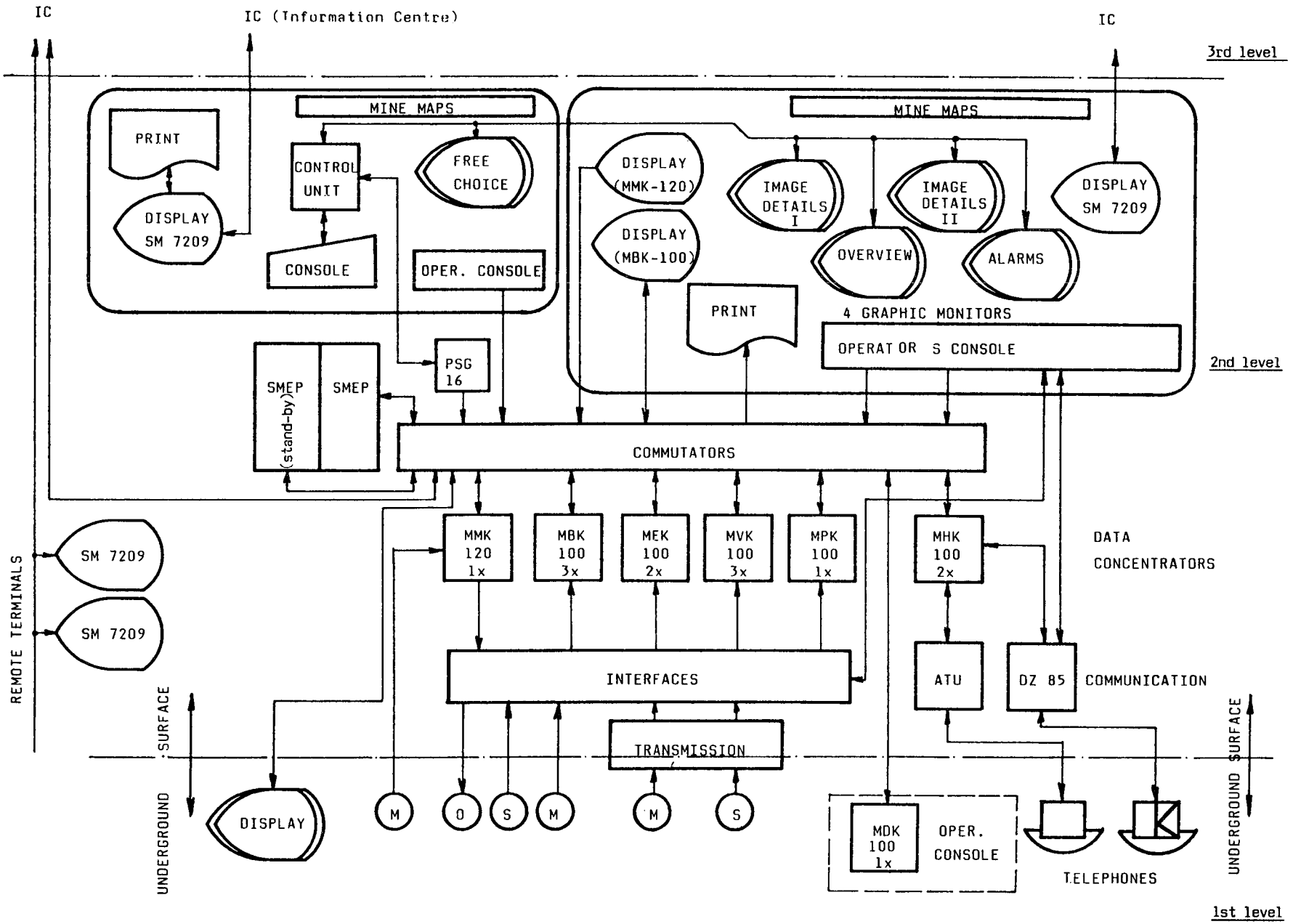


FIGURE 5: BLOCK DIAGRAM OF MULTILEVEL CONTROL SYSTEM

SURVEILLANCE DE L'ENVIRONNEMENT MINIER DANS LES HOUILLERES SOUTERRAINES DE TCHECOSLOVAQUIE

O. Hlavica et T. Back

Notre rapport examine la surveillance de l'atmosphère minière et la transmission de ces données vers la surface. Un système méthanométrique intrinsèquement sûr est habituellement un système autonome continu, indépendant des réseaux principaux des houillères. Des systèmes séparés surveillent la vélocité de l'air, le CO et le CO_2, la pression statique, les différences de pression, ainsi que le méthane présent dans les systèmes de drainage. Nous soulignons l'importance que présente la concentration de CO, car elle est le signe d'une combustion spontanée (endogène). Nous décrivons le fonctionnement effectif d'une mine en activité et exprimons le besoin d'une coopération internationale.

DIE WETTERÜBERWACHUNG IN TSCHECHISCHEN GRUBEN

O. Hlavica, T. Back

Die Überwachung des Grubenwetters und die Übertragung der Informationen nach übertage wird beschrieben. Ein eigensicheres Methanometersystem ist normalerweise ein autonomes, fortlaufendes System, welches unabhängig von der Haupteinspeisung der Grube betrieben wird. Verschiedene Systeme überwachen die Wettergeschwindigkeit, CO- und CO_2-Werte, den statistischen Druck, den Druckunterschied, sowie das Methangas in der Gasabsaugung. Besonders wird dabei auf den CO-Gehalt als Anzeichen einer Selbstentzündung geachtet. Die Anwendung in einer aktiven Grube wird beschrieben und die Notwendigkeit der internationalen Zusammenarbeit betont.

НАБЛЮДЕНИЕ ЗА АТМОСФЕРОЙ В ПОДЗЕМНЫХ УГОЛЬНЫХ ШАХТАХ ЧЕХОСЛОВАКИИ

О. Хлавика и Т. Бак

В докладе описывается наблюдение за шахтной атмосферой и передача данных на поверхность. На угольных шахтах обычно используется автономная метанометрическая система, не зависимая от других систем. Отдельные системы следят за скоростью воздушных потоков, содержанием CO и CO_2, статическим давлением, перепадами давления, а также за содержанием метана в дегазационной системе. Особое внимание уделяется содержанию CO, который является индикатором самопроизвольного возгорания. Описываются реальные ситуации в действующих шахтах. Отмечается необходимость международного сотрудничества.

捷克斯勒伐克地下煤矿环境监测

O。赫拉维卡 及 T。贝克

本文叙述矿井空气的监测和与地面连络的情形。一个本质安全的甲烷测定系统通常是一个自备连续的系统与矿身并无关连。在排气系统中的空气速度，一氧化碳和二氧化碳，静压力，压力差和甲烷量都由分别的系统来监测。我们特别重视一氧化碳的浓度因为它是自发（内生）燃烧的指标。这里叙述现行矿井的实际操作情形。本文表示出国际合作的需要。

MONITORING AND CONTROL OF GAS CONTENTS AND AIRSTREAMS IN GASSY UNDERGROUND COAL MINES

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ABSTRACT

This paper presents the results of research into selection of an optimum system for monitoring gas, fire, and ventilation parameters in the Stara Jama Mine, a brown coal mine of the Zenica Coal Basin in Yugoslavia. It discusses the need to investigate, and logistics of applying automatic controls for gas content and ventilation in coal mines with methane. Remote monitoring of mine air, and ventilation automation systems applied and considered at Stara Jama are presented.

INTRODUCTION

Extraction of coal in underground coal mines is burdened with a number of natural and technological problems which present a danger to life and health of the miners and material resources. Occurrence of explosive, toxic, and suffocating gases; formation of explosive coal dust, oxidation processes, and spontaneous underground fires; sudden outbursts of methane, coal, and surrounding rocks; and increase in temperature and humidity of mine air, all add to the complexity of mine ventilation as a crucial element for safe work.

The scientific and professional experience with catastrophes which occurred in the coal mines of Kakanj, Breza, and Zenica, identifies instability of and disturbances in the ventilation system as the main causes of the formation of methane-concentrations. The explosive concentrations were often ignited by an electric spark. The catastrophes were usually associated with secondary circumstances such as insufficient control of gas contents and ventilation, lack of modern ventilation monitoring systems, inadequate ventilation facilities, large and uncontrolled air losses through isolating structures and gobs, and improper handling and maintenance of electrical equipment. Solutions for these complex natural and technological problems require long-term and costly scientific research. Cooperative research between the Institute and Faculty of Geology and Mining in Tuzla and the U. S. Bureau of Mines in Pittsburgh under the Yugoslav-United States Joint Fund for Scientific and Technological Cooperation, was initiated on conditions governing brown coal extraction, occurrences of explosive and other gases, and applied ventilation systems. The purpose was to select an optimum gas, fire, and ventilation monitoring system. Such systems are not manufactured in Yugoslavia, however those of European manufacturers have been applied in the country for the last 5 years. The selection of a monitoring system for the Zenica mine was based on

knowledge of the European and U.S. systems, as well as on experience with their application. The entire research was based on the Stara Jama brown coal mine of the Zenica mine complex as a typical case example of the problems and dangers from methane and underground fires. This paper presents the research results from this study to find a solution for stable and safe ventilation by means of automatic monitoring and control of air distribution in underground workings, thereby reducing the possibility of accumulation of explosive methane-air mixtures.

BASIC DATA ON STARA JAMA COAL MINE

Stara Jama is one of four active underground mines operating within the Zenica brown coal mine complex, and occupies a central part of the Zenica coal basin. The mine was started in 1881. The annual output of the mine is 400,000 tons of commercial coal. The mine was developed with two ventilation slopes and a production shaft. Seven coal seams of various thicknesses and quality have been developed. Currently only the northern part of the mine is active with working of the main coal seam 730 to 911 m deep. The 14 m thick main coal seam is mined in sections by mechanized longwall (Fig. 1), in which the roof seam is mined first, then the floor seam, and medium seam. Presently, the floor seam and medium seam are being mined by a longwall face 150 m wide and up to 2,000 m long.

A larger fault separates the Stara Jama deposit from the adjacent mines, while smaller faults determine the size of the mine section and have a considerable effect on selection of the mining technology. These faults may be related to an increase in the methane emission when the mining is done in their vicinity or through them.

Gas content in a deposit, spontaneous combustion of coal, and spontaneous combustion and explosiveness of coal dust are important to ventilation in a producing area. Methane, hydrogen sulfide, nitrogen oxide (from equipment), and carbon monoxide (from fires) are liberated at the

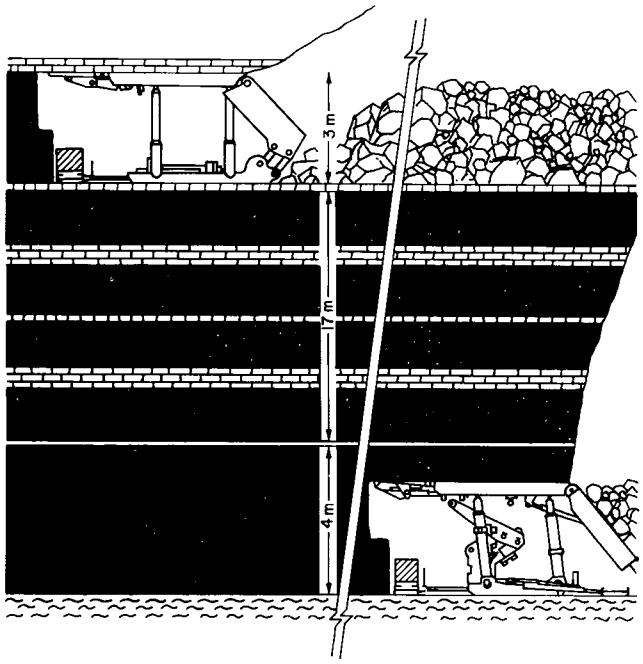


FIGURE 1. - Mining of the main coal seam of Stara Jama Mine

Stara Jama coal mine. The largest emissions in quantity and concentration are methane, whereas emission of other gases are much lower and less frequent.

The coal seams and the surrounding rocks of Stara Jama are significant methane carriers, whose migration is affected by the structure of the deposit. The local tectonics effect is related to the micro-distribution of methane within the deposit, and is considerable in highly disturbed deposits. The Podbrezje fault on the northern side has a 300-400 m displacement which brought the overlying seam and the main coal seam in direct contact with deep floor clay sediments (known as good isolators), thus preventing the migration of methane in this direction. The Stranjani fault caused similar methane isolation in another area. These faults are great barriers to methane migration and natural degasification of coal seams. Other faults further complicate the problems related to methane.

The methane emission in Stara Jama occurs through exhalation from coal seams, release from crevices due to mining, and in the form of "blowers" (release of methane under pressure). For mine safety, methane concentrations in all underground workings are divided into three categories:

Category 1: up to 0.5% safe level.

Category 2: 0.6-1.0% warning level--take corrective action.

Category 3: above 1.0% alarm level--removal of miners, deenergize equipment.

Methane emissions may cause concentrations from 0 to over 1.0%, therefore all danger categories of 1, 2, and 3 can be expected. Total emission of methane is approximately 20 m^3/min. The coal seams are also very prone to spontaneous combustion. Natural inflammability index by Olpinsky ranges from 78 to 110° C/min. The lower explosiveness limit of dust of the main coal seam in the coal dust-air mixture is 170 g/m^3, and 150 g/m^3 in the coal dust-air with 1.5% methane mixture.

The mine is ventilated by one intake and two return airways. The central and western parts of the mine are ventilated by the diagonal system, and the southern part is ventilated by the central system. There are three ventilation sections, southern, western, and central (Fig. 2). The main intake is the main production shaft which feeds to the productive (western) part by the haulage road. The airflow in the production shaft branches off by levels. Air is taken from the haulage road to ventilate the depot and explosive storeroom. The airflow from this section and the main level leaves the mine by the ventilation shaft.

The airflow for the productive part in the main haulage road splits in two parts, one for ventilation of a development entry in the central part and one for entries for longwall development in the western part. The return airflow leaves through the various airways and central slope.

Basic parameters of the ventilation network are:

- Main fan pressure on the central slope: h_1 = 2,747 Pa
- Main fan pressure on the ventilation shaft: h_2 = 343 Pa
- Air quantity in the production part: Q_1 = 2,862 m^3/min
- Air quantity through the ventilation shaft: Q_2 = 515 m^3/min
- Total equivalent mine opening: A_u = 1.377 m^2.

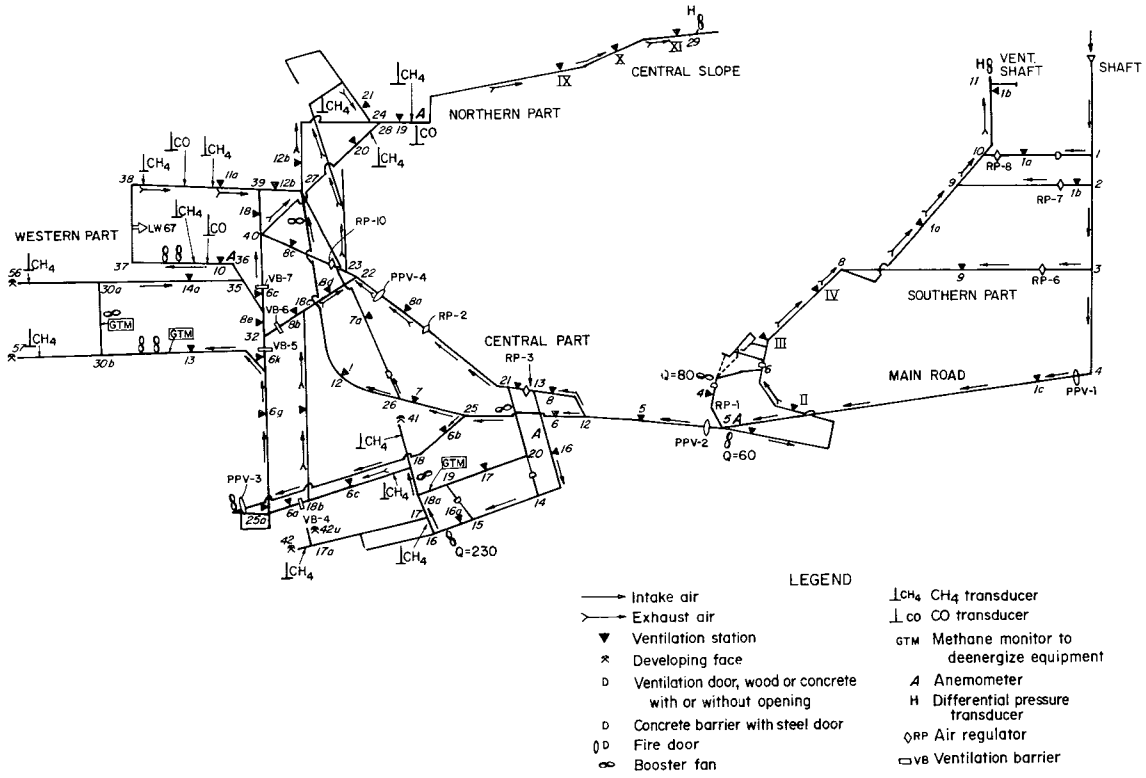


FIGURE 2. - Ventilation diagram of Stara Jama with location of sensors

BASIC REQUIREMENTS OF REMOTE MONITORING SYSTEMS

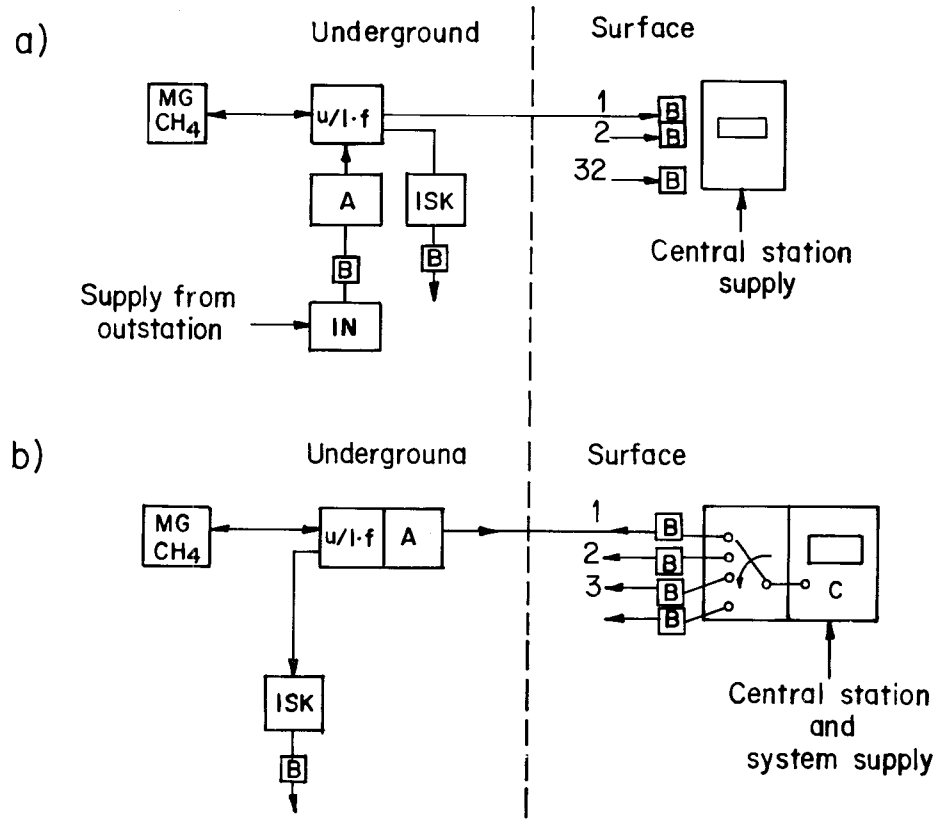
Three basic functions of mine monitoring systems are (1) sensing and measurement of certain parameters (detection), (2) data transmission (telemetry), and (3) data analysis and display.

Specific mine requirements of these functions determine the basic structure of the system (Fig. 3) which incorporates transducers, which monitors and measures the parameters, and converts them into electrical signals. The system must also include outstations, electrical installations for transmission of data from underground transducers to central station or computer on the surface, and the central station or computer on the surface, where the received signal is analyzed, converted to measured value, and displayed or filed.

Data reception and display of the data may be continuous for all connected transducers, or discontinuous-cyclic, for one transducer at a time. Transducers contain current circuits and measuring elements (head, sensor), and circuits for conversion of current or voltage signal received from the measuring head into the corresponding code signal for transmission to the central station or computer. The measuring head may be in the same enclosure with the converter unit, or placed in a separate enclosure.

The measuring head output can be a simple status indication, called a binary, contact closure or status output (high-low, open-closed), or it can be a continuously variable function of time, called analog output (air velocity, methane concentration, etc.). The analog outputs can provide much more information so they are used for monitoring the environmental parameters, whereas the simple status outputs are used to monitor the equipment parameters.

Transmission of data from the transducers to the central station



KEY

IN	Supply source
A	Battery
U/I	U/I or U/f converter (encoder)
ISK	Switch-off relay
MG	Measuring head(transducer) CH_4
C	Central station
B	Barrier

FIGURE 3. - Basic structure of mine monitoring system

on the surface is done by special cables (pairs) for each transducer, or several cables are connected to the outstation in which the signals are encoded and transmitted in a common cable to the central station. In selection of a method for transmission of several signals in a common cable, the system technically identified as frequency domain multiplexing offers certain advantages. It makes possible a simultaneous and uninterrupted transmission of all data within the overall frequency range of the system, whereas, when time domain multiplexing is used, the signals are received and transmitted cyclically, at time intervals necessary to receive and analyze the data from all measuring points.

The central station or computer on the surface should provide the following:

(1) Recording, display, analysis, and storage of data received from the underground transducers. This should be provided for methane, carbon monoxide, carbon dioxide, and oxygen contained in the mine air, velocities of the main or

separate airstreams, temperature of mine air, and positions of air regulators and ventilation doors.

(2) Signaling of excess over the predetermined thresholds of measuring data in two stages (warning and accident condition).

(3) Automatic deenergization of equipment in the hazardous area or in the whole mine, depending on design solutions or requirements of technical standards.

(4) Checking of each line and the entire system for proper functioning, and signaling of defects found in the system.

As the mine monitoring systems should increase the safety of mines and miners, one of the requirements imposed on them is reliability of the system itself and a constant transmission of accurate data. Reliability is particularly important because of the seriousness of problems that could develop in an inflammable and explosive environment. The systems must provide early detection of fires, and provide rapid warning and deenergization of electrical equipment.

The above stated circumstances require a proper selection of a monitoring system, its installation, operation, and maintenance. A deficiency in any of these stages, regardless of whether it is a fault of the manufacturer or the user, may lead to serious consequences.

Reliability of mine monitoring systems in inflammable and explosive methane-air mixtures necessitates an intrinsically safe explosion-proof design (Exia) of the entire underground part of the system. Since the system is connected to surface and underground electrical networks which are not intrinsically safe circuits, it is necessary to install barriers on all lines connecting the system to other networks, to secure a separation of intrinsically safe circuits from those that are not.

Guaranteeing a continuous supply of data on the monitored parameters requires a duplication or even triplication of certain elements, or the entire system. The duplication should be made in such a way that one breakdown or defect would not cause a failure of the entire system. This is a much more expensive solution, but at the same time it insures that no alarm signal may pass unnoticed. Both the software and the hardware part of the system must be reliable. Based on the analyses made by U. S. Bureau of Mines research, a conclusion was made that the central processor unit (CPU) on the surface should be duplicated and that transducers should also be duplicated. The data transmission systems should be designed with double independent cable installed in separate entries to prevent the signal loss in case of damage to one cable by fire or roof fall.

Another requirement is the accuracy of received data. Sensitivity of a system to transmission of incorrect data depends on several factors such as type of cable, local field of electromagnetic noise, cable length and shape of the signal. Research at the U. S. Bureau of Mines showed that the maximum transmission distance for one undetected random error/year ranges from 2,100 m to 10,950 m in an average noise field, and from 160 m to 965 m in an estimated maximum noise field. Therefore, in mine monitoring data communication systems, one must take into account the fact that transmission of undetected errors will sometimes occur.

Reliability of the monitoring system software program is also a factor of considerable importance, since it is difficult to be certain that all errors and defects have been detected and removed from the software. Optimally, to reduce the chance of software errors, software should be developed by two independent programmers, and then

requirements of Yugoslav

- reading of measurements on

run on two separate processors to compare the results and determine whether all possible bugs and errors have been removed from the program.

In addition, it is necessary to stress the importance of transducer accuracy regarding their capability to eliminate the effect of other parameters when measuring desired parameter. Important also are the quality of all system elements with respect to operation in the mine environment. Modern design technology must be practiced

1.0 to 24.0 m/s in ventilation tubing, and

- differential pressure from 0.0 to 5.0 kPa.

Basically, the system consists of:

(1) transducers installed in the mine for measurement of the above parameters,

(2) transmission of outputs, individually or directly from the transducers to the central station; and

- automatic switching to standby battery supply in case of power failure in the network;

- capability of testing of the standby supply and adjusted alarm levels;

- multiplex transmission of signals from underground to the dispatching center on the surface;

- explosion-proof design of transducers and cabling.

In addition, the methane transducers should be able to automatically deenergize the electrical equipment in the endangered part of the mine, or the whole mine.

The following should be provided by the central station at the dispatching center on the surface:

- monitoring and processing of data from the mine;

- alphanumeric display of transducer outputs during normal and alarm conditions;

- color display of mine and measuring point schematics;

- recording and storage of received data in a data base;

- checking of data transmission lines for proper performance including failure, short-circuit or switch-off.

Based on the above requirements and analysis of the existing CTT 63/40 U system, it was concluded that the existing system does not provide

- installation of a new system from Sieger, Great Britian, and EI Nis, Yugoslavia, and utilization of the existing system for other mines;

- installation of a new system of U.S. production, which would fully comply with the defined requirements.

The new CGA system is considerably improved with respect to the CTT 63/40 U and has the following features:

- All transducers have local (on the transducers themselves) digital display of the measured parameters, local adjustment of the alarm level, and local alarming if the threshold limit is exceeded.

- Multiplex transmission of 8 output signals to the surface by a paired cable.

- The signals from the mine are received and processed in a microcomputer on the surface, which is connected to the control board, display panel, two alphanumeric printers, and a central control unit.

The British Sieger system also provides all the required features, while data transmission to the surface and a computer with full processing and display are produced by EI Nis. This system operates successfully in some Yugoslav mines.

The American system has all the required features. The computer can be connected to the control board, display units, and printers.

monitoring, will represent an integral solution to the safety of mines and miners.

LOCATIONS OF MEASURING POINTS

It is important that the monitored data accurately reflect the status of the mine environment. Fulfillment of this requirement necessitates the proper arrangement of transducers for the monitored parameters in the system of underground mine entries.

Measuring point locations generally are along the underground entries and points in an entry cross-section.

Location of the measuring points along the underground entries is usually determined by requirements of regulations for control of a particular parameter, such as ventilation method, and actual degree of hazard. Location of the measuring points in cross sections of underground entries are determined by physical properties of the measured parameter with respect to ventilation air, arrangement of equipment within the entry, or entry characteristics affecting the point at which the parameter is to be monitored.

In principle, the methane transducers are placed in the main intake and the main return ventilation entries, intake and return entries of longwall faces, return entries of development faces, and other locations where there is a need for continuous monitoring.

Carbon monoxide transducers function as a means for protection of miners from being poisoned by carbon monoxide and as early detectors of spontaneous underground fires. They are located in the area where the oxidation process may be expected, such as in the vicinity of mined-out areas, intake and return entries of longwall faces, and in the vicinity of faults. Another area of placement is near equipment locations such as in belt haulage entries. They are placed in the middle of entries.

Air velocity transducers (anemometers) are placed in all major branches of the ventilation network. Location in a cross section of an entry is at 2/3 of the entry height.

Differential pressure transducers are placed before the main fan to measure the pressure difference in the main airflow, and in other places where the pressure difference ought to be controlled.

The location of transducers of gas, fire, and ventilation for Stara Jama followed the above rules. The arrangement is as follows (Fig. 2):

(a) methane transducers

- in all separately ventilated underground mine entries (at 3 and 20 m distance from the face on the opposite side of the entry from the ventilating tubing);
- in return airflow of ventilation sections;
- in intake and return airflow of longwall face (at 20 m from the face in haulage road and airway);
- in areas of the ventilation system with separate ventilation of workings;
- in main intake entry of the mine;
- in other underground mine entries in which special methane conditions might occur.

(b) carbon monoxide transducers

- in intake and return entry of longwall faces;
- in main intake entry in the mine;

- in main return entry of the mine;
- in return entry of the productive part of the mine;
- in return entry of the longwall panel GPP-VI;
- in return entry of southern part of the mine.

(c) air velocity transducers

- in intake and return entry from longwall face;
- in intake and return entry of ventilating sections;
- in intake entry of productive part in the mine;
- in intake entry of southern part of the mine;
- in intake entry of deep section of the mine;
- in intake entry of the longwall panels;
- in return entry of separately ventilated workings or in the separate tubing at 20 m from its end in cases where controller of the separate ventilation has not been installed.

(d) air temperature transducers

- in intake and return entry of longwall faces;
- in main intake entry of the mine;
- in main return entry of the mine;
- in intake entry of productive part of the mine.

(e) differential pressure transducers

- at main ventilation fans.

According to the position of mining operation in the mine ventilation system, scope of mining operation and gas parameters, the arrangement of transducers must be adjusted to provide full monitoring of gas contents and airflow.

CONCLUSION

Application of a monitoring system for gas contents and airflow in coal mines with methane represents a considerable contribution to securing safe mining conditions. Environmental monitoring must accurately represent the actual conditions in the mine. It is important to select a system which best provides for monitoring of the specific mining conditions and dangers in the mine. Data obtained by this system make an excellent basis for determination, and timely application of safety measures. It is planned that a monitoring system will be purchased by Stara Jama Mine for its operations which meets the discussed requirements.

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SURVEILLANCE ET CONTROLE DE LA TENEUR EN GAS ET DE L'ECOULEMENT D'AIR DANS LES MINES GRISOUTEUSES SOUTERRAINES DE CHARBON

Milutin Vukic, Jeffrey H. Welsh

La modernisation rapide des mines de charbon grâce à l'introduction d'équipements électriques à haute tension, de vastes réseaux d'appareils et d'installations, ainsi que de l'exploitation à grande échelle de charbon à des profondeurs de plus en plus importantes, a eu pour conséquence l'apparition de gaz explosifs, toxiques et suffocants, de phénomènes d'oxydation et de combustion spontanée, ainsi que l'élévation des températures et de l'humidité de l'air minier. En raison de tous ces facteurs, la ventilation des mines est devenue un élément crucial, de plus en plus complexe, d'un processus technologique visant à donner aux mineurs des conditions de travail sûres.

Une recherche scientifique a été effectuée sur le fonctionalisme et la fiabilité des systèmes de surveillance du gaz et de la ventilation; cette recherche porte sur les conditions spécifiques de la technologie minière, sur la ventilation et sur les teneurs en méthane des filons de charbon. Des systèmes intrinsèquement sûrs ont été proposés, ainsi qu'une définition des paramètres de gaz, de flammes et de ventilation, et des meilleurs emplacements des appareils de mesure.

ÜBERWACHUNG UND KONTROLLE DES GASGEHALTES IN GRUBENWETTERN IN GASHALTIGEN KOHLEGRUBEN

Milutin Vukic, Jeffrey H. Welsh

Die rapide Modernisierung der Gruben durch die Einführung von elektrischen Hochleistungsausrüstungen, große verkettete Einrichtungen und Installationen, konzentrierte, groß angelegte Kohleproduktion in immer größeren Tiefen, das Auftreten von explosiven, toxischen und erstickenden Gasen, Oxidationsvorgänge und Selbstentzündungen, sowie die steigende Temperatur und Feuchtigkeit der Grubenluft machen die Bewetterung der Grube als kritische technologische Sicherungsmaßnahme für den Arbeitsplatz immer schwieriger.

Eine wissenschaftliche Untersuchung der Funktionalität und Zuverlässigkeit der angewendeten Methangas- und Wetterungsüberwachungs-Einrichtungen wurde im Hinblick auf spezifische Bedingungen durchgeführt, die die Technologie des Abbaues und der Ventilation, sowie Methangasgehalte der Flöze beeinflussen. Eigensichere Einrichtungen und die Definition von Gas-, Feuer- und Bewetterungsparametern, sowie die optimale Verteilung von Einrichtungen zur Erfassung derselben werden vorgeschlagen.

СЛЕЖЕНИЕ И КОНТРОЛЬ НАД УРОВНЕМ СОДЕРЖАНИЯ ГАЗА И ВОЗДУШНЫМИ ПОТОКАМИ В ПОДЗЕМНЫХ УГОЛЬНЫХ ШАХТАХ С ВЫСОКИМ СОДЕРЖАНИЕМ ГАЗА

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Описываются условия ускоренной модернизации угольных шахт путем введения мощного электрического оборудования и сети приспособлений и установок, а также концентрированная и крупномасштабная добыча угля на более глубоких горизонтах. Указывается значение таких факторов, как появление взрывоопасных, токсичных и удушающих газов; процессы окисления и произвольного возгорания в шахтах, а также повышение температуры и влажности воздуха в шахтах. Технология строительства вентиляционных систем в шахтах становится все более сложной, и значение вентиляции для обеспечения безопасности труда шахтеров увеличивается.

Было проведено исследование функциональности и надежности систем слежения за содержанием газа и вентиляцией. Особое внимание было уделено специфическим условиям, воияющим на технологию добычи, вентиляцию и содержание метана в угольных пластах. Предлагается введение сложных систем, обеспечивающих безопасность, а также определение газовых, пожарных и вентиляционных параметров и оптимальное местоположение датчиков, использующихся для их измерения.

瓦斯型地下煤矿中煤气及空气流的监测与控制

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强力电气设备的引进使煤矿高速地现代化起来，设施和设备的大型网络化，集中型煤炭的大规模高深度开发，瓦斯爆炸，中毒和窒息，煤矿内的自发性爆炸以及矿内空气湿度与温度的增高，这些都使得矿工安全作业的矿井通风问题愈来愈复杂.

本文科学地探讨了在各种不同的采矿方式，通风设备和煤层甲烷含量情况下，气体和通风监测系统的使用性能与可靠性. 本文建议使用内在性安全的系统，同时确定了气，火和通风参数的定义以及测量这些参数仪器摆设的最佳位置.

COMPUTER AIDED UNDERGROUND MINE FIRE LOCATION

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ABSTRACT

Sophisticated electronic fire detection systems can improve safety conditions in underground mines by providing early warning of fire emergencies. However, such systems are seldom capable of providing the specific location of a fire - only its general vicinity. Without specific knowledge of the fire's location, mine personnel are unable to optimize miner escape, fire fighting, and rescue team activities.

The U.S. Bureau of Mines, as a part of research to improve fire detection technology for underground mines, has devised a means for estimating the location of a mine fire based on information obtained from a few, widely scattered fire detectors. The system comprises three major elements: a network of underground fire detectors, an electronic data communications system linking the detectors to the surface, and a computer programed with appropriate process control and ventilation network analysis software.

This paper provides a brief description of the algorithm for determining the location of an underground mine fire, the results of a computer simulation case study applying the algorithm to a mine network, and the results of an in-mine simulated fire experiment to help delineate certain key operational characteristics in a typical production mine setting.

INTRODUCTION

Underground mine fires are a serious threat to the safety of miners and can result in significant damage to mine facilities and loss of valuable mineral resources. However significant progress has been achieved in recent years to improve underground fire safety. Advancements in fire prevention, detection, warning, suppression, and self rescuer technology have all contributed to this improvement. During an underground fire emergency, time is a critical factor due to the limited number of escapeways underground and the speed with which underground workings can be contaminated by products of combustion (POC). Improved fire detection is important because early warning of fire allows underground personnel greater time to initiate evacuation and fire control activities.

Previous efforts to improve fire detection technology have focused on the detection instruments themselves, as well as their utilization in sophisticated mine monitoring networks. One deficiency of existing fire detection technology, however, is the inability to determine the location of a fire in a complex mine network. Through careful analysis of a detection system output, personnel with knowledge of the mine's ventilation could at best, determine the general area where a fire is located.

Knowing the location of an underground mine fire is quite important and can significantly reduce associated risks. If the location is known, safe escape routes or refuge areas can be selected, rescue teams can concentrate searches where survivors will most likely be found, and fire fighters can select the most efficient route to the fire and the most effective fire fighting strategy. Also, once the location is known, other related aspects of the fire such as intensity, fuel, and growth rate can be inferred.

The simplest solution to the problem of quickly and safely locating a fire's source is to install a fire detector in every network branch in the mine. Through a straightforward application of logic, the branch in which the fire is located could be identified. However as most mines comprise hundreds of branches, installation of a detector in every branch would be cost prohibitive.

A more practical approach involves integration of a fire detection system with a ventilation network analysis computer model. In the event of fire, products of combustion (POC) flow along predictable courses in a mine, following the ventilation. It is hypothesized that by carefully noting the pattern and timing of alarms from a few strategically placed POC detectors with reference to known ventilation flows, it should be possible to determine the network branch in which a fire is located.

Although fires can themselves alter ventilation flow conditions, thus interfering with such a determination, it is further hypothesized that during the early stages of a fire the initial distribution of POC would precede ventilation disturbances. This would be true for most mine combustibles. The exception is material having a high flame spread rate such as most combustible liquids. In cases involving such material, ventilation disturbances could occur rapidly. Therefore, in areas such as fuel bays, fuel transfer points, and shops, dedicated, fast-response fire detection devices would be required in order for the fire's location to be reliably determined.

Based on these hypotheses, the Bureau of Mines developed an algorithm to utilize the real time outputs from a system of strategically placed underground detectors as inputs to a computer model for determining the location of an underground mine fire. The algorithm was evaluated by numerical analysis of the idealized flow of combustion products through a simulated mine network. Results of this analysis were quite promising, with the algorithm identifying the correct network branch or pair of adjacent network branches for nearly every potential fire location.

Although these computer simulations were encouraging, the capability of the algorithm to function successfully in an actual mine setting was still not known. These simulations incorporate certain simplifying assumptions regarding fire detection system operation and ventilation network flow. The successful application of the fire location algorithm depends on the validity of these assumptions, which can only be determined through full scale in mine testing.

Accordingly, plans were developed for such in-mine fire testing to evaluate the functional capabilities and limitations of the fire location algorithm. Due to the number and size of the fires which would be required to fully test the algorithm, it was determined that testing should not be conducted at an operating mine.

As an alternative, several experimental mines were examined for their suitability as test sites, ultimately resulting in the

selection of the Waldo mine near Magdelena, New Mexico. The Waldo mine is an inactive ASARCO property which has been leased to the New Mexico Institute of Mining and Technology (New Mexico Tech) for use as a mining research facility. It comprises about 20 miles of workings over four accessible levels with utilities and track haulage, and was considered suitably complex to realistically represent an actual multilevel hard rock mine. Through cooperation with the Mining and Geological Engineering Department at New Mexico Tech, repair and restoration of the mine was initiated in preparation for fire testing.

As this repair and restoration work was expected to last 15 to 18 months however, it was determined that an interim series of tests should be conducted. The objective of these tests was to gain further operational experience with the algorithm, particularly with respect to two key characteristics of ventilation network flow: constant fluctuations in flow and the thermal effects of a fire.

Ventilation flow conditions within a mine are influenced by a multitude of factors, some which are controllable (fan settings, bulkhead locations), some which are not (skip travel in shafts, muck train movement in drifts), and some in between (leaking vent tubing, vent doors inadvertently left open). The combined effect of these factors is that air flow at a given location, in the context of ventilation network analysis, is in reality, the average of a constantly fluctuating flow. Velocity variations of $\pm$ 60 pct or more over time periods of a few minutes are not unusual in some mines. This degree of uncertainty suggests a potential difficulty in implementing the fire location algorithm, which is based on the measurement and analysis of the sequence and timing of ventilation events, often to a precision of less than one minute. However as described below, the algorithm is based on the arrival of POC at two or more locations. Hence, the precise flow conditions at a single location are not critical. Rather, it is the flow conditions at a number of locations, relative to each other, which is important. It is hypothesized that the small yet constant fluctuations in flow which do occur will affect a given area within the mine in a similar fashion, and that consequently, the relative variation in flow between two or more locations within that area, will be very small.

The thermal effects of a fire can affect mine ventilation flow. As noted above, it is hypothesized that the initial distribution of combustion products would precede the thermal effects produced during a fire's early stages, however simulation analysis alone could not provide definitive verification of this hypothesis.

These interim tests were conducted at the Homestake mine in Lead, SD using SF_6 tracer gas as a surrogate for combustion products. This report provides a brief description of the fire location algorithm, a summary of the computer simulation tests, and the results of the SF_6 tracer gas studies.

ACKNOWLEDGMENTS

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DESCRIPTION OF FIRE LOCATION ALGORITHM

The algorithm involves three steps, as follows:

1. Instrumentation. Electronic instrumentation is installed at strategic underground locations.

Environmental parameters to be monitored include POC such as CO and CO_2, air temperature, velocity and direction, and barometric pressure. The number and placement of POC detectors is critical and is determined iteratively in a process described below. The detectors are linked through appropriate telemetry to a master control.

2. Computer Simulation. In the event of fire underground, the current state of the mine's ventilation is simulated using mine ventilation network analysis software (Edwards and Greuer 1982). The ventilation monitoring instruments (temperature, velocity, direction and pressure) provide real-time input to the network analysis program, thereby insuring a high degree of accuracy. Potential fire locations are determined through an analytic procedure which identifies all upstream network branches which are common to the detectors in alarm. As POC waves reach additional detectors, potential fire locations are re-defined (i.e.: impossible locations are eliminated). Ventilation travel times are then calculated from every network branch which is a potential fire location to every network branch in which a fire detector is located. The travel times are determined using calculated air velocities and branch lengths. Next, predicted combustion product arrival times (CPAT) are calculated from every potential fire location to each detector. CPAT is the elapsed time beginning with the arrival of POC at the first alarming detector (defined as arrival time = 0 for that detector) until subsequent detectors begin to alarm. For example, if POC reach detector "A" first, detector "B" 3.5 min later and detector "C" 2.6 min after that, the respective CPAT would be 0 for "A", 3.5 for "B", and 6.1 for "C". The CPAT are then stored in a data array.

3. Fire Location Determination. The computer engages in an array scanning routine wherein the real-time pattern of incoming detector alarms is compared to the CPAT stored in the data array. The data array fire location corresponding to the CPAT pattern which best matches the pattern of real-time incoming detector alarms is the most probable location of the actual fire.

The iterative process for determining proper detector number and placement involves three steps. First, initial detector locations are chosen arbitrarily based on such criteria as the availability of power or accessibility of a given site. Next, the data array is created through computer modeling as described above. Finally, this data array is scanned to identify entries having similar arrival time patterns. Adjustments to the number and placement of detectors are made and the last two steps repeated until no similar arrival time patterns are produced. Through this process, potential fire locations can be associated with a recognizable pattern of combustion product arrival times at the various detector locations.

At this juncture, it is important to note a characteristic of the fire location algorithm which, under certain circumstances, may make it impossible to distinguish between two or more network branches which have equal possibility of being the fire's true location. A necessary condition for the proper function of the algorithm is that the branch in which the fire is located terminate at a junction which splits the air into two independent paths leading to respective detectors. At junctions where separate splits of air merge, each split loses its individual identity (perfect mixing at the junction is assumed). From the vantage point of a downstream observer (i.e.: detector), it is impossible to determine from which split the combustion products

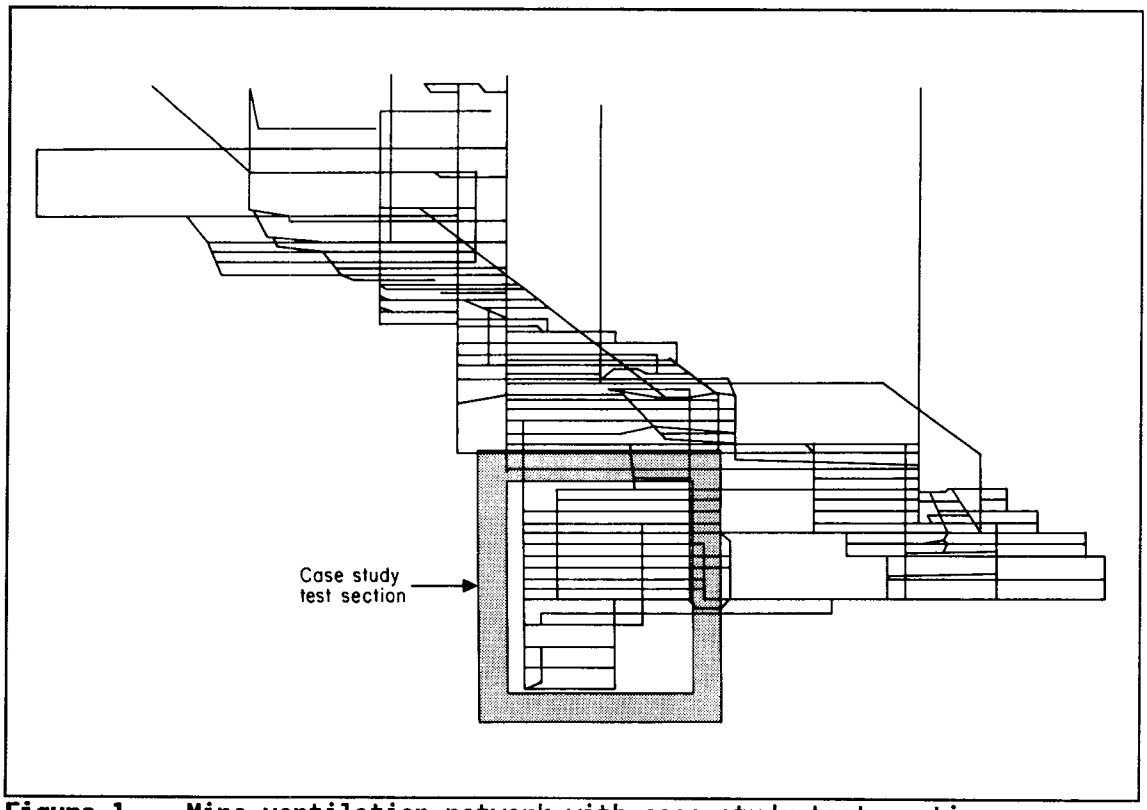


Figure 1. - Mine ventilation network with case study test section indicated

originated.

As a result of this characteristic, the algorithm is unable to resolve fire location in two situations. The first situation is where two or more branches enter a junction and two or more branches exit the same junction. If detectors are appropriately located in independent paths downstream of the junction, the algorithm would indicate equal probability of the fire occurring in each branch entering the junction. This outcome still represents a significant improvement over fire bossing, however, as the search could be limited to those branches entering the junction.

The second situation occurs where two or more branches combine at a junction to produce a single branch. In this situation, the condition requiring two independent paths to respective detectors cannot be satisfied, hence the fire's location remains uncertain. However this outcome would be limited to the return side of the ventilation system. Unless workings are ventilated with recirculated air (in which case, specialized fire detection would be provided for that area) this limitation would therefore not significantly diminish the overall value of the algorithm.

COMPUTER SIMULATION TESTS

To exercise the fire location algorithm described above, a hypothetical case study experiment was performed (Laage, Pomroy, and Bartholomew 1988). The objective of the experiment was to determine whether detector locations could be selected which would indeed result in a recognizable pattern of combustion product arrival times for every potential fire location. In order to test the algorithm under as realistic circumstances as possible, the subject of the experiment was a portion of the ventilation network from the Homestake gold mine in

Lead, SD, USA. The case study experiment was performed on a section of the mine comprising 83 branches and 48 junctions (figure 1). The experiment began by applying conventional network analysis techniques to determine air flows, velocities, and resulting branch times (time required for air to travel the entire length of a branch). Next, locations for fire detectors were specified. Using the technique described above, initial locations were selected, followed by two iterations to determine their final locations.

Even with the detectors optimally located, resolution of fire location was not possible in all cases, illustrating the algorithm's limitations described above (figure 2). Three distinct zones within the test section were produced. In the major zone, fire location was resolved to a single branch or pair or adjoining branches, representing the fully successful case. In the second zone, consisting of five network branches, identical arrival time patterns could not be eliminated without placing a separate detector in each branch. This zone represents the case on the return side of a ventilation system where two or more splits merge into a single split. The last zone consisted of independent network branches which led outside the test section. Appropriately positioned detectors outside the test section would have provided satisfactory resolution of fire location in these branches.

DISCUSSION OF COMPUTER SIMULATION TEST RESULTS

The case study test demonstrates the operation of the fire location algorithm. Several observations relating to the results of this case study experiment are worthy of note and are discussed below.

First, this test illustrates the need to maintain a high degree of control over mine ventilation parameters for the fire location system to perform properly. In some instances, the basis for specifying one network branch over another as the probable fire location was a difference in CPAT of less than 0.5 min. If knowledge of ventilation flow parameters is only approximate, the resulting inaccuracies would preclude a proper determination of fire location.

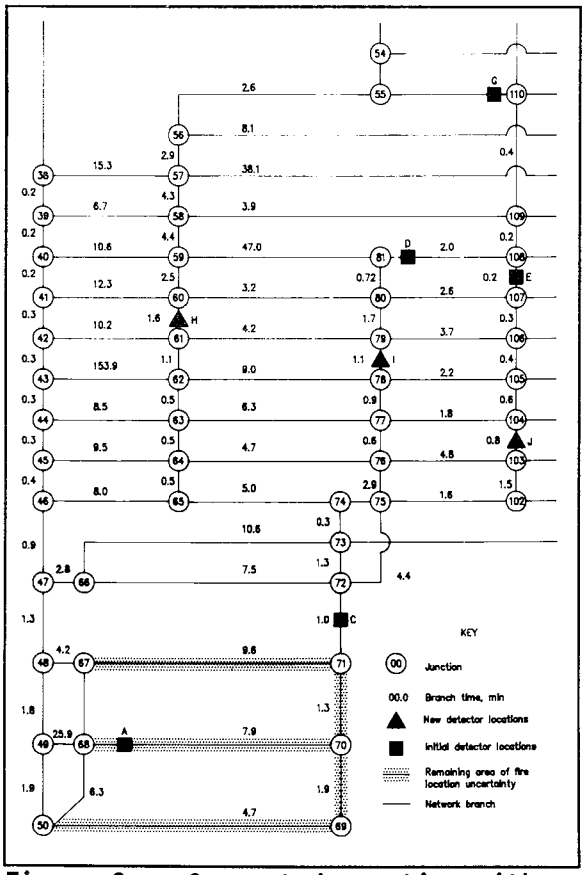


Figure 2. - Case study section with initial and new detector locations and remaining area of fire location uncertainty.

Second, this test illustrates the importance of proper detector placement. With detectors in the initial seven locations, a large "blind" area was produced (comprising 27 branches) where the location of the fire could not be specified. Only after detectors B and F were moved (to locations H and I) and detector J added, was the

system capable of performing properly. In larger and more complex networks, several detector relocation iterations could be necessary. It should also be noted that if mine conditions prevented placement of detectors in the required locations (no power, inaccessible, etc.), a system performance decrement could be expected.

Third, this test illustrates the need for a high level of detection system reliability for the fire location function to perform properly. The loss of one or two detectors (power outage, telemetry failure, zero or span drift, cell failure, etc.) would seriously compromise the ability of the system to determine the correct fire location. In addition to systematic detector and telemetry inspection and maintenance, consideration should be given to installing multiple detectors at each detector outstation to provide a backup detection capability in case of detector failure.

Finally, certain enhancements to the system are possible which could potentially improve its speed and accuracy. One obvious enhancement is the inclusion of POC concentrations in the fire location scheme. If the pattern of POC concentration was tracked, the arrival of second and third "waves" of POC (air which had originated at the fire site and traveled to the detector over well defined but slower paths than that accounting for the first arrival) could be noted. The pattern of second and third arrivals, observed at a single detector or at multiple detector locations, could help define a unique fire location where the same detectors, using first arrival times alone, would leave blind areas.

SF_6 TRACER GAS EXPERIMENTS

Tracer gas analysis using SF_6 has become an accepted method for qualitative and quantitative evaluation of industrial ventilation systems, and is particularly useful in analyzing mine ventilation networks. (Timko and Thimons 1982) In these experiments, tracer gas was released to simulate the generation of combustion products from a fire. At selected locations downstream from the release point, the air stream was manually sampled, simulating the function of a fire detector at that location. The samples were then returned to the laboratory and, following analysis to determine the arrival times of the tracer gas, the differential arrival times for each location were reconstructed.

The objectives of the tracer gas testing were twofold; to determine the effect on differential arrival times of the small yet constant and unavoidable fluctuations in ventilation flows and to determine whether a small heat source in the ventilation stream would appreciably alter differential arrival times. Differential arrival times which do not vary from test to test would indicate that the two hypotheses described above are valid and support the fire location algorithm.

Six separate tests were conducted. In all cases, the release point and the two sample points remained fixed. In the first four tests, tracer gas was released under ambient mine conditions. In the last two tests, electrical heaters generating a total of 10 KW were energized at the release point. The heaters simulated the thermal effect of a small fire to determine whether a change in differential arrival times at the sampling points could be detected.

The release point was on the mine's 5600 level approximately 79.25 m from the no. 6 winze (figure 3). Sample point number 1 was on the 5600 level 472.44 m downstream from the release point (figure 4). Sample point number 2 was on the 5900 level at the bottom of a raise which interconnects the 5600, 5750,

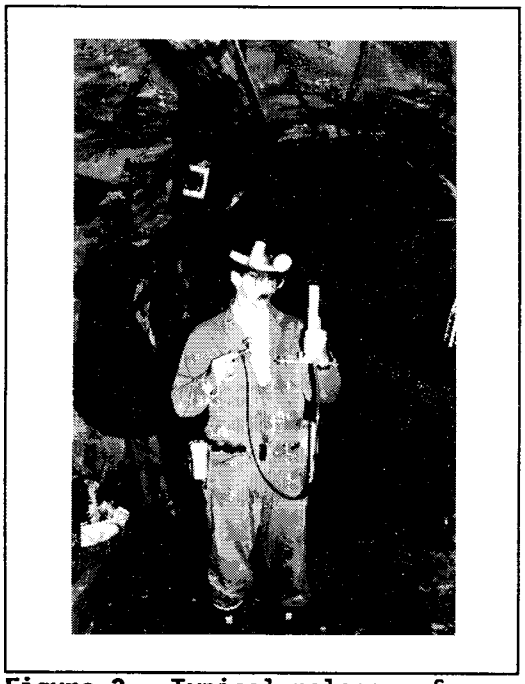


Figure 3 - Typical release of tracer gas.

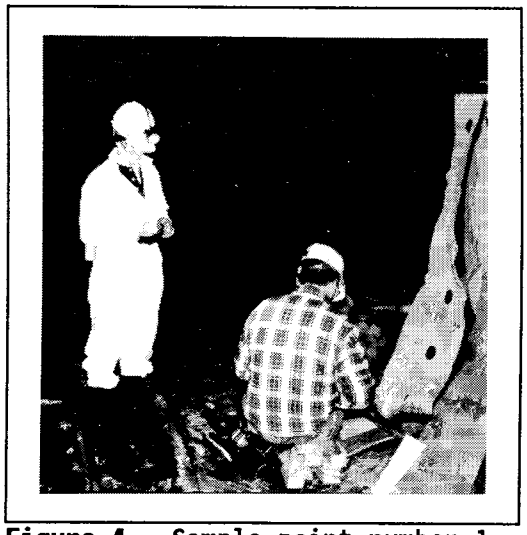


Figure 4 - Sample point number 1.

and 5900 levels (figure 5). Sample point number 2 was approximately 455.68 m from the release point.

The experimental procedure was identical for each test, and all six tests were conducted during the mine's day shift. At a predetermined time, the tracer gas was released, and the air sampling begun. The samples were collected by puncturing a re-sealable bladder in evacuated blood collection vials with a hypodermic needle (figure 6). The maximum sampling rate possible without compromising safety or data integrity was found to be one sample every 15 sec. Air sampling continued at that rate for 15 minutes at each sampling point. Following a 1 hr waiting period for the SF_6 to purge, the procedure was then repeated until the planned six tests were completed. A total of 720 samples were collected and analyzed. The analysis was performed on an S-Cubed gas chromatograph model 215-BGC capable of detecting SF_6 in the range of 1×10^{-12} to 1×10^{-8} parts concentration SF_6 per part of air (Reference to specific trade names is made for identification only and does not imply endorsement by the Bureau of Mines).

DISCUSSION OF SF_6 TRACER GAS EXPERIMENTS

Upon analyzing the data, results of two of the tracer gas tests were found to be unusable. An accidental release of SF_6 occurred while changing a regulator shortly before test 3, producing sufficient tracer gas contamination of the mine air to render meaningless, the samples of the subsequent intentional release. Unfortunately, due to lack of communication ability between the release point and the sampling points, the test could not be cancelled. The final test, number 6, also showed evidence of tracer gas contamination of the mine and the results were not defensible. This build up of the tracer gas was probably due to diffusion from a



Figure 5 - Sample point number 2.

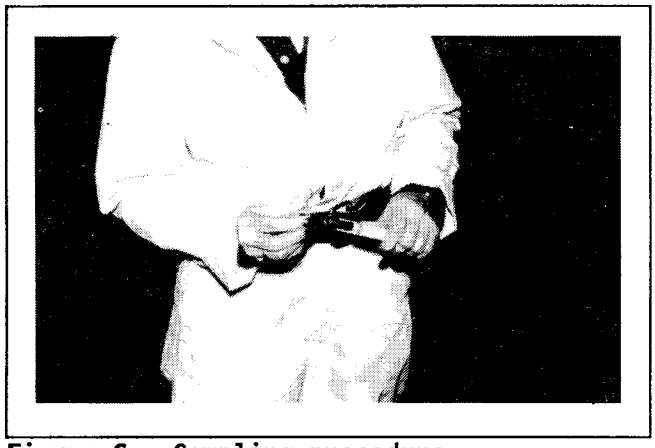


Figure 6 - Sampling procedure.

known, low volume recirculation near sample point number 1.Tracer gas arrival times were satisfactorily determined for the remaining four experiments. The concentrations for each time period at each sample point were graphed and interpreted to insure that arrival of tracer gas observed was not due to some aberration or background SF_6 levels (figure 7). Concentration levels were encountered which were much greater than the calibrated range of the gas chromatograph, which was 1 to 1,000 parts per trillion (PPT). These were assigned the arbitrary value of 2,000 PPT for graphical purposes. A typical graph is shown in figure 5.

Since sampling was not continuous (one sample every 15 sec), the actual arrival time of the tracer gas at a given sampling point could not be resolved to a precision less than 15 sec. As a result, differential arrival times could not be precisely calculated. However in each case, the spike in SF_6 concentration corresponding to the arrival of the tracer gas cloud at each sampling point was easily distinguishable on the graphs and could be resolved to a single pair of samples adjacent in time. By comparing the 15 sec ranges within which the samples were known to have arrived, a 30 sec differential arrival time range was calculated for each test. The ranges within which the tracer gas arrived at each sample point and the arrival time differential for each test are shown in Table 1.

Table 1: Tracer Gas Arrival Times and Differential Arrival Times

Test	Sample point	Arrival time, sec	Differential arrival time sec
1	1	360 - 375	
1	2	270 - 285	75 -105
2	1	285 - 300	
2	2	210 - 225	60 - 90
4	1	315 - 330	
4	2	240 - 255	60 - 90
5	1	315 - 330	
5	2	240 - 255	60 - 90

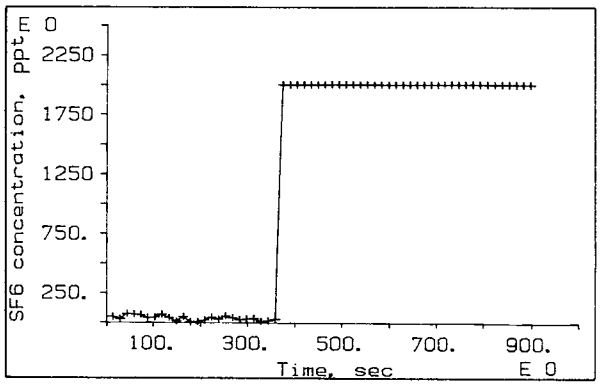


Figure 7. - Tracer gas arrival for test 1 sample point number 1.

As noted above, differential arrival times which do not vary from test to test indicate the validity of the subject hypotheses. Within the precision attainable through the experimental method used in these tests, no variance in differential arrival times could be established.

CONCLUSIONS

A technique for determining the location of an underground mine fire using a system of underground fire detectors and a specially designed ventilation network analysis computer model has been presented. The technique is simple and straightforward and can be implemented using off-the-shelf, mine worthy detection, telemetry, and computer hardware. The technique has certain limitations relating to ventilation network configuration, however the limitations affect only small parts of a network, and thus do not detract significantly from its many potential benefits.

Although the findings of the case study were favorable, the reader should be cautioned that this approach to mine ventilation and fire safety analysis is meant to supplement, and not supplant, the traditional decision making processes at a mine. This system should be regarded as a source of heretofore unavailable information which, when integrated with other relevant data, can form the basis for important ventilation decisions. It should not be used as the sole data source for an automated closed-loop feedback system of ventilation controls. The complexity of modern mine ventilation systems and the compounding effect of the sometimes unpredictable human element dictate that such controls be exercised only by teams of trained and experienced experts. The value of the fire location system is in the information it supplies to human decision makers.

The ever present threat of mine fires requires decisive action to protect against losses of life and property. Application of advanced technology such as the fire location system herein presented can assist in this endeavor.

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LOCALISATION ASSISTEE PAR ORDINATEUR DES INCENDIES MINIERS SOUTERRAINS

L.W. Laage , W.H. Pomroy et A.M. Bartholomew

Dans toute l'histoire écrite, les incendies miniers ont toujours constitué des problèmes désastreux. L'expérience a démontré que l'on peut contrôler plus facilement les incendies lorsqu'ils sont détectés à leurs débuts; en outre, l'on est également mieux à même de sélectionner des voies de sauvetage appropriées. Les progrès récents de la technologie de détection des incendies miniers ont permis d'installer des systèmes fiables; il n'est toutefois pas encore pratique d'installer des détecteurs dans tous les emplacements. Pour des raisons de sécurité et économiques, les mines abandonnées et les emplacements dangereux excluent l'installation de détecteurs. Il s'ensuit donc que s'il est possible de détecter et de localiser certains incendies à leurs débuts et de pouvoir en général les combattre de manière efficace, d'autres incendies, même détectés à leurs débuts, peuvent aboutir à un désastre si on ne les localise pas dans un bref délai.

Notre rapport examine les récents travaux du Bureau ayant pour objectif de développer une stratégie de détection d'incendies miniers en temps réel en utilisant un nombre minimum de détecteurs placés de manière stratégique et en interprétant des données par ordinateur. Il présente le développement d'un programme d'ordinateur exécutant la tâche de localisation de l'incendie en temps réel et comprend des simulations informatisées illustrant et validant l'algorithme.

COMPUTER-GESTÜTZTE AUFSPÜRUNG VON BRANDSTELLEN IN UNTERTAGEGRUBEN.

L. W. Laage, W. H. Pomroy, A. M. Bartholomew

Von jeher waren untertage auftretende Brände ein katastrophales Problem. Durch Experimente wurde gezeigt, daß frühzeitig entdeckte Feuer wesentlich leichter zu kontrollieren sind und eine vernünftigere Wahl von Fluchtwegen erlauben. Neue Fortschritte in der Technologie zur Aufspürung von Gruben-Feuern ermöglichen die Einrichtung verläßlicher Systeme. Dabei ist jedoch die Installation von Feuermeldern an allen Orten praktisch heute noch nicht durchführbar. Aufgegebene Grubenbaue und unsichere Gebiete schließen die Anlage von Sensoren aus sicherheitlichen und wirtschaftlichen Gründen aus. Das Endresultat ist, daß einige Feuer früh erkannt, lokalisiert und effektiv bekämpft werden können, während andere Feuer, auch wenn sie früh entdeckt werden, bei unbekannter Lage eine Katastrophe zur Folge haben können.

Der Vortrag beschreibt neuere Arbeiten des Bureau of Mines zur Entwicklung einer Strategie zur Aufdeckung von Bränden in Echtzeit, wobei ein Minimum von an ausgesuchten Orten installierten Sensoren benutzt wurde, deren Daten durch Computer interpretiert wurden. Die Entwicklung eines Computerprogrammes zur Durchführung einer Echtzeitanalyse für die Auffindung von Feuern wird diskutiert, und Computersimulationen werden zur Illustration und Bestätigung des Algorithmus herangezogen.

НАХОЖДЕНИЕ ПОЖАРА В ПОДЗЕМНОЙ ШАХТЕ С ПОМОЩЬЮ ЭВМ

Л.У. Лааге, У.Х. Помрой и А.М. Бартоломью

Исторически, пожары в шахтах всегда были страшным несчастьем. Опыт показывает, что гораздо легче бороться с пожаром и проще найти выходы из опасных участков, если пожар обнаружен в самом зачатке. Успехи в области разработки технологии обнаружения пожаров в шахтах позволяют установить надежную систему обнаружения, однако установка детекторов в каждой точке попрежнему остается нерентабельной. Заброшенные штреки и опасные места не позволяют установить датчики из-за опасности и неэкономичности. В результате некоторые пожары обнаруживаются и локализуются достаточно рано, что обычно способствует их эффективному укрощению, в то время, как другие пожары, хоть и вовремя обнаруженные, могут кончиться трагедией, поскольку их местоположение не становится быстро известным. Настоящий доклад описывает успехи Бюро по выработке стратегии точного нахождения пожара в шахте в короткое время, используя минимальное количество выборочно установленных датчиков в сочетании с интерпретацией данных на ЭВМ. Обсуждается разработка вычислительной программы, способной выполнить задачу точного нахождения пожара в короткое время. В порядке иллюстрации и доказательства алгоритма применяется машинное моделирование.

计算机辅助地下矿火定位

L. W. 雷格， W. H. 帕姆罗埃 及 A. M. 巴淑罗缪

有史以来矿火总是一个灾害问题。经验指出如果矿火能被及早觉察，控制就比较容易，而且可以选好适当的逃脱路线。近来矿火侦察技术的进步已经使得这种设备非常可靠，但在每一个场所都装有侦测器却是不符合实际的事。从安全和经济的观点来看，废巷道和不安全场所当然不宜装设传感器。结果是有些矿火能够很早侦察定位，而易于控制，另一些矿火虽能够及早侦察，但位置不能很快测定，便会发生灾难。

本文叙述矿业局最近的研究，我们发展出一种策略，用很少的适当地位放置的传感器和计算机连接起来分析数据，可以在实时确定矿火的位置。这里讨论一个用来实时确定矿火定位的计算机程式，并且用计算机模拟来解释和证实地下矿火定位程式的算法。

РАЗВИТИЕ ТЕПЛОВЫХ МЕТОДОВ ГАЗОВОГО АНАЛИЗА ШАХТНОЙ МЕТАНОМЕТРИИ И ЭКСПЛОЗИМЕТРИИ

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Контроль содержания рудничных газов на шахтах Советского Союза производится на основе термокаталитического принципа с использованием низкотемпературных каталитических элементов. Теоретические и экспериментальные исследования ИГД им. А.А.Скочинского по изучению процессов тепломассообмена на таких элементах послужили основой для разработки метода косвенной метрологической поверки стационарных шахтных метанометров без применения поверочных газовых смесей и автоматической коррекции показаний датчиков метана, принципов управления процессами тепломассопереноса в эксплозиметрических датчиках, обеспечивающих снижение погрешности измерения в несколько раз. Разработано новое поколение термокаталитических датчиков горючих газов, отличающихся высоким быстродействием (постоянная времени 0,2 с), малым потреблением энергии (потребляемая мощность 150 милливатт), улучшенными эксплуатационными характеристиками.

Для контроля высоких концентраций метана в дегазационных газопроводах дальнейшее развитие получил термокондуктометрический принцип газового анализа, основанный на использовании литого платинового микропровода в кварциодной изоляции в качестве чувствительного элемента. Разработан метод линеаризации выходного сигнала термокондуктометрического датчика на уровне первичного преобразователя без корректирующих схемотехнических решений, изготовлены и испытаны опытные образцы приборов реализующих разработанный метод.

Автоматический контроль довзрывных концентраций метана на шахтах Советского Союза осуществляется с помощью газоаналитической аппаратуры и приборов, основанных на тер-

мохимическом (термокаталитическом) принципе действия. В них применяются датчики с низкотемпературными каталитически активными термопреобразовательными элементами, к которым контролируемый воздух подводится естественным путем за счет диффузии. В настоящее время в угледобывающей отрасли страны в эксплуатации находится около 20 тыс. стационарных газоанализаторов и около 100 тыс. переносных метанометров, основанных на термохимическом принципе. Этот же принцип использован в переносных эксплозиметрах, которыми оснащены в основном подразделения горноспасателей.

Указанные аппаратура и приборы хорошо зарекомендовали себя в шахтных условиях. Они отличаются высокой надежностью, простотой конструкции, относительно невысокой себестоимостью.

Вместе с тем их широкое распространение на угольных шахтах обуславливает ряд специфических требований, которые необходимо учитывать при разработке новых средств шахтной метанометрии и эксплозиметрии.

Так обслуживание стационарной метанометрической аппаратуры при относительно большом количестве датчиков, расположенных в разветвленной сети горных выработок, сопряжено с определенными сложностями, связанными с периодической выдачей на поверхность этих датчиков для проверки правильности их показаний или доставкой в шахту емкостей с чистым воздухом и поверочной газовой смесью. Отсюда вытекает целесообразность создания такой аппаратуры, в которой бы имелась возможность автоматически контролировать правильность показаний датчиков без применения баллонов с чистым воздухом и поверочными газовыми смесями, а также автоматически осуществлять диагностику ее состояния и необходимую коррекцию выходного сигнала.

При разработке новых средств контроля метана необходимо стремиться к уменьшению инерционности и потребляемой мощности датчиков. Снижение инерционности позволяет сократить величину динамической погрешности защитного отключения электроэнергии в контролируемой выработке по газовому фактору, являющегося основной функцией шахтной стационарной метанометрической аппаратуры. А уменьшение мощности, потребляемой датчиком, дает возможность увеличить длительность работы переносного метанометра без перезарядки аккумуляторов при небольших значениях массы и габаритных размеров прибора. Эти же задачи являются актуальными при

разработке новых эксплозиметров. Кроме того эксплозиметры нуждаются в существенном сокращении статической погрешности измерения, обусловленной неодинаковостью тепловых эффектов сгорания на термокаталитическом преобразовательном элементе различных газов при одном и том же значении их содержания, взятом в долях от соответствующего каждому газу нижнего концентрационного предела взрываемости.

С вопросом сокращения статической погрешности приходится сталкиваться и при разработке приборов и аппаратуры основанных на термокондуктометрическом принципе и предназначенных для измерения концентрации метана в шахтных дегазационных трубопроводах. Это связано с тем, что функция преобразования термокондуктометрического чувствительного элемента является нелинейной по своей физической сущности. Создание для таких метанометров датчика с линейной статической характеристикой позволило бы существенно упростить их схемотехническую часть за счет устранения корректирующих звеньев и, следовательно, повысить надежность приборов. Решение указанных выше задач, связанных с совершенствованием метанометров и эксплозиметров, было получено за последние годы в институте горного дела им.А.А. Скочинского. Они основаны на результатах исследований процессов тепломассообмена в термохимических и термокондуктометрических датчиках, а также на применении для изготовления термопреобразовательных элементов остеклованного платинового микропровода [I - 4].

Такой провод позволяет получить элементы различной конфигурации весьма малых размеров, у которых потребляемая мощность не превышает 750 милливатт, а постоянная времени разогрева составляет 0,2 с.

На основе применения этих элементов в термохимическом мостовом датчике разработан метод его косвенной метрологической поверки [I, 2]. Он базируется на использовании метана, непосредственно находящегося в рудничной атмосфере, и не требует доставки в шахту баллонов с поверочными газовыми смесями. В основу метода положены формирование и последующий анализ процесса газообмена в реакционной камере датчика при условии, что сопротивление переносу метана через фильтр, установленный на входе в камеру, обеспечивает при установившемся режиме газообмена снижение концентрации метана в ее объеме по сравнению с концентрацией в рудничной атмосфере в 2-3 раза.

В целях создания переходного процесса предложено на время, достаточное для выравнивания концентрации метана в реакционной камере с концентрацией в окружающей среде, снизить напряжение питания мостовой схемы и исключить тем самым протекание реакции окисления на каталитически активном элементе. После этого необходимо восстановить нормальное напряжение питания, в результате чего концентрация метана в камере станет уменьшаться до своего установившегося значения. Если параметры датчика выбраны так, что постоянная времени разогрева преобразовательного элемента много меньше постоянной времени газообмена в объеме реакционной камеры, то уравнение переходного процесса, обусловленного снижением концентрации метана в результате его выжигания имеет вид:

$$S = 10^{-2} \gamma Q_м k_п \left[\frac{C_{мА}}{1+\gamma\varphi} + C_{мА}\left(1 - \frac{1}{1+\gamma\varphi}\right) e^{-\frac{\tau}{T}} \right] \qquad (I)$$

где S – выходной сигнал датчика при протекании переходного процесса газообмена в результате частичного выжигания метана в объеме реакционной камеры, В;

γ – производительность термохимического преобразовательного элемента при окислении метана, $м^3/с$;

$Q_м$ – удельная теплота сгорания метана, $Дж/м^3$;

Кп – коэффициент передачи измерительно-преобразовательной схемы $В \cdot с/Дж$;

φ – сопротивление газообменного фильтра диффузионному переносу метана, $с/м^3$;

Сма – концентрация метана в рудничной атмосфере в объемных долях, %;

τ – текущее время, отсчитываемое от момента восстановления напряжения на мосту, с;

Т – постоянная времени газообмена в объеме реакционной камеры, С.

$$T = \frac{\varphi V_к}{1+\gamma\varphi} \qquad (2)$$

где $V_к$ – объем реакционной камеры, $м^3$.

Сигнал, выдаваемый датчиком в начале переходного процесса, когда $\tau = 0$, составит

$$S_н = 10^{-2} C_{мА} \gamma Q_м k_п \qquad (3)$$

а сигнал, выдаваемый при установившемся режиме газообмена, когда $\tau \to \infty$, равен

$$S_у = 10^{-2} C_{мА} \gamma Q_м k_п \frac{1}{1+\gamma\varphi} \qquad (4)$$

Считая для каждой конструкции датчика величины Кп и $V_к$ неизменными, из выражений (2)-(4) определяются значения μ и φ которые могут меняться в процессе эксплуатации метанометра, и вычисляется истинная концентрация метана. Для этого следует лишь предварительно сформировать переходный процесс газообмена и измерить при его протекании значения S н, S у и Т.

Отметим, что формулы (I), (3), (4) не учитывают величину возможного ухода нуля датчика, которую легко можно оценить при его работе на пониженном напряжении.

Данные, полученные при реализации рассмотренного метода, могут быть реализованы как для метрологической поверки датчиков стационарной аппаратуры, так и для автоматической коррекции ее показаний.

Использованное здесь управление степенью выгорания газов может быть применено и для повышения точности эксплозиметрических датчиков [3].

Покажем это на примере окисления двух газов, отличающихся между собой различной реакционной способностью.

При окислении на термохимическом элементе каждого из газов в отдельности выходные сигналы аналогично (4) составили бы:

$$S_л = 10^{-2} C_л \mu_л Q_л k_л k_п \qquad (5)$$

$$S_т = 10^{-2} C_т \mu_т Q_т k_т k_п \qquad (6)$$

Здесь и далее величины, отмеченные индексом "Л" относятся к газу, который легче вступает в реакцию окисления, а индексом "Т" - к газу, который окислить тяжелее.

Безразмерные коэффициенты Кл и Кт характеризуют степени выжигания соответственно газа легче вступающего в реакцию окисления и более трудно окисляемого газа.

Дополнительно введем обозначения:

$$n^*_л = 10^2 C_л / C^*_л \qquad (7)$$

$$n^*_т = 10^2 C_т / C^*_т \qquad (8)$$

$$w = C^*_т / C^*_л \qquad (9)$$

где n^*л, n^*т - соответственно концентрация "легкоокисляемого" и "трудноокисляемого" газов, выраженные в долях от их нижних концентрационных пределов взрываемости, % ;

$C_л^*$, $C_т^*$ – соответственно концентрация "легкоокисляемого" и "трудноокисляемого" газов, равные их нижним концентрационным пределам взрываемости, и выраженные в объемных долях, %.

Выполним подставку (7)– (9) в (5), (6):

$$S_л = 10^{-4} \gamma_л Q_л K_л K_п R_л^* C_л^* \qquad (10)$$

$$S_т = 10^{-4} \gamma_т Q_т K_т K_п R_т^* C_л^* w \qquad (11)$$

Для того, чтобы совпали между собой характеристика датчика по каждому из двух газов необходимо обеспечить условие:

$$S_л = S_т \quad \text{при} \quad R_л^* = R_т^*$$

Как следует из (10), (11) для соблюдения этого условия нужно чтобы параметры датчика отвечали следующему требованию:

$$K_л / K_т = \gamma_т Q_т w / (\gamma_л Q_л) \qquad (12)$$

В ИГД им. А.А.Скочинского выполнен также комплекс исследований по совершенствованию мостовых термокондуктометрических датчиков, предназначенных для измерения высоких (до 100 %) концентраций метана.

Выполненные исследования показали, что статическая характеристика такого моста при включении рабочего и компенсационного элементов в общую ветвь описывается выражением: [4]:

$$U_{вых} = 0{,}75 U_н [1 - (1 + 10^{-2} k_л C_м)^{-1/3}], \qquad (13)$$

где $U_{вых}$ – напряжение в измерительной диагонали моста, В;

$U_н$ – напряжение начального разогрева термокондуктометрического преобразовательного элемента, В;

$$k_л = \frac{\lambda_м - \lambda_в}{\lambda_в}$$

$\lambda_м, \lambda_в$ – соответственно коэффициенты теплопроводности метана и воздуха, Вт/(м °С).

Из (13) следует, что для линеаризации выходного сигнала датчика можно управлять величиной напряжения питания моста в зависимости от концентрации метана, что адекватно управлению напряжением начального разогрева термокондуктометрического элемента. Этого относительно просто достичь в датчике путем подачи напряжения питания на мост через нелинейный делитель напряжения.

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DEVELOPMENT OF THERMAL METHODS OF GAS ANALYSIS FOR MINE METHANOMETRY AND EXPLOSIMETRY

E. Karpov and B. Basovsky

In USSR coal mines, concentrations of mine gases are monitored using low-temperature catalytic elements. Theoretical and experimental investigations conducted at the A. A. Skochinsky Mining Institute to study heat mass transfer resulted in methods of indirect metrological testing of stationary mine methanometers without testing of gas mixtures to be used; condition monitoring of stationary instruments involving automatic correction of methane sensor readings; controlling the processes of heat mass transfer in explosimetric sensors.

The above methods led to a new generation of thermo-catalytic sensors of combustible gases, featuring high response time (time constant is 0.2 sec.), low power consumption (power consumption is 150 milliwatt), better operating characteristics.

Thermoconductometric gas analysis, based on the use of a cast platinum microwire with quartz insulation, was developed further for monitoring high methane concentrations in gas drainage pipelines. The linearization of the thermoconductometric sensor output signal at the level of a primary transducer without any correcting circuit engineering solutions has been developed; prototypes of instruments based on the above method have been manufactured and tested.

DEVELOPPEMENT DE METHODES THERMIQUES D'ANALYSE DES GAZ POUR LA METHANOMETRIE ET L'EXPLOSIMETRIE MINIERES

E. Karpov et B. Basovsky

Dans les mines de charbon de l'URSS, on surveille les concentrations de gaz miniers au moyen d'éléments catalyseurs à basses températures. Des recherches théoriques et expérimentales effectuées à l'Institut des Mines A.A. Skotchinsky pour étudier la transmission des masses thermiques ont permis de développer des méthodes d'essais métrologiques indirects sur des méthanomètres miniers fixes sans effectuer d'essais sur les mélanges gazeux; elles ont en outre permis d'établir des méthodes de surveillance d'instruments fixes avec correction automatique des lectures des détecteurs de méthane, ainsi que des méthodes de contrôle des processus de transmission des masses thermiques dans les détecteurs explosimétriques.

Toutes ces méthodes ont permis de créer une nouvelle génération de détecteurs thermo-catalytiques des gaz combustibles, avec des temps de réponse élevés (la constante de temps est de 0,2 secondes), une faible consommation énergétique (150 milliwatts) et de meilleures caractéristiques de fonctionnement.

L'analyse thermoconductométrique des gaz, basée sur l'utilisation d'un microfil de platine moulé avec isolant à quartz, a été approfondie plus encore pour surveiller les concentrations élevées de méthane dans les conduites de drainage des gaz. On a développé la linéarisation du signal de sortie du détecteur thermo-conductométrique au niveau d'un transducteur primaire sans solutions d'ingénierie de circuits de correction. Enfin, des prototypes d'instruments ont été fabriqués sur la base de la méthode décrite ci-dessus, puis testés.

ENTWICKLUNG VON THERMISCHEN METHODEN DER GASANALYSE FÜR GRUBENGASMESSUNGEN UND FÜR DIE BESTIMMUNG DER EXPLOSIVITÄT

E. Karpov, B. Basovsky

In sowjetischen Kohlegruben wird die Grubengaskonzentration mit Niedertemperatur-Katalysatorelementen überwacht. Theoretische und experimentelle Nachforschungen, die im AA Skochinsky Bergbauinstitut durchgeführt wurden und die Wärme-Masse-Übertragung analysierten, führten zu Methoden der indirekten metrologischen Überprüfung von stationären Grubenmethanometern, wobei die zu benutzenden Gasmischungen nicht geprüft wurden, sowie zu der Zustandsüberwachung der stationären Instrumente, wobei die Methanometeranzeigen automatisch korrigiert wurden, und zu der Steuerung des Prozesses der Wärme-Masse-Übertragung in Explosionssensoren.

Diese Methode hatte die Entstehung einer neuen Generation von thermo-katalytischen Sensoren für brennbare Gase zur Folge, welche eine hohe Reaktionsgeschwindigkeit (Zeitkonstante gleich 0.25), einen geringen Stromverbrauch (Stromverbrauch gleich 150 mW) und bessere Betriebseigenschaften aufweisen.

Die thermo-conductometrische Gasanalyse, die auf der Verwendung eines gegossenen Platin-Microdrahtes mit Quarzisolation basiert, wurde für die Überwachung von hohen Gaskonzentrationen in Gasabsaugleitungen weiterentwickelt. Die Linearisierung des thermo-conductometrischen Ausgangssignals auf das Niveau eines primären Umwandlers ohne jeglichen Korrektur-Schaltkreis wurde entwickelt, und auf Grund dieser Methode wurden Prototypen von Instrumenten hergestellt und getestet.

矿井甲烷试验及爆炸试验的气体分析热力方法的发展

E。卡卜夫　及　B。巴索夫士基

苏联煤矿里矿井沼气浓度用低温催化剂来侦测。A。A。司各青士基煤矿学院在热传导及物质传递方面所做的理论及实验研究推导出矿内甲烷间接计量方法，而不须要做混合气体的试验；同时在固定式的自动校正甲烷传感器读数仪器作了测定；并在爆炸试验传感器里控制了热传导及物质传递。以上所述的方法为燃烧气体热力催化传感器开辟了一个新的纪元，使得作用时间加快（时间常数为0。2秒）动力消耗减低（动力消耗为150毫瓦），作用效果较佳。用铸造的细铂丝以石英绝缘的热传导气体分析仪并发展为测定排气管系统里甲烷高含量之用。主变换器等极上热传导传感器输出信号的线性化不用任何校正电路已经发展成功；上述的这些仪器均已制成并试验过。

ANALYSIS OF THE DYNAMIC BEHAVIOR OF ELECTRONICALLY REGULATED POWER SUPPLIES AS A SUBSTITUTE FOR TESTING INTRINSIC SAFETY BY SPARK-IGNITION TESTS IN EXPLOSIVE MIXTURES

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Intrinsically safe power supplies of category ib according to IEC 79-11 (current limiting by semiconductors) have in comparison to category ia (current limiting by resistors) the advantage of higher power output and lower source impedance. The disadvantages are

1 The electronic current limiters have to be retested with the spark test apparatus (IEC 79-3) when used with nonresistive loads, because e.g. inductances (even when diodeshunted) can change the behavior of the electronic regulators in such a way that ignition is possible.

2 If a component that influences the dynamic behavior has to be modified, a new spark-ignition test has to be made. If the modified power supply is to be installed in already existing systems, many tests have to be repeated.

The above disadvantages reduce the flexibility of intrinsically safe process control technology.

One possibility for solving this problem is to analyze the dynamic behavior of the electronically regulated power supplies. The results are then used to decide whether or not a new spark-ignition test for intrinsic safety has to be made.

In this way it is possible to reduce the times for designing and testing intrinsically safe power supplies.

INTRODUCTION

The intensive automation and remote control in modern coal mines has inducedced an exponential increase of the number of electrical apparatus with type of protection "i" -intrinsic safety. This causes the need for high-powered intrinsically safe power supplies, being able to supply as many apparatus as possible without galvanic isolation or to operate over great distances. Circuits are called intrinsically safe, if the elements in the circuit cannot cause ignition by heat and the possibility of spark ingition has been excluded by test with the spark test apparatus as defined in IEC 79-3 (1). The intrinsic safety of a power supply with current limitation by resistors can simply be assessed by using published reference curves. It is reasonable to summarize the essential components for intrinsically safe power supplies in a standard as it has been done in the UK (2). The intrinsic safety of the classical "ia"- supply (Figure 1) with the static characteristics according to Figure 2 can be affected by the load only, when the maximum allowable values for capacity or inductance are exceeded.
However, just a capacitor in parallel to the output (indicated with dashed lines in figure 1) makes the assessment more complicated. If the spark test apparatus is the only load, the capacitor will by reducing the slewing rate of the voltage, simulate intrinsic safety for higher electric values, as has been shown by Gray (3) and Zborovsky (4).

This spark quenching effect of the capacitor only seems to increase safety. This is demonstrated by inserting a resistor between power supply and spark test apparatus. Then the slewing rate of the voltage will be increased in the moment of breaking the contact and the sparks will become more invendive.

When testing intrinsic safety of resistive circuits, it can become reasonable to remove those capacitors.

On the other hand it has been shown by Schebsdat (11), that a capacitor has a negative effect on the intrinsic safety of inductive loads.
The advantage of power supplies with resistive current limitation is that their intrinsic safety depends on only a few components. The disadvantages are:

- low utilizable output current
- short circuit current considerably above rated current
- high power dissipation
- high source impedance

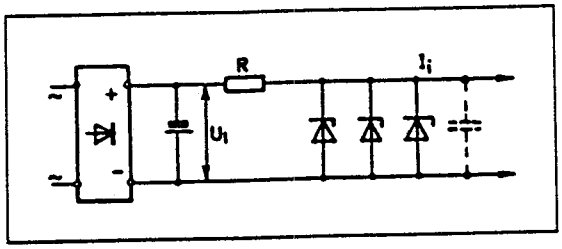


Figure 1. "ia" power supply

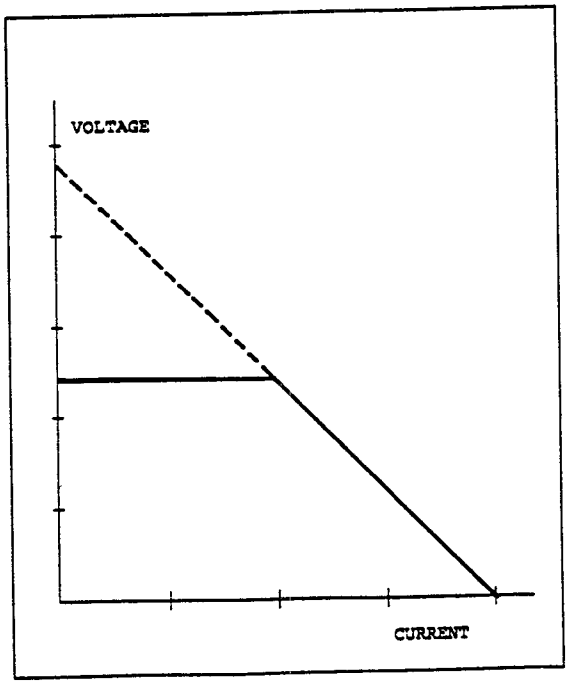


Figure 2. Output characteristic of an "ia" power supply

ELECTRONIC CURRENT LIMITATION: MORE POWER, BUT LOWER IGNITION VALUES

By using electronic devices to limit the current or spark duration an attempt has been made to avoid the disadvantages of resistive current limitation. References can be found in (5) to (10), some of which refer to investigations in 1963 and 1965.

Already Kravchenko (6) predicted by calculation, that intrinsically safe power supplies with electronic current limitation might have ignition values, which are 1/3 of those for resistive current limitation with regard to the open loop output voltage.
This has been confirmed by calculation and by experimental work, done by Tomlinson and Widginton (12), Johannsmeyer (13) and Monomakhoff (14).

Comparison of the minimum ignition values for hydrogen-air found independently by Monomakhoff and Johannsmeyer shows a very good agreement. For this reason in the following the results in (14) will be used to calculate the intrinsically safe values for voltage and current.

For the 12 V-power supply frequently used in coal mines the following facts can be derived:

For power supplies with zener diode stabilized voltage and resistive current limitation, a maximum rated current of around 0,5 A can be assumed to be realistic. Experience in Europe shows that it is possible to construct 12 V-power supplies with electronic current limitation which have currents up to 2.4 A and are intrinsically safe if tested with the spark test apparatus.
By calculation it can be shown that for 12.0 V the theoretical margin for the intrinsic safety of supplies with a rectangular characteristic is 2.3 A. If output voltage is further increased above the characteristic spark voltage of around 11 V, the theoretical limit for the safe current is drastically reduced. Table 1 gives some values, which are calculated from the formula given in (14).

U/V					
12,0	12,5	13,0	14,0	15,0	24,0
I/A					
2,31	1,49	1,09	0,70	0,45	0,11

Table 1: Voltages and currents for intrinsically safe power supplies with electronic current limitation for use in coal mines susceptible to firedamp

The values have been calculated by applying the safety factor of 1,5 to the current according to IEC 79-11. An additional condition for intrinsic safety is that at the moment of breaking the current there are no voltage transients which could within 100 - 200 µs produce sparks with energies above the minimum ignition energy.

THE INFLUENCE OF THE LOAD ON THE INTRINSIC SAFETY OF POWER SUPPLIES WITH ELECTRONIC CURRENT LIMITATION

At least since the basic investigations in (12) it should be recognised that the intrinsic safety of electronically limited circuits depends much more on the load than in the case of resistive limitation, especially if the electrical values are above those which have been found by experiment as minimum igniting values for rectangular sources. Of course it is possible to determine maximum allowable values for inductance and capacity for those power supplies. But there is a need for additional tests.
Diode shunted inductances can affect the regulation characteristic of components which are triggered by current alterations in such a way that incendive sparks occur.
Even electronic regulators which are situated between the supply and the sparking device can have a similar effect by suppressing the voltage and current transients caused by the spark.

Diode shunted coils (solenoids, relays) and regulators are very often used in underground intrinsically safe apparatus. This leads to a large number of tests, because for every particular load a spark-ignition test has to be made.

HIGH EFFORT FOR TESTING AFTER CIRCUIT MODIFICATION OF POWER SUPPLIES

Power supplies, which have been manufactured for a long time, and which are in use in many different intrinsically safe systems, have been tested with a large number of different complex loads including different types of cables.
A major problem arises, if one of these power supplies has to be modified. This can happen, if a semiconducting component is not longer available and has to be replaced by a similar component which does not have the same properties. If it is not possible to prove that the new component has the same static and dynamic values as the original one, the output characteristic of the power supply has to be checked. If the static values are higher than before, all previous tests have to be repeated.
If the static values are unchanged or lower, the dynamic behavior has to be checked. If it is not possible to investigate the possible changes in the dynamic characteristics, all the spark-ignition tests have to be repeated.

ANALYSIS OF THE DYNAMIC BEHAVIOR OF ELECTRONICALLY REGULATED POWER SUPPLIES

An analysis of the dynamic behavior can hardly be done by using the contact mechanism of the spark test apparatus, because of its stochastic behavior relative to resistance and frequency.

Consequently, the analysis has to be made with a substitute load which simulates dynamic events. Most suitable for this purpose are electronic load modules, which can be regarded as resistors whose value can be modulated. They can be controlled with frequencies up to some 100 kHz.
Practical experiments with sinoidal and triangular modulation signals give characteristic current-voltage curves for different frequencies.
Figures 3 and 4 show the output characteristic of an intrinsically safe power supply for two different load frequencies with triangular modulation. For this purpose square wave pulses for the control of the load module are not very useful, because then only very few points on the vertical branch of the

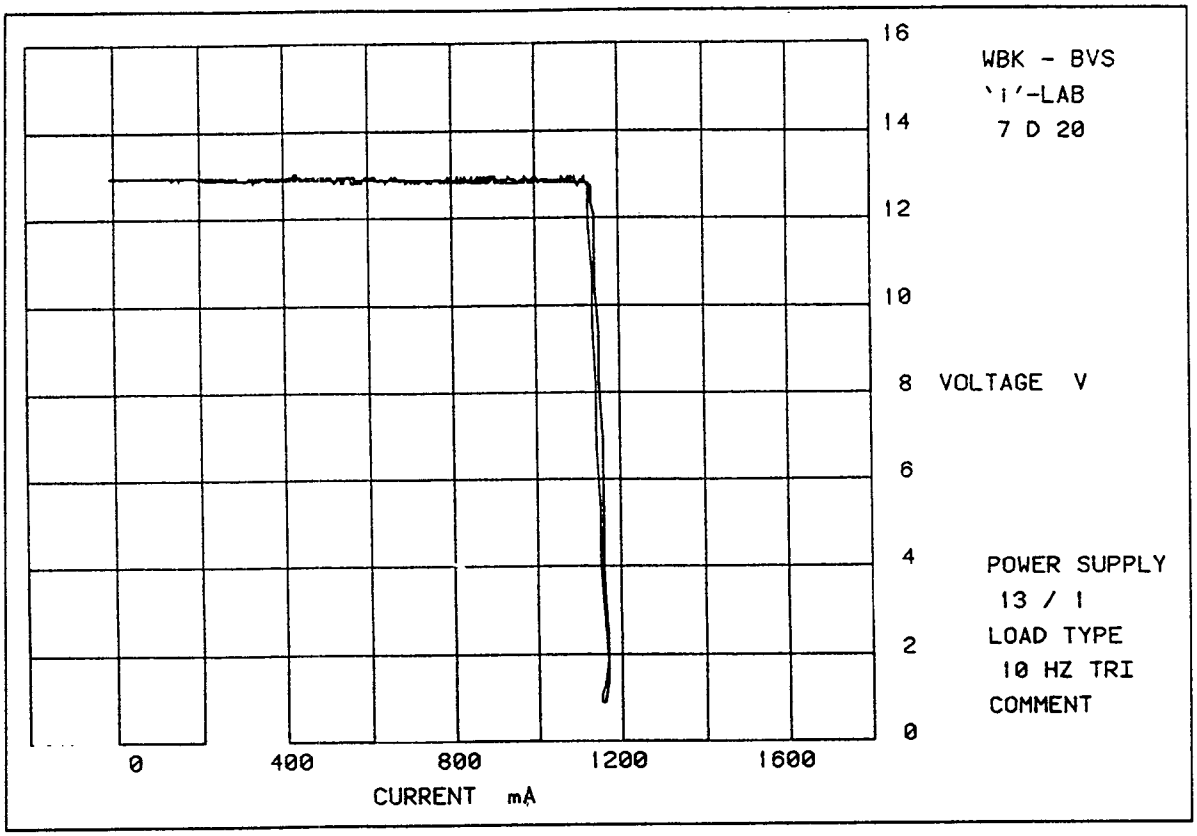


Figure 3. I/U-characteristic at 10 Hz

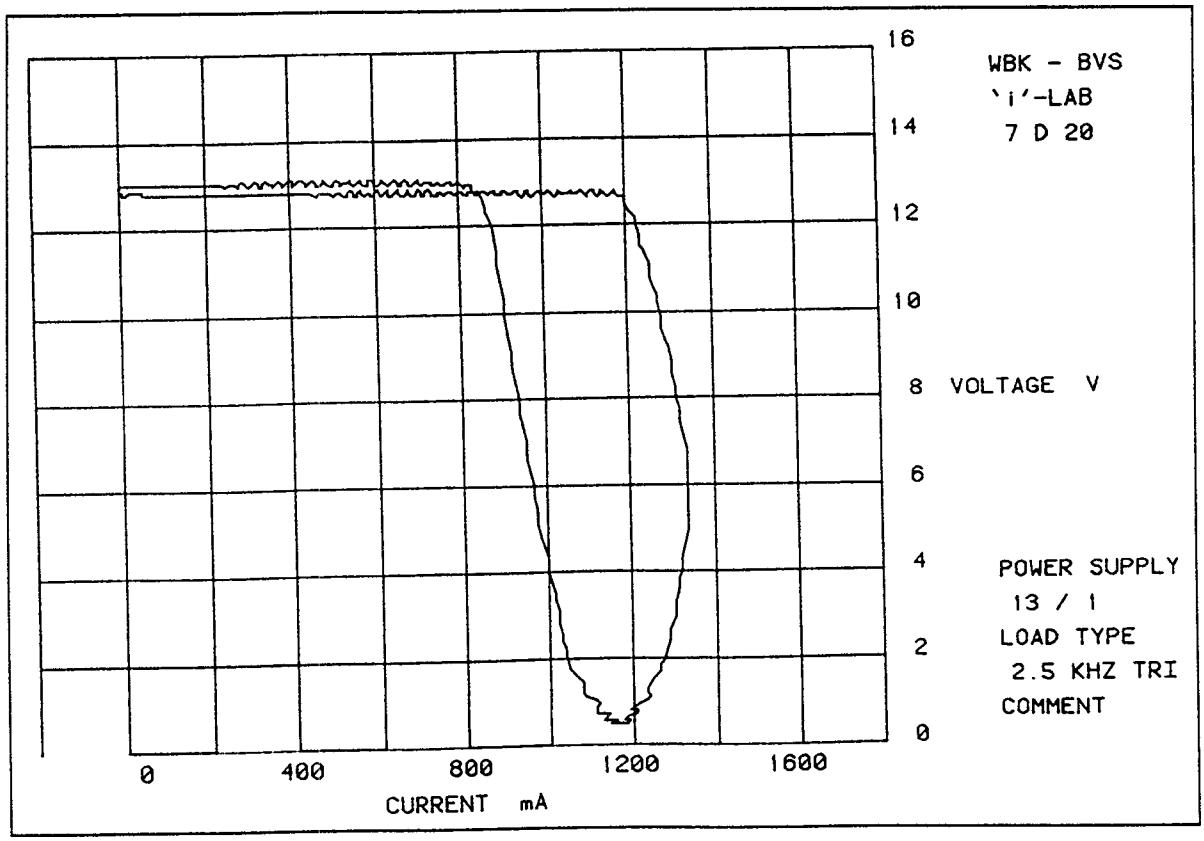


Figure 4. I/U-characteristic at 2.5 kHz

characteristic will be determined. If this method were used to analyze the possible changes in the dynamic behavior, it would be necessary to screen the output characteristic over the whole frequency range of interest (DC - 10 kHz, for switching regulators - 100 kHz). This is the reason why this method has been rejected. But it can be used as a simple way to detect instabilities in the regulators.

In addition one has to realize that testing intrinsic safety is normally based on a transient change in the load, so the work was concentrated on the load modulated with square wave rectangular pulses.

In the first step the investigation only covered the typical electronically stabilized power supply without switching-off or shutdown devices and no switching regulators.

COMPARATIVE MEASUREMENTS ON POWER SUPPLIES WITH ELECTRONIC CURRENT LIMITATION

Figure 5 shows the basic design of a linear regulator with an output characteristic according to figure 6. In the region A the current regulator is inactive, in the region B the voltage regulator is inactive. It is proposed to choose two measuring points MP on the characteristic, which should be placed 5 to 10 % under the maximum values for current and voltage respectively.

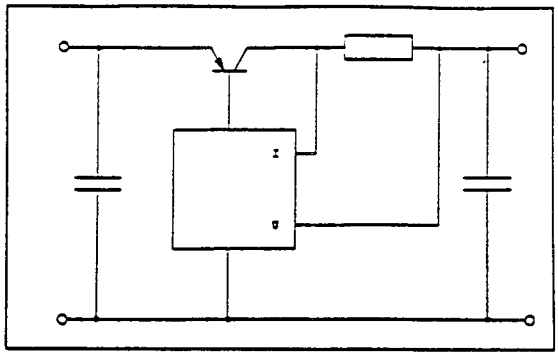


Figure 5. Linear Regulator

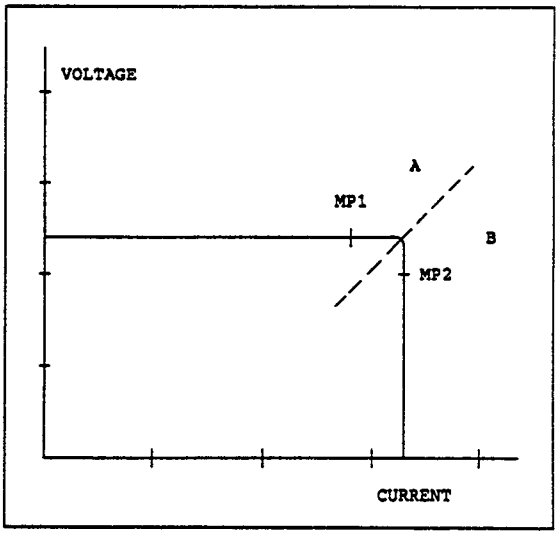


Figure 6. Output characteristic of an "ib" power supply

For analyzing the dynamic behavior the following tests are proposed:

Voltage regulation - load type 1

The output is switched between MP1 and open circuit. At measuring point MP1 the regulator is nearly completely turned on, because the supply delivers nearly the maximum possible power.

The voltage increase over the static value of the output voltage when switching to the open circuit is the transient response of the voltage regulator. The voltage increase depends also on inductances which might be part of the circuit. Figure 7 and 8 show the dynamic response of the current and voltage and the corresponding I/U - characteristic.

Voltage regulation - load type 2

The output is switched between MP2 and open circuit. In measuring point MP2 the regulator already limits the current. The voltage increase when switching to open circuit is caused by the behavior with load type 1 and in addition by the transition of the regulator from a current limiting to a voltage limiting state. Figures 9 and 10 give the typical results.

Current regulation - load type 3

The output is switched between MP2 and short circuit. The increase above the maximum static value when switching to short circuit depends on the delay time

of the current regulator. This current increase can in addition be elevated by capacitors in parallel to the output. Figure 11 and 12 show the characteristic behavior.

Current regulation - load type 4

The output is switched between MP1 and short circuit. At the measuring point MP1 the regulator is acting as voltage regulator.
The current increase above the maximum static value is caused by load type 3 and in addition influenced by the delay time caused by the transition from voltage to current limitation. Figure 13 and 14 show the results for this load type. Every single type of intrinsic safe power supply will have its typical responses for this four load types.

If the manufacturer applies for testing and certification of a modified circuit design, the modified power supply should undergo the same tests. If no significant deviations from the original behavior can be found, no retesting with the spark test apparatus has to be done. These dynamic load tests can also be done by the manufacturer himself to speed up the approval procedure.

ANALYSIS OF THE DYNAMIC BEHAVIOR BY MEASURING OF THE FREQUENCY RESPONSE OF THE SOURCE IMPEDANCE

If the output-characteristics of a power supply are almost linear for the load types 1 and 3, the test with a step function can be replaced by recording the frequency response of the source impedance of the power supply. For this test it is necessary to have an electronic load module and a vector analyzer. Guidance on this method is given in (15).

CONCLUSION

The dynamic behavior of intrinsically safe power supplies with electronic regulators can be tested with typical types of load.

Detection of changes in the dynamic behavior can be used to simplify the testing procedure needed in the following cases:

- use of modified components in certified supplies
- testing the influence of variation of the mains voltage
- testing the influence of different environmental temperatures.

If no significant changes in the dynamic behavior can be found, it is not necessary to carry out the appropriate series of tests with explosive mixtures and the spark test apparatus for intrinsically safe circuits.

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(2) BS 6182:1982 Intrinsically safe power supplies for use in coal mines, British Standards Institution

(3) Gray, R., 1981, "The effect of shunt capacitance on the intrinsic safety of power supplies", HSE Sheffield, RLSD Section paper

(4) Zborovsky, Zouzanna K., (1981), "New concepts and experimental results on the ignition limits of CR circuits"

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(6) Kravchenko, V.S., 1976 "Intrinsic Safety in Electric Ciruits, chepter 8 - Means of increasing the power in intrinsically safe systems"

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(11) Schebsdat, F., 1978, Zündverhalten von Netzwerken eigensicherer Stromkreise mit konzentriert auftretender Kapazität und Induktivität bei Gleichstrom, Physikalisch-Technische Bundesanstalt, Braunschweig, PTB-Bericht W-11

(12) Tomlinson, R., and Widginton, D.W., 1978, "Effects of slewing rate when testing intrinsically safe power supplies, HSE Technical Paper 2

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(14) Monomakhoff, A., Halama, S., and Bonnec, P., 1981, "The use of calculation in the determination of the intrinsic safety of power supplies." XIX International Conference on Mines Safety Research, Katowice,1981

(15) Pause, H., 1986, Quellimpedanz als Optimierungskriterium, Auslegung von Regelschaltungen für Schaltnetzgeräte, Elektronik 15

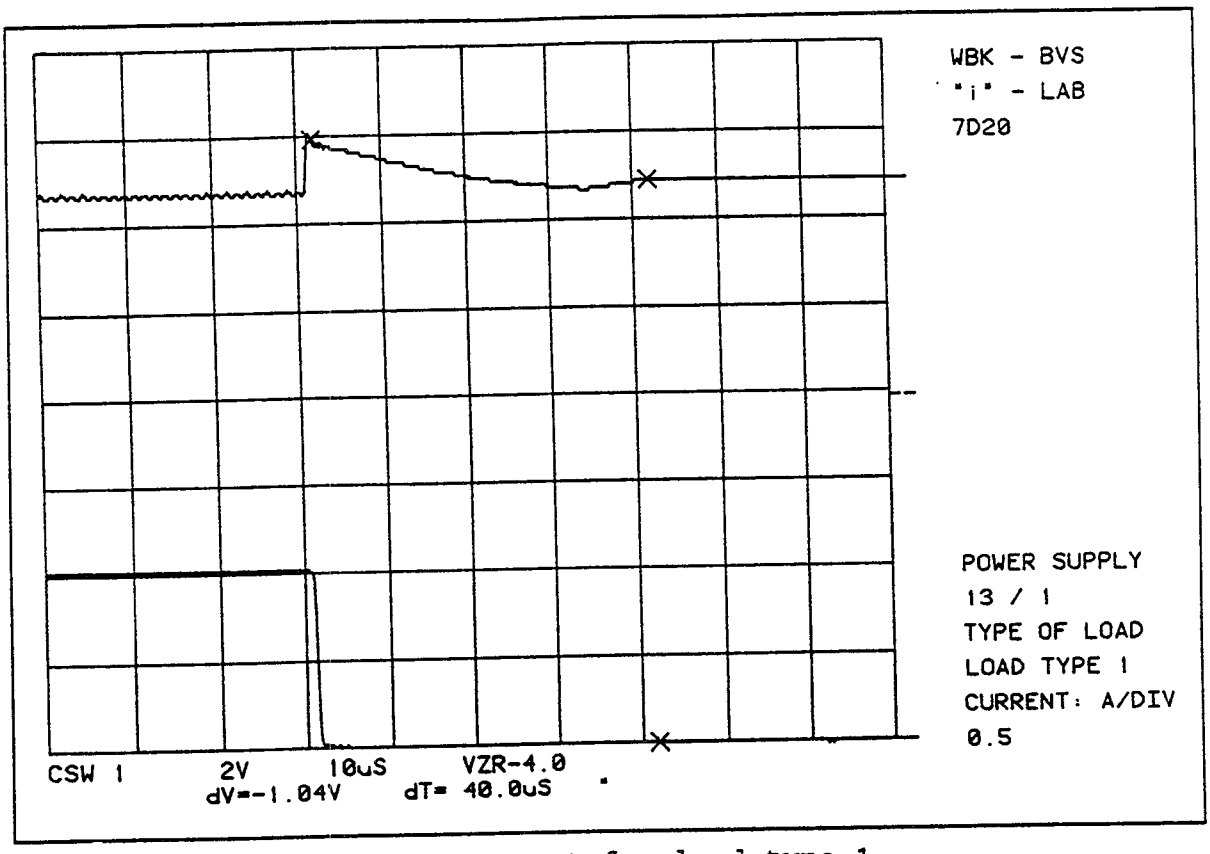


Figure 7. Voltage and current for load type 1

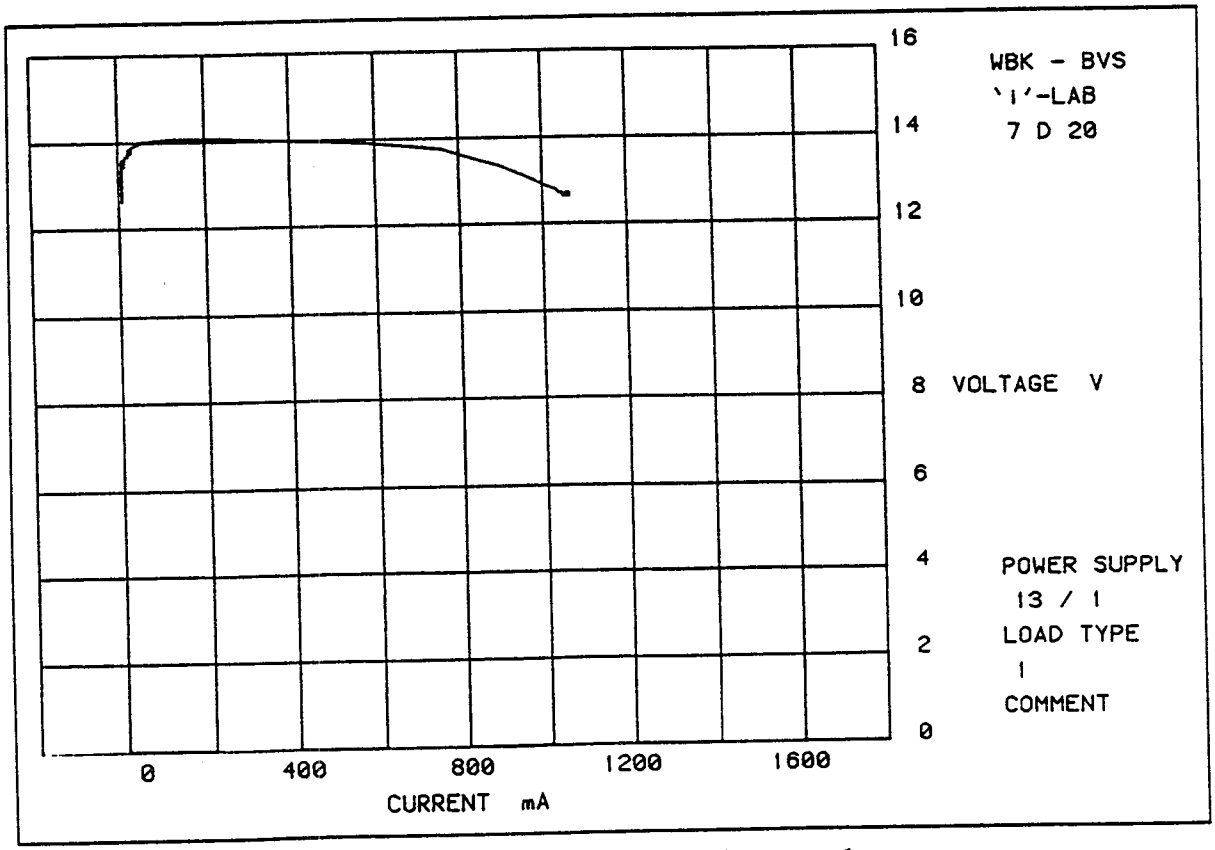


Figure 8. I/U-characteristic for load type 1

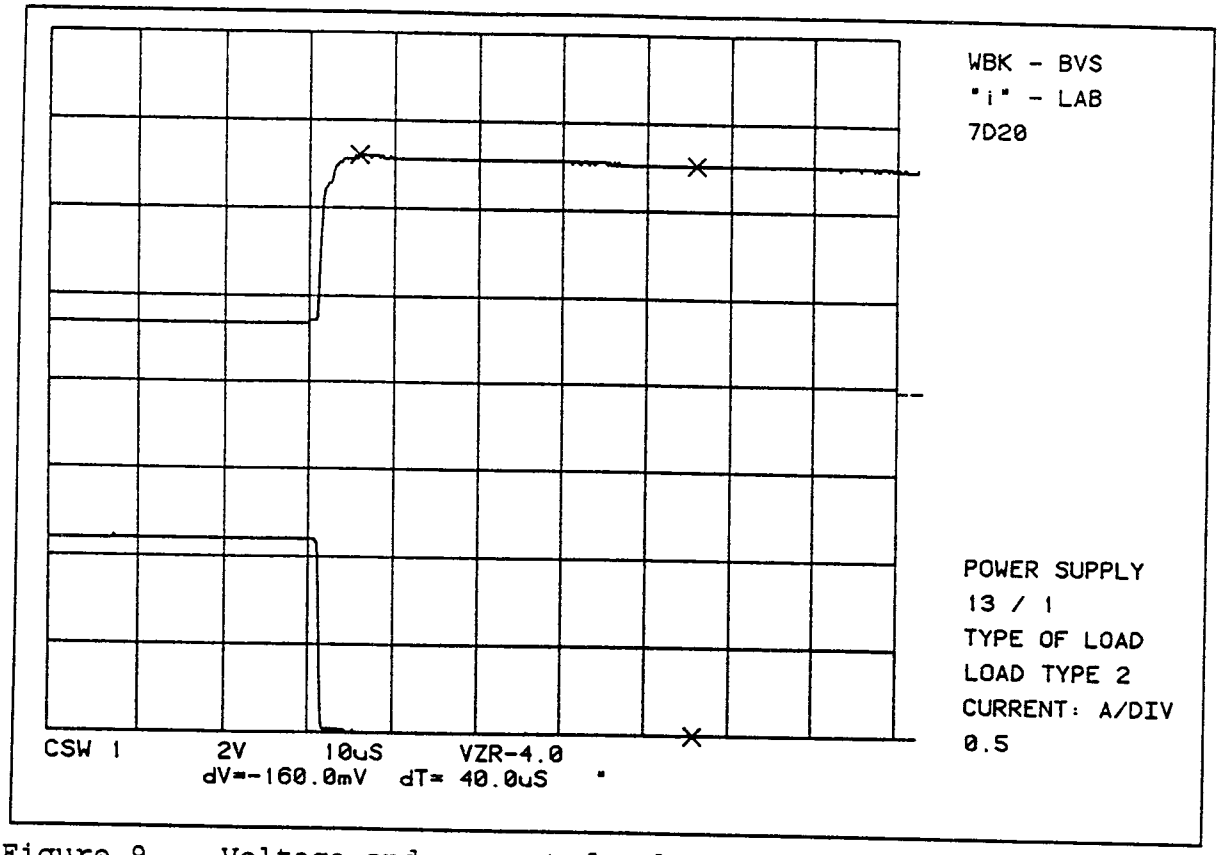


Figure 9. Voltage and current for load type 2

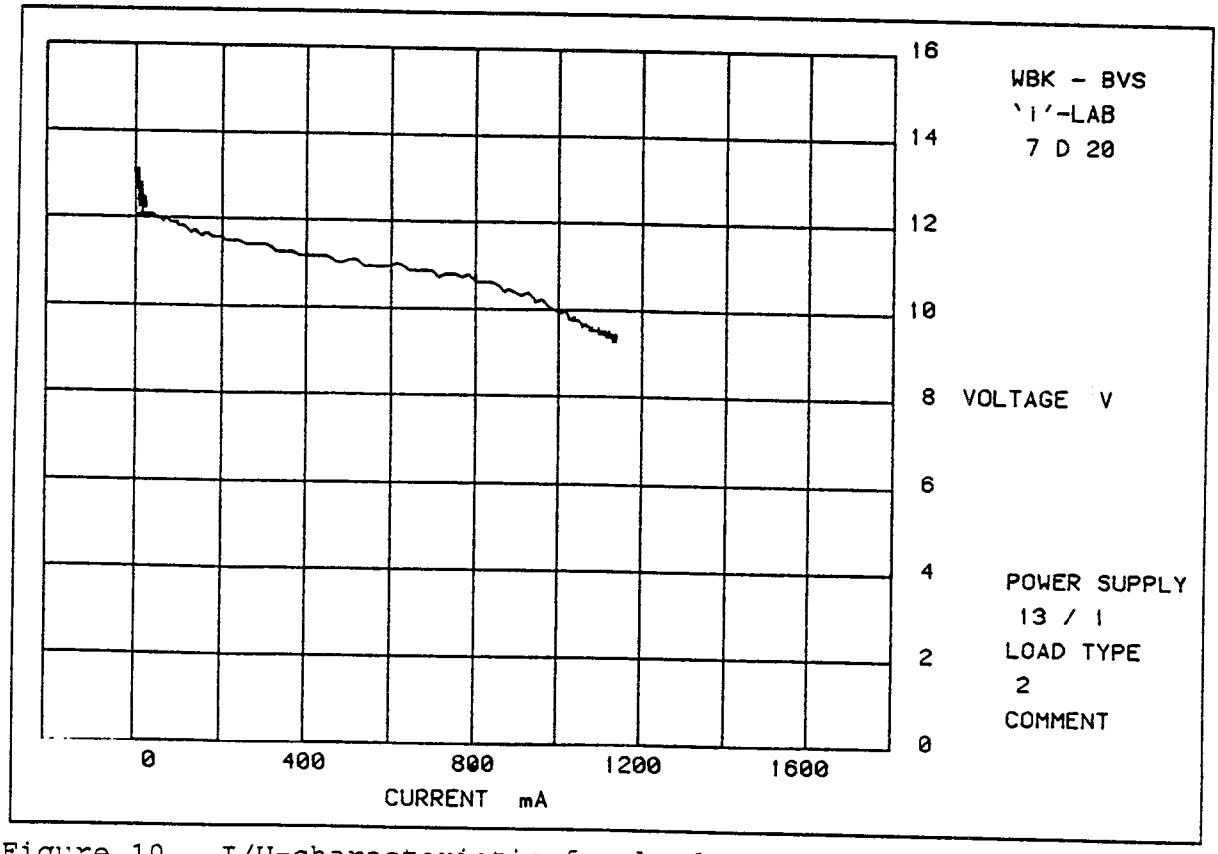


Figure 10. I/U-characteristic for load type 2

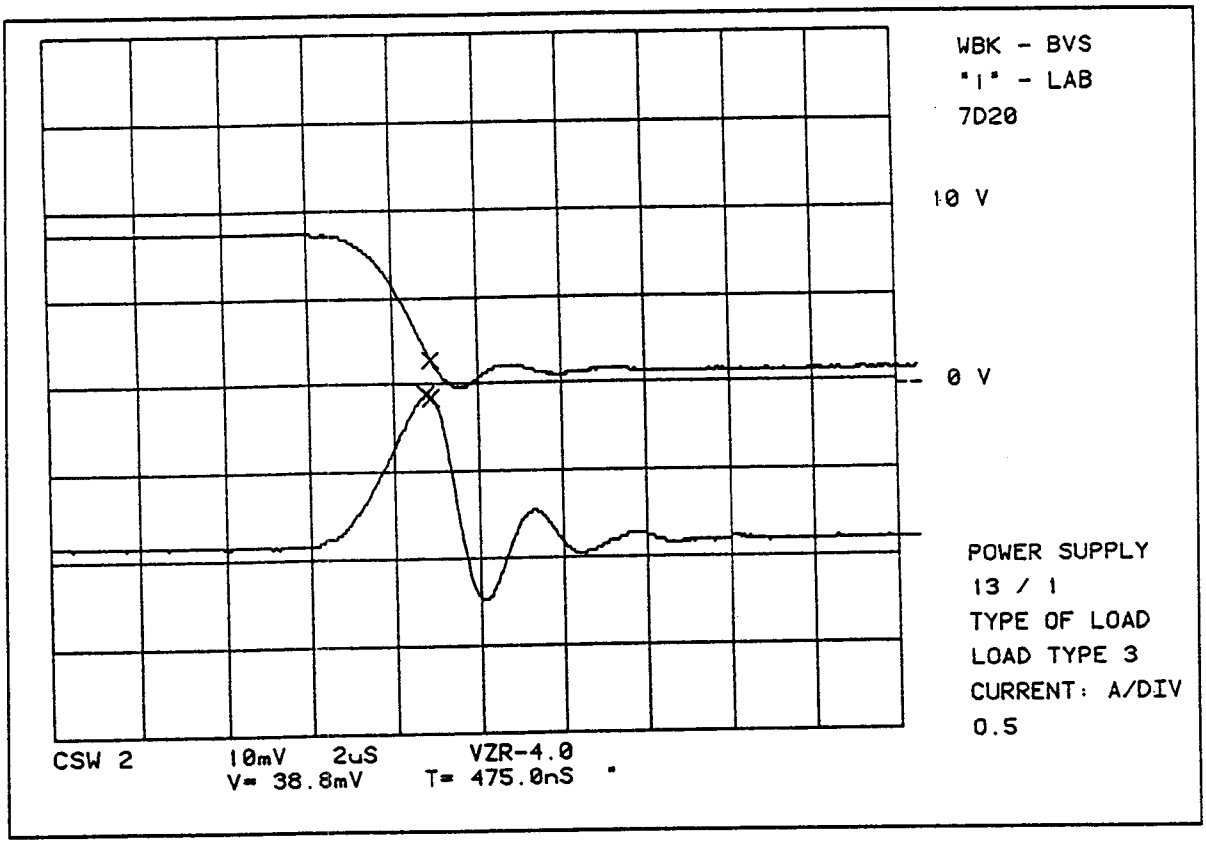


Figure 11. Voltage and current for load type 3

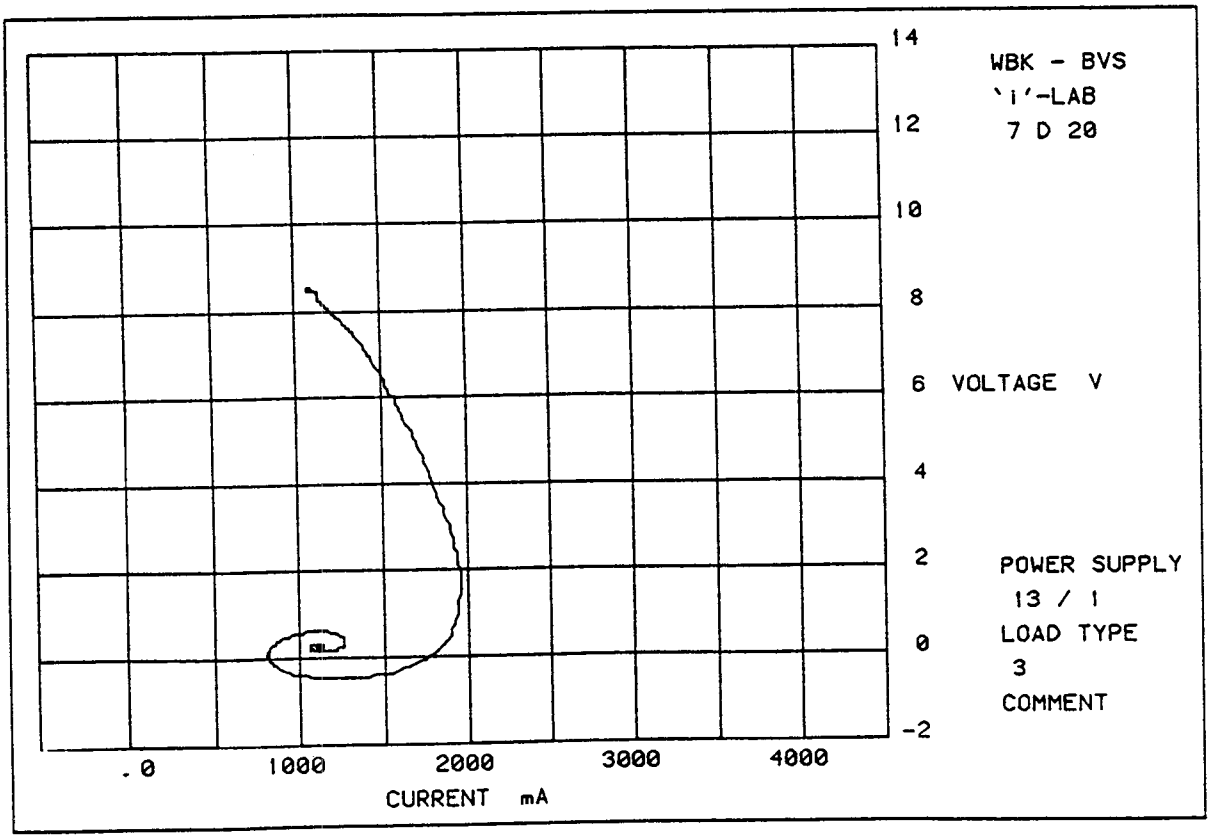


Figure 12 I/U-characteristic for load type 3

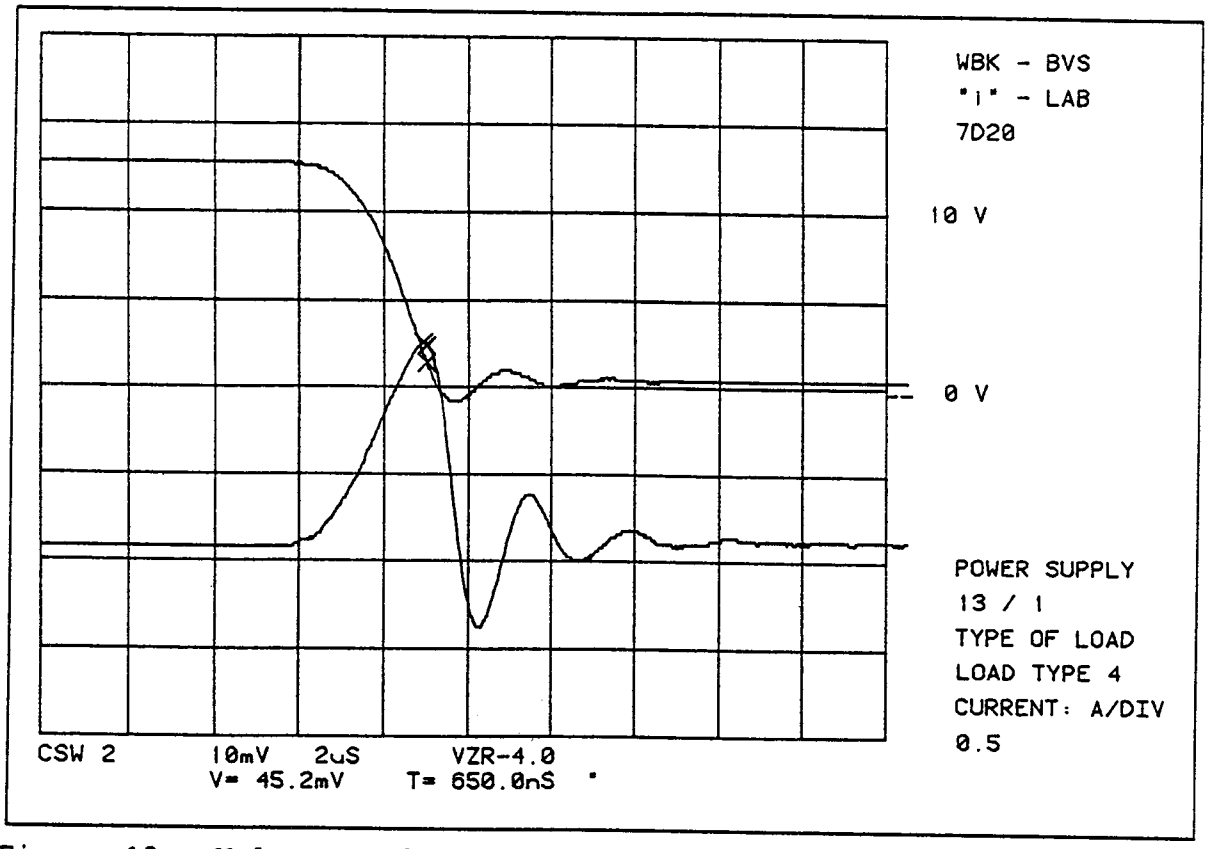


Figure 13. Voltage and current for load type 4

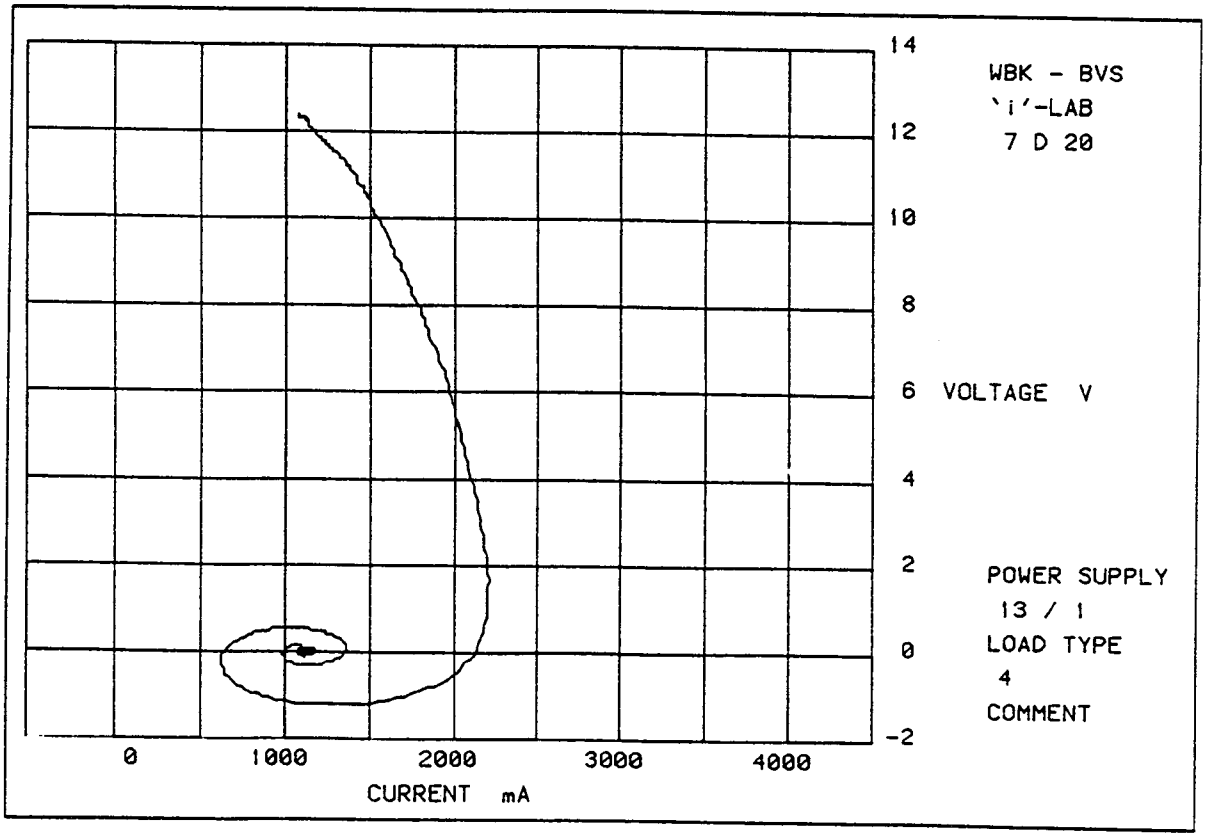


Figure 14. I/U-characteristic for load type 4

UNTERSUCHUNG DES DYNAMISCHEN VERHALTENS ELEKTRONISCH GEREGELTER STROMVERSORGUNGSGERÄTE ALS ERSATZ FÜR DIE PRUFÜNG DER EIGENSICHERHEIT IN EXPLOSIONSFÄHIGEN GEMISCHEN

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Eigensichere Stromversorgungsgeräte der Kategorie ib nach IEC 79-11 (Strombegrenzung mit Halbleitern) bieten gegenüber der Kategorie ia (Strombegrenzung durch Widerstände) den Vorteil höherer Ausgangsleistung und niedrigeren Innenwiderstandes. Die Nachteile sind:

1 Die elektronischen Strombegrenzungen müssen mit nicht-ohmschen Verbrauchern erneut mit dem Funkenprüfgerät nach IEC 79-3 geprüft werden, weil z.B. Induktivitäten (auch mit Shuntdioden) das Verhalten der elektronischen Regeleinrichtungen bis zur Zündfähigkeit verändern können.

2 Muß ein Bauteil gewechselt werden, das das dynamische Verhalten verändern kann, ist ebenfalls eine erneute Prüfung erforderlich. Soll das geänderte Stromversorgungsgerät auch in bereits installierten Anlagen eingesetzt werden, sind viele Prüfungen zu wiederholen.

Die oben geschilderten Nachteile führen zu einer Verringerung der Flexibilität der eigensicheren Prozeßleittechnik.

Eine Möglichkeit der Abhilfe bietet die Untersuchung des dynamischen Verhaltens elektronisch geregelter Stromversorgungsgeräte. Die Ergebnisse sind dann die Grundlage für die Entscheidung, ob eine erneute explosionstechnische Untersuchung der Eigensicherheit notwendig ist oder nicht. Hierdurch besteht die Möglichkeit zu drastischer Reduzierung von Entwicklungs- und Prüfzeiten.

EIGENSCHAFTEN DER ia-STROMVERSORGUNGSGERÄTE

Die weitgehende Automatisierung und Fernüberwachung moderner Steinkohlenbergwerke ist gekoppelt mit einem exponentiellen Anstieg des Einsatzes elektronischer Geräte in der Zündschutzart Eigensicherheit. Dies erzeugt ein Bedürfnis nach möglichst leistungsstarken eigensicheren Stromversorgungsgeräten, um möglichst viele Geräte (ohne galvanische Trennung) aus einem Stromkreis versorgen zu können oder um große Entfernungen zu überbrücken.

Stromkreise werden vereinbarungsgemäß dann als eigensicher bezeichnet, wenn die Elemente im Kreis keine Wärmezündung verursachen können und die Funkenzündung durch Prüfung mit dem Funkenprüfgerät nach IEC 79-3 (1) ausgeschlossen wird.

Die Eigensicherheit klassischer Netzgeräte mit Strombegrenzung durch Widerstände läßt sich auch einfach nach publizierten Zündgrenzkurven beurteilen. In Großbritannien hat man sinnvollerweise für bestimmte Spannungen die für die Eigensicherheit wesentlichen Bauteile der Netzgeräte in einer Norm (2) zusammengefaßt.

Die Eigensicherheit der klassischen "ia"-Stromversorgung nach Bild 1 mit der statischen Kennlinie nach Bild 2 kann von der Last nur durch Überschreitung der maximal zulässigen Werte für Kapazität oder Induktivität aufgehoben werden.

Schon ein Kondensator im Ausgang (im Bild 2 gestrichelt dargestellt) macht die Beurteilung schwieriger.

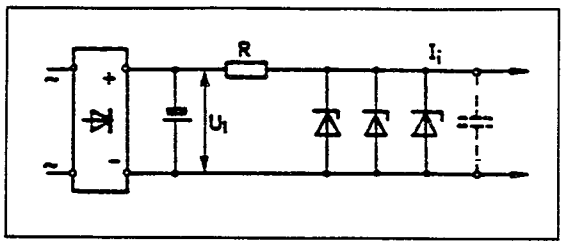


Bild 1.
"ia" Stromversorgungsgerät

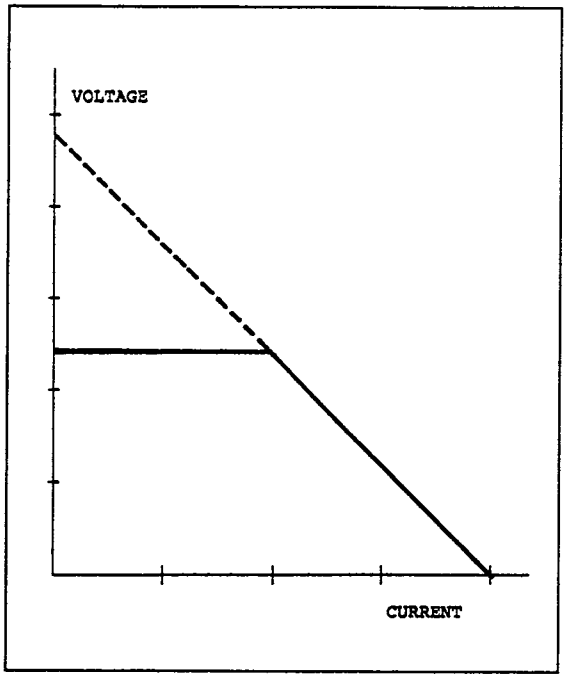


Bild 2. Ausgangskennlinie von "ia" Stromversorgungsgeräten

Wird als Last nur das Funkenprüfgerät angeschlossen, kann der Kondensator durch Verzögerung der Spannungswiederkehr höhere elektrische Werte als ohne Kondensator eigensicher erscheinen lassen, wie z.B. Gray (3) und Zborovsky (4) zeigen.

Daß dieser funkenlöschende Effekt die Sicherheit nur scheinbar erhöht, läßt sich zeigen, indem man zwischen Netzgerät und Funkenprüfgerät einen Widerstand schaltet. Durch Erhöhung der Anstiegsgeschwindigkeit der Spannung bei der Kontaktöffnung werden die Funken wieder zündfähiger.

Für die Prüfung der Eigensicherheit werden derartige Kondensatoren zweckmäßigerweise zunächst entfernt.

Für für die Anschaltung induktiver Verbraucher hat der Kondensator einen negativen Einfluß auf die Eigensicherheit, wie z. B. Schebsdat (11) nachwies.

Die Stromversorgungsgeräte mit ohmscher Strombegrenzung haben den Vorteil, daß die Eigensicherheit nur von wenigen Bauteilen beeinflußt werden. Die Nachteile sind jedoch:

- niedrige nutzbarer Ausgangsstrom
- Kurzschlußstrom größer als Nennstrom
- hohe Verlustleistung
- hoher Innenwiderstand

ELEKTRONISCHE STROMBEGRENZUNG: MEHR LEISTUNG ABER AUCH NIEDRIGERE ZÜNDWERTE

Die Nachteile der ohmschen Strombegrenzung hat man durch Verwendung elektronischer Begrenzungseinrichtungen für Strom oder Funkendauer zu vermeiden versucht. Entsprechende Hinweise finden sich z.B. in (5) bis (10), die teilweise auf Arbeiten in 1963 und 1965 zurückverweisen.

Schon Krawtschenko (6) wies rechnerisch nach, daß eigensichere Netzgeräte mit elektronischer Spannungs- und Strombegrenzung Zündwerte aufweisen, die bezogen auf die Leerlaufspannung um den Faktor 3 unter den Werten bei ohmscher Strombegrenzung liegen können.

Dies wurde von Tomlinson und Widginton (12), Johannsmeyer (13) und Monomakhoff (14) theoretisch und durch Zündversuche untermauert.

Bei Vergleich der von Monomakhoff und Johannsmeyer unabhängig voneinander gefundenen Zündgrenzkurven für Wasserstoff-Luft-Gemische ergibt sich eine sehr gute Übereinstimmung. Deswegen werden im Folgenden die Angaben in (14) für die Berechnung der Zündgrenze benutzt.

Betrachtet man die im Bergbau häufig verwendeten 12 V-Netzgeräte, ergibt sich folgendes Bild:

Für Netzgeräte mit Z-Dioden-Stabilisierung der Spannung und Strombegrenzung durch Widerstände ist ein maximaler Betriebsstrom von etwa 0,5 A realisierbar.

Wie Erfahrungen in Europa zeigen, kann man elektronisch begrenzte Geräte bauen, die bei 12 V Ströme von 2 bis 2,4 A liefern und bei Prüfung allein mit dem Funkenprüfgerät eigensicher sind.

Eine rechnerische Überprüfung dieser Werte ergibt, daß für 12,0 V die Stromgrenze für eigensichere Rechteckquellen bei 2,3 A liegt.

Sobald die Ausgangsspannung noch weiter über die charakteristische Funkenbrennspannung von ca. 11 V hinaus angehoben wird, sinkt der theoretisch ermittelte Wert für den eigensicheren Ausgangsstrom schnell ab. Tabelle 1 gibt einige Werte, die nach (12) berechnet wurden:

Tabelle 1: Spannungen und Ströme für elektronisch strombegrenzte eigensichere Stromversorgungsgeräte, für grubengasgefährdete Bereiche

U/V	12,0	12,5	13,0	14,0	15,0	24,0
I/A	2,31	1,49	1,09	0,70	0,45	0,11

Diese Werte beziehen sich auf die Anwendung des Sicherheitsfaktors nach IEC 79-11 auf den Strom.

Weitere Voraussetzung für die Eigensicherheit ist allerdings, daß im Augenblick der Stromunterbrechung keine Spannungsspitzen auftreten, die innerhalb von ca. 100 bis 200 µs Funken und Energien über der minimal nötigen Zündenergie erzeugen können.

BEEINFLUSSUNG DER EIGENSICHERHEIT ELEKTRONISCH STROMBEGRENZTER STROMVERSORGUNGSGERÄTE DURCH DIE LAST

Spätestens seit den grundlegenden Versuchen in (12) müßte auch allgemein bekannt sein, daß die Eigensicherheit elektronisch begrenzter Stromkreise wesentlich stärker von der angeschlossenen Last abhängig ist als bei ohmsch begrenzten Kreisen, vor allem dann, wenn die Ausgangswerte, die sich aus theoretischen Überlegungen und praktischen Versuchen ergebenden niedrigsten Werte überschreiten.

Zwar lassen sich die derartigen Netzgeräte Maximalwerte von Induktivität und Kapazität bestimmen. Jedoch sind darüber hinaus zusätzliche Untersuchungen notwendig.

Durch mit Dioden beschaltete Induktivitäten können die Regeleigenschaften von Stufen, die auf Stromänderungen ansprechen, soweit verändert werden, daß zündfähige Funken entstehen.

Auch elektronische Regeleinrichtungen, die zwischen Netzgerät und funkender Stelle liegen, können denselben Effekt haben durch Unterdrückung der vom Funken verursachten Strom- und Spannungsänderung.

Da sowohl beschaltete Induktivitäten (Ventilmagnete, Relais) als auch Regelschaltungen (Dreibein-Regler) in den Geräten unter Tage häufig verwendet werden, ergibt sich zwangsläufig ein besonders hoher Prüfaufwand, da für jeden speziellen Lastfall eine Funkenprüfung durchzuführen ist.

HOHER PRÜFAUFWAND BEI SCHALTUNGSÄNDERUNGEN AN DEN NETZGERÄTEN

Netzgeräte, die über Jahre unverändert gebaut und in den verschiedensten eigensicheren Anlagen verwendet werden, werden somit im Laufe der Zeit mit den unterschiedlichsten komplexen Lastfällen einschließlich verschiedener Leitungsbauarten geprüft.

Ein Kardinalproblem entsteht dann, wenn dieses Netzgerät geändert werden muß. Dies kann notwendig werden, wenn ein Halbleiterbauelement nicht mehr lieferbar ist und durch ein ähnliches ersetzt werden muß, das jedoch abweichende Daten hat. Wenn nicht nachgewiesen werden kann, daß das neue Bauteil statisch und dynamisch dieselben Werte wie das alte aufweist, muß die Ausgangskennlinie des Netzgerätes überprüft werden. Haben sich die statischen Werte erhöht, sind die vorangegangenen Prüfungen in der Regel zu wiederholen.

Sind die statischen Werte unverändert oder niedriger, ist zu prüfen, ob sich das dynamische Verhalten verändert hat.

Kann die eventuelle Änderung des dynamischen Verhaltens nicht überprüft werden, sind alle Funkenprüfungen zu wiederholen.

UNTERSUCHUNGEN DER DYNAMISCHEN EIGENSCHAFTEN

Die Untersuchung des dynamischen Verhaltens kann nur schwer unter Verwendung der Kontakt-Vorrichtung des Funkenprüfgerätes erfolgen, da es ein Gebilde mit stochastischem Verhalten bezüglich Widerstand und Frequenz darstellt.

Also ist die Untersuchung mit einer Ersatzlast vorzunehmen, die dynamische Vorgänge simuliert. Hierzu eignen sich elektronische Lastmodule, die man sich als elektronisch modulierbare Widerstände vorstellen kann. Sie können mit Frequenzen bis zu einigen 10 kHz angesteuert werden.

Praktische Versuche mit sinus- und dreieckförmigen Lastsignalen ergaben für verschiedene Frequenzen charakteristische I/U-Kennlinien.

Bild 3 und 4 zeigen die Ausgangskennlinie eines Netzgerätes für zwei verschiedene Lastfrequenzen bei Ansteuerung mit dreieckförmigem Signal. Eine Ansteuerung mit Rechteck-Impulsen kommt hier nicht in Frage, da dann zuwenig Punkte auf dem vertikalen Ast der Kennlinie abgetastet werden.

Da für die Untersuchung der Veränderung des dynamischen Verhaltens die Veränderung der I/U-Kennlinie im gesamten interessierenden Frequenzbereich (DC - ca. 10 kHz, für Schaltregler bis ca. 100 kHz) geprüft werden müßte, wurde dieser Weg nicht weiter verfolgt. Allerdings können mit dieser Methode auf einfache Weise Instabilitäten der Reglerschaltung erkannt werden.

Da zudem für die Untersuchung der Eigensicherheit in der Regel von einer sprunghaften Veränderung der Last ausgegangen wird, konzentrierten sich die Arbeiten auf Belastung mit einer Sprungfunktion.

Weiterhin wurde für die Untersuchung zunächst nur die klassische elektronisch stabilisierte Stromversorgung betrachtet, also keine Verwendung von Abschalteinrichtungen und Kurzschließer sowie keine getakteten Stromversorgungen.

VERGLEICHSMESSUNGEN AN STROMVERSORGUNGSGERÄTEN MIT ELEKTRONISCHER STROMBEGRENZUNG

Bild 5 zeigt die Prinzipschaltung eines Linearreglers mit der Ausgangskennlinie gemäß Bild 6.

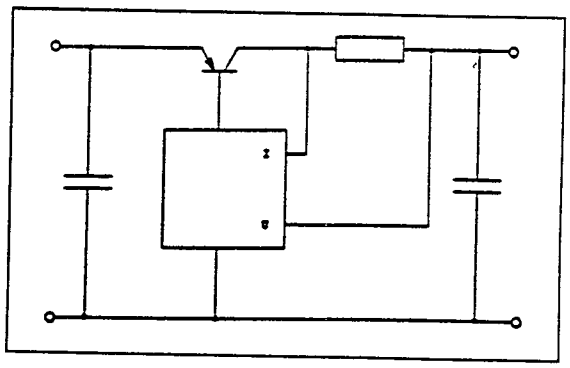


Bild 5. Linearregler

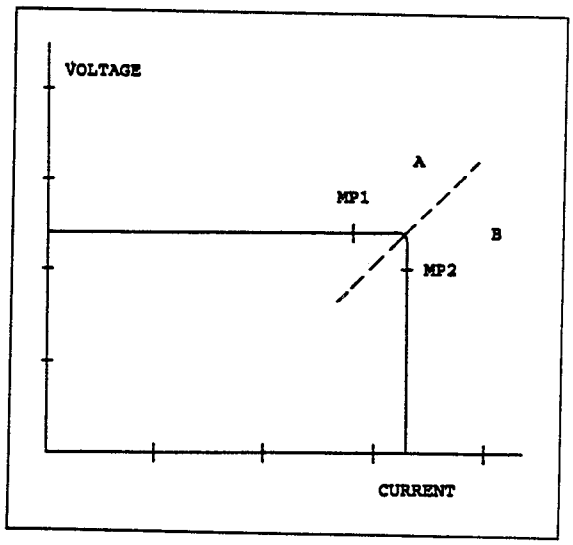


Bild 6. Ausgangskennlinie eines "ib" Stromversorgungsgerätes

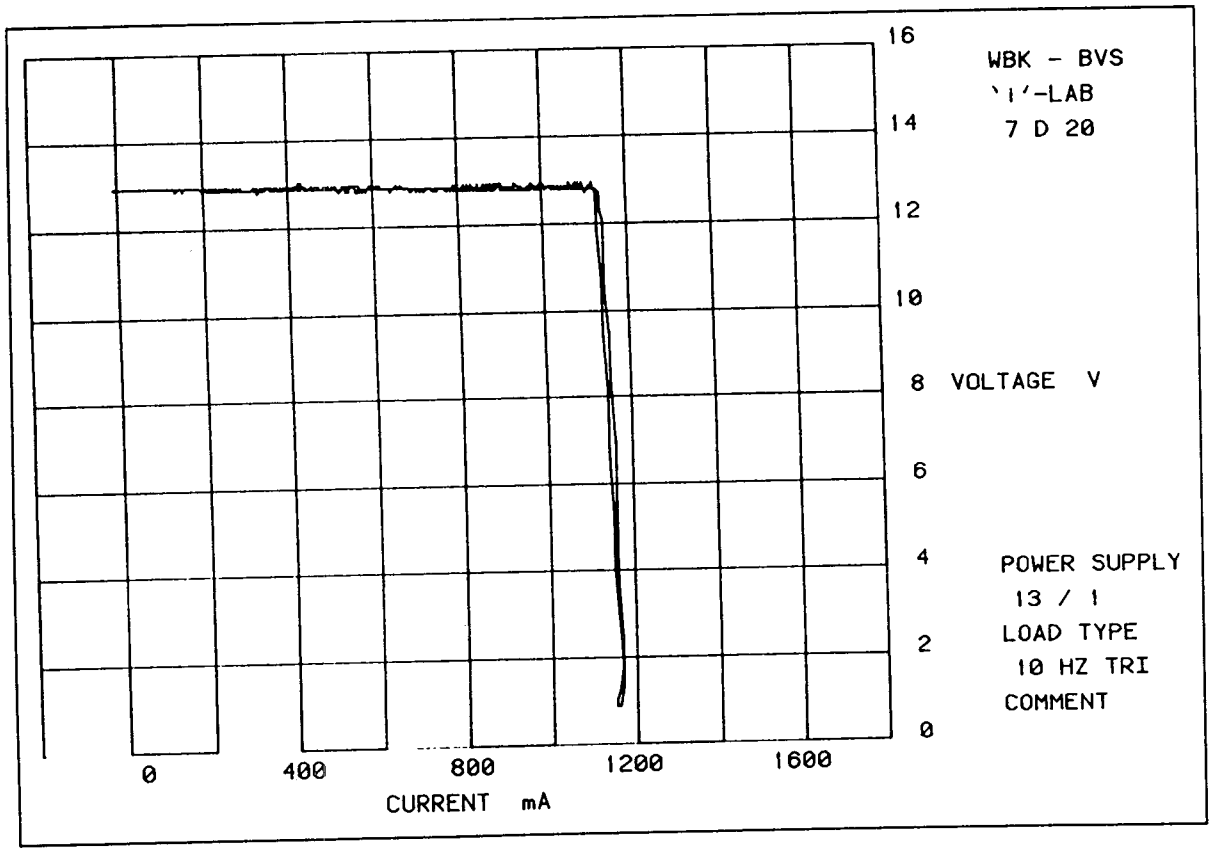


Bild 3. I/U-Kennlinie bei 10 Hz

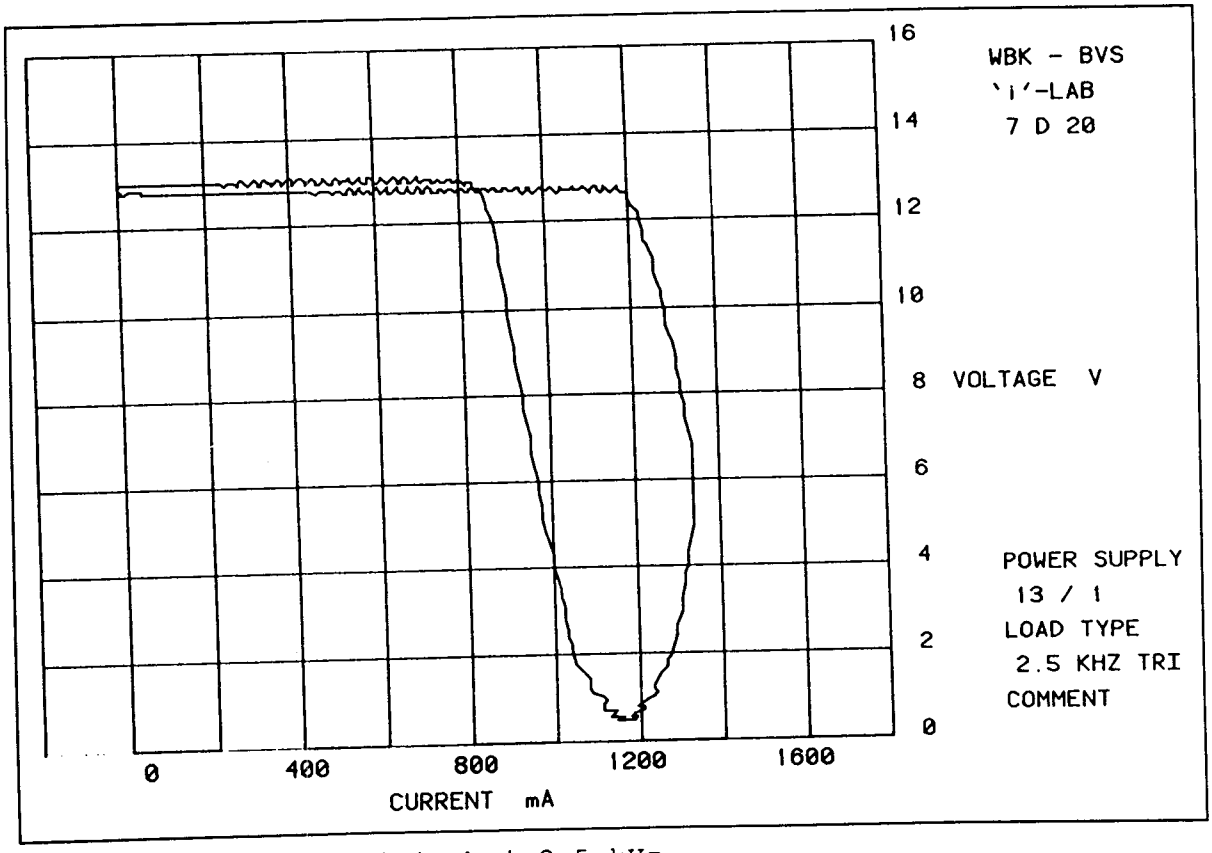


Bild 4. I/U-Kennlinie bei 2.5 kHz

Im Bereich A ist die Strombegrenzung inaktiv. Im Bereich B ist die Spannungsbegrenzung inaktiv. Der Meßpunkt MP sollte die Bereichsgrenze um 5 bis 10 % über- bzw. unterschreiten. Für die Untersuchung der dynamischen Eigenschaften werden die folgenden Messungen vorgeschlagen:

Spannungsregelung - Lastfall 1

Der Ausgang wird zwischen MP1 und Leerlauf geschaltet.

Im Meßpunkt MP1 ist der Linearregler stark durchgesteuert, da er fast das Maximum seiner vordefinierten Nennleistung abgibt.

Die Spannungserhöhung über den statischen Wert von U_{max} beim Sprung in den Leerlauf ist die Sprungantwort des Spannungsreglers; sie wird auch von eventuell nachgeschalteten Induktivitäten beeinflußt. Bild 7 und 8 zeigen die Sprungantwort von Spannung und Strom eines typischen ib-Netzgerätes sowie die entsprechende I/U-Kennlinie.

Spannungsregelung - Lastfall 2

Der Ausgang wird zwischen MP2 und Leerlauf geschaltet.

Im Meßpunkt MP2 befindet sich der Linearregler bereits in der Strombegrenzung. Die Spannungserhöhung über U_{max} statisch beim Sprung in den Leerlauf ist auf die Übergangszeit des Reglers von Strom- auf Spannungsbegrenzung zurückzuführen. Bild 9 und 10 zeigen ein typisches Meßergebnis.

Stromregelung - Lastfall 3

Der Ausgang wird zwischen Kurzschluß und MP2 geschaltet.

Die Stromerhöhung über I_{max} statisch beim Sprung in den Kurzschluß ist auf die Ausregelzeit des Stromreglers, aber auch auf die ebentuell nachgeschalteten Kapazitäten zurückzuführen. Bild 11 und 12 zeigen das typische Verhalten.

Stromregelung - Lastfall 4

Der Ausgang wird zwischen Kurzschluß und MP1 geschaltet.

Im Meßpunkt MP1 befindet sich der Linearregler in der Spannungsbegrenzung.

Die Stromerhöhung über I_{max} statisch beim Sprung in den Kurzschluß ist auf den Lastfall 3 als auch auf die Übergangszeit des Reglers von Spannungs- auf Strombegrenzung, zurückzuführen.

Bild 13 und 14 zeigen die Verläufe von Strom und Spannung über der Zeit sowie die zugehörige I/U-Kennlinie für diesen Lastfall.

Jeder Typ von Stromversorgungsgerät wird sein typisches Übergangsverhalten haben. Wenn ein Hersteller die Prüfung und Bescheinigung einer geänderten Schaltungsauslegung beantragt, sollte das modifizierte Stromversorgungsgerät denselben Tests unterzogen werden. Wenn keine signifikanten Abweichungen vom ursprünglichen Verhalten gefunden werden können, ist eine erneute Prüfung mit dem Funkenprüfgerät unnötig. Diese Untersuchungen mit dynamischer Belastung können auch vom Hersteller selbst vorgenommen werden, um die Prüfprozedur zu beschleunigen.

UNTERSUCHUNG DES DYNAMISCHEN VERHALTENS DURCH MESSUNG DES FREQUENZGANGES DER IMPEDANZ DER STROMQUELLE

Sind die Kennlinien eines Stromversorgungsgerätes für die Lastfälle 1 und 3 hinreichend linear und steht ein Vektoranalysator zur Verfügung, kann auch statt der Sprungantwort von Spannung und Strom der Frequenzgang des Innenwiderstandes des Netzgerätes untersucht werden. Näheres zu dieser Methode kann (15) entnommen werden.

ZUSAMMENFASSUNG

Das dynamische Verhalten eigensicherer Stromversorgungsgeräte mit elektronischen Reglern kann mit typischen Lastfällen untersucht werden.

Die Erkennung von Veränderungen der dynamischen Eigenschaften kann benutzt werden, um die Prüfungsabläufe zu beschleunigen, insbesondere für folgende Fälle:

- Einsatz geänderter Bauteile
- Einfluß von Schwankungen der Eingangsspannung
- Einfluß unterschiedlicher Umgebungstemperaturen.

Werden keine signifikanten Veränderungen im dynamischen Verhalten beobachtet, kann auf die entsprechenden explosionstechnischen Versuchsreihen zur Überprüfung der Eigensicherheit verzichtet werden.

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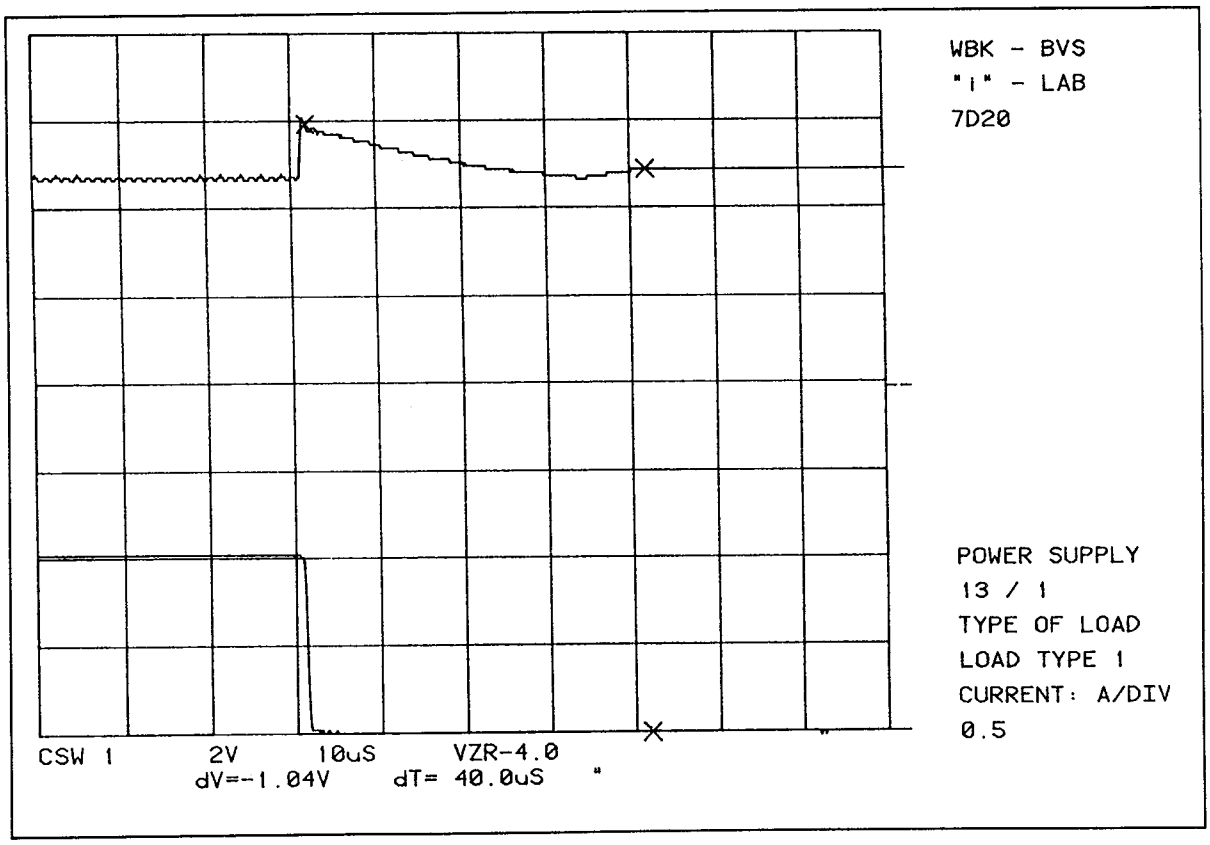


Bild 7. Spannung und Strom zum Lastfall 1

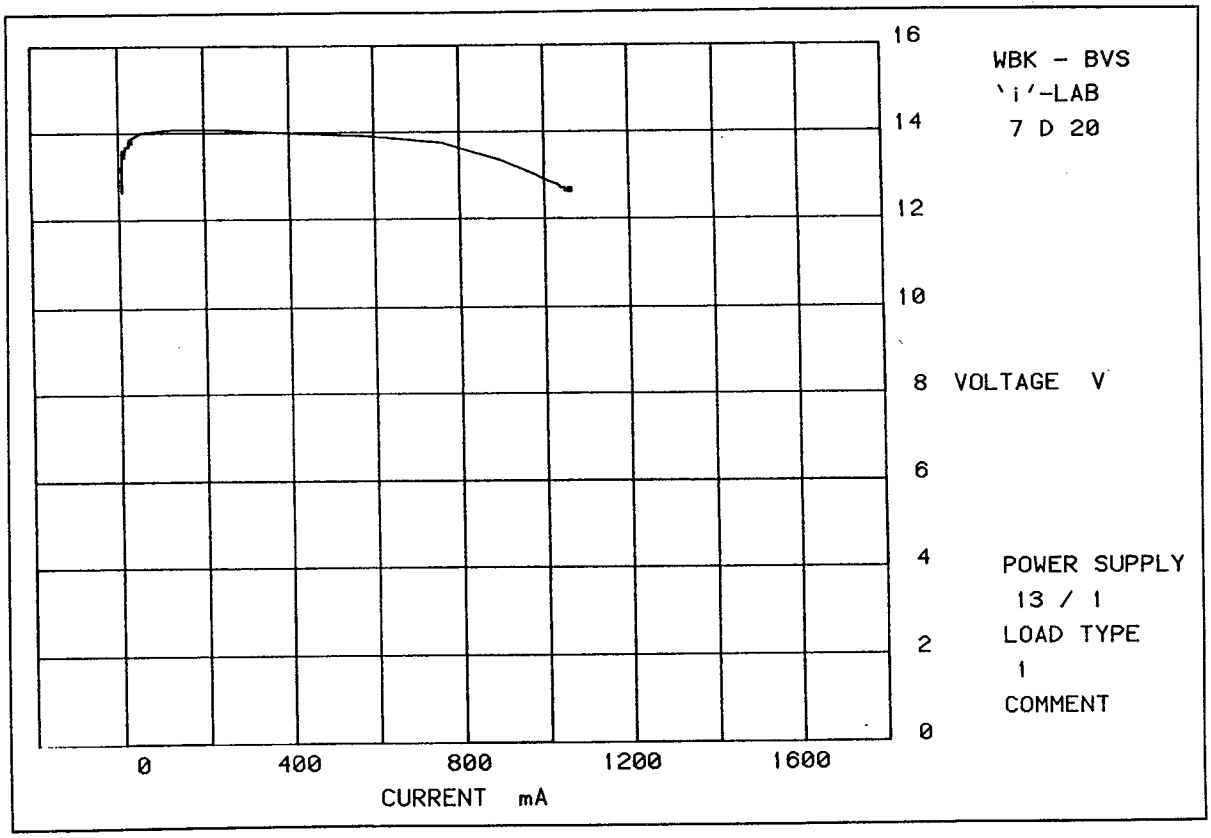


Bild 8. I/U-Kennlinie zum Lastfall 1

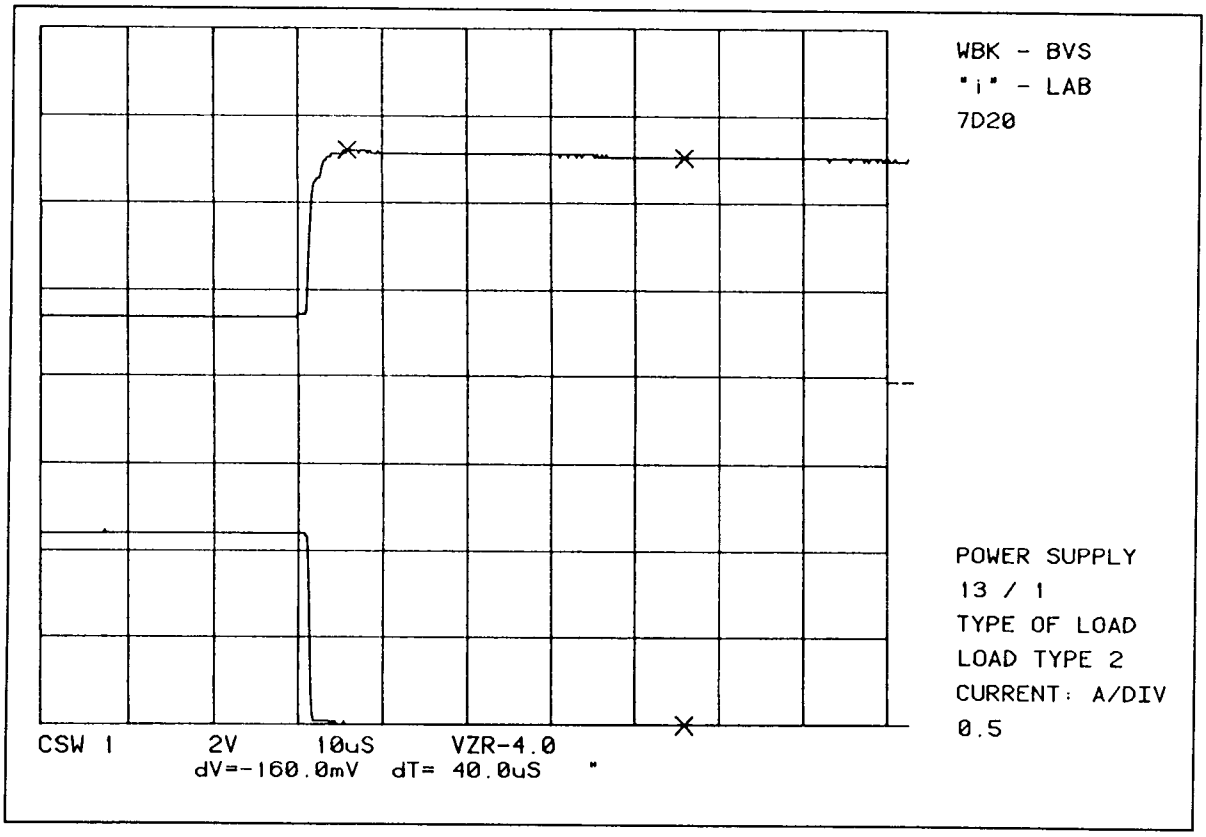


Bild 9. Spannung und Strom zum Lastfall 2

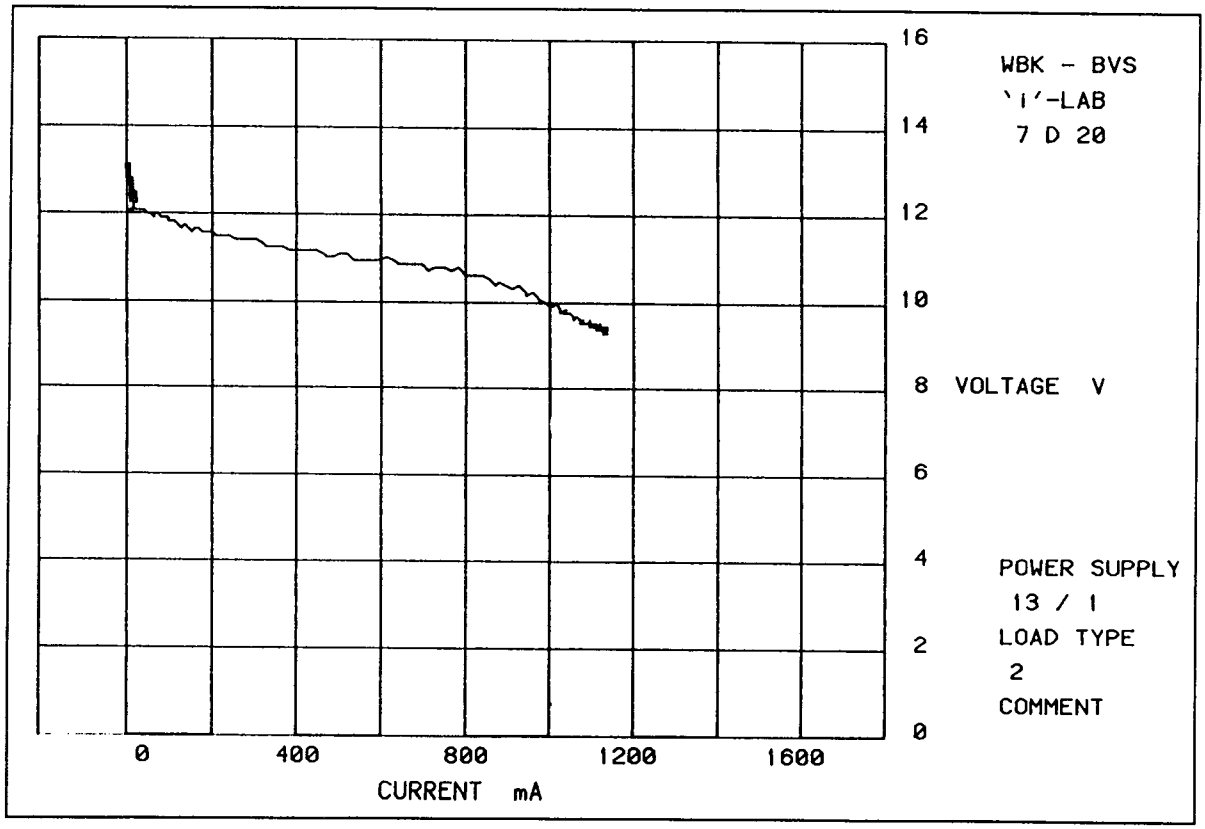


Bild 10. I/U-Kennlinie zum Lastfall 2

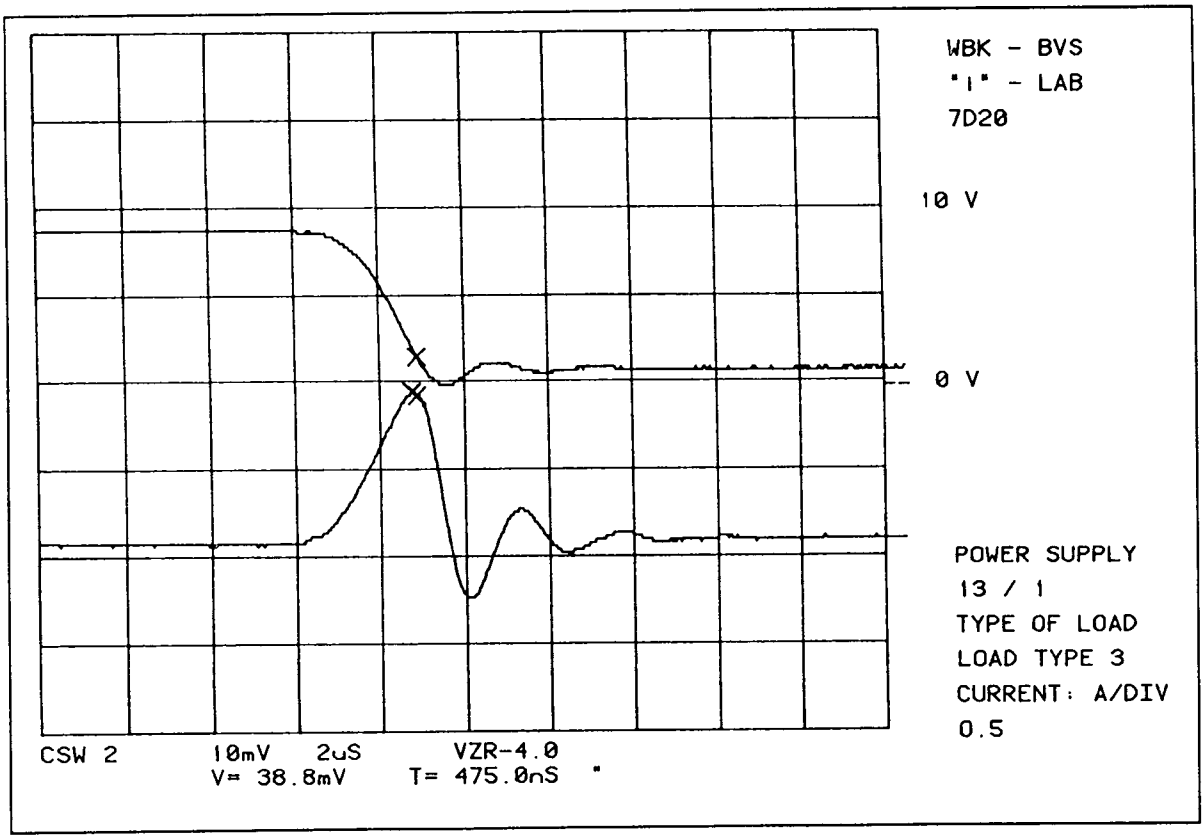


Bild 11. Spannung und Strom zum Lastfall 3

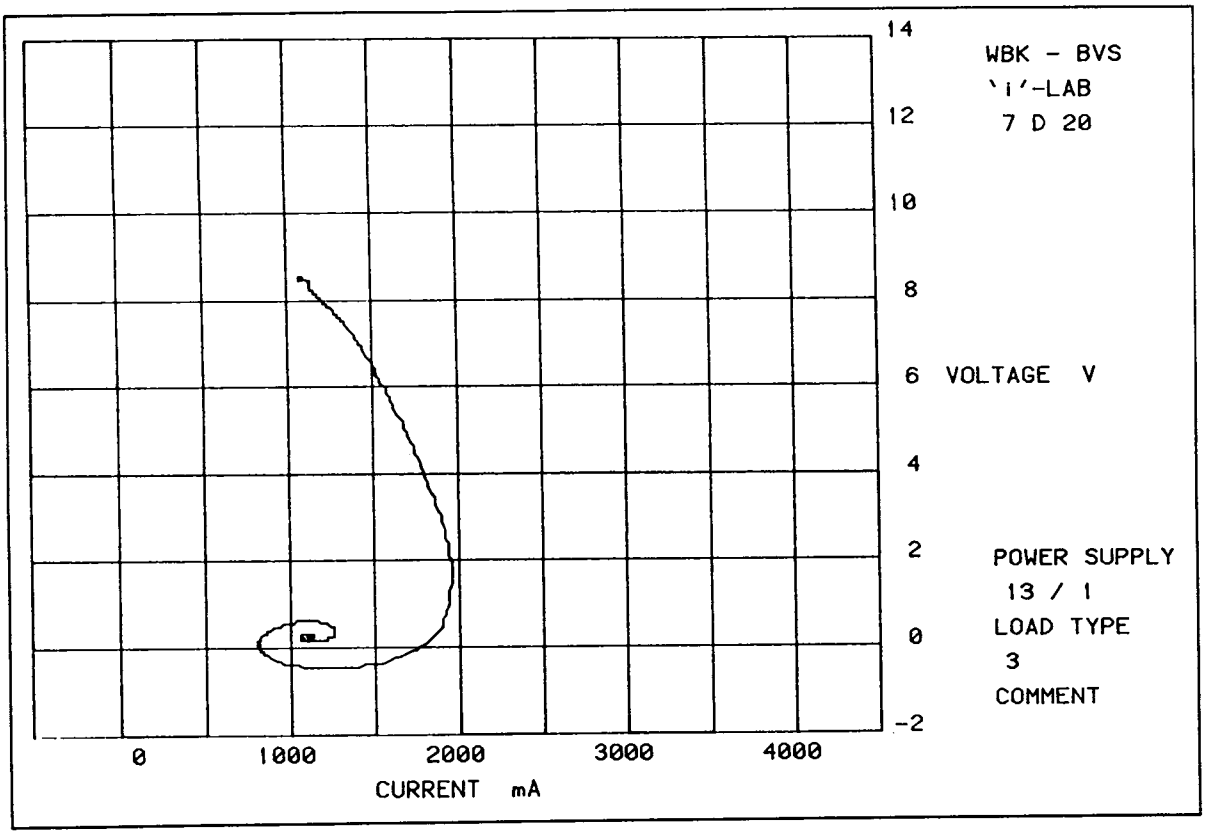


Bild 12. I/U-Kennlinie zum Lastfall 3

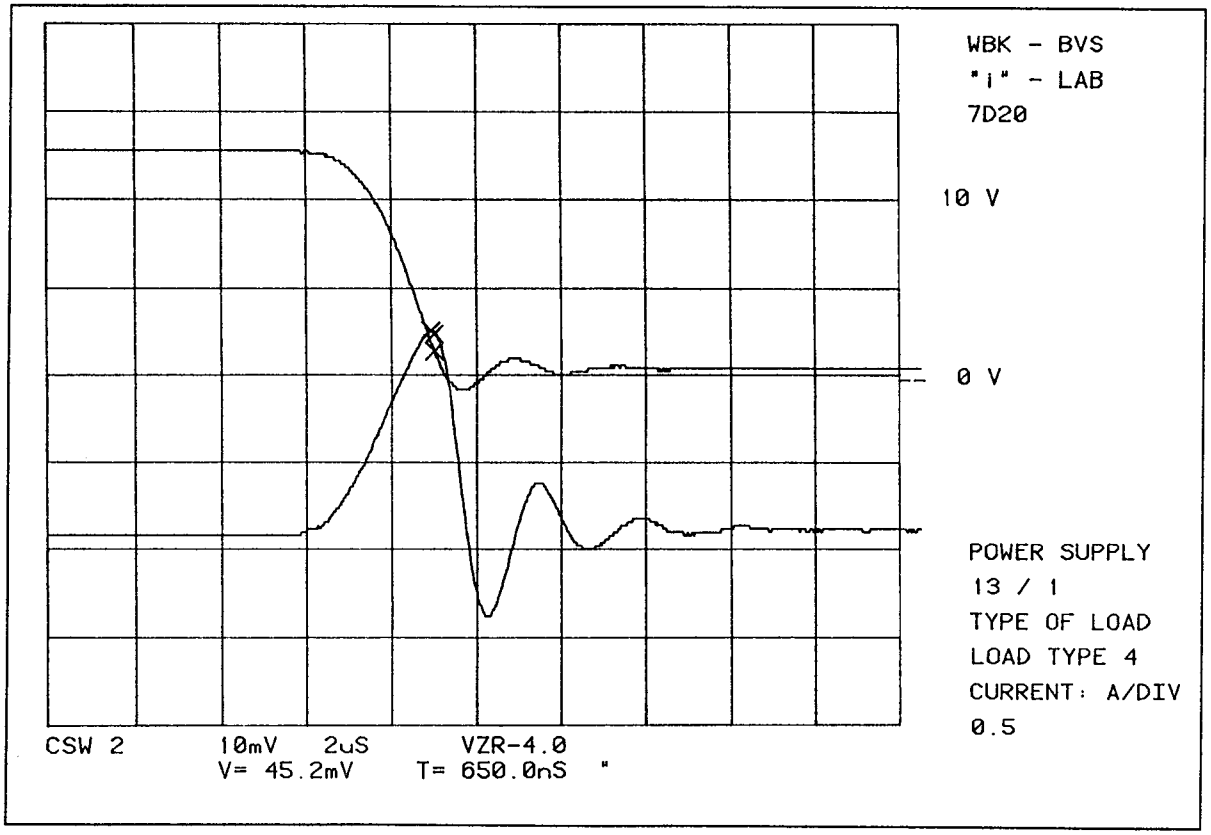


Bild 13. Spannung und Strom zum Lastfall 4

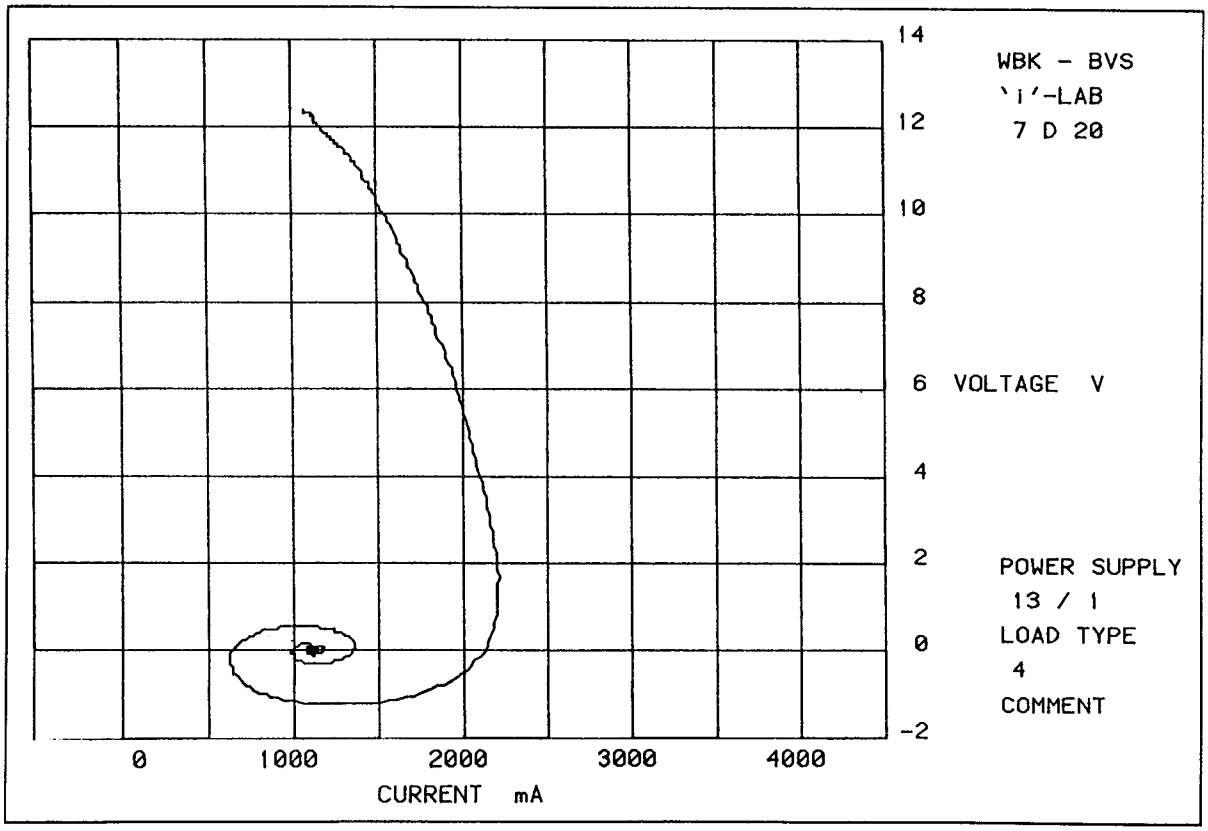


Bild 14. I/U-Kennlinie zum Lastfall 4

ANALYSE DU COMPORTEMENT DYNAMIQUE DES ALIMENTATIONS ENERGETIQUES STABILISEES ELECTRONIQUEMENT DESTINEES A REMPLACER LES ESSAIS D'ALLUMAGE DES MELANGES EXPLOSIFS.

W.G. Dill

Les alimentations énergétiques intrinsèquement sûres, de la catégorie ib, classification IEC 79-11 (limite du courant par des semi-conducteurs) ont été comparées à celles de la catégorie ia (limite du courant par des résistances). Leurs avantages consistent en un débit plus important et en une impédance d'entrée plus faible.

Leurs inconvénients sont les suivants:

1 Les limiteurs de courant électroniques doivent faire l'objet de nouveaux essais au moyen du dispositif d'essai d'allumage (IEC 79-2) lorsqu'ils sont utilisés avec des charges non-ohmiques, car les inductances (même lorsqu'elles sont dérivées par diode) peuvent modifier le comportement des régulateurs électroniques, et provoquer l'inflammation.

2 Lorsqu'on modifie un élément changeant le comportement dynamique, il faut effectuer un nouvel essai d'allumage. Si l'alimentation énergétique modifiée est déjà installée dans des systèmes existants, il faut répéter de nombreux essais.

Les inconvénients décrits ci-dessus diminuent la souplesse de la technologie de contrôle, dont le processus est intrinsèquement sûr.

Une solution à ce problème consiste à analyser le comportement dynamique des alimentations énergétiques stabilisées électroniquement. Selon les résultats, on peut prendre la décision d'effectuer ou non un nouvel essai d'allumage pour tester la sécurité intrinsèque.

Il est ainsi possible de diminuer les délais de conception et d'essais d'alimentations énergétiques intrinsèquement sûres.

АНАЛИЗ ИЗМЕНЕНИЙ ПОВЕДЕНИЯ ИСТОЧНИКОВ ПИТАНИЯ С ЭЛЕКТРОННЫМ РЕГУЛИРОВАНИЕМ, ИСПОЛЬЗУЮЩИХСЯ В КАЧЕСТВЕ БЕЗОПАСНЫХ ЗАМЕНИТЕЛЕЙ В ИСПЫТАНИЯХ ВЗРЫВООПАСНЫХ СМЕСЕЙ ИСКРОВЫМ МЕТОДОМ

В.Г. Дилл

Степень безопасности источников питания категории ib, в соответствии с кодом 79-11 МЭК (ток, ограничиваемый полупроводниками), сравнивалась с категорией ia (ток, ограничиваемый сопротивлениями), преимуществом которой является более высокая мощность и низкий уровень полного сопротивления. Были определены следующие недостатки:

1. Электронные ограничители тока должны перепроверяться на искровых испытательных аппаратах (МЭК 79-2), когда используются неомические нагрузки, потому что индуктивность, даже когда используется диодный шунт, может изменить поведение электронного регулятора до предела шкалы и вызвать возгорание.
2. В случае модификации компонента, влияющего на динамические характеристики, искровое испытание должно быть повторено.

Описанные выше недостатки сужают сферу применения в принципе безопасной технологии. Одной из возможностей решения этой проблемы является анализ изменений поведения источников питания с электронным регулированием. Полученные данные послужат основой для принятия решения о необходимости проведения нового искрового испытания. Использование этого метода даст возможность сократить количество времени, затрачиваемое на конструирование и испытания безопасных источников питания.

用电子调节电源的动力状态分析来代替在爆炸性混合气体中举行火花试验以测定真正安全性

W．G．迪尔

根据 IEC 79－11 的 ib 类（用半导体限止电流）真正安全的电源比 ia 类（用电阻限止电流）有较高的动力输出和较低的电源阻抗，但弊点是：

1．在非欧姆载荷时电子电流限止器须用火花试验仪器（IEC 79－2）再做试验，因为例如电感（即使用二极管并联）可以改变电子调节器的性能使它有产生点火的可能

2．如果要变更影响动力状态的零件，也必须要做火花试验．若变更的电源装在已存在的系统里，很多试验都要重做．

上述的弊点减少了真正安全的过程控制技术应用范围．

一种解决这问题的办法是来分析电子调节电源的动力状态．我们可以用这分析结果作为根据，以决定是否有必要做确定真正安全性的火花试验．

这样我们便可能减少真正安全的电源设计和试验的次数．

EARLY DETECTION OF MOTOR FAILURE

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ABSTRACT

A system capable of monitoring mine electrical power systems and detecting component failure in early stages could significantly improve power system safety and availability. Such monitoring would require a method of evaluating electrical features, calculated from terminal values, for indications of component deterioration. Research conducted by the U.S. Bureau of Mines is examining the use of mathematical models to aid in this evaluation, by creating polynomial networks that can indicate deteriorated conditions in cable-connected motor systems. This process uses laboratory "training" data to select the electrical features most significant for accurately modeling cable-motor system conditions, and forms mathematical expressions relating these features to the presence and severity of deterioration. Models developed thus far can process readily measured terminal information and quantify deterioration power and current to within 3% of motor full load values.

INTRODUCTION

Background

Past Bureau of Mines contract research examining mine electrical system safety investigated the possible benefits of using continuous monitoring techniques to evaluate the condition of power system components (Morley and Kohler, 1981). Initial efforts considered components such as ac and dc motors, power cables, transformers, rectifiers, and capacitor banks. This work concluded that early detection of mine electrical system component deterioration could significantly improve system safety and availability, based on the following findings:

- Numerous electrical-system-related accidents and significant equipment downtime result from electrical component failure.
- Safety and availability are closely related, both being characteristic of well designed and maintained power systems.
- Availability is a function of maintainability and is, therefore, directly related to electrical system maintenance.
- Time-dependent deterioration, primarily dielectric (insulation) breakdown, accounts for the majority of conditions that result in electrical failure or safety hazards.
- A method of detecting time dependent deterioration in its early stages would, therefore, greatly enhance maintenance procedures.

Additional results of this research were the proposal of a methodology for early deterioration detection,

development of a computer-based system to detect and analyze the electrical signatures of component deterioration, identification of numerous characteristics for electrical system deterioration, and confirmation of these characteristics by laboratory and field data collection. An independent research program continued this work, with further analysis of the relationships between electrical component deterioration mechanisms and their resulting signatures, and the development of a classification algorithm for cable insulation deterioration on cable-connected motor systems (Kohler, 1983). Recent Bureau of Mines-funded efforts in this area addressed the analysis and improvement of the above algorithm, produced several computer models to simulate power system and induction motor operation under deteriorated conditions, and identified characteristics associated with deterioration involving windings in induction motors (Kohler, and Trutt, 1987).

The research programs just described have produced information that can be used to form a theoretical basis for the detection of low-level insulation deterioration. Most importantly, the response of numerous electrical features to various levels and types of cable and motor deterioration has been documented. Upon completion of the most recent work, however, this information had not yet been successfully implemented in a practical deterioration detection system. Although some generalizations have been made regarding specific features, and a deterioration classification algorithm was developed for a limited number of cable-connected motor system conditions, a method to detect the presence and level of electrical deterioration is not yet available. This situation prompted the initiation of in-house research at the Bureau of Mines to develop such a system.

Adaptive Learning Network[1] Approach to Failure Detection

Information available from past research suggests that a mathematical relationship or group of relationships can be used to identify electrical deterioration, using measurable parameters; but the identification of the appropriate input features and derivation of suitable relationships is a complex task. Present Bureau of Mines in-house research utilizes an empirical data mathematical modeling approach to this problem, employing commercially available data-modeling software and laboratory-generated power system information to create the necessary relationship(s).

PNETTR-4X,[2] a modeling software package, forms polynomial networks, called adaptive learning networks (ALN's), from input training data using an unsupervised learning process (General, 1986). Input data are typically multiple sets of measured or calculated quantities which represent a physical process to be modeled. For this research, input data are electrical features extracted from laboratory systems undergoing simulated deterioration. PNETTR selects those features that are most significant for modeling motor operation and creates an ALN relating them to some dependent variable, in this case deterioration severity. Fig. 1 illustrates the use of this methodology to create the basis for a deterioration detection system. A flexible modeling package such as PNETTR offers the advantages

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1"Adaptive Learning Network" and "ALN" are registered trademarks of the General Research Corp., McLean, VA.

2"PNETTR-4" is a registered trademark of the General Research Corp., McLean, VA.

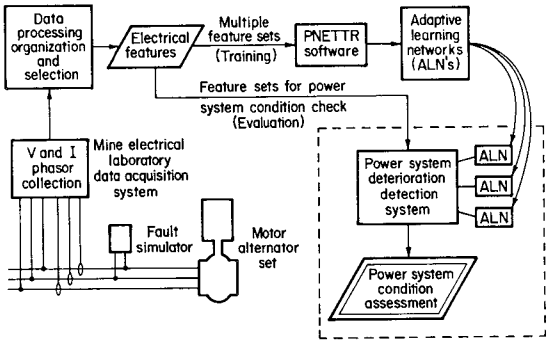


FIGURE 1. - Methodology for creation of deterioration-detection ALN's.

of evaluating and selecting the most appropriate input variables, and creating a polynomial network without assuming a specific model structure.

This paper describes various aspects of the subject research, including data collection, PNETTR software use, and actual use of ALN's for deterioration detection. Finally, conclusions are presented for work thus far, and future plans are outlined.

PNETTR-4X MODELING SOFTWARE

General

PNETTR-4X (Polynomial NETwork TRainer version 4X) is a data-modeling software package marketed by the General Research Corp.,[3] which can create a polynomial network, or ALN, to represent a physical process (General, 1986). PNETTR uses an empirical database and an unsupervised learning method to build ALN's, selecting only those input features that are necessary for accurate modeling.

The database required for modeling must include quantities to be used for both dependent and independent variables. ALN's are the mathematical functions that relate some or all of the available independent variables to one or more dependent variables. Although the overall modeling process is unsupervised, the user can significantly alter the final form of an ALN. This is accomplished through numerous control options which can restrict modeling flexibility, limit use of database information, and affect network evaluations that are part of the modeling process. Note that the term "adaptive learning network" indicates only that the polynomial network adapts to the database during the modeling process. Once formed, a given ALN is fixed.

ALN's have been used successfully in numerous fields to model physical processes from observable variables. Several examples are signal processing, pattern recognition, materials testing, geophysical sciences, process control, flight control, health sciences, and economics.

PNETTR assumes no particular model structure, but creates series-parallel networks of polynomials and generates coefficients solely from a database. The individual elements of the networks are polynomials, each with up to three different input values from any of several sources. Fig. 2 illustrates several possible element connections within

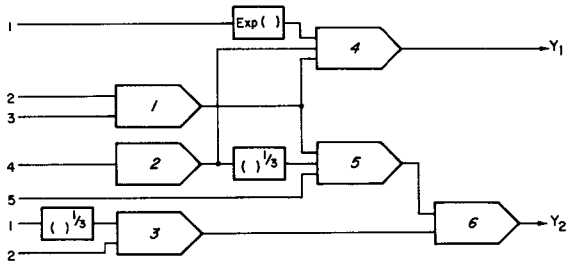


FIGURE 2. - Possible element connections for an ALN.

[3]Use of names of manufacturers is for identification purposes only and does not imply endorsement by the Bureau of Mines.

an ALN (General, 1986). Element inputs can come from the following sources:

- Original input variables
- Outputs from elements on any preceding layer
- Outputs from networks being constructed for the modeling of different dependent variables from the same input database
- Exponentials of any of the above candidate inputs
- Cube roots of any of the above candidate inputs.

The terms of each polynomial are taken from the following expression:

$$Y = A_0 + A_1X_1 + A_2X_2 + A_3X_3 + A_4X_1X_2 + A_5X_1X_3 + A_6X_2X_3 + A_7X_1^2 + A_8X_2^2 + A_9X_3^2 + A_{10}X_1^3 + A_{11}X_2^3 + A_{12}X_3^3$$

Usually, only a few of the possible terms will be used in an element, but both the networking of elements and the selection of polynomial terms can range from quite simple to extremely complex.

PNETTR will accept up to 999 candidate independent input variables, although 40 or fewer typically provide the most efficient modeling. Dependent variables are limited to 14, and polynomial networks may have no more than 16 layers.

Input Information

PNETTR requires an input database containing multiple observations, usually of measurable quantities, from a process to be modeled (General, 1986). Each observation consists of one or more dependent variables and independent variables that may mathematically predict the dependent variable(s). The independent variables or "features" are almost always measured or calculated quantities that characterize the physical process under consideration. Dependent variables can be measured quantities which represent some state or condition for the process, or arbitrary values denoting "classes" of conditions. Other information can be placed within the database, such as variable uncertainties (measurement inaccuracies), observation weights, and class names.

Although PNETTR will isolate the variables that are most significant for a desired model, a thorough knowledge of the physical process in question and careful selection of observations and variables will allow focusing of the modeling process and improve results. Observations, for example, should be representative for conditions or times of primary interest, while candidate independent variables should not include unlikely predictors, highly correlated quantities, or values with poor accuracy.

Modeling Control and Evaluation

PNETTR-4X as used for this research runs through a command procedure called PNETTR, and a control file must be supplied for each PNETTR run to indicate the database file name and size, as well as numerous conditions and constraints for the run (General, 1986). PNETTR, using the data provided and working under the conditions specified, constructs and evaluates ALN's in a layer-by-layer training procedure. An essential element of such training is a method to select and rank candidate models at all stages of construction. PNETTR evaluates each candidate model by calculating a value called the predicted squared error (PSE). The PSE is the sum of three values: the training squared error, the overfit penalty term, and the model uncertainty term (General, 1986).

The training squared error is the average of the sum of squared differences between actual (observed) dependent variables and the dependent variable values

predicted by an ALN, for the training observations used. The overfit penalty term is the product of a user-specified multiplier and the ratio of the estimated number of coefficients for an ALN to the number of training observations. The user can control model complexity (prevent overly simple or overfit models) by appropriate selection of the multiplier, called sigma prior. Sigma prior is an a priori estimate of the standard deviation of model-prediction error and can cause overfitting if too small, or an overly simple model if too large. The uncertainty term, as the name implies, accounts for input uncertainties as they are transmitted through successive ALN layers during construction.

This has been a simplified description of ALN evaluation by PNETTR and does not include complexities introduced by factors such as varying observation weights or intermediate operators in model networks. In general, however, the PSE value is the primary indicator of model quality, with the lowest PSE normally indicating the best ALN.

The PNETTR modeling process begins by the formation and evaluation of single-input elements as dependent variable predictors. The PSE's of these elements are used to rank the effectiveness (as predictors) of individual independent variables and to select candidate inputs for multiple-input elements in the first layer. PSE's from these multiple-input elements are then similarly used to choose the best of the first-layer elements from which to pass outputs to the second layer. Subsequent layers are formed and evaluated, using candidate inputs as described earlier, until an output with the lowest PSE is found.

PNETTR generates two output files upon completion of a run. One is a full description of the modeling and ALN evaluation process; the second is a listing of the best ALN, coded as a FORTRAN subroutine.

Summary

The preceding description of PNETTR-4X described the purpose, utility, structure, and actual use of this modeling software. The discussion has been general with respect to applications, but the information presented is important for an understanding of the use of ALN's in empirical modeling of deteriorated cable-connected motor systems.

DATA COLLECTION

General

The use of empirical information for modeling cable or motor deterioration necessitates laboratory data collection under well controlled conditions. Laboratory facilities required for such data collection include an appropriate power system, instrumentation to monitor the system, and some method to collect and organize the desired information. Specific requirements for this research follow:

- A variable 3-phase power supply;
- 3-phase motors on motor-alternator sets;
- A resistive load bank to load alternators;
- A deterioration simulator;
- Instrumentation to monitor 3-phase motor and load bank electrical signals;
- Computer hardware and software to collect, process, and store data.

The following sections briefly describe the equipment and software developed to generate empirical data for modeling.

Laboratory Equipment

The motor-alternator sets used for experimentation require 230- or 460-V, 3-phase power. The Mine

Electrical Laboratory (MEL) at the Pittsburgh Research Center (PRC) of the Bureau of Mines is equipped with a Hypotronics MTC 750C motor test station, which can supply infinitely adjustable voltage levels up to 13.8 kV, with a continuous capacity of 750 kVA. The output of this system is wired as a resistance-grounded wye source. The motors used for tests are Reliance Electric, Duty Master AC, design B units, in 5-hp (194T frame) and 10-hp (215T frame). These motors are coupled axially to Onan, single-phase, 60-Hz, 120/240-V, 17.5-kW alternators.

A bank of resistors is used to load the alternators driven by the test motors. The single-phase alternators power the load bank through a motor-driven variable transformer, which allows remote control, quick load changes, and precise motor loading.

In order to simulate varying levels of current leakage (insulation deterioration) across various points on a motor system, a bank of resistors is used, connected to the motor leads through a variable transformer. The variable transformer allows precise control of leakage current levels while permitting simulation of several different deterioration types (paths) having different voltage levels. The deterioration simulator is primarily resistive, with a power factor of 0.995.

The actual collection of data from the laboratory cable-connected motor system requires instrumentation to sense and condition electrical signals from the system terminals and transmit them to a data-acquisition computer. Voltage dividers reduce line-to-neutral voltages to a level compatible with downstream signal-transmission equipment, and clamp-on-type current transformers monitor line currents on all three phases. The single-phase outputs of the alternators also require instrumentation in order to monitor motor load levels. Laboratory instrumentation and the MEL data collection computer system are connected by a multichannel optical fiber transmission system. Each channel consists of a transmitter, an optical fiber, and a receiver which transmits instrumentation signals to the analog-to-digital converter in the data acquisition computer system. The fiber optic link allows location of the data collection computer system in a clean environment remote from the testing laboratory, and also reduces the chance of instrument interaction through grounding loops, or damaging transient overvoltages at the analog-to-digital converter input.

The MEL has a DEC LSI 11/73 computer equipped with a Data Translation DT 3362/3369 analog-to-digital converter for laboratory data acquisition. This unit is linked to the PRC VAX computer cluster through a DECnet communications network system, which allows transfer of information and commands between computers.

Fig. 3 shows the overall layout of the laboratory power system equipment and data-acquisition hardware.

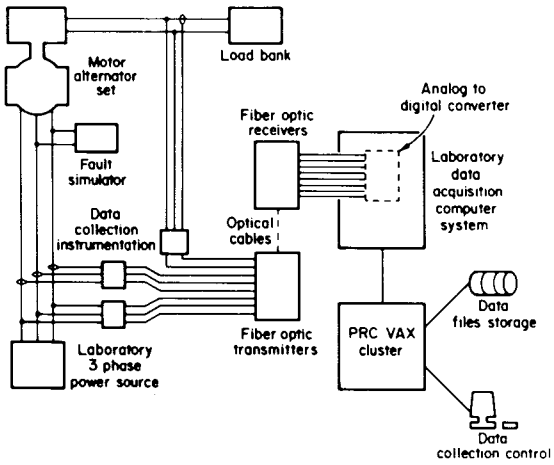


FIGURE 3. - Layout of MEL power system equipment and data-acquisition hardware.

Data Acquisition Software

The extensive laboratory data collection necessary for empirical modeling requires software that enables researchers to sample

instrumentation signals, process the signals to obtain desired electrical values, sort and select specific information, and create data files for input to PNETTR. Appropriate software must be capable of handling large volumes of data but allow flexibility in selecting sampling characteristics, calculation of electrical parameters, and data file creation. The unique software written by Bureau personnel to fulfill these requirements utilizes the database system INGRES and is made up of three distinct programs, ADC, CALC, and CHOOSE, which run on the VAX cluster.

ADC executes analog-to-digital conversion through the MEL LSI 11/73, records acquisition parameters in the database, and stores voltage and current raw data in separate files. Sampling and test-condition information can be specified by the user and is stored in the database with each sampling run of ADC until needed in later stages of processing, such as engineering unit calculations, per-unit-value conversions, and deterioration classification. (The term "per unit" or "pu" refers to the expression of electrical quantities as the ratio of a given value to a base value. Base values for this work are the nameplate voltage and current ratings of the motor in question).

CALC forms the second stage of data handling, by calculating engineering values from ADC raw-data files. Program features provide flexibility by allowing the use of all or specific segments of a raw-data sample, in any combination. The selected raw-data segment(s) are processed using a Fast Fourier Transform routine, and the resulting information is stored in the database as a calculated-data entry.

The program CHOOSE uses calculated-data entries to create a data file for input to PNETTR. Entries can be selected manually one at a time, or in groups according to any one of several entry characteristics. Information from the calculated-data entries (measured values from the laboratory tests) is at this point used to derive the electrical features that will be used as candidate input variables for modeling. The program creates two versions of a PNETTR database; one is a binary file for actual PNETTR input, and the other is an ASCII file for printout and review.

Using these three programs, the overall process consists of sampling and storage of power system measurements as raw data, calculation of engineering values for these measurements, calculation of numerous electrical features from the basic power system measurements, and placement of these features into a database file suitable for PNETTR.

MOTOR FAILURE DETECTION

General

Prior to data collection and empirical modeling of power system operation, a number of basic decisions must be made regarding the components involved, power system conditions, specific modeling techniques, and overall approach to monitoring. It is therefore necessary to describe the major assumptions and options selected, which have focused efforts for the subject research, as well as the logic behind the selections.

The components targeted in this research are 3-phase squirrel cage induction motors, which are the most widely used electric machines in mining operations. Power conductors (cables) are also considered, in order to include all conductors between monitoring instrumentation and motor terminals. Electrical failure of these components will often originate with the breakdown of insulation and subsequent leakage of current through an abnormal path, eventually resulting in a fault (Morley and Kohler, 1981). Insulation breakdown may be due to any of numerous deterioration mechanisms arising from electrical, thermal, mechanical, or environmental stresses. The development of

leakage current paths however, is the common result of all these processes, and is the effect that most directly leads to electrical failure of cables and motors. Since the subject research deals only with squirrel cage induction motors, deterioration of brushes, slip rings, commutators, and similar motor components is not considered; nor is mechanical deterioration such as bearing wear and stator or rotor structural failure. The process of insulation breakdown is used in this research as the basis for quantifying progressive deterioration of cables and induction motors, with deterioration severity indicated by the level of current flowing and power dissipated in an abnormal path. Such leakage current can seldom be located, much less measured, in the field, but it is readily controlled and measured in laboratory simulations. Therefore, laboratory simulations can be used to establish an empirical relationship between deterioration current and power levels and readily measured electrical values, and this relationship can be employed to detect low-level (otherwise undetectable) deterioration in the field. Although leakage levels may be an accurate gauge of insulation deterioration, no relationship is yet available to correlate leakage to a useful measurement of deterioration in a cable or motor. Such a relationship will be essential to accurately estimate useful life remaining for failing equipment, but cannot be developed without a suitable method to detect and quantify leakage in field data collection or laboratory destructive testing. Accordingly, the focus of this initial research is development of an accurate detection method, which will allow verification of the overall failure-prediction concept in subsequent research.

The laboratory hardware and software described in the previous sections are used to generate data for the modeling process. Necessary information, such as voltage and current phasors for each phase, load information, and deterioration characteristics, is sampled during steady-state operation of a cable-connected motor system. Steady-state conditions were selected for data acquisition in order to simplify sampling techniques both in laboratory work and in eventual field implementation of a monitoring system. All electrical features for each acquisition are derived from voltage and current phasors for a specific set of system conditions, and are candidate input variables to the modeling process, while the deterioration levels corresponding to system conditions are dependent variables for the sample.

Supplying observations that cover the full range of cable-motor system conditions in question, for a given range of deterioration levels, allows PNETTR to model system operation as a continuous function relating specific electrical parameters to deterioration levels. This particular modeling approach was selected because deterioration current and power levels have a mathematical relationship to specific electrical features, in contrast to the arbitrary dependent variables often used in classification techniques. In addition it does not require an a priori selection of a cutoff deterioration level to define a good or bad condition, and will provide trend information prior to failure, which will be essential in using such monitoring systems to accurately anticipate failures. Beyond the points just listed, ALN's created with PNETTR should ideally determine deterioration presence and level independent of deterioration type and location or motor loading. A suitable ALN should also be applicable to cable-motor systems with components different from those on which the training data were generated, within reasonable size and design limitations. This ALN "generality" is an important concept for actual application of cable-motor system monitoring, in that evaluation would not require a "no deterioration" base line with which to compare subsequent samples, nor would a specific motor need

extensive laboratory testing to be successfully monitored in the field.

First stages of current research involved PNETTR modeling of data from cable-motor systems with specific and limited conditions, typically with only one parameter such as deterioration type, location, and severity, or motor loading, being varied for each investigation. When the effect of various parameters on ALN performance had been determined, modeling of more general cases was attempted, ultimately including all possible system-condition combinations. Upon verification of modeling accuracy for widely varying system conditions, several ALN's were tested for applicability to motors different from the unit used to generate the training data.

The following sections detail the procedures used for empirical modeling with PNETTR, results of this modeling, and a summary of important ALN characteristics.

Input Data

As previously mentioned, all input information originates from laboratory cable-motor system voltage and current phasors, or is input directly during data file creation. The leakage current levels simulated for laboratory tests are entered directly during data acquisition for each sample taken, and along with calculated deterioration power become the dependent variables in PNETTR data files. All independent variables are derived from 3-phase voltage and current values and include an echo of voltage and current, system impedance, symmetrical component values for voltage, current, and impedance, system power levels, and power factor. For each of the complex values used, four separate quantities are placed in the data file: rectangular X component, rectangular Y component, polar magnitude, and polar angle. This broadens the number of candidate input variables from which PNETTR may choose and reduces the chance of missing subtle feature variations that may prove valuable for modeling. An additional input variable is the magnitude of power output by the alternator in each test, which directly represents motor load level. All of the above described information is available for each individual laboratory sample taken, and each sample typically represents one of many different conditions simulated. Further, one or more data file entries or "observations" for PNETTR can be created from a single sample.

For most PNETTR modeling runs in this research, one raw data set was taken for each specific condition within the range of conditions under consideration in a given investigation. From each set, four observations were derived for input to PNETTR; the first two or three were designated for ALN training, and the last was reserved as part of an independent ALN-evaluation data set. In this way, ALN performance can be evaluated with previously unseen data from the full range of conditions represented in training.

Failure Detection ALN's

As previously described, PNETTR modeling was initially conducted using data generated under limited and well controlled variation of cable-motor system conditions, varying only one major parameter for each attempt, and gradually increasing complexity. Eventually, models were created to represent a wide range of motor system operating conditions, including different deterioration (insulation leakage) positions and levels, and different motor load levels. Due to the nature of empirical modeling, a complete cross-section of situations that may occur must be included in ALN training in order to accurately model the full range of conditions in question.

Data collection for these investigations was carried out using a motor alternator set with a 5-hp, design B, dual-voltage, 3-phase induction motor, operating at 460 V and 7.0 A full load current, with

delta-connected windings. Motor load was varied among four distinct levels, producing motor line currents of 3.85 A (no load), 4.90 A, 5.95 A, and 7.00 A (full load).

Data generation for this research simulated insulation leakage across all possible combinations of several locations on the cable/motor system. Connection points included phase conductors (motor line terminals), midpoints of each delta-connected winding within the motor, and the power system neutral or ground. A variable resistance was placed between selected points to create a leakage current, as illustrated in figure 4, including simulation of all the following paths.

AB	BC	CA
AG	BG	CG
Aa	Ba	Ca
Ab	Bb	Cb
Ac	Bc	Cc
ab	bc	ca
aG	bG	cG

Seven deterioration current levels were used at each of the above positions, ranging from 0.00 pu to 0.30 pu at 0.05 pu increments (pu base is motor rated full load current). Power dissipated at the point of deterioration was then derived from these values, and both current and power used as dependent variables in the modeling process. For all possible test parameter combinations, this results in 588 distinct system conditions represented in the modeling process.

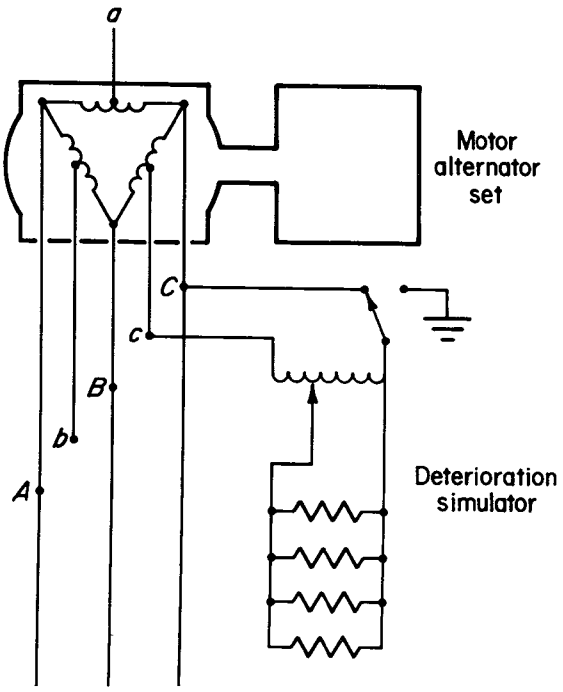


FIGURE 4. - Deterioration simulation for laboratory data collection.

One raw data set was taken for each distinct laboratory system condition and subsequently divided and processed to yield four PNETTR observations, the first two or three of which were used for training and the last for ALN evaluation.

Data sets were created as described above, and processed using PNETTR. Several modeling controls were employed to optimize the resulting ALN's, including elimination of inappropriate independent variables, reducing the maximum number of ALN layers, and specifying lower sigma prior values. The PSE value was used as a relative indication of model quality among different versions of the same ALN, while the rms error for an evaluation data set indicated accuracy in determining deterioration level. The latter is the rms of residuals between the actual and model-calculated deterioration levels for a set of observations not used in training of the ALN (refer to section describing PNETTR software).

Modeling attempts indicated that ALN's can detect and quantify low level deterioration on a cable/motor system independent of deterioration type and level, as well as motor load level. The most accurate ALN's developed thus far, covering all the aforementioned 588 different system conditions, are shown below.

Deterioration Current ALN:

$$A_1 + A_2X_1 + A_3X_2 + A_4X_3 + A_5X_1X_2 + A_6X_2X_3 + A_7X_1^2 + A_8X_2^2 + A_9X_2^3 = X_4$$

$$A_{10} + A_{11}X_4 + A_{12}X_5 = Y_1$$

Deterioration Power ALN:

$$A_{13} + A_{14}X_6 + A_{15}X_7 + A_{16}X_7^2 = Y_2$$

X_1 = zero sequence current polar magnitude

X_2 = exponential of negative sequence impedance polar magnitude

X_3 = zero sequence current y-coordinate

X_4 = intermediate output, first layer of Current ALN

X_5 = negative sequence current polar angle

X_6 = negative sequence current polar magnitude

X_7 = negative sequence impedance polar angle

Y_1 = deterioration current level

Y_2 = deterioration power level

The rms errors for calculation of deterioration power and current for an independent evaluation data set processed using the above ALN's, are 0.011 pu and 0.031 pu, respectively.

The next step in evaluation of these ALN's is the use of laboratory-generated evaluation data sets, from cable-motor systems different from those used for training. The investigation used data from systems containing the following three motors.

1. Same 5-hp motor used for training, but operating at 230 V and 14.0-A full load current
2. A 5-hp motor different from the unit used in training, operated at 460 V and 6.8 A full load current.
3. A 10-hp motor operating at 460 V and 12.2 A full load current.

The results of evaluation of these data sets using the subject ALN's are shown below.

Motor	Power rms error	Current rms error
1	0.056 pu	0.027 pu
2	0.027 pu	0.067 pu
3	0.055 pu	0.082 pu

Whether errors of several percent (of motor line current or power) will be acceptable for actual deterioration-detection systems will depend on the as yet unknown characteristics of cables and induction motors prior to complete failure. Based on the fact that 0.20 to 0.30 pu leakage current levels can exist in motors without immediate obvious effects however, ALN resolution as high as 0.05 pu may well be acceptable for low-level deterioration monitoring.

Further work is required to extend the applicability and improve the accuracy of failure detection ALN's, as well as establish a predictable relationship between simulated deterioration and actual motor failures. The foregoing results, however, are a major step in development of algorithms that can be used to detect and quantify insulation leakage in motor systems.

Summary

The important findings of the foregoing discussion follow:

- PNETTR modeling of cable-motor systems can create functions, ALN's, that will accurately detect and quantify insulation leakage from measurable electrical parameters, even for data samples not used to train the ALN.
- This modeling can be used successfully to detect and quantify insulation leakage without regard for deterioration type, position, and severity, or motor load level.
- Appropriate ALN's can be used, within limits, to detect deterioration on systems with motors different from that on the original system used for model training. That is, the ALN's have

a certain degree of generality, an essential quality for practical field applications of deterioration-detection techniques.

CONCLUSIONS

Overall, the accuracy and generality of ALN's developed thus far strongly suggest that such models can serve as the basis for a power system condition monitoring system. This conclusion is based on the successful modeling of cable-connected motor system deterioration independent of wide-ranging system conditions, and on model generality beyond the specific system used in training.

As previously discussed, the actual field application of an ALN-based monitoring system should clarify and define numerous aspects of the overall concept. Among these are the verification of the relationship between gradual insulation breakdown and ultimate component failure, collection of the resulting trend information which will allow anticipation of failure, defining the accuracy and resolution needed for sampling and processing to provide adequate evaluations, and selecting the appropriate format and frequency for condition reports.

Efforts are now underway to implement such ALN's in a prototype motor-deterioration detection system. This initial application uses a distributed-processing network on board a continuous miner to monitor electrical parameters and evaluate the insulation condition of all motors on the mining machine.

Although results of PNETTR modeling have established the feasibility of ALN use for early deterioration detection, several areas must be further investigated and developed prior to any field implementation of power system condition monitoring. Those presently scheduled for further study are listed below.

- Better define minimum data-sampling criteria that will yield adequate accuracy and resolution.
- Check on possible improvement in evaluation accuracy from the use of multiple ALN's for each evaluation, in a two- or three-step evaluation process.
- Overall accuracy of a fully general ALN should be optimized.
- Generality beyond motors used for training must be further verified and improved, and limits identified.
- Devise practical techniques to account for bus voltage imbalances, which would otherwise hinder condition evaluation accuracy.
- Use destructive motor testing to better define the relationship between insulation deterioration and ultimate failure.

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DETECTION A L'AVANCE DES PANNES DE MOTEUR

G.T. Homce

Un dispositif capable de surveiller les systèmes électriques utilisés dans les mines et de détecter dans leurs stades initiaux les pannes des éléments qui les composent peut améliorer de manière significative la sécurité et la fiabilité de ces systèmes. Ce type de dispositif de surveillance requiert l'emploi d'une méthode d'évaluation des caractéristiques électriques, calculée à partir des valeurs terminales, permettant d'indiquer le niveau de détérioration des éléments. Le Bureau des Mines étudie l'utilisation de modèles mathématiques pour aider à effectuer cette évaluation; pour ce faire, il a conçu des réseaux polynomes indiquant les détériorations des systèmes de moteurs reliés par des câbles. Ce processus utilise des données "d'instruction" de laboratoire afin de sélectionner les caractéristiques électriques les plus significatives pour représenter exactement l'état des systèmes câbles-moteurs; il établit des formules mathématiques associant ces caractéristiques à la présence et à la sévérité des détériorations. Les modèles développés jusqu'à maintenant peuvent traiter les informations, qui sont promptement évaluées par les terminaux, et quantifier la force de détérioration et le courant à 3% près des coefficients de charges complètes du moteur.

FRÜHERKENNUNG EINES MOTORSCHADENS

G. T. Homce

Ein System, das in der Lage ist, das elektrische System einer Grube zu überwachen und das Versagen von Komponenten im Frühstadium zu erkennen, würde in großem Maße die Sicherheit und Verfügbarkeit des Systems verbessern. Eine solche Überwachung würde eine Methode zur Beurteilung elektrischer Merkmale, welche aus Eingangswerten errechnet würden, erfordern, um eine Verschlechterung der Komponenten erkennen zu lassen. Das U.S. Bureau of Mines untersucht die mögliche Verwendung von mathematischen Modellen, die diese Beurteilung vereinfachen könnten, und zwar durch die Erzeugung polynominaler Netzwerke, die in der Lage sind, sich verschlechternde Zustände von fest angeschlossenen Motorsystemen anzuzeigen. Dieser Prozeß verwendet Labor-"Trainings"-Daten, um die wichtigsten elektrischen Merkmale für eine akkurate Modellierung der Umstände in einem Kabel-Motor-System auszuwählen, und erstellt mathematische Formeln, welche diese Eigenschaften mit dem Auftreten und dem Grad der Verschlechterung in Beziehung setzen. Die bisher entwickelten Modelle können sehr schnell gemessene Eingabewerte verarbeiten und die Verlustleistung und Ströme auf bis zu 3% der Motorvollastwerte quantifizieren.

РАННЕЕ ОБНАРУЖЕНИЕ НЕИСПРАВНОСТИ ДВИГАТЕЛЯ

Г.Т. Хомс

Система, способная управлять шахтными силовыми электрическими системами и обнаруживать поломки компонентов на ранних стадиях, может значительно улучшить безопасность и готовность силовой системы. Такая система потребует метода оценки электрических параметров, рассчитываемых по величинам на клеммах, для определения ухудшения состояния компонентов. Исследования, проводимые Горным Бюро США, рассматривают использование математических моделей для такой оценки посредством создания полиномиальной сети, которая покажет ухудшение состояния соединенных кабелем моторных систем. Этот процесс использует "пробные" лабораторные данные для отбора электрических параметров, наиболее важных для точного моделирования состояния системы "кабель-мотор", и создает математические выражения, "привязывающие" эти параметры к наличию и серьезности ухудшения. Разработанные модели могут легко считывать информацию с клемм и количественно оценивать уменьшение мощности и тока с точностью до 3% от мощности.

马达故障的及早察觉

G。T。霍母士

侦测动力系统及察觉先期的故障可以保障矿里动力设备的安全和工作效率。这种侦测须要用端点值计算来估计电力性能的方法，以判定是否其中零件失效。美国矿业局所作的研究便是用数学模型来辅助这种估计的。数学模型用多项式网路显示电线连接的马达系统里面的失效情形。步骤是从实验室“训练”数据选择能模拟接线的马达系统情形的电力特征。并找出数学公式来表达失效的发生和严重性及这些电力特征的关系。现在这种模型可以处理量得的端点数据并且将失效马力和电流数量化到马达满负荷百分之三的精度。

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ROOFBOLTING IN THE UK

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SYNOPSIS

Coal industries around the World are struggling for financial viability. The key component to be controlled is cost of production. It was recognised that roofbolting offered the possibility of reducing these costs, both by reducing material costs and increasing drivage rates. In addition, repair costs can be reduced if roadway support conditions are improved. Hence it was decided to investigate the potential of roofbolting, and to introduce it to UK mines where appropriate, always providing safety standards were not compromised.

Much of the World's coal comes from mines where the normal roadway support is roofbolts. This is particularly true of the USA, Australia and South Africa. It was clear that to effect the introduction of roofbolting to the UK in the most efficient manner, the experience overseas should be recognised and preferably utilised through active collaboration rather than a mere literature survey. It was decided to capitalise on these links and to seek active assistance from the US Bureau of Mines (USBM) and the Australian Coal Industries Research Laboratories Ltd (ACIRL). This paper provides a description of the introduction of bolting to the UK, and stands as an example of the value of international collaboration to mining.

1. INTRODUCTION

In the world of coal mining, it is common knowledge that British Coal are investing heavily in roofbolting trials, but it is mistakenly assumed that the introduction of roofbolting to the UK is recent. In fact, bolting using mechanical shell anchors was first used in the UK in 1946. Large scale trials of mechanical anchor bolts and grouts were conducted in 1952, and by 1959 over 60 miles of roadway in British Coal mines were solely supported by roofbolts. However, the basic unreliability of mechanically anchored bolts in the soft rock coal measures of the UK, meant that their use had died out by the mid 60's.

In the 1970's bolting as a means of strata reinforcement gained popularity to the extent that by 1982 British Coal's annual usage of steel rebar and wooden dowels was 330,000 and almost one million respectively. Following the strike of 1984-85, management within British Coal realised the

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potential benefits that could be offered by roofbolts as a general means of support. Drivage rates could be increased, material costs reduced, and repair requirements diminished. It was decided to investigate the potential of roofbolting, and to introduce it to UK mines where appropriate, always providing safety standards were not compromised.

Many of the world's profitable coal mines have roofbolts as the sole means of support throughout most of the roadways. British Coal already had excellent links with mining companies and research organisations throughout the world, and it was decided to capitalise on these links and seek active assistance from a number of organisations. This paper describes the outcome.

Section 2 details the knowledge base which British Coal had in the early 80's. In section 3 we describe collaborative work with the US Bureau of Mines (USBM), Australian Coal Industries Research Laboratories Ltd (ACIRL), and Cerchar. In section 4 we define the strategy that was determined in part as a result of the collaborative work, and finally in section 5 we summarise the results.

2. THE UK KNOWLEDGE BASE

Although the British Coal abandoned roofbolts as a sole means of support in the 60's, other countries continued down this line. New technologies associated with bolting were continuously examined by British Coal engineers, but for a different purpose - reinforcement and repair.

The benefits of strengthening weak and broken strata using this technology gradually became more widespread and initiated a whole new phase of developments. In 1970, the concept of full column dowelling was introduced using wooden dowels and capsule resins. This was followed in 1972 by resin injection dowelling. Throughout the following decade, strata reinforcement became a routine technique.

Most production is carried out by longwall faces using total caving techniques. Sometimes the roof above the face breaks down and requires remedial action. The extent of these problems can range from a few centimetres of roof dropping over a few metres in length, to cavities well over 100 cubic metres in volume requiring specialised cavity filling techniques. In simple cases, a row of wooden dowels inserted in the coal seam of the affected area is sufficient to hold the roof until the powered supports can be advanced underneath the freshly exposed roof. Sometimes steel dowels are used to secure the face of the cavity. In very severe cases, longhole injection methods may be employed. As deterioration tends to be time dependent and longhole injection tends to be time consuming, this method is usually a last resort.

The trend was towards the greater usage of full column dowelling, both steel and wood, at the expense of the older point anchor methods. In addition, resin injection techniques flourished in the early 1970's, peaking in 1977. The decline in the use of injection resins can be attributed to the relatively high level of expertise required.

An incident occurred at a British Colliery in 1974 which demonstrated the flammable nature of injection resins. The risk was deemed to be highest with these materials stored underground in their uncured

state. Accordingly, work was carried out by the manufacturers to develop safer alternatives. By 1978 this work was complete. All injection resins used since that time are gypsum or plaster based.

In 1976 a Working Group on the Introduction of Non-Metallic Materials Underground was formed to assess the potential risks associated with all existing and new non-metallic materials. In 1977, the Working Group decided that polyester resins capsules, then being used, were not acceptable for use underground. In their uncured state, they did not conform to British Coal standards with regard to products of combustion and flammability. However, it was decided to permit the use of these capsules with certain restrictions, and to encourage the development of safer capsules. This lead to the introduction of both low hazard polyester capsules, and inorganic capsules. It is probable that, at the time, British Coal were alone in requiring flame-resistant grout materials. The reduction in strength of the most popular capsule, water extended polyester, was recognised but not considered to be critical.

In the majority of cases, wooden dowels were used as the reinforcement member especially in machine-cut zones. Glass-fibre reinforced wooden dowels were also introduced in 1982 as these exhibited a four-fold increase in shear strength and were still machine-cuttable. Their use was restricted to particularly severe face repair applications.

Steel dowels were covered by a British Coal Specification. Rebar steel of 19 mm, 25 mm and 32 mm diameter mainly in 1.8 m lengths was usual. They were generally used in strata outside the machine cut sections where they could add considerable strength. The parent metal was standard reinforcing bar to BS4449. The minimum yield and elongation characteristics were considered to be ideal for essentially short term requirements of repair and reinforcement applications using full-column techniques.

By the end of the 1970's, bolting as means of strata reinforcement had become a routine mining technique. Once again management began to investigate the use of bolts as a means of support. Systematic trials were carried out at several collieries and these trials proved the benefits that could be gained.

However, there were two drawbacks: firstly, the installation technology was not then fully developed; and secondly, the widespread impetus to increase efficiency had not reached the levels required to overcome the natural resistance to this novel technology.

This reluctance was due to two main reasons. Bolts do not offer visual support and cannot be seen to be failing. Also, from an operational viewpoint, the introduction of bolting into the tunnelling cycle is an extra task to be performed. The use of lightweight supports in conjunction with bolts was offered as an alternative to conventional supports to attempt to reduce the overall task requirement.

3. HELP FROM OVERSEAS

Over the last three years British Coal has increased the application of rock bolting in its mines. This initiative is based on the new technology provided by full-column resin bonded bolt systems which are proven by routine application overseas.

Experience from abroad has been fully utilised by British Coal. Collaboration has taken place between the United States Bureau of Mines (USBM) and the Australian Coal Industries Research Laboratories (ACIRL). Similarly an appraisal of European methods has been made, and liaison with Centre d'Etudes et Recherches de Charbonnages de France (CERCHAR) has resulted in the use of their computer software as part of the design process employed by British Coal.

3.1 USBM

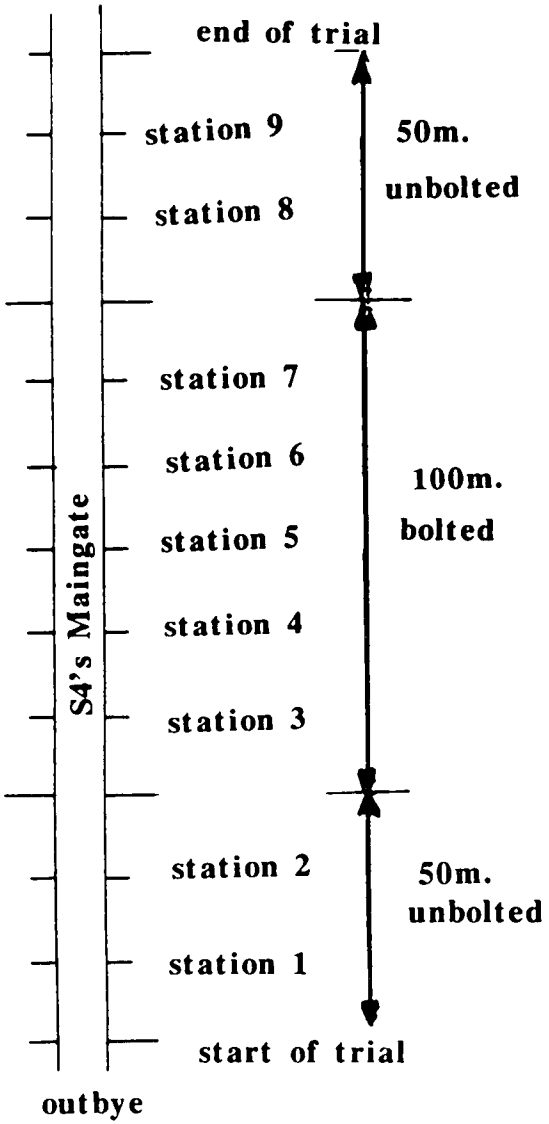
Liaison with the USBM resulted in two underground demonstrations using American experience and techniques. These were at Betws mine in South Wales and at Ellington mine in the North East of England. The liaison is ongoing although the underground installations are now complete. The knowledge imported from the USBM as a result of this liaison provided very real progress when British Coal was early in the learning process. The installations at the two collieries were exemplary in the standard of underground installation and provided flagships against which other practitioners could compare results. Basic but good installation procedure was defined to the benefit of the industry and many practical lessons were learnt. The importance of bolt/hole annulus size, spin time, resin type, hole length and straightness, drill bit type and many other parameters were placed in their correct perspective and importance. New equipment and instruments were introduced to the repertoire of the research team within British Coal. Essentially the basic parameters of a good bolt installation were defined.

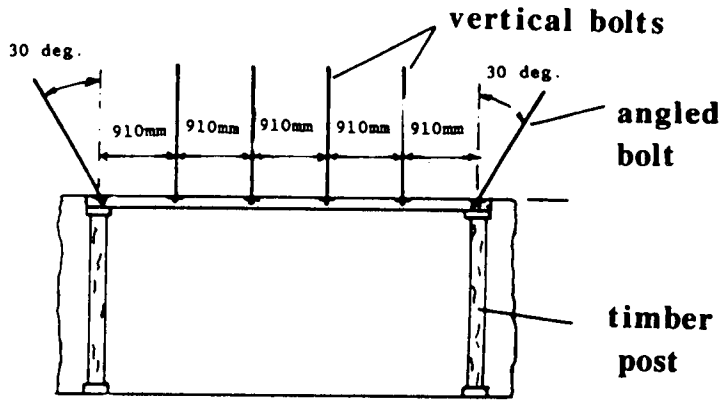
The demonstration installations which have been executed at Betws mine and Ellington consisted of a comparison between bolted and unbolted sections of a fully monitored roadway. A typical example is shown in Figure 1. The free standing support was required to be set and the efficiency of the bolt system could only be demonstrated by measurement of low (or zero) load on the free standing supports.

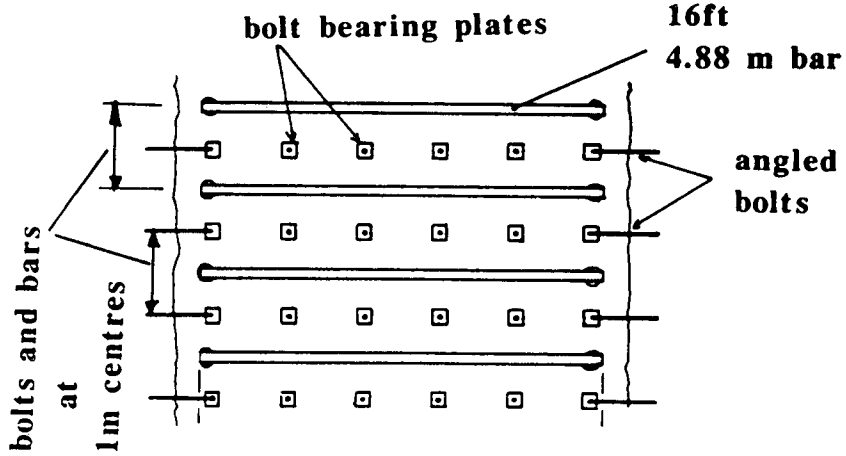
The current status is that the face has now entered the trial zone at Betws (a retreat gate roadway). At Ellington, the free standing supports have been removed in part of the trial roadway, and further support removal with remote monitoring is planned for the near future.

3.2 ACIRL

Australian experience has also been utilised and imported. This collaboration is more recent and currently very active. Visits to Australia by representatives from British Coal have established that many Australian mines have very similar conditions to those in the UK. The research expertise of ACIRL is being exploited in Wistow Mine (Selby Complex) to demonstrate that UK conditions can benefit from roof bolting to the same extent as mines in







Demonstration Site USBM Liaison: Betws S4's

Figure 1

Australia. The underground demonstration utilises 100% Australian equipment so that bolts, resin, straps and every aspect recreate the Australian technology in the UK. Australian experts are present on site for the duration of the trial and are accompanied by British Coal counterparts. The use of detailed monitoring methods is one of the major contributions from the demonstration. The data provided by these research techniques amply illustrate the efficacy of the bolt system to the management, men and inspectorate. This is important since bolts are routinely used as a sole means of support in Australia and the UK collaboration is designed to demonstrate that this is also technically possible in the UK.

The trial at Wistow Mine is underway at the time of writing this paper. The site is in a retreat face gate roadway and extends for the full length of the gate. The drivage is rectangular, in-seam and would normally have been supported on steet sets. Currently the trial has covered some 200 metres of drivage under bolted roof employing bolts, mesh and straps. Rib bolting has also been employed with fibre glass dowels on the face side rib. Free standing support at the start of the drivage was props and bars. This requirement was gradually relaxed as the drivage progressed, such that only one wooden end of the 2 mm thick roof strap. This therefore means that the site is solely supported by bolts.

The major problems have been non-technical. The gradual removal of the free standing support was only possible with the co-operation of the workforce, management, and the UK inspectorate. The factor which allowed this to occur was the high quality monitoring data which was explained in detail to all parties involved. This data unequivocably verified that the bolt system was operating well within design limits, with maximum roof bolt loads of 50 kN and maximum rib bolt loads of 160 kN. Similarly roof dilation, monitored to up to 7.5 metres into the roof, was measured to an accuracy of 0.10 mm. This indicated that roof movement of less than 2 mm was taking place, well within the elastic limit of the rock.

3.3 Design Methods

British Coal has examined various theoretical and empirical design methods for roof bolts as a sole means of support. These included the methods of Daws (1986 a,b), Bieniawski (1973, 1974, 1976, 1979) and CERCHAR. However, these methods are by their nature unable to take account of more than a selected few of the many geotechnical variables involved, and cannot be expected always to give a reliable answer in the complex mine environment. This is not to say that they should not be used but it is important to recognise the limitations of any design approach or predictive technique. Because of this a study has been made of the different methods and their respective recommendations have been compared in a series of case studies.

Daws has applied the hypothesis of a zone of compression around a rockbolt to coalmine roadway design. The only rock engineering property required to

use the method is the internal angle of friction. As such the Daws method is viewed as simplistic. When compared with the output of the CERCHAR computer program the Daws method usually suggests longer bolts but a wider spacing.

The method due to Bieniawski is based on his well known classification system which uses five major parameters:-

Strength of the intact rock
Drill core quality
Spacing of joints
Condition of joints
Groundwater condition.

Additionally, the joint orientation, in-situ stress ratio, and method of excavation are also used in mining applications. This classification system has been extended to give recommended bolt lengths and spacings.

CERCHAR have developed an empirical design method which is available as a computer program called PC bolting. The program is based on rock bolting experience in French mines and gives advice on monitoring and supplementary support requirements (if any). The input data includes:

Strata type and thickness
Physical properties
Fracture pattern and weathering
Roadway geometry and life
Groundwater conditions
Rock stress conditions.

The program has been found to give useful advice in a number of cases but some changes were identified as desirable to adapt the program to UK conditions. These have been incorporated by CERCHAR at the request of British Coal.

Physical modelling using a 1/50 th scale test rig has been much exploited by HQTD over a number of years. This has now been adapted to model the use of roof bolts and has been well reported. The technique is time consuming however; between one and two weeks are required to complete a model. The shortcomings of this method are therefore the time (hence cost) and the requirement for a highly skilled technician and the dependence on the accurate physical representation of the strata. Considerable success has been achieved using the method particularly as a qualitative guide.

All of the above methods have been, and still are, used by HQTD to help design roof bolt systems. However, increased experience has indicated that the most successful design data comes from monitoring the behaviour of roof bolt systems underground and the use of this data to modify the system itself. The methods of monitoring most useful in this respect have been identified as those used by ACIRL.

It is also recognised that the bolt system should be part of an overall mining concept. Initial use of bolts within British Coal tended to apply this technology to old mining situations. Hence bolting was eagerly applied to advancing gateroads, advance headings, roadways to be used twice. Frequently this approach gave some tangible benefit. However, such applications were always in conjunction with free standing support. It is now becoming recognised that sole support using roof bolts is most likely to be successful in situations where the mine opening is small, the stress regime is favourable, and where

the roadway is required for use once only with minimum massive ground movement due to adjacent excavation, ie, on retreat face gate roadways.

The design of bolt systems for essentially in-seam, rectangular, retreat gate roadways is now being based on measurement underground. Increasingly, the state of stress within the ground is becoming a dominant consideration, and features which modify the stress field (adjacent workings, fault systems, channels etc) are considered in the light of experience. This approach to design therefore relies less on pre-conceived models and equations, which can miss an essential 'site specific' point, but more on the application of rock mechanics analysis applied to sound and thorough monitoring and measurement. The method is directly comparable to the 'New Austrian Tunelling Method'. Its drawback is that it requires expertise to be available on site.

4. FORMULATING A STRATEGY

Currently, roof support by bolts in the UK is part of a hybrid system. That is, the free standing supports (arches) are placed additional to the roof bolts. Many examples exist where improved conditions result from this marriage. Fundamental differences in the two approaches exist however. Roof bolts are a stiff system, that is, deformation of the bolt is small, amounting to a maximum of typically 2 mm/metre (or 0.2% strain) if the bolt system is satisfactory. Arches are a low modulus system. That is, they accept relatively large deformations (more than 100 mm) as they offer restraint to the ground. In many instances voids around the support will mean that the ground has moved more than this before intimate contact with the arch occurs. Hence it can be proposed that if the arch takes weight the bolt system has failed, or at least partially failed. Therefore to maximise the benefit from bolt systems the design should be based on sole (or primary) support by the bolts.

The early trials involving overseas technology clearly identified the potential that roofbolting offered to British Coal. To further the application in a safe and cost effective manner clearly demanded a strategic overview with a supporting plan. The strategic overview provided for

i) Objectives
ii) Timetable
iii)Methodology
iv) Materials and equipment
v) Training
vi) Monitoring of Progress

It was recognised at an early stage that the role of the Mines Inspectorate would be vital, and that the key to successful relations with them was a well-defined and sensible plan.

The detailed objectives emphasised a step-by-step approach from bolting as a means of supplementary support to bolting a sole means of support. Specifically, some of the more important objectives were (and remain)

- to establish design criteria which will enable the correct choice of roofbolt support design to be applied in prevailing conditions

- to organise the expertise and resources available to British Coal in order to effect in the optimum manner applications of the new technology with appropriate technology transfer and training.

- to evaluate and control the introduction of all equipment and associated materials

- to establish safe and cost beneficial methods.

The timetable covers such aspects as evaluation of equipment and materials including surface trials and laboratory testing; the provision of technical support including design methods, monitoring and training; and a series of underground trials including work with ACIRL and the USBM.

The methodology for bolting at any particular site was established and tasks to implement this methodology were identified as shown below. These actions are the responsibility of various parts of the British Coal organisation including Headquarters Technical Department (HQTD), central purchasing, and groups of collieries called "Areas".

Task	Action to Enable this Task to be Done
(a) Assessment of the necessary support. This may require the use of geotechnical assessments and bolting trials.	HQTD to devise a scheme for assessment of any site. In some instances, Areas will be able to carry out appropriate geotechnical assessment themselves, in other cases it may need to come from a specialist source, eg HQTD or specialist company. Where appropriate HQTD will provide guidelines and training for Areas. HQTD will maintain a central core of expertise. Ideally, bolting parameters should be considered at the mine layout planning stage. This is especially important with regard to in-situ and mining induced stress fields.
(b) Economic assessment - does the proposed support plan make economic sense based both on up-front money, and life cycle costs?	Devise a user friendly means of economic assessment which Areas can use and which also takes into account future benefits such as reduced repair costs and possible re-use of roadways.
(c) Detailed design including the obtaining of necessary approvals.	HQTD needs to use currently available design procedures and modify or change these as necessary in light of wider experience.

(d) Given the geotechnical assessment the correct equipment needs to be specified and made available. The appropriate consumables (bolt, resins, etc) must be obtained.	Appropriate approval schemes need to be established. A few lead companies should be encouraged to offer turn-key services, including consumables. HQTD needs to continue overseeing field trials of new equipment. It is vital that all concerned appreciate the need for quality control. Area/ colliery management must seek it; purchasing must enforce it.
(e) The correct installation procedure must be adopted and rigorously enforced by management.	Appropriate training courses must be provided for both the management and the general workforce. In some instances training will be provided by the supplier of the turn-key service.
(f) There is a need to monitor and analyse the results. In some instances it may be necessary to change the bolting plan as bolting proceeds.	HQTD must establish a central data base and the procedure for feedback of information from Areas. HQTD will use this information to improve and update the whole methodology. HQTD will need to continue with the provision of specialist advice. Areas should co-ordinate their bolting activity and periodically make that information available to HQTD.

In order to ensure that all materials and equipment are safe and cost-effective, British Coal have rigorously applied various Acceptance Schemes. These procedures have been established to cover not only the basic functioning of equipment and materials, but also safety. It is recognised that in striving for sole support by bolts the emphasis should be on the integrity of the support system. This therefore means that the very best mechanical properties have to be attained in both resins and steel. British Coal is now seeking ways of achieving properties equivalent to those used abroad and may well have to introduce methods of handling straight polyester resins which allow equivalent or higher safety to that offered by the current flame retardant versions if flame retarded resins cannot be developed to equivalent mechanical standards. The current flame retarded resins used by British Coal are of lower strength and stiffness than straight polyester and cannot be manufactured with the same precision on viscosity and gel time. This means that practical installation difficulties can occur underground.

This presents an interesting safety issue. In changing the purpose and use of resins underground, the primary risk in respect to safety has changed, ie, the consideration given to fire risk has become secondary to the risk due to roof falls in sole support roof systems. This does not mean that overall safety has to be jeopardised however.

Safe handling methods for straight polyester resins may actually result in a much reduced risk of fire. Furthermore, underground storage of large quantities of resin for prolonged periods is less likely if it is in constant use as opposed to being held as insurance against a potential repair requirement.

Roofbolting as a means of primary or supplementary support requires high standards of design, installation practice and monitoring. These high standards must be backed up by both high quality materials and workmanship. Because achievement of these standards is dependent on the quality of the workmanship involved, it is essential that all levels of management and workmen are adequately trained. Training requirements can be divided into three, namely those for:

(a) Roofbolting Specialist Engineers;

(b) Area and Colliery Management; and

(c) Operators.

Various training courses have been initiated and the sound practices that have resulted have helped ensure the success of bolting.

As developments occur, it is essential that all levels within the industry are aware of the implications. At the forefront of the application of bolting technology are specialists who ensure that a data base is compiled so that all experience gained is recorded and evaluated on a rational basis. This information disseminated on a regular basis to all involved.

5. RESULTS AND CONCLUSIONS

Roofbolting is being applied at an ever increasing number of sites and since the resurgence began in 1985, several hundred sites have been bolted. The majority of these have produced cost effective results, and safety standards have been sacrified at none. Indeed, quite the reverse. It is becoming recognised that correct bolting provides a superior method of strata well beyond that obtained by conventional, passive support systems.

This rapid advancement has been due in no small measure to the overseas experience. Many British Coal engineers have visited mines in Australia and the USA, and personnel from these countries have been directly involved in UK mines. Their interaction with high calibre staff in the UK has permitted a rapid transfer of knowledge and technology. In particular overseas personnel have identified the need for:

i) training of all levels of personnel involved
ii) management of installation procedures
iii) use of correct equipment and materials
iv) geotechnical assessment and design procedures
v) monitoring of results and adaption of the bolting pattern as a result
vi) planning the appropriate form of strata control very early in the mine planning scheme.

A wealth of knowledge now exists in the UK from specialists in HQTD, through the Area bolting teams, to first class labour at

the collieries. However we continue to learn from the vast experience of our friends and colleagues overseas. We hope that they too can learn from our experience.

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BOULONNAGE DES TOITS EN GRANDE-BRETAGNE

D.J. Buchanan, C.J. Morris, S.R. Newson

Les industries minières du monde entier, luttant pour conserver une fiabilité financière, doivent principalement contrôler leurs coûts de production. Le boulonnage des toits permet de les réduire en diminuant les coûts de matériel et en augmentant la cadence de creusement. L'amélioration des systèmes de soutènement des voies permet de réduire également les coûts de réparation. Il a donc été décidé d'examiner le potentiel du boulonnage des toits et de l'introduire dans les mines britanniques là où il s'avère nécessaire, à condition de ne jamais compromettre les normes de sécurité.

Une grande partie du charbon mondial provient de mines dans lesquelles les boulonnages de toits forment les soutènements de voies habituels, surtout aux Etats-Unis, en Australie et en Afrique du Sud. Il est évident que pour mettre en place ce système en Grande-Bretagne de la manière la plus efficace possible, il faut prendre en considération l'expérience outre-mer et l'utiliser de préférence par une collaboration active plutôt que par une simple étude de documents. Il a été décidé de tabler sur cette collaboration et de demander l'assistance du Bureau des Mines Américain (USBM) et des Laboratoires de Recherches des Industries du Charbon Australien (ACIRL). Le présent rapport, qui décrit l'introduction du boulonnage de toit en Grande-Bretagne, démontre la valeur d'une collaboration minière internationale.

ANKERAUSBAU IN GROßBRITANNIEN

D. J. Buchanan, C. J. Morris, S. R. Newson

Weltweit kämpft die Kohleindustrie um ihren finanziellen Fortbestand, wobei Produktionskosten eine große Rolle spielen. Der Ankerausbau bot eine deutliche Möglichkeit, diese Kosten zu verringern, und zwar durch Reduktion der Materialkosten und Erhöhung der Vortriebsraten. Zusätzlich können die Unterhaltungskosten der Strecken durch verbesserte Ausbaubedingungen verringert werden. Es wurde daher beschlossen, das Potential von Ankerbauten zu untersuchen und sie, wo dies möglich war, in britischen Gruben einzuführen, wobei zu jeder Zeit dafür gesorgt werden muß, daß die Sicherheitsstandards nicht verletzt werden.

Ein großer Teil der gewonnenen Kohle kommt aus Gruben, in denen der normale Streckenausbau aus Ankern besteht. Das trifft besonders für die USA, Australien und Südafrika zu. Offensichtlich sollten die Erfahrungen aus anderen Ländern anerkannt und vorzugsweise auch durch aktive Zusammenarbeit genutzt werden, anstatt sich auf reine Beschreibungen in der Literatur zu beschränken, um die Einführung der Ankerbauten in britischen Gruben so wirksam wie möglich zu gestalten. Es wurde beschlossen, diese Verbindungen zu nutzen und um die aktive Unterstützung des U.S. Bureau of Mines (USBM) und der Australian Coal Industries Research Laboratories Ltd. (ACIRL) zu bitten. Der Vortrag gibt eine Beschreibung der Einführung des Ankerausbaues in Großbritannien und bestätigt die Vorzüge internationaler Zusammenarbeit im Bergbau.

КРЕПЛЕНИЕ КРОВЛИ НА АНГЛИЙСКИХ УГОЛЬНЫХ ШАХТАХ

Д.Дж. Бьюкенан, К.Дж. Моррис, С.Р. Ньюсон

Угольная промышленность во всем мире борется с финансовыми трудностями. Главный компонент успеха в сохранении жизнеспособности угольной промышленности - это сокращение стоимости добычи. Признанным является тот факт, что крепление кровли может быть одним из способов снижения стоимости добычи, как посредством сокращения стоимости материалов, так и путем увеличения скорости проходки. Может также быть сокращена стоимость ремонтных работ, если улучшить состояние крепи штреков. Вследствие этого, было решено изучить возможности улучшения крепления кровли, и, где возможно, ознакомить английских шахтеров с результатами этого исследования, всегда при этом соблюдая нормы безопасности.

Большинство каменного угля в мире добывается в шахтах, где крепление кровли осуществляется анкерными болтами, особенно это относится к США, Австралии и Южной Африке. Для того, чтобы эффективно ввести крепление кровли такого типа в Англии, необходимо изучить опыт этих стран и предпочтительно использовать его путем активного сотрудничества, а не просто посредством изучения литературы. Было решено использовать профессиональные контакты и запросить помощи у Горного Бюро США и у исследовательских лабораторий угольной промышленности Австралии. Настоящий доклад описывает начальные шаги по введению анкерного крепления в Англии, а также представляет собой пример международного сотрудничества в горной промышленности

联合王国的顶板锚杆问题

D. J. 布肯南, C. J. 莫里斯 及 S. R. 牛森

世界上所有的煤炭工业都在挣扎求经济上的生存，主要能控制的部分便是生产成本．我们认清了顶板锚杆从减低材料费用和增高开采量这两方面有可能缩减成本．此外，如果车行道情形改善，修理费用也可大减．所以我们决定调查研究顶板锚杆在联合王国煤矿上能够发挥的用途，而且在不妨碍安全标准的情况下尽量应用．

世界上的煤矿里面的车行道很多都是用顶板锚杆支撑的，特别是美国，澳大利亚和南非．如果要引进顶板锚杆到联合王国，很清楚的是要认清其他国家这方面的经验并且积极地技术合作，而不能仅仅地做点文献的检索而已．我们决定加强这方面的联系，求得美国矿业局（USBM）和澳大利亚煤炭工业研究有限公司（ACIRL）的协助．本文叙述锚杆作业介绍到联合王国的经过，以显示国际合作在煤矿上的价值．

TENDENCIES IN DEVELOPMENT OF ROADWAY AND CHAMBER SUPPORT IN POLISH COAL MINING INDUSTRY

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ABSTRACT

In 1982, the average depth of development operations performed by the Mines Development Establishment was 615 m. The depth increased at the rate of approximately 12 m/year. It is expected that the average depth of development operations will reach 826 m by the year 2000, even though most operations are performed below 900 m or even 1,100 m. Before the year 2000, the cross section of roadways, and especially cross headings, will grow rapidly due to ventilation requirements at great depth and the mechanization of tunnelling operations in case of roadways. The approximate cross section of cross-headings was 16.2 m^2 in 1980 and 16.8 m^2 in 1982. It is expected to reach at least 25 m^2 by 2000. Under such conditions, the primary task is to select proper preventive measures and maintenance for roadways and chambers, if possible, with minimum consumption of labour and materials. Throughout the last decade, the range of support designs has been enlarged significantly. Not long ago, the mining industry employed essentially two types of support: roadway arch support of KS/KO-21 bars and brickwork lining for chambers. At present, a series of modern arch and shield support designs are used. The design of closed-profile support and roadway drivage technologies has become more advanced. The paper presents tendencies in development of support designs which should be more commonly applied in Polish mining construction, based on the present knowledge of phenomena occurring in strata disturbed by mining operations.

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TENDANCES DU DEVELOPPEMENT DES SUPPORTS DES VOIES ET DES CHAMBRES DANS L'INDUSTRIE CHARBONNIERE DE LA POLOGNE

J. Maloszewski et K. Rulka

En 1982, la profondeur moyenne des opérations de développement effectuées par l'Etablissement du Développement Minier était de 615 m. Elle s'est accrue au rythme de 12 m par an environ. L'on prévoit qu'en l'an 2000, elle atteindra 826 m, même si la plupart de ces opérations s'effectuent au-dessous de 900 m ou même de 1 100 m. Avant l'an 2000, les coupes transversales des voies, et les galeries transversales en particulier, croîtront rapidement en raison des besoins de ventilation dans les profondeurs importantes et de la mécanisation des opérations de percement des tunnels des voies. La coupe transversale approximative des galeries transversales était de 16,2 m^2 en 1980 et de 16,8 m^2 en 1982. On prévoit qu'en l'an 2000 elle atteindra 25 m^2 au moins. Dans de telles conditions, la tâche principale est de décider des mesures et de la maintenance préventives appropriées des galeries et des chambres, en utilisant si possible un minimum de main d'oeuvre et de matériel. Au cours de la dernière décennie on a accru de manière significative le nombre des divers moyens de soutènement. Il y a peu de temps, l'industrie minière employait essentiellement deux types de soutènements: soutènements des voûtes de voies de 21 barres KS/KO et parois de maçonnerie en briques pour les chambres. A l'heure actuelle, on utilise une série de types modernes de soutènements de voûtes et d'écrans protecteurs. La conception des technologies de soutènements à profils fermés et de drivage des galeries s'est modernisée. Notre rapport présente les tendances du développement de certains types de soutènements; ceux-ci seront appliqués de manière plus générale aux constructions minières polonaises, sur la base des connaissances actuelles des phénomènes se produisant dans les strates perturbées par les excavations minières.

DIE ENTWICKLUNGSTENDENZEN IM STRECKEN- UND HOHLRAUMAUSBAU IM POLNISCHEN BERGBAU

J. Maloscewski, K. Rulka

1982 lag die durchschnittliche Tiefe der Aufschlußarbeiten in den Kohlegruben bei 615 m. Dieser Wert erhöht sich mit einer Rate von ungefähr 12 m pro Jahr. Es wird erwartet, daß die durchschnittliche Tiefe der Aufschlußarbeiten im Jahre 2000 826 m erreicht, obwohl die meisten Betriebe unter 900 m oder sogar 1,100 m liegen werden. Vor dem Jahr 2000 wird der Querschnitt der Strecken, besonders der Querschläge, rapide anwachsen, was sich mit den Bewetterungsanforderungen in großen Tiefen der Mechanisierung der Vortriebe erklären läßt. Der ungefähre Querschnitt der Querschläge betrug im Jahre 1980 16,2 m^2, im Jahre 1982, 16,8 m^2. Es wird erwartet, daß im Jahre 2000 ein Querschnitt von mindestens 25 m^2 erreicht wird. Unter diesen Bedingungen ist es am wichtigsten, die angemessenen Vorkehrungen zur Erhaltung der Strecken und Räume auszuwählen, die, wenn möglich, einen Minimalbedarf an Material und Arbeitsaufwand haben sollten. Im letzten Jahrzehnt hat sich die Auswahl der verschiedenen Ausbaukonstruktionen beträchtlich erweitert. Vor nicht allzulanger Zeit kamen im Bergbau nur zwei Arten von Ausbau zum Einsatz: Streckenbogenausbau mit KS/KO-21 Profilen und Mauerwerk zur Auskleidung von Hohlräumen. Heute werden eine Reihe moderner Bogen- und Schildausbaue verwendet. Die Konstruktion der Vollprofil-Ausbaue, sowie die Streckenvortriebstechnik sind fortgeschritten. Der Vortrag beschreibt die Tendenzen in der Entwicklung von Ausbauen, welche in vermehrtem Maße im polnischen Bergbau zum Einsatz kommen werden und sich auf die Kenntnisse der Phänomene, hervorgerufen durch aubbaubedingte Gebirgsstörung, stützen.

ТЕНДЕНЦИИ В РАЗРАБОТКЕ КРЕПЛЕНИЯ ШТРЕКОВ И ЗАБОЕВ В ПОЛЬСКОЙ УГОЛЬНОЙ ПРОМЫШЛЕННОСТИ

Ж. Малошевский и К. Рулка

В 1982 году средняя глубина разработки в шахтах составляла 615 метров. Глубина разработки увеличивалась примерно на 12 метров в год. Ожидается, что в 2000 году средняя глубина забоев достигнет 826 метров, хотя большинство работ будет производиться на глубине 900 метров или даже 1100 метров. В течение ближайших лет, до двухтысячного года, будут быстро возрастать площади поперечных сечений штреков и, особенно, забоев в связи с повышенными требованиями к вентиляции на больших глубинах и механизацией проходческих работ в штреках. Площадь поперечного сечения забоев составляла примерно 16.2 квадратных метра в 1980 году и 16.8 квадратных метра в 1982 году. Ожидается, что к 2000-му году эта величина достигнет 25 квадратных метров. В этих условиях основной задачей является разработка адекватных предохранительных мер и мероприятий по обслуживанию штреков и забоев при минимальной затрате труда и материалов. За последнее десятилетие значительно увеличилось разнообразие креплений. Не так давно в шахтах применялись два типа креплений: арочные опоры KS/KO-21 и кирпичная облицовка забоев. В настоящее время используется много разновидностей арок и опор кровли. Наиболее прогрессивным является применение низкопрофильных опор для штреков и прогонов. Доклад описывает тенденции в развитии новых видов крепи, пригодных для широкого применения в строительстве шахт в Польше, с учетом особенностей поведения пластов, нарушенных угольными разработками.

波兰煤矿工业中坑道和矿井支撑发展的趋势

J。马绍索维士基 及 K。鲁尔卡

１９８２年煤矿开发区开采的平均深度为６１５公尺。这深度的增加量约为每年１２公尺。到２０００年时估计开采的平均深度将达８２６公尺，虽然很多开采已达９００公尺甚至１１００公尺。到达２０００之前，由于在大深度的通风需要和机械化开采关系坑道（特别是交叉坑道）的截面势必将日益增加。１９８０年交叉坑道的截面约为１６。２平方公尺，１９８２年则为１６。８公尺。估计到２０００年至少将达到２５平方公尺。在这种情形下，主要工作是选择坑道和矿井需要最少的人工和材料的防护措施。过去十年中支撑的设计范围曾有显著的扩展。不久以前煤矿工业主要用下列两种支撑方法：坑道拱门用的ＫＳ／ＫＯ－２１ 杆件和矿井砖石衬层。现在各种新式拱门和防护支撑设计已被采用。闭式断面支撑设计已被采用。闭式断面支撑设计和坑道开筑祝技术现在十分先进。

本文根据最新煤层开采现象的知识讨论波兰煤矿建造时应采用得支撑设计的发展趋势。

EVALUATION OF PIT SLOPE STABILITY

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ABSTRACT

An overview of the Indian coal production trends has been presented out-lining the increasingly significant role of opencast mining at greater depths, requiring closer attention to the problems of pit-slope stability.

Various factors influencing the stability of slopes in open-pit mines have been briefly discussed and a case study of Umrer opencast mine in India has been presented. The general approach to the evaluation of pit-slope stability has been discussed for the specific geo-mining conditions obtaining at the Umrer opencast mine. Impact of mining depth, physico-mechanical properties of rocks and presence of tectonic faults on the final quarry batter slope has been brought out.

AN OVERVIEW

Energy demand in India is going to rise steeply in the years to come and coal sector will be meeting more than 60% of the commercial energy requirement inspite of substantial oil and gas discoveries. The perspective plan envisages a production of 417 million tonnes of coal by the turn of the century, a major share being contributed by open pit mining. Table No. 1 shows the trend of coal production in India and Coal India Limited (CIL) and the share of opencast mining in the total coal production of CIL.

It can be seen from Table No. 1, that in 13 years since nationalization, the production from opencast mines of CIL has increased five-fold from 20.77 million tonnes in 1974-75 to 100.00 m.tes in 1987-88. The share of opencast in the total production of CIL has gone up from 26% to 63%. Trends are similar for the entire coal production of India since Coal India Limited contributes and will continue to contribute about 90% towards India's total coal production.

The phenomenal increase in the production from opencast mines in the post nationalization era has been achieved as a result of enlarging the scope of modern means of mechanization. Introduction of a wide range of open pit mining equipment viz. electric rope shovels of up to 20 cu.m. bucket capacity, off-the-highway trucks of up to 170 T payload capacity and draglines of up to 32 cu.m. bucket capacity and 96 metres boom length, have brought in more and more coal reserves into the ambit of opencast mining with ever-increasing cut-off stripping ratios. The trend is likely to continue and will involve mining of coal at greater depths by opencast method.

Average depth of mining is likely to increase from 80-100 metres at present to 200-300 metres in the future. Large open-pit mines up to 14 MTY production capacity have been envisaged up to 450 metres depth of operations. Stability of open pit slopes has assumed considerable significance in this context, since change of pit slope by even one degree will involve saving or

Table 1: Trend of coal production in India & Coal India Ltd.

Year	Coal Production in M. Tes				Percentage Share of Opencast in Total Coal Production of CIL
	All India	Coal India Limited			
		O/C	U/G	Total	
1974-75	88.41	20.77	58.22	78.99	26.3
75-76	99.69	23.77	65.21	88.98	26.7
76-77	100.03	23.65	65.83	89.48	26.4
77-78	100.98	24.91	64.05	88.96	28.0
78-79	101.97	28.92	61.03	90.05	32.1
79-80	104.00	32.32	59.12	91.44	35.4
80-81	114.01	39.36	60.90	100.26	39.3
81-82	124.29	45.67	63.27	104.94	41.9
82-83	130.58	53.59	61.18	114.77	46.7
83-84	134.26	60.02	61.39	121.41	49.4
84-85	147.43	70.10	60.70	130.80	53.6
85-86	154.25	74.13	59.98	153.11	55.3
86-87	165.69	84.70	60.07	144.77	58.5
87-88	179.78	100.00	59.05	159.05	62.9
89-90	226.00	119.10	77.20	196.30	60.7
94-95	325.00	180.00	105.00	285.00	63.2
99-00	417.00	235.00	135.00	370.00	63.5

otherwise of very large quantities of overburden to be removed in mines of this size and depth, and consequent reduction or increase in the stripping ratio. It can often decide the viability of a project specially when a succession of coal seams are involved and increasing depth does not necessarily mean increased stripping ratio. Indian mine planners are gripped with the problems associated with slope stability which was of marginal significance hitherto.

GENERAL

The stability of a slope is generally determined by a number of influencing factors out of which the most important in an open-cast mine are the following:

- effect of water
- structure and properties of the rocks
- slope geometry
- time factor

Though the ground water level is brought down below the lowest point of the open-pit, it is not always possible to totally dewater a mine. It is all the more important to consider this particular aspect in Indian conditions where duration of monsoon is prolonged and the strata normally becomes saturated with water.

The geological structure, alternating layers of material, inclination and faulting have distinctive influence on slope stability. Indian coal deposits are rarely flat though usually gently dipping and stratified nature of the deposits makes it imperative that the heterogenous nature of the rocks be taken into account in any evaluation of slope stability. Faulting zones are zones of weakness due to tectonic stress. As a matter of precaution, special measures should be enforced while approaching tectonically disturbed areas. Slope height, length and slope angle in addition to material properties and hydrological conditions are influenced by the dimensions of the

mining machinery and the mine plan. Thus it is important to know them well in advance to specify the equipment in accordance with the stability requirements and to adapt the mine plan.

Unlike slope stability in more or less permanent construction works, in mining the time during which a slope is left unsupported plays an important role. Due to mining, the ground is relieved of weight and can react with heaving and/or lateral movement. The time-span during which the side slope is kept open can be reduced if dumping is organized to immediately follow mining. Such slopes can be steeper inclined because they are open for a shorter period of time. Calculation methods developed take into account the length of the open pit mine slope and the time factor and result in producing additional reserves, previously thought to be not mineable or alternatively the stripping ratio can be reduced.

UMRER OPENCAST MINE - A CASE STUDY

Mining activities at Umrer opencast mine were started in the year 1963. The present depth of the quarry is 55 to 70 M. At the beginning of mining operations, overburden rocks were dumped in external dumps along the pit periphery and the back-filling is being done by dragline sidecasting followed by dumping by off-the-highway trucks from the upper shovel benches.

GEOLOGICAL STRUCTURE AND STRATIGRAPHY

The coal deposit at Umrer is of semiellipsoidal structure extending 3.5 Kms. from East to West and 1.4 Kms. from North to South. The coal deposit is bounded by incrops of coal seams on north, East and West and by a major tectonic fault F1 on the South.

The rocks in the Umrer coal field consists of Lameta, Kamthis, Barkars and Talchir formations which correspond to creataceous, lower triasic, permean and permocarboniferous age, respectively. The entire deposit is covered by alluvium i.e. clay and sandy soil of recent geological age with a tendency to swell and has shear strength in the lower range.

Schematic stratagraphy of Umrer coalfield is shown in Figure 1.

Middle carbonaceous section of the Barakar formation contains three coal seams Top, Middle and Bottom. The dip range varies from 5° to 8° in the main sector of the coalfield. In eastern sector, the dip angle increases up to 10° and near tectonic fault F2, it reaches up to 14°. Partings between the coal seams consist mainly of shales. Medium hard sand stones also occur in small quantities. Top arnaceous section of the baraker formation consists of mainly fine to coarse grained sand stones of below medium to medium hardness.

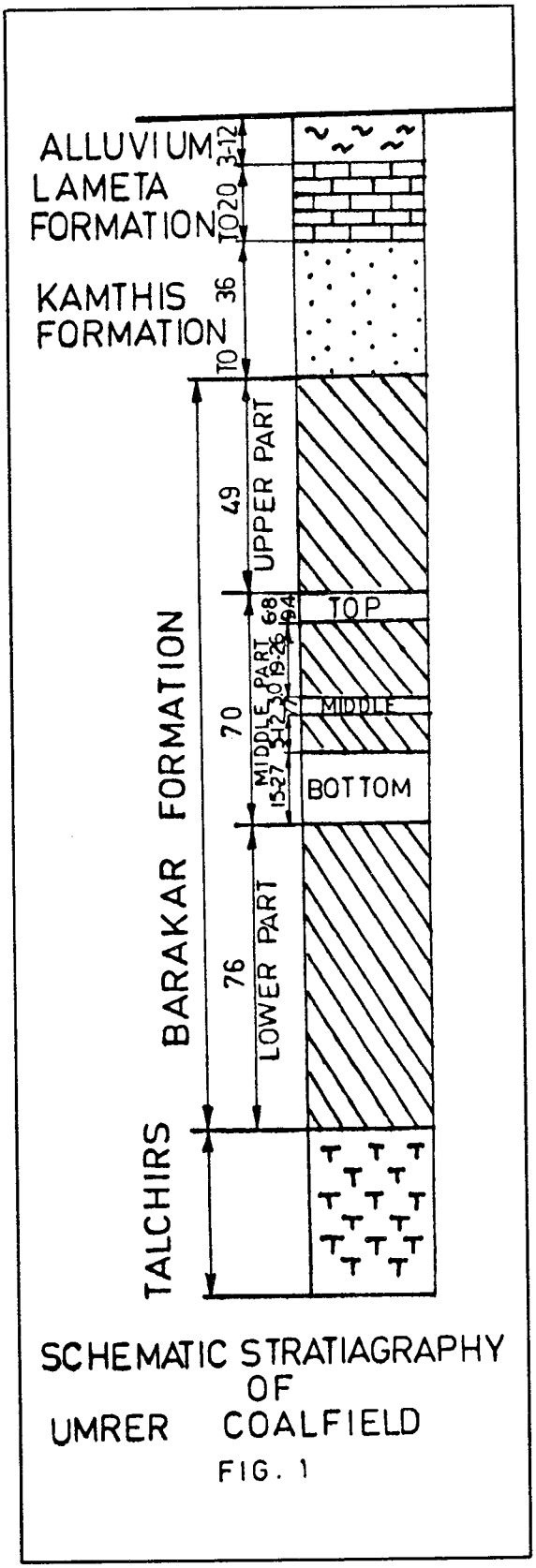
A total number of 12 tectonic faults have been identified by geological exploration and mine workings. All the faults are generally steep.

GROUND WATER INFORMATION

The hydrogeological investigations carried out in 1984-85 reveal that confined acquifer conditions are present in alluvial cover and in fractures of lameta and Kamthi sandstones. Similarly the movement of ground water in the Barakars is governed by the grain-size, porosity and fissures present in it. The acquifer system of coal bearing rocks of Barakar formation can be considered as a composite one. However, this composite system has been sub-divided into three semi-confined acquifers based on geological stratification.

The main source of recharge of the acquifers is the monsoon water that percolates through the soil cover to the formations underneath. The Amb river cutting through the soil cover and lameta seems to have no effect on the underlying formations. The

tectonic faults also do not show any effect on the water level of the acquifers.



SCHEMATIC STRATIAGRAPHY OF UMRER COALFIELD

FIG. 1

SELECTION OF DESIGN PARAMETERS

Six-numbers geo-engineering bore holes spaced at 475 M to 740 M were drilled to determine the detailed lithological composition of strata for testing of physico-mechanical properties. These boreholes were drilled from surface to a depth of 20 M below the floor of the Bottom seam.

The samples were tested for geo-engineering properties at Central Mining Research Station, Dhanbad and Regional Engineering College, Nagpur. On the basis of recommendations of the above institutions, the following values were adopted:

Volume density	- 1.9-2.0 T/m^3
Moisture	- 20-25%
Lower limit of plasticity	- 26-27%
Plasticity Index	- 45-48
Cohesion in mass	- 1-2 T/M
Angle of internal friction	- 9-10°
Cohesion at the contact with underlying rocks	- 0.8-1.0 T/M
Angle of internal friction at the contact with underlying rocks	- 9-10°

On the basis of field observations of rock exposures at Umrer opencast mine it was ascertained that the average distance between two vertical cracks varies from 0.18 m to 0.83 m and the distance between horizontal cracks varies from 0.2 m to 0.36 m. Based on this a structural weakening factor (C_{oc}) of 0.08 has been adopted.

Cohesion of rocks in situ was thus established by the following relationship:

$K_m = Koe.\eta$ T/M^3

EVALUATION OF FINAL PIT SLOPE STABILITY

The main aim of the evaluation of final pit slope was to determine values of stable slope angles which

will prevent the possible failures of slopes or safety zone left on the surface against the Amb river on the east, south and west sides of the quarry. Seven nos. geo-engineering cross-sections were drawn at a spacing of approximately 430 m. Results of six geo-engineering bore holes CMUM-1 to CMUM-6 were used for plotting these cross-sections.

Considering the importance of flood protection embankment near the final pit slope for safe mining operations and the need for long term stability of the pit-slope, a safety factor n=1.4 was adopted.

Based on the given characteristics of rock strength, the most probable position of the max. stressed surface was drawn (ref. figure 3). Necessary parameters were determined using the following relationships:

$$\theta = \frac{\pi}{4} + \frac{1}{2}(\rho - \rho') - \frac{1}{2}\arcsin\frac{\sin\rho'}{\sin\rho}$$

$$H_{90} = \frac{2K}{r}\cot(45^\circ - \rho/2), \text{metres}$$

$$W_n = 45^\circ + \rho/2, \text{ degrees}$$

$$a = \frac{2H[1 \cot\alpha.\tan(\alpha+\rho/2)] - 2H_{90}}{\cot\varepsilon - \tan\left(\frac{\alpha+\rho}{2}\right)} \text{m.}$$

Where $\varepsilon = 45^\circ - \rho/2$ degrees

Graphical relationship between the slope height and the width of the failure prison is shown in Figure 2.

The weighted average values of p and k for the most stressed surface were determined by the following formulas:

$$\rho = \frac{\tan\rho\ б\ S + \cdots\cdots + \tan\rho i\ бi\ Si}{\Sigma\ бi\ Si}$$

$$K = \frac{Kmi\,Si + \cdots\cdots + Kmi\,Si}{Si + \cdots + Si}$$

Where $бi = hi\sqrt{\cos^2\varphi}$

The weighted average values of p and k were modified taking into account the safety factor for the purpose of adopting the design values of rock characteristics.

$$\text{Thus } \rho n = \rho/n \text{ \& } Kn = K/n$$

As per design characteristics and the value of Hgo, the conditional height of the slope was determined using the relationship:

$$H' = \frac{H}{H_{90}}, \text{ metres}$$

Figure 4 shows the relationship between height and slope angle. For the given values of H' and design parameter P_n, the slope angle was determined. Slope profile was corrected on the geo-engineering cross-section based on the determined value of slope angle. Base length of the stress prism AD=L was ascertained for the adopted value of safety factor for verifying the slope profile arrived at above. For this purpose, a design surface was drawn (Ref. Figure 5). The prism of active pressure was divided into a number of slices for which the normal and tangential components of the weight of slices were determined using the standard equations:

$$Ni = Pi\cos\psi i$$

$$Ti = Pi\sin\psi i$$

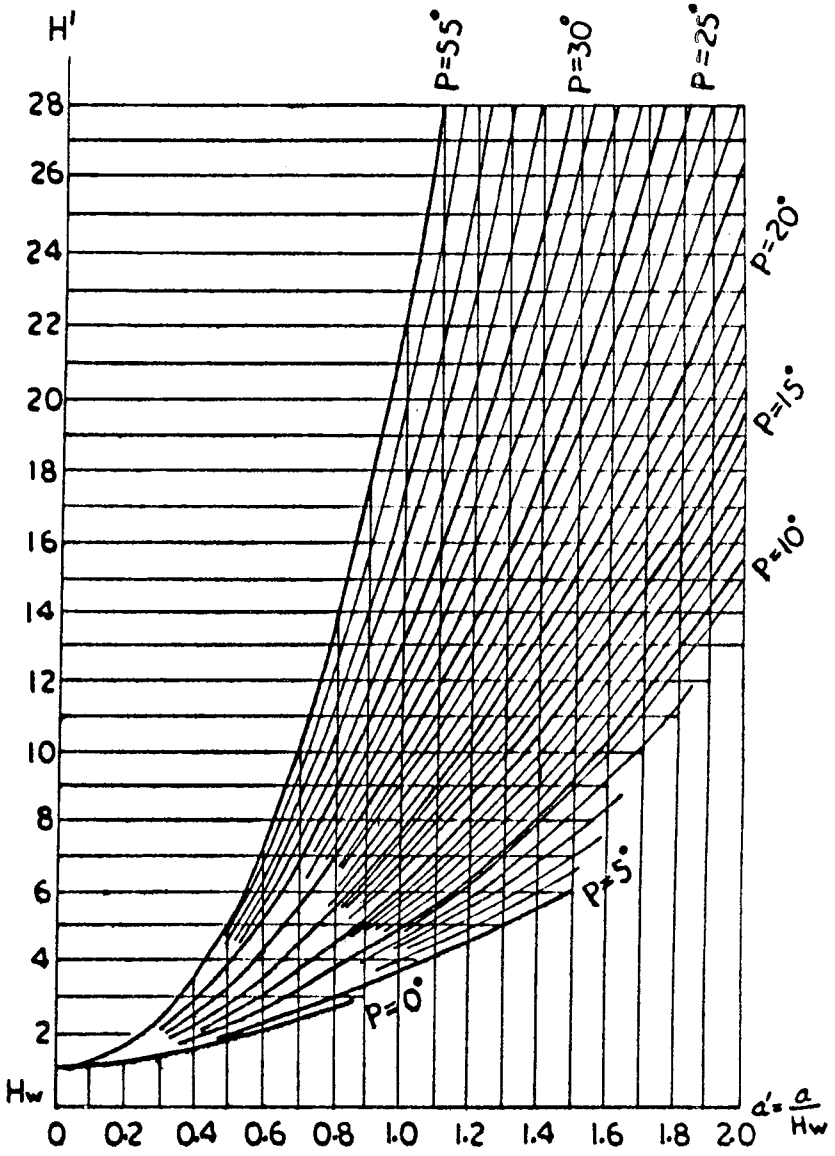
Where $Pi = hi.\gamma.ai$

Reaction between the prism of active pressure and stress prism was determined by the formula

$$E = \Sigma Ti - \tan\rho ni\ \Sigma Ni - Kn.l$$

Base length AD of the prism of stress AFD was determined by the formula:

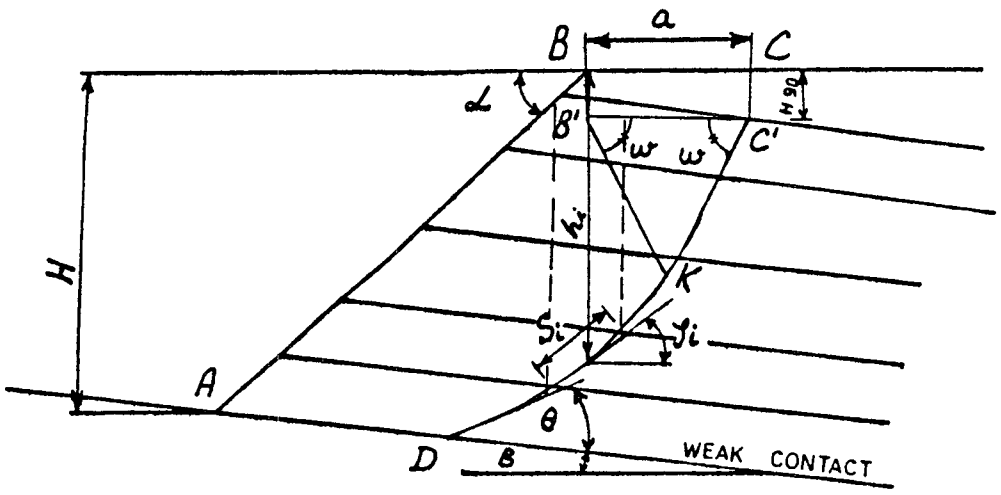
$$L = \frac{E\sin\theta(\cot\theta - \tan\rho' n)}{K'n + \frac{1}{2}h.\gamma.\cos^2\beta(\tan\rho' n - \tan\beta)} \text{m.}$$



GRAPH OF SLOPE HEIGHT AND WIDTH OF THE FAILURE PRISM

* GRAPH OF THE SLOPE HEIGHT AND WIDTH OF THE FAILURE PRISM IS DRAWN IN RECTANGULAR CO-ORDINATES, ALONG THE AXIS OF ABSCISOAE-CONDITIONAL VALUE OF THE WIDTH OF FAILURE PRISM AND ALONG THE AXIS OF ORDINATES-CONDITIONAL HEIGHT OF THE SLOPE GRAPH IS USED IN THE FOLLOWING WAY. USING PHYSIO-MECHANICAL CHARACTERISTICS IS DETERMINED AND CONDITIONAL HEIGHT AFTER THIS THE INTERSECTION POINT WITH THE GIVEN CURVE FOR DIFFERENT VALUES OF AND ALONG AXIS OF ABSCISOAE THE CONDITIONAL WIDTH OF THE FAILURE PRISM IS 10.5 DETERMINED. THE REAL VALUE OF THE FAILURE PRISM WILL BE

Figure 2



α - Angle of Quarry better slope, degree.

β - Dip angle of rock layers, degree.

θ - Angle between the tangent of slip surface and weak contact of rock layer, degree.

H_{90} - Height of vertical crack of separation, M.

ω - Angle of inclination of slip surface on the upper part of slope side degree.

a - Width of potential failure prism, M.

H - Height of slope side M.

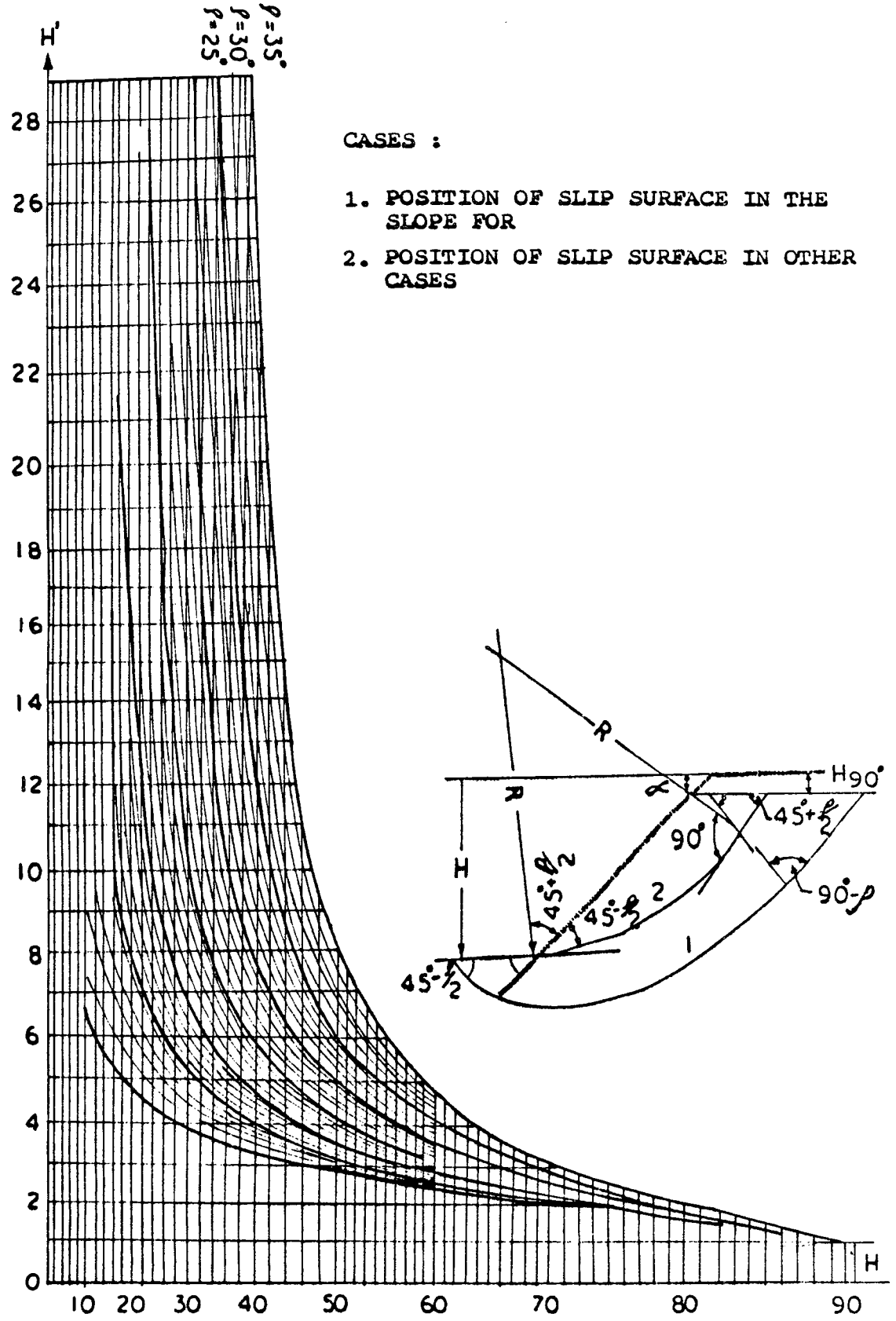
hi - Middle height of the sector having same lithological composition M.

Si - Sector of the most stressed surface having same lithological composition M.

γ_i - Angle of inclination of the base of the sector, degree.

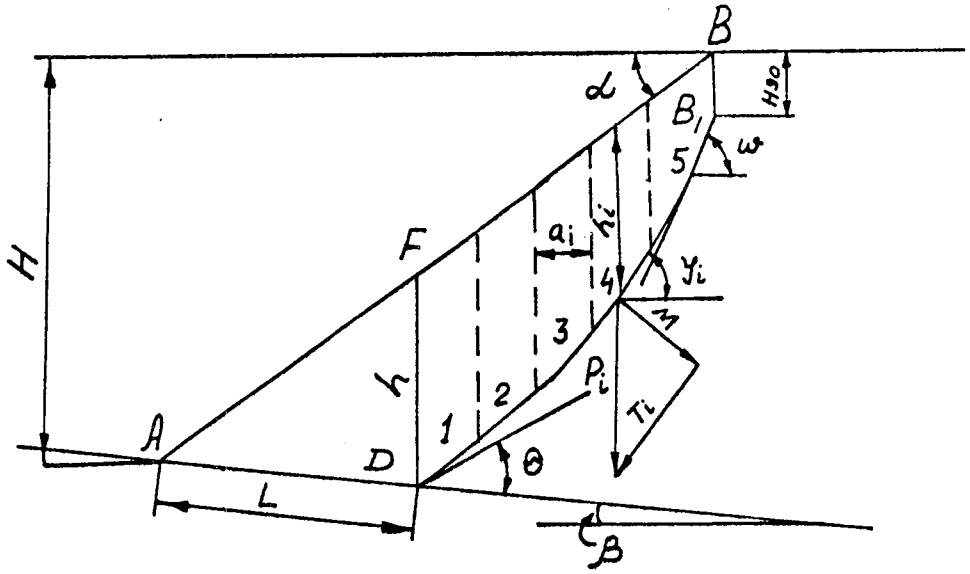
Scheme illustracting the tentative position of the most stressed surface in slope side with weak contact on the base.

Figure 3



GRAPH SHOWING THE RELATION BETWEEN THE HEIGHT OF PLAIN SLOPE AND SLOPE ANGLE FOR DIFFERENT CALCULATED VALUES

Figure 4



α - Slope angle degree.
H - Height of slope side, M.
β - Dip angle of rock layers.
FBB,D - Prism of active pressure.
AFD - Prism of stress.
L - Base length of the prism of stress M.
1.2,3,4,5 - Number of slicon of the prism of active pressure.
Pi - Weight of slices, T.
Ni - Normal component of the weight of the slice, T.
Ti - Tangential component of the weight of slice, T.
hi - Height of the slice, M.
ai - Width of the slice, M.
h - Adjustment height between prism of active pressure and prism of stress, M.
ω - Angle of inclination of the slip surface, degree.
φ_i - Angle of inclination of the base of slice, degree.
θ - Angle between the tangent of slip surface and weak contact, degree.
H_{90} - Height of vertical crack of seperation, M.

Diagram illustracting determination of the stability of plane slope with weak contact on the base.

Figure 5

The determined value of L was marked on the design cross-section and the profile of pit slope was corrected and finally the ultimate slope angle of the quarry was confirmed.

The above analysis was carried out for all the geo-engineering cross-sections and the resultant stable slope angles are tabulated below:

Cross Section No.	Quarry Depth in Metres	Stable Slope Angle in Degrees
1-1	55	31.5°
2-2	93	36.5°
3-3	120	36°
4-4	134	30°
5-5	97	33°
6-6	64	39°
7-7	48	42°
7'-7'	54	29°
7"-7"	52	26°

CONCLUSION

The final quarry batter slope for Umrer opencast mine varies from 26° to 42° depending upon the height of slope, physico-mechanical properties of rock strata and presence of tectonic faults.

In future, it will be necessary to check slope angles based on physico-mechanical properties of rocks, contact between rock layers and nature of tectonic faults as met during actual mining operations.

It will also be necessary to carry out systematic observations for all types of deformations of the batter to determine measures required for protections of slope.

APPENDIX 1

NOMENCLATURE

Coe - Structural weakening factor

Km - Cohesion of rocks in situ

K - Cohesion of rocks in test sample

η - Safety factor

θ - Angle between the contact and tangent to the slip surface

ρ - Angle of internal friction

ρ' - Angle of internal friction along the weak contact

H_{90} - Height of the vertical crack of separation

ωn - Angle of inclination of slip surface from horizontal

a - Width of the potential failure prism

H - Height of slope

α - Slope angle

β - dip angle of strata

hi - Av. height of the sector having same lithological composition.

γ_i - Angle of inclination of the base of the sector

σi - Approx. value of normal stress in the middle of corresponding sectors of the most stressed surface

Si - Sectors of the most stressed surface

H' - Conditional height of the slope

Ni - Normal component of the weight force

Ti - Tangential component of the weight force

r - Volume weight

Pi - Weight of the slice

ρ_{hi} - Angle of internal friction of the slice

l - Length of slip surface within limits of the prism of active pressure

h - contact height between the prisms of active pressure and stress

EVALUATION DE LA STABILITE DES PENTES DE PUITS

Mahip Singh, S.M. Kolay, N. Khurana

Nous présentons une vue générale des tendances de la production de charbon en Inde. Elle met en évidence le rôle de plus en plus significatif des exploitations à ciel ouvert plus profondes, ce qui exige une plus grande attention vis-à-vis des problèmes de stabilité des pentes des puits.

Nous donnons une brève description de divers facteurs influençant la stabilité des pentes des mines à ciel ouvert et présentons une étude de cas de la mine indienne à puits ouvert Umrer. Nous effectuons une évaluation globale de la stabilité des pentes des puits en considérant les conditions géo-minières spécifiques de la mine Umrer. Nous mettons en évidence l'influence de la profondeur des excavations, des propriétés physico-mécaniques des roches ainsi que de la présence de fissures tectoniques sur l'inclinaison des pentes des carrières.

BEWERTUNG DER STABILITÄT VON BÖSCHUNGEN.

Mahip Singh, S. M. Kolay, N. Khurana

Der Vortrag berichtet über Trends in der Kohleproduktion in Indien, insbesondere die wachsende Rolle der Tagebaue in größeren Tiefen, die eine besondere Aufmerksamkeit auf die Stabilität der Böschung erforderlich machen.

Verschiedene Faktoren, die Einfluß auf die Stabilität von Böschungen ausüben, werden beschrieben und eine Fallstudie an dem Umrer - Tagebau in Indien wird vorgestellt. Zur Beurteilung der Stabilität von Böschungen werden die spezifischen geologischen Bedingungen im Umrer - Tagebau berücksichtigt. Dabei werden auch der Einfluß der Abbautiefe, der physikalisch - mechanischen Eigenschaften des Gesteines sowie der von tektonischen Störungen auf die letzte Abbauböschung untersucht.

ОЦЕНКА УСТОЙЧИВОСТИ СКЛОНА КАРЬЕРА

Махип Сингх, С.М. Колеу и Н. Хурана

В докладе представляется обзор направлений, в которых развивается индийская угольная промышленность. Подчеркивается возрастающая роль карьерной добычи угля на больших глубинах, что требует серьезного внимания к проблеме устойчивости склона карьера.

Кратко обсуждаются различные факторы, влияющие на устойчивость склонов в открытых карьерах, и приводится пример Умрерского карьера в Индии. Описывается общий подход к оценке устойчивости склонов в применении к геологическим и горным условиям, существующим на Умрерском карьере. Обсуждается влияние глубины добычи, физико-механические свойства пород и присутствие тектонических разломов на склон карьера.

矿井倾度稳定性的评定

马赫帕 辛格，S. M. 柯来 及 N. 库拉拿

本文综述印度煤炭生产的趋势，大体讨论到深井露天采矿的重要地位，在这种情形下，需要密切地考虑到矿井倾度的稳定性。

这里简略地讨论影响开井倾度的稳定性，并对印度欧姆勒露天开采煤矿实情加以叙述。在欧姆勒露天开采煤矿特具的地矿情况下，我们述及矿井倾度稳定性的一般评定方法。矿深，岩石的物理机械性质和地壳断层对最后露天矿场井坡度倾斜度的重大影响亦在此指出。

FRACTURE STUDIES OF MINING STEEL AND MINING COMPONENTS

C. E. Nicholson and P. F. Heyes
Health and Safety Executive, Research and Laboratory Services Division, UK

ABSTRACT

Steel containing 0.15% carbon and 1.5% manganese has been used extensively throughout the past thirty years for haulage and suspension components in the UK mining industry. Initially this material was chosen for its resistance to strain ageing, allowing it to be used as a replacement for wrought iron in many applications. Studies undertaken in order to characterise the fatigue and fracture performance of this steel are described and the influence of variables such as heat treatment, environment and temperature is outlined.

Laboratory testing of components is discussed with special reference to static and dynamic testing of wire ropes, fatigue problems associated with mine auxiliary fans and degradation of high-pressure wire-braided hydraulic hoses. The use of the results to improve the safety of components in the mining industry is discussed.

1 INTRODUCTION

This paper, is divided into two sections: the first is concerned with the properties of mining steel, in this case 1.5% manganese low carbon steel, determined largely from small test-pieces; the second considers the performance and testing of actual components from the mining industry. The work described was carried out either within the Research and Laboratory Services Division of HSE, or at UK Universities under extramural contract.

2 MINING STEEL - GRADE 150M12

2.1 Background

In the past, and throughout most of the first half of this century wrought iron was the material used most frequently for the manufacture of colliery haulage and suspension gear in the UK mining industry. This material was fabricated into a variety of component shapes using traditional blacksmithing methods. There were, however, several disadvantages in using wrought iron which include: variability in properties both within a single component and also between components; welding difficulties relatively low strength which could not be improved by heat treatment and, most importantly, the susceptibility to strain age embrittlement in service. Strain age embrittlement occurs with time when plastically deformed components of wrought iron and mild steel are allowed to continue in service, resulting in an increase in hardness and a loss of ductility. In wrought iron components, for example suspension chain, this phenomenon had led to many service failures, some of which were due to incorrectly applied remedial heat treatment. From a safety point of view it was readily apparent that wrought iron was unreliable.

In order to overcome the problem of strain age embrittlement, and to provide components of higher strength, a low carbon steel containing 1.5% manganese was proposed originally for haulage and suspension components in 1936. By 1950, with improvements in welding techniques, this steel was being widely used in the UK mining industry. This material became known as grade 150M12 steel when British Standard 2772: "Specification for iron

and steel for colliery haulage and winding equipment. Part 2. Wrought steel" was revised in 1977

The basic mechanical properties, including tensile strength, notch impact toughness and S/N curve (stress range v number of cycles to failure) fatigue data were established for this steel in the 1950s and 1960s. Subsequently however, developments in engineering science revealed that, in order to predict fully the performance of a component under a variety of service conditions and environments, the determination of additional material properties was required.

Hence in 1980, to ensure the continued safety of 150M12 steel components in service and to permit the efficient design of other components, RLSD undertook a comprehensive programme of research aimed at characterising the fatigue and fracture behaviour of 150M12 steel in a variety of service conditions.

2.2 Subjects Studied

The following subjects were studied during the period 1980-1988:

a) fatigue crack initiation

b) fatigue thresholds ΔK_{th}, (the level of stress intensity range below which fatigue cracks will not grow) and fatigue crack propagation

c) effect of heat treatment on fatigue properties

d) effect of environment on fatigue performance

e) effect of waveform and frequency on fatigue performance

f) fracture toughness under static loading conditions

The results of these investigations have been widely reported in international journals and at conferences. Two PhD theses and numerous HSE internal reports have been written. A brief summary of each package of work is presented below.

2.2.1 Fatigue crack initiation

Although a great deal of effort had been devoted worldwide to the study of fatigue cracks initiating at geometrical discontinuities and artificial notches, at the time this work commenced little was known about the role of metallurgical discontinuities e.g inclusions in the initiation of fatigue cracks. A technique by which very small fatigue cracks in smooth surfaces could be detected and measured was therefore developed (Jutla et al 1984) and values of the threshold stress intensity range, below which fatigue cracks would not propagate, were determined for a variety of loading conditions. The nucleation of fatigue cracks at inclusions was clearly demonstrated (Figure 1) and the growth behaviour of a fatigue crack near the threshold level was shown to be dependent on crack length (Jutla et al 1986). It was also shown that threshold values for short cracks were significantly lower than for long cracks. Subsequently a great deal of effort has been devoted worldwide to the study of short cracks.

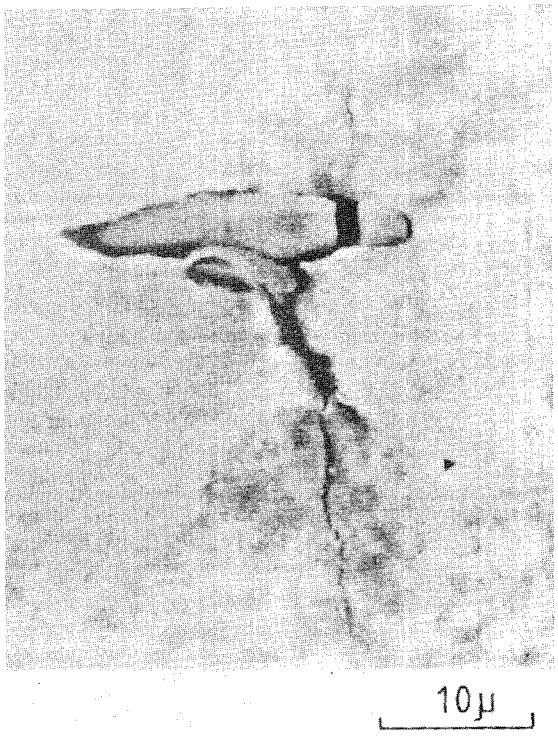


Figure 1.

2.2.2 Fatigue threshold and crack propagation

At the time this work began there were several experimental test methods available which purported to measure the fatigue threshold in terms of stress intensity range. A detailed study of the factors which influenced the fatigue threshold in 150M12 mining steel was carried out (Cadman et al 1980, 1982 and 1983A) and this highlighted the importance of the ΔK reduction rate $d(\Delta K)/da$ at which the stress intensity range was reduced with respect to crack length, during the threshold determination (Figure 2). This led to the development of a technique (Cadman & Bann 1986) which gave reliable and repeatable results, and this was then used to study the effect of stress range, orientation and heat treated condition on the fatigue threshold and the near threshold fatigue crack growth rate in mining steel (Cadman et al 1983B and 1983C). Subsequently the test technique was computerised and used to study the fatigue behaviour of other steels. It was found that the influence of manganese sulphide particles was to retard the fatigue crack growth rate and a mechanism for this retardation was proposed.

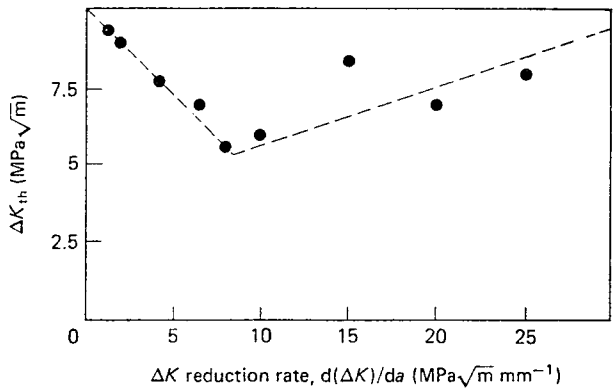


Figure 2

In a separate study (Blacktop et al 1982) it was established that, when 150M12 steel has been pre-strained, e.g by an accidental overload, the value of the fatigue threshold stress intensity range for a crack which develops subsequently is substantially reduced at low values of stress ratio (ratio of minimum applied stress to maximum applied stress). No effect was observed at high stress ratios.

2.2.3 Effect of heat treatment on fatigue performance

British Standard 2772:Part 2 specifies that 150M12 steel is to be used in the hardened and tempered condition and a range of austenitizing temperature (870-910°C) and tempering temperature (550-650°C) is quoted. Since different heat treatment temperatures give rise to variations in mechanical properties, a programme of work was undertaken to establish the extent to which fatigue crack growth rate could vary with heat treatment temperatures within the specified ranges (Geary & Bann 1988). Fatigue crack growth data is used, together with fracture toughness data, to assess the residual life of components containing cracks in order to establish safe operating conditions and appropriate inspection intervals. The results of this work showed that the variation in fatigue crack growth rate with respect to heat treatment temperatures was small and within the scatter band usually encountered in other low carbon and low alloy steels

2.2.4 Effect of environment on fatigue performance

Some UK coal mines are subject to ingress of water containing appreciable amounts of sodium chloride and hydrogen sulphide. Hence some components may be subjected to highly corrosive conditions together with cyclic loading. It is therefore important to quantify the combined effect of these two operational conditions.

Fatigue thresholds and fatigue crack growth rates were determined for 150M12 mining steel in air and saline environments, using a computerised test procedure (Anderson 1988). The results showed that measured threshold values were lower in aqueous saline solutions than in air and the stress ratio effect (i.e the decrease of the threshold stress intensity range as the stress ratio increases) was slightly reduced. In addition, testing in a saline

solution gave increased scatter in both measured thresholds and crack growth rates. Tests carried out in actual minewater, containing appreciable concentrations of sulphide ions, did not give reliable results because an accumulation of iron sulphide in the crack resulted in an underestimation of actual crack length using the potential drop monitoring technique.

Further work is underway using an alternative crack length measurement technique (back-face strain gauging) which will not be influenced by the test environment.

2.2.5 Effect of waveform and frequency on fatigue performance

In service, conditions are such that components are rarely subjected to the simple sinusoidal loading usually applied in laboratory testing. In addition, the frequency of loading usually varies with the application. In order to fully categorise fatigue behaviour it is necessary to know the influence of both waveform and frequency.

100 cycles/second, for both sine wave and square wave loading conditions (Bristow 1986). In addition, fatigue crack tip plasticity was measured using light interference microscopy. The effect of loading frequency, in the range 1 to 100 cycles/second, on fatigue crack growth rate was found to be small, but more pronounced at low rates of growth. Waveform variations did not have any consistent effect on growth rate.

This work also included consideration of the defect structure associated with fatigue damage and this involved a review of the methods used to measure the crack tip plastic zone size. The shape and severity of the fatigue crack tip plastic zone was shown clearly on interference photomicrographs (Figure 3). In many cases, a graph of fringe number versus distance showed a well defined change in gradient which indicated the true extent of the plastic zone. These data can now be used to predict stress intensity factors at fatigue crack tips.

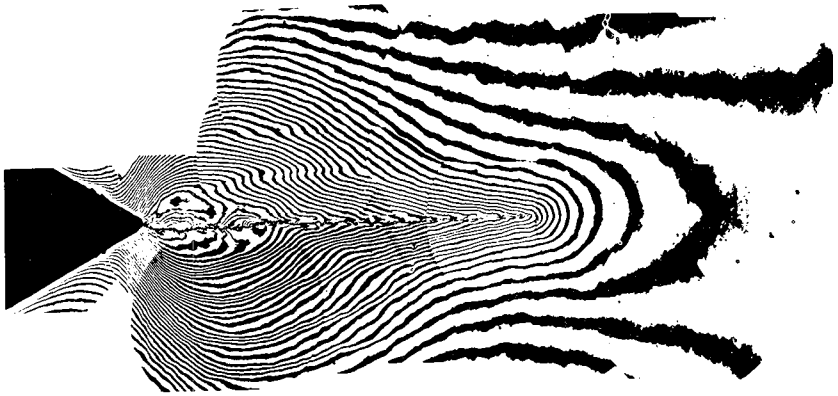


Figure 3

Fatigue tests were carried out in air at a range of frequency, between 1 and

2.2.6 Fracture Toughness

Fracture toughness data are used to explain how a component behaves in the presence of a crack-like defect and they are essential in carrying out defect assessments. There are several criteria by which fracture toughness may be assessed. It was demonstrated that, for a relatively ductile material such as 150M12 steel, "crack opening displacement" (COD) is the most appropriate fracture toughness parameter. The influence of thickness, orientation and test temperature on COD under quasi-static load conditions was assessed in the course of a comprehensive fracture programme using three-point bend fracture toughness specimens made from 150M12 plate material (Geary & Bann 1987). Results showed that 150M12 possessed a high level of fracture toughness which is comparable with other carbon and low alloy steels of an equivalent strength level.

Haulage and suspension components are occasionally subjected to impulsive loads which will influence the fracture toughness of the material from which they are constructed and may substantially reduce the factor of safety. These effects are being studied currently using 150M12 steel and data obtained will be used to assess the integrity of a suspension component under impulsive loading conditions, e.g. a hold-fast situation.

3 COMPONENT STUDIES

3.1 Auxiliary Ventilation Fans

In the UK mining industry there have been a considerable number of failures of welded blades on the impellers of centrifugal fans used for auxiliary ventilation. In this application, a very high degree of reliability is required in view of the potential for catastrophic results, as happened for example at Golborne (1979) and Houghton Main (1975). Failures have occurred in fans of differing design, from different manufacturers, and numerous investigations have been carried out in several laboratories. In all cases failure was attributed to fatigue cracking at the weld joining the blade to the impeller hub, an example is shown in Figure 4. These cracks were often initiated at the toe of the blade/hub weld but, occasionally they originated at the root of the weld, Figure 5. In some cases fatigue cracks were associated with corrosion or weld defects. Since these fans operate at high speed, typically 3000 rpm, it is possible for fatigue cracks to initiate and grow to a critical length in the interval between inspections.

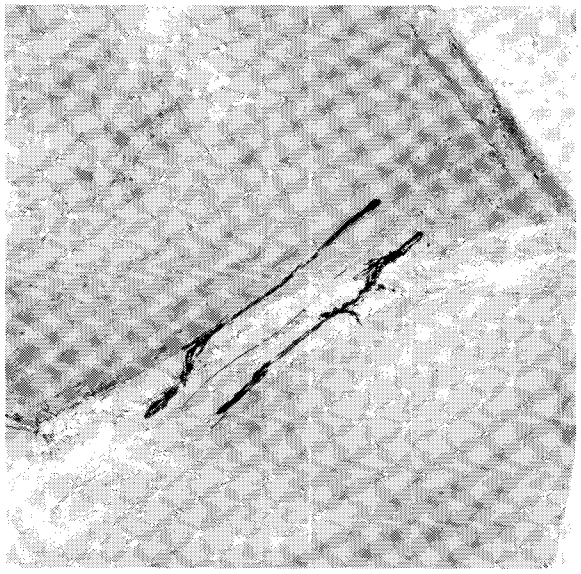


Figure 4

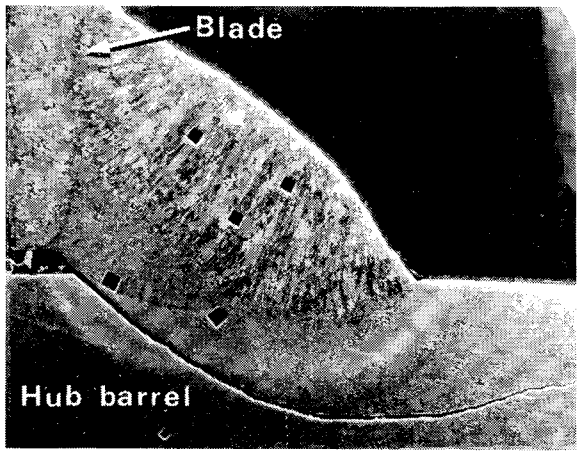


Figure 5

Early work involved a theoretical analysis of the stresses in the blades of a particularly suspect design of fan but the results did not agree with strain gauge data obtained by the manufacturer. In an attempt to resolve this discrepancy a finite element analysis was then undertaken.

This revealed that:

a) in some locations on the blade centripetal forces produced a high steady state stress, approaching the yield stress of the blade material

b) the superimposed cyclic stress was small.

c) there was considerable torque loading at the blade/hub junction.

d) once initiated, cracks propagated in a biaxial stress field.

It was clear that the situation was complex and fatigue cracking was occurring at high frequency and high mean stress but with a low amplitude cyclic load. Since 100,000,000 cycles of stress could occur in only 25 days a simple S/N (stress/number of cycles) curve approach was not applicable. One suggestion for the source of the cyclic stress was aerodynamic pulses travelling down the length of the ventilation ducts.

More recently a survey of failures was carried out by British Coal and all cracked fan impellers were sent to RLSD for metallurgical examination. The location of cracks, the microstructure in which they initiated and propagated, and the influence of welding defects was studied. It was confirmed that, although the majority of fatigue cracks initiated in the heat affected zone at the toe of the blade/hub welds, cracks frequently propagated into parent material rather than remaining parallel to the weld and in the heat affected zone. This is a consequence of the complex nature of the stress field into which the fatigue cracks tend to grow. However, it was not possible to determine whether the problem is associated with cracks which initiate early in the life of the component and grow relatively slowly to critical size, or whether the majority of cycles is expended in initiating a crack which, once formed, propagates very rapidly.

A research project is currently underway in order to study initiation and propagation of fatigue cracks more closely. In the first series of experiments the initiation and growth of small fatigue cracks, under both torsion and uniaxial tension, is being carried out on steel suitably heat treated to simulate the heat affected zone microstructures encountered in service. The formation and growth of small cracks is being studied using a replication technique developed originally at the University of Sheffield. The type of cracks formed (categorised as type I, II, or III using fracture mechanics terminology) and the relationship between crack growth and microstructure is being assessed for both tensile and torsional loading using a variety of mean stress levels consistent with service conditions. The influence of welding defects on initiation behaviour will be considered and tests will be performed using a spark eroded slit to simulate a defect. The final phase of this current work will investigate the growth of long cracks under biaxial loading conditions using a cruciform specimen (Figure 6) subjected to two loads applied independently at right angles. Crack extension will be monitored by an electrical resistance method (potential drop) and optically using a travelling microscope.

The current work is intended to demonstrate how the traditionally separate disciplines of fatigue crack initiation and fatigue crack growth may be combined to predict the service life of a component, in this case a fan impeller. Inspection intervals can then be chosen so that crack detection occurs before failure. The data obtained will be applicable to many of the current designs of fan impeller and will provide a basis for an improved design.

3.2 High Pressure Hydraulic Hoses

In 1981 a miner was killed when a high pressure steel braided hose, carrying hydraulic fluid underground, ruptured near where he was working. The burst region is shown in Figure 7. There have been numerous other incidents where serious permanent injury has resulted from bursting of hydraulic hoses, typically 25mm outside diameter and operating at 2500 psi. An

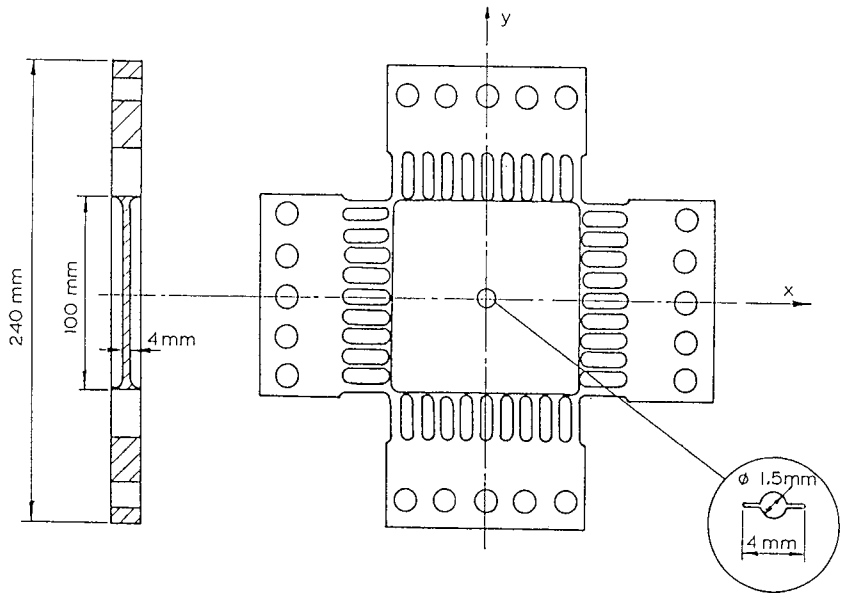


Figure 6

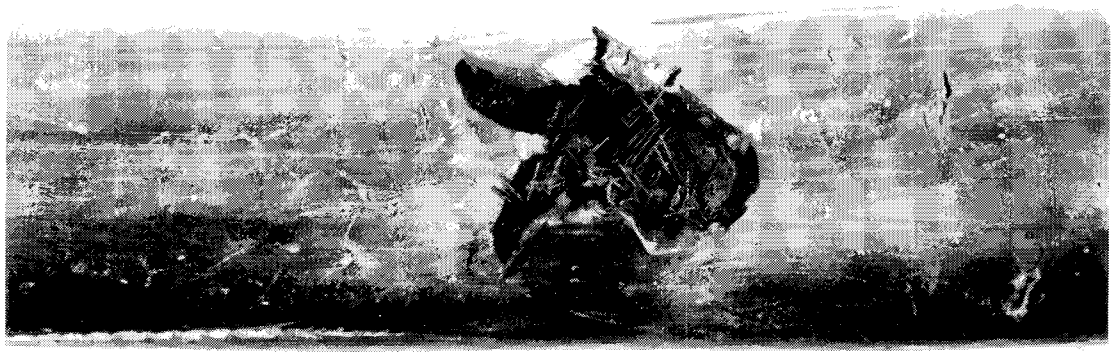


Figure 7

investigation of the 1981 incident, (Nicholson & Wilson 1983A) attributed the failure to weakening of the reinforcing steel braid by corrosion at the site of localised mechanical damage to the outer rubber cover. Some UK minewaters have a high sodium chloride content and are highly corrosive.

Hydraulic hoses of this type usually consist of five layers: an external rubber layer, two layers of steel braiding separated by a layer of insulating rubber, and an internal layer of rubber. When the external layer of rubber is damaged, for example by abrasion, the outer layer of braiding may be exposed to corrosive attack. The most serious corrosion does not necessarily occur at regions where damage to the external rubber layer is extensive and it has been shown that a small cut in the external layer is potentially worse than an area where a relatively large area of rubber has been removed. Corrosive liquid is more easily retained in a slit and crevice corrosion of the braiding can occur. Unfortunately localised damage is much more difficult to detect on routine inspection.

The investigation (Nicholson & Wilson 1983B) disclosed three mechanisms by

which a corrosive liquid could subsequently reach the inner layer of braiding:

a) by direct contact between the inner and outer layers of braiding when the intermediate rubber layer has ruptured.

b) by capillary action along the identification thread, incorporated during hose construction.

c) by transport along the interface between regions of rubber which have been extruded from the internal layer of rubber through the braiding.

Research showed that, in extreme cases, corrosion of the braiding could reduce the burst pressure to below the working pressure in as little as 150 days. RLSD recommended that the use of stainless steel braiding and/or the use of a more abrasion resistant outer cover would drastically reduce the number of incidents of this type. Both of these options have been adopted by hose manufacturers and a number of new designs have recently been proposed.

3.3 Wire Rope Work

Under some circumstances colliery winding ropes may be subjected to impulsive loading, for example when a hold-fast situation occurs. However, very little information on the effects of high strain rates on the performance of winding ropes and their terminations is available. An appropriate programme of work was therefore commenced at RLSD in 1985. Before specifying tests for the study of impulsive loading a preliminary project on the effects of quasi-static overloads was carried out using 2 metric lengths of 19 mm locked-coil wire rope terminated with white metal sockets (Robinson 1986). The effects of quasi-static overload on permanent extension, energy absorption, diameter, lay length and socket draw were investigated. Tests were carried out on a 4 MN wire rope testing machine and an optical extensometer was used to measure extension.

A relationship between quasi-static axial overload and permanent extension was established. Hence if the permanent extension on a wire rope in service can be measured, then the quasi-static overload to which the rope has been subjected may be deduced. From the latter the loss in energy absorption capacity may be estimated.

Measurement of change in rope diameter did not provide a satisfactory indication of overload, the results being dependant on the location of the measurement and the degree of ovality. Similarly the detection of overload using the lay length was not reliable since the change over one lay length was too small. The use of a longer gauge length would require indelible markings on the rope. This would not be a practicable proposition in the workplace. It was not possible to establish a relationship between socket draw and overload, although it was shown that draw was greater in new sockets.

Testing of wire ropes under impulsive loading conditions is currently in progress on the Impact Test Track at RLSD. Initially 2m lengths were tested for comparison with the quasi-static data. Preliminary results have suggested that when an overload occurs under impulsive loading conditions, in this case at a velocity of 6m/sec, permanent extension does not occur uniformly along the entire length of the rope. Further tests will be conducted using 8m and 50m lengths of rope at test velocities up to 11m/sec.

4. CONCLUSIONS

4.1 The fatigue performance of the steel used most widely in the UK mining industry for haulage and suspension components has been thoroughly researched. The results confirm the suitability of this steel for mining applications and, in conjunction with fracture toughness data, they enable defect assessments and residual life calculations to be carried out with a high degree of confidence. The effects of variables, such as frequency and environment, have been assessed to allow predictions of fatigue life to be refined.

4.2 The results of HSE's fatigue studies of 150M12 steel have been widely publicised and thus have advanced the general understanding of the fatigue process. The various projects have provided valuable information on which standard test methods may be based.

4.3 The problem of fatigue cracking in auxiliary ventilation fans has been studied and, when complete, results will enable safe inspection intervals to be specified. Recommendations for design changes will be made also.

4.4 The causes of failure in high pressure hydraulic hoses have been determined and reported. Design recommendations have been adopted by manufacturers.

4.5 The response of colliery winding ropes to overloads, under low and high strain rates conditions, is being assessed. A technique for relating permanent extension to overload has been proposed for impulsive loads.

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ETUDES DES FRACTURES DES ACIERS DE MINES ET DES ELEMENTS MINIERS

C.E. Nicholson, P.F. Heyes

De l'acier contenant 0,15% de carbone et 1,5% de manganèse est utilisé depuis trente ans de manière intensive pour fabriquer des éléments de roulage et de suspension destinés à l'industrie minière britannique. Au départ, cet acier a été choisi pour sa résistance à la déformation causée par le vieillissement, ce qui permet de l'utiliser pour remplacer le fer forgé dans de nombreuses applications. Ce rapport présente les études effectuées pour évaluer les performances de déformation et de fractures de l'acier, ainsi que l'influence de facteurs variables tels que le traitement par la chaleur, l'environnement et la température.

Il examine les essais d'éléments en laboratoire, en particulier, les essais statiques et dynamiques des câbles métalliques, les problèmes de déformation des ventilateurs miniers auxiliaires, la détérioration des tuyaux hydrauliques en fils émaillés à haute pression et les essais de structures à grande échelle. Il décrit l'influence des résultats sur la sécurité dans l'industrie minière.

BRUCHSTUDIEN AN BERGBAUSTAHL UND BERGBAUKOMPONENTEN.

C. E. Nicholson, P. F. Heyes

Seit ungefähr 30 Jahren wird Stahl mit einem Kohlenstoffgehalt von 0,15% und einem Mangangehalt von 1,5% allgemein für Förderer und Aufhänge-Einrichtungen in der Bergbauindustrie von Großbritannien verwendet. Dieses Material wurde zunächst wegen seiner guten Alterungsbeständigkeit gewählt, wegen derer es in zahlreichen Anwendungen als Ersatz von Schmiedestahl benutzt werden konnte. Verschiedene Studien, welche die Ermüdung und das Bruch-Verhalten der Stähle zu charakterisieren versuchen, werden vorgestellt, und der Einfluß einiger Variablen, so wie Wärmebehandlung, Umwelt und Temperatur, wird beschrieben.

Laborexperimente werden besonders im Hinblick auf statistische und dynamische Tests an Drahtseilen, auf Ermüdungsprobleme an Sonderbewetterungs-Ventilatoren, auf den Verschleiß von Hochdruckschläuchen mit Stahlgeflecht und auf umfassende Tests an Strukturen diskutiert. Der Einfluß der Ergebnisse auf die Sicherheit in der Bergbauindustrie wird beschrieben.

ИЗУЧЕНИЕ ТРЕЩИН В СТАЛИ, ИСПОЛЬЗУЕМОЙ ПРИ СТРОИТЕЛЬСТВЕ ШАХТ

К.Е. Николсон и П.Ф. Хейз

Сталь, содержащая 0.15% углерода и 1.5% марганца, широко использовалась в последние 30 лет английской горной промышленностью в откаточных и подвесных компонентах. В начале, сталь была выбрана из-за сопротивляемости деформационному старению, что позволило во многих случаях применять ее вместо кованого железа. Описываются результаты исследований поведения стали в отношении усталости и возникновения трещин, а также влияние таких переменных, как термообработка, окружающая среда и температура.

Обсуждаются результаты лабораторных испытаний, в которых специальное внимание обращалось на статичное и динамичное испытание кабеля, проблемы усталости в фенах вспомогательной вентиляции в шахтах, деградация гидравлических шлангов высокого давления, обмотанных кабелем, а также полное испытание конструкций. Описывается воздействие результатов исследования на безопасность в горной промышленности.

矿用钢料和矿用零件的断裂研究

C．E．尼可生　及　P．F．海斯

含百分之0．15碳量和百分之1．5锰量的钢料过去三十年来在联合王国的煤矿里应用得非常广泛．起初这种材料的选择是由于它对应变老化的抵抗力，使它可以在很多应用方面代替熟铁．这里我们叙述怎样表示它疲劳和断裂的性质，并且约略地讨论热处理，环境和温度这些因素的影响．

本文讨论零件的实验室试验，例如钢丝绳的静力和动力试验，矿用附属通风扇的疲劳问题，高压钢丝编织的水管的剥蚀，和实尺度结构试验等．试验结果对矿业上安全问题的影响亦在此讨论到．

DEVELOPMENT AND TESTING OF A MEASURING INSTRUMENT FOR THE DETERMINATION OF CORROSION AND BREAKS OF STEEL CORDS IN STEEL CORD CONVEYOR BELTS

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ABSTRACT

Belt conveyors are important transportation facilities in which faults and damages have to be detected as early as possible for repair and maintenance, especially in the case no parallel facilities exist. Corrosion and break damages in the steel cords of conveyor belts can possibly lead to belt rupture. Because belts are also used for manriding, rupture must be prevented by all means.

In a research project, subsidized by the state of Northrhine-Westphalia, the Tremonia Experimental Mine, in conjunction with the Micro-Epsilon-Meßtechnik, has developed up to service stage and tested a belt monitoring equipment for the detection of break and corrosion damages of steel ropes. The present paper describes the arrangement of the measuring instrument and gives details of test measurements on a test belt at Tremonia Experimental Mine and on conveyor belts in active mines. With the measuring diagramms faults were located and analyzed, and, for documentation, recorded with a portable x-ray apparatus.

INTRODUCTION

Today, conveyor belts of great length and capacity are used in many main entries underground. Just like in shaft hoisting, in most cases no parallel way exist, where, in case of fault, the hoisted material could be directed to. Many belt conveyors underground are used too for manriding. The high demands with regard to correct functioning and safe manriding must be met by appropriate monitoring and maintenance measures. Quite a number of monitoring and control installations serve to prevent or early detect faults and disturbances in the operation of the belt conveyors, so that appropriate repair and maintenance work could be carried out.

Many of the main belt conveyors are equipped with steel cord belts. This paper deals with corrosion and break damages of the ropes vulcanized into rubber, so damages cannot be directly determined. Corrosion damages can occur after longer service time, when the cover layer is damaged and water penetrates into the steel cords leading to uncontrolled corrosion. Figures 1 and 2 show x-ray photographs of such corrosion damages. Damages can also occur during transportation of the belt coil, during mounting, or operation by falling sharp edged parts. By this, the steel cords in the belt can be damaged or broken without necessarily visible damages of the surface layer. Figures 3 and 4 show x -ray photographs of rope breaks with only small gaps.

Corrosion damages and breaks in the steel cords are only partly visible at the rubber cover layer, which makes it difficult to determine precisely the size of the damage. The corrosion and break damages lead to a reduction in tensile strength and in extreme cases to a rupture of the conveying belt. Serious damages, which present a safety risk, have to be avoided for man riding, but also because of operational disturbances

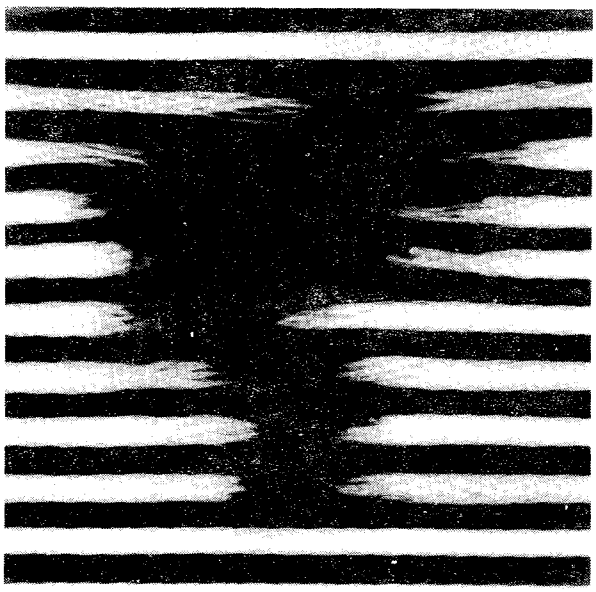


Figure 1. X-ray photograph of a corrosion damage

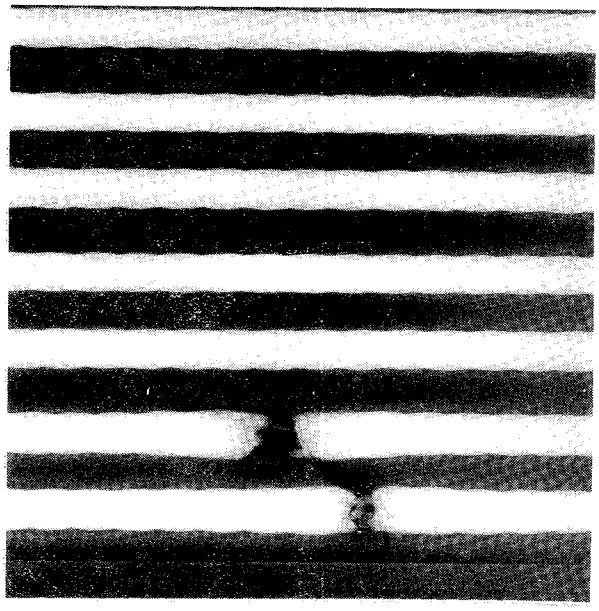


Figure 3. X-ray photograph of a steel cord break

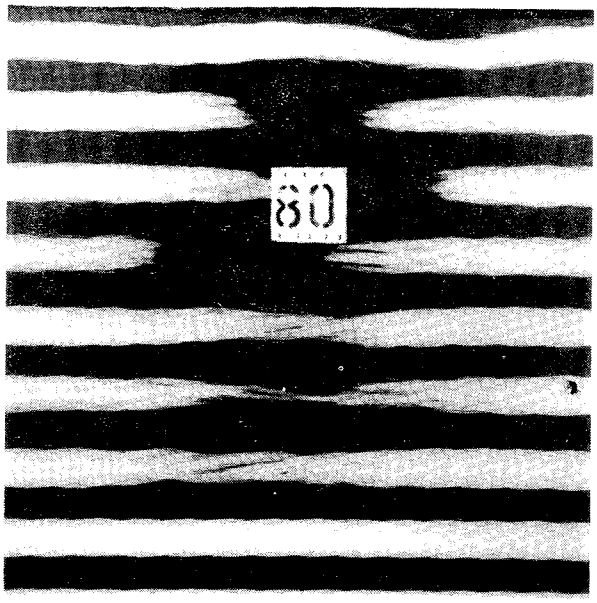


Figure 2. X-ray photograph of a corrosion damage

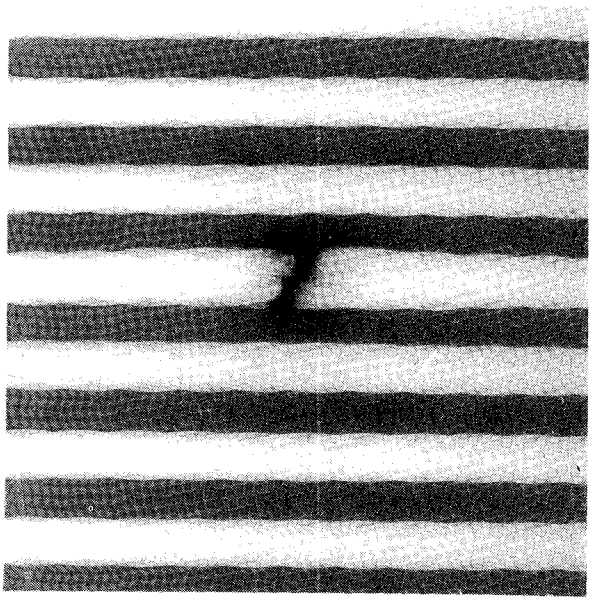


Figure 4. X-ray photograph of a steel cord break

and greater subsequent damages. Therefore, the monitoring of belt conveyors has to be of such conditions that a rupture of belt is avoided, and safety and economic risks are minimized by appropriate repair and maintenance work.

In the last years Tremonia Experimental Mine has tested measuring methods for monitoring steel cord damages and has, in a research project, subsidized by the State of Northrhine-Westphalia, developed a measuring instrument up to service stage. The measuring instrument, as well as measurements on the test belt of Tremonia Experimental Mine and belt conveyors underground are described in the following.

TEST BELT OF TREMONIA EXPERIMENTAL MINE

Different measuring methods and instruments and the effect of their modification could not be assessed with justifiable efforts in current underground measurements. On the one hand underground measurements are too time-consuming and expensive, on the other hand there is a great number of different types of faults that has been x-rayed underground in few cases only. Therefore a small belt conveyor with 15 m distance between the axes was installed at Tremonia Experimental Mine (Fig. 5).

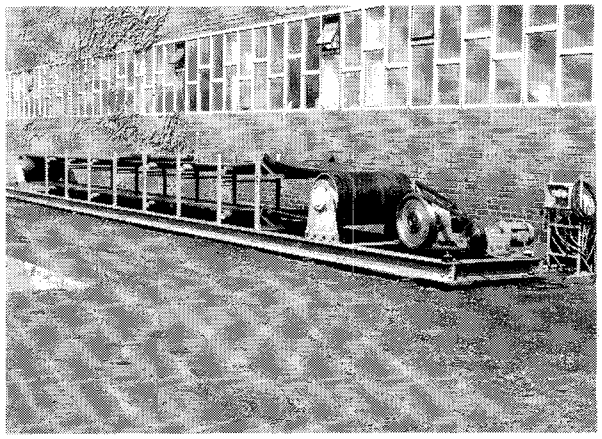


Figure 5. Test belt installation

The steel cords have defined defects (Fig. 6), ranging from a broken steel cord with facing ends (defect 1) to large size defects (defect 8) and different combinations of faults. This test belt is regarded as measuring reference, which allows comparison of different measuring methods, assessment of improvements of measuring instruments, and monitoring of correct functioning of a measuring instrument.

MICRO-EPSILON BELT MONITORING SYSTEM

The measuring system had to meet the following requirements:

- quick survey of the state of damage of the steel cords over the whole length of the belt,
- detection of large area corrosion damages as well as steel cord breaks with small gaps, to correctly determine loss in strength,
- appropriate design for underground use,
- flameproofness
- applicability for automatic data monitoring.

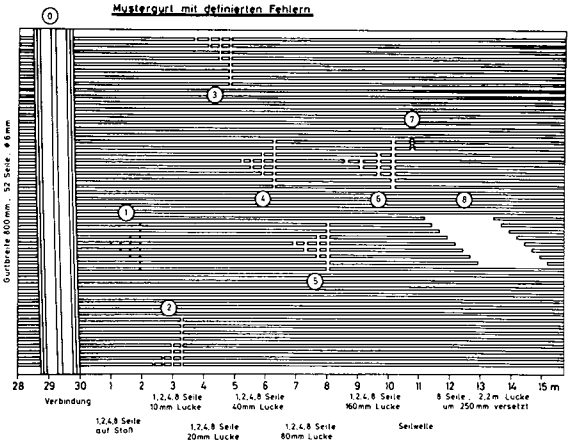


Figure 6a. Defects in the test belt (1st half)

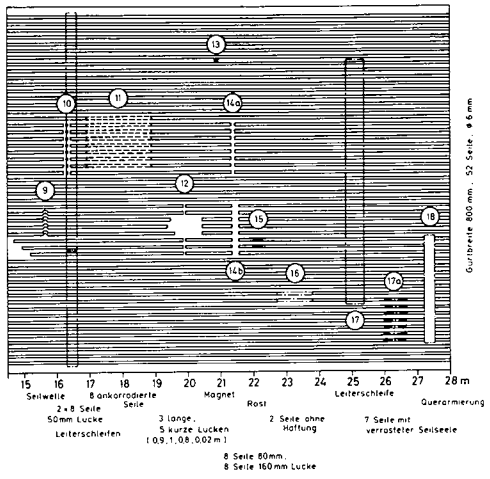


Figure 6b. Defects in the test belt (2nd half)

An instrument for measuring steel cord damages has been developed by

Tremonia Experimental Mine in conjunction with Micro-Epsilon-Meßtechnik, Ortenburg. The belt monitoring system consists of several transmitting coils in a beam positioned above the belt. The electromagnetic fields, radiated with high frequeny, are received by receiving coils in a second beam which is fixed under the belt. Figure 7 is a schematic representation. The measuring principle is based on the reduction of electric fields by electrically conductive materials. These are, in the case of steel cord conveyor belts, the steel cords imbedded in the rubber. The steel cords act as screen between the transmitting and receiving beams. Steel cord gaps, caused by corrosion or breaking of the steel cords, transmit more energy, which is displayed, proportional to the missing quantity of steel.

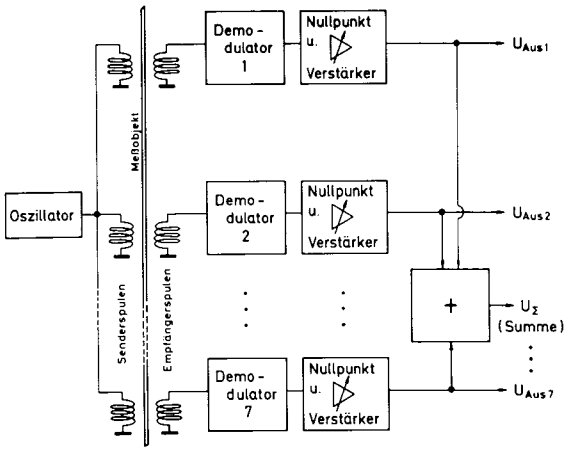


Figure 7. Block diagram of the Micro-Epsilon belt testing system

All electronic devices are incorporated into the transmitting and receiving beams. The actual version of the measuring apparatus (Fig. 8) is applicable on steel cord conveyor belts up to a width of 1600 mm and delivers measuring signals for 7 channels of 240 mm width each, and an intergrated signal for the whole belt width. To emphasize steel cord breaks with very small gaps, the output signals are directed parallel over the "break detectors", incorporated into the measuring beam. They are a

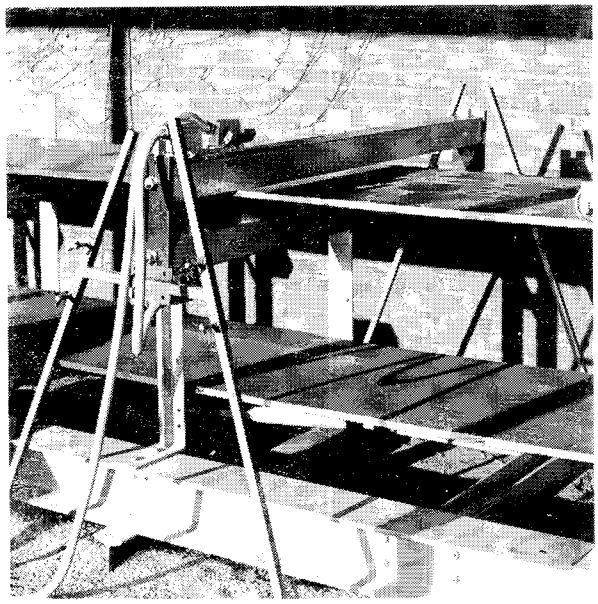


Figure 8. Micro-Epsilon belt testing system, type SN 1002

specific filter/amplifier combination that also produces analogue output signals. Therefore, 16 analoguous measuring signals are in a single common outgoing connector.

TEST MEASUREMENTS

The laboratory tests were carried out with pieces of conveyor belts and with test boards, on which steel cord pieces were put into grooves; this corresponds to the standard configuration of steel cord conveyor belts. With this arrangement the linearity of the belt monitoring system was examined, also the calibration, the influence of unprecisions in the installation and the requirements to the zero point and amplification settings.

A result of these tests was that all belt qualities can be measured with one amplifier setting. Before a measurement zero point adjustment is

checked in the laboratory and need not readjusted during the measurement. The linearity of the system is satisfactory (Fig. 9).

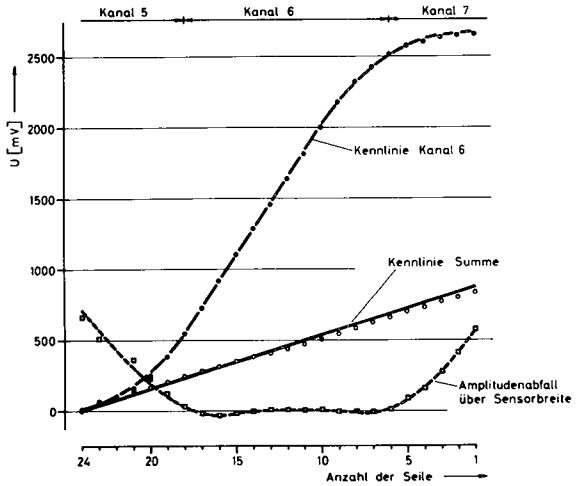


Figure 9. Calibration characteristics of the Micro-Epsilon belt testing system

The sensor coils are arranged in such a way that a linear characteristic line in the sum track is obtained by favourable overlapping of the channels in the boarder region. In the case of enlargement of the steel cord gaps in the belt (Fig. 10) it is noted that for the test belt the maximum amplitude is reached with approximately 100 mm steel cord gaps, in other samples with even smaller gaps. Below steel cord gaps of about 50 mm there is a clear decrease in amplitudes. In the case of steel cord gaps, with touching break ends, still one tenth of the maximum amplitude is reached. The width of impulse in the diagramm corresponds to a measuring beam width of 100 mm (figure 11), it increases according to the size of the steel cord gap. It is recognized in Figure 11 that the full amplitude is reached for steel cord gaps of about 100 mm.

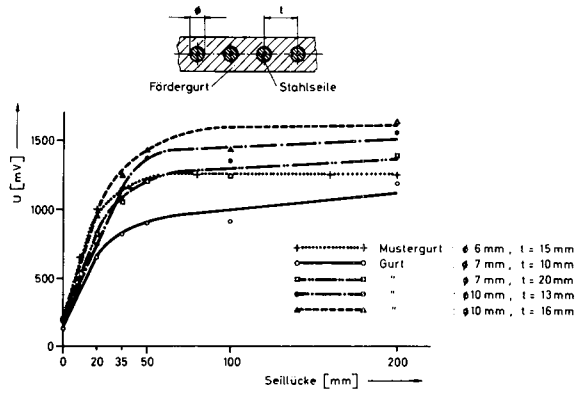


Figure 10. Calibration characteristics of enlarged gaps in 8 steel cords

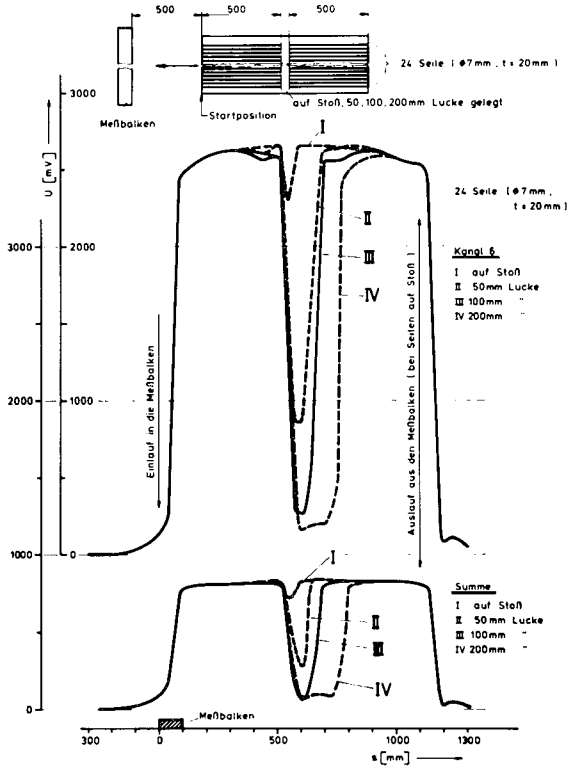


Figure 11. Passing of steel cord gaps through the belt testing installation

The results, given in Figure 12 of measurements carried out on the test belt of Tremonia Experimental Mine

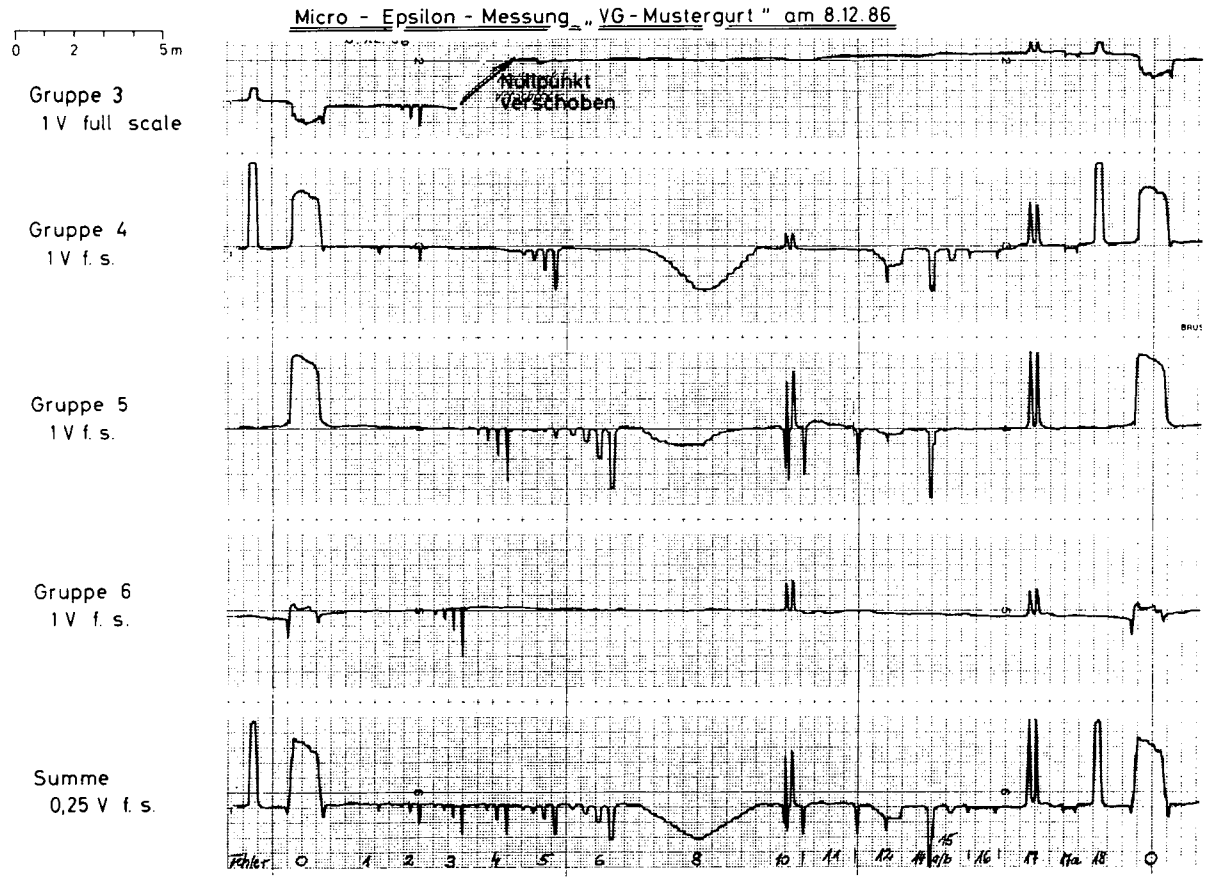


Figure 12. Measuring diagramm of the test belt, group and sum channels

show deflections proportional to the steel mass. Here the base line corresponds to the steel mass of the intact belt. Upward deflections represent a greater mass; as an example: in the belt splice (defect 0) the steel cords overlap. Downward deflections represent loss in steel mass, defect 8 is a very distinct example. All defects on the test belt are shown in the diagramm. The plotter setting was chosen in such a way that big defects are represented proportional to the amplitude, whereas small defects are hardly visible. This compromise in plotter setting means that in the normal diagramm smallest defects can hardly be recognized. By means of a specific filter and amplifyer combination that has been integrated as auxiliary electronic equipment parallel to the output signals, very small defects are shown clearly. In Figure 13 the output signals of the auxiliary electronics, named break detector are shown. In this figure even the smallest defect (no. 1, break of a steel cord without gap) is visible. In the case of bigger defects overamplifications are registered; such defects can better be seen in Figure 12.

In underground test measurements, carried out in the last two years, mainly little defects with small steel cord gaps were detected. Figure 14 shows a diagramm with details of an underground measurement. In the sum signal and in the individual channels several signal peaks are noted corresponding to ruptures in one or two steel cords. In defect 40 a 2.5 m long part of one steel cord is missing. In Figure 15 the signals of Figure 14 are presented as output signals of the break detector. In this figure defects are shown as distinct deflections.

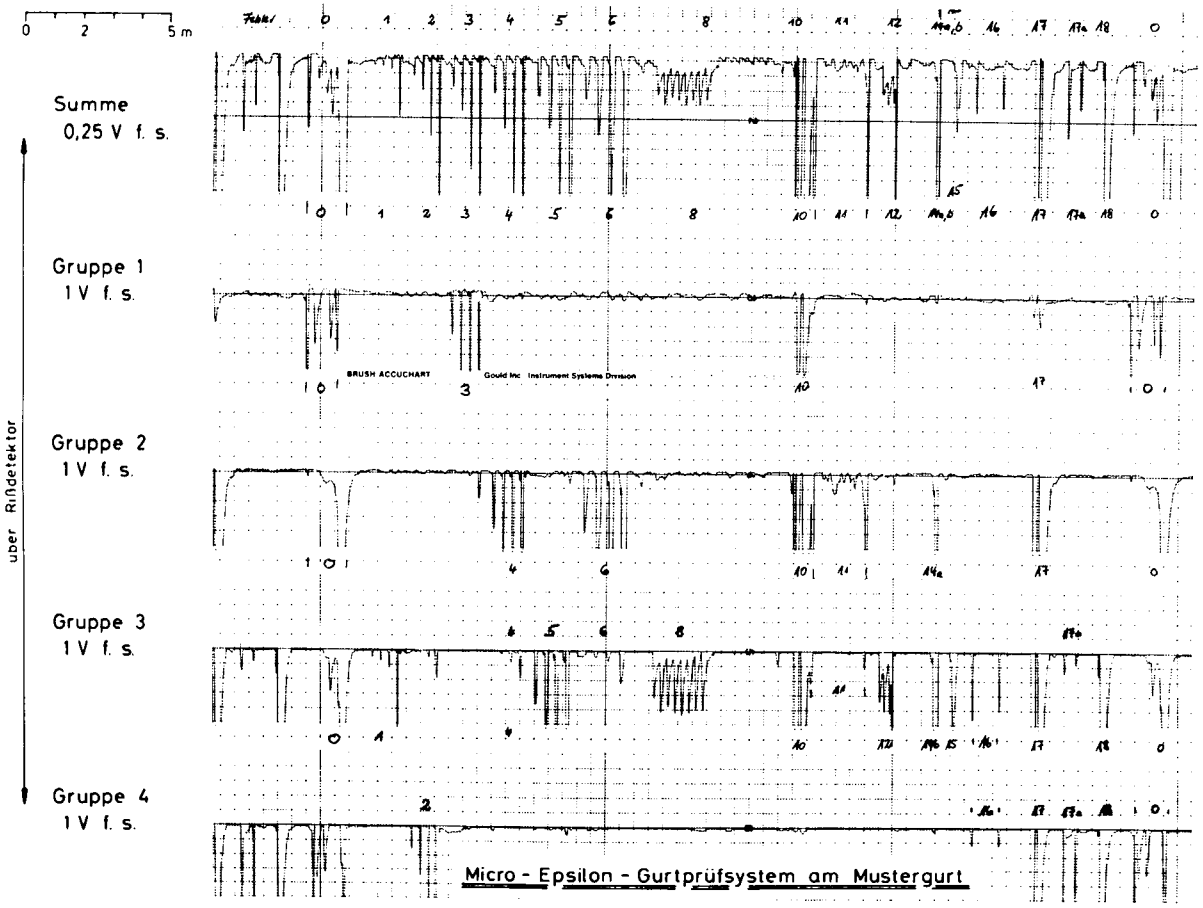


Figure 13. Measuring diagram of the test belt, output signals of the break detector

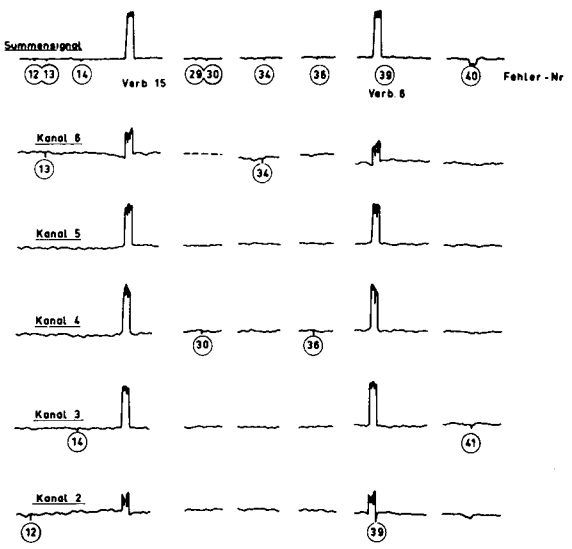


Figure 14. Measuring diagram of an underground measurement (details)

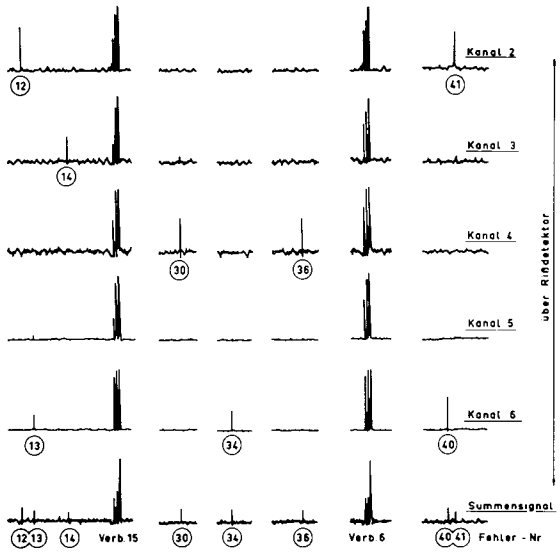


Figure 15. Measuring diagram of an underground measurement (details)

FAULT DETECTION AT AN UNDERGROUND CONVEYOR BELT

The measuring diagrams give a survey of damages of the steel cords of steel cord conveyor belts. The position of the defects in the several kilometers long belts is determined in longitudinal direction corresponding to the scale of the print-out to the actual length (e.g. 1:1000) and in the transversal direction corresponding to the position of the individual channels. Underground a defective part is measured with a measuring tape in proportion to the next belt splice. However, a defect is not exactly located because of unprecise reading of the diagram measuring with the measuring tape, and also in the transversal direction because of the track width of 260 mm. The location of the fault is only exactly determined, if the cover plate is damaged in the surveyed section. In the case the cover plate is not damaged, or if the rubber damage has nothing to do with the steel cord damage, the exact location of the defect is carried out by other means. Small breaks are exactly recorded by a portable instrument by Micro-Epsilon-Meßtechnik or by magnitizing the steel cords. At the break points of steel cords magnetic south and north poles are generated, the magnetic fields of which are measured by a residual field magnetometer.

For close examination of the damage the point of defect is either opened or an x-ray photograph is taken. With a portable x-ray apparatus (Figure 16), called "Inspector", Tremonia Experimental Mine has recorded several defects. Figures 17 und 18 show some of the defects. The precision of location of defects is shown in these figures. The damages were only partly determined from the outside by damages of the cover plate. They were measured so exactly that the x-ray photographs could be recorded with a normal 9x13-cm-instant-picture cassette.

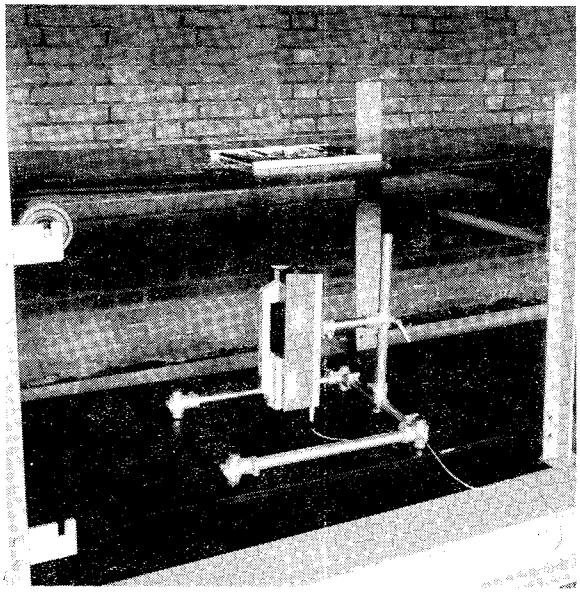


Figure 16. X-ray-Inspector at the test belt

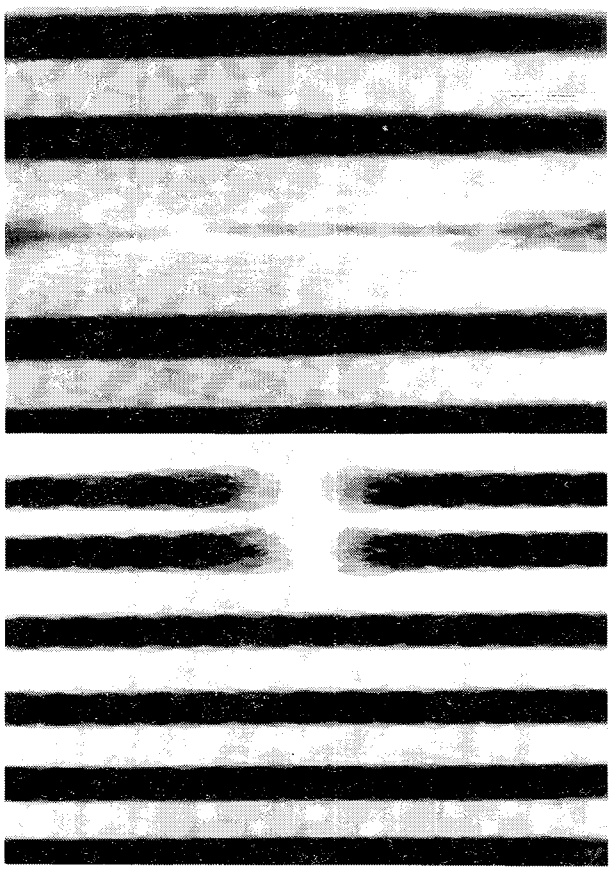


Figure 17. Defects at an underground conveyor belt

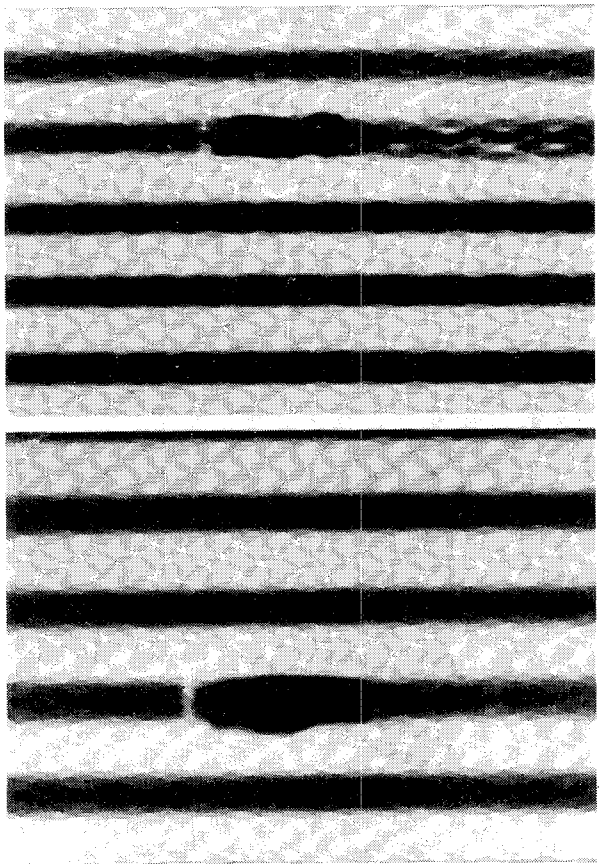


Figure 18. Defects at an underground conveyor belt

DISCUSSION OF THE RESULTS

The x-ray photographs show that the testing of steel cord conveyor belts on corrosion and breaks can be carried out with the Micro-Epsilon-belt-monitoring system leading to very precise results even for very small damages. Defect analysis with the measuring diagramm and precise location of damages on the often kilometers long conveyor belts underground have been tested. The mounting of the belt monitoring installation on two supports is easy.

The measuring system for use in underground coal mines is designed to be intrinsically safe. However, the certificate of conformity has not been issued so that underground measurements can only be carried out with an approval of exemption. After the certification it is planned to record the measuring data with an intrinsically safe measuring data storage device, and to interpret them at the surface. A further development deals with the possibility of continuously monitoring steel cord conveyor belts.

MESSGERÄTEENTWICKLUNG UND ERPROBUNG FÜR DIE PRÜFUNG VON STAHLSEILFÖRDERGURTEN AUF KORROSIONS UND BRUCHSCHÄDEN IN DEN SEILEN

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ZUSAMMENFASSUNG

Fördergurtanlagen sind Haupttransportadern, bei denen, vor allem wenn keine Parallelwege vorhanden sind, Störungen und Fehler möglichst frühzeitig erkannt werden müssen, so daß gezielte Wartungs-und Reparaturarbeiten eingeleitet werden können. Bei Stahlseilfördergurten können Bruch- und Korrosionsschäden in den Seilen auftreten, die in extremen Fällen zum Riß des Fördergurtes führen können. Da diese Förderanlagen auch zur Personenfahrung benutzt werden, muß die Überwachung so gestaltet werden, daß ein Gurtriß auf jeden Fall vermieden wird.

Im Rahmen eines Forschungsvorhabens des Landes Nordrhein-Westfalen entwickelte und erprobte die Versuchsgrubengesellschaft mbH zusammen mit der Firma Micro-Epsilon-Meßtechnik ein Gurtprüfsystem zur Erfassung derartiger Bruch-und Korrosionsschäden in den Seilen bis zur Betriebsreife. Der Beitrag beschreibt den Meßgeräteaufbau sowie Erprobungsmessungen am Mustergurt der Versuchsgrube Tremonia und an verschiedenen Förderbändern unter Tage. An Hand der Meßdiagramme wurden Fehler analysiert, am Förderband aufgesucht und in mehreren Fällen zur Dokumentation mit einem tragbaren Röntgengerät aufgenommen.

EINLEITUNG

Fördergurtanlagen großer Länge mit hohen Förderleistungen sind in vielen Hauptförderstrecken unter Tage eingesetzt. Wie bei der Schachtförderung sind keine Parallelwege vorhanden, über die das Fördergut bei Störungen der Förderanlagen umgeleitet werden könnte. Viele dieser Fördergurtanlagen werden unter Tage auch zur Fahrung durch die Bergleute benutzt. Die geforderte hohe Verfügbarkeit zur Förderung sowie die sicherheitlichen Anforderungen bei der Fahrung müssen durch geeignete Überwachungs- und Wartungsmaßnahmen gewährleistet werden. Eine ganze Reihe von Überwachungseinrichtungen und Kontrollmaßnahmen dient dazu, Störungen und Fehler beim Betrieb der Fördergurtanlagen zu vermeiden oder frühzeitig zu erkennen, um dann gezielt Wartungs- und Reparaturarbeiten einleiten zu können.

Viele der Hauptförderanlagen sind mit Stahlseilfördergurten ausgerüstet. Dieser Bericht befaßt sich mit der Erfassung von Korrosions- und Bruchschäden der Seile, die im Gummi einvulkanisiert sind und daher nicht direkt im Hinblick auf Schäden kontrolliert werden können. Mit Korrosionsschäden muß man nach längerer Betriebszeit rechnen, da bei Beschädigungen der Deckschicht Wasser in die Stahlseile eindringen kann und damit eine unkontrollierbare Korrosion beginnt. Die Abbildungen 1 und 2 zeigen Röntgenaufnahmen solcher Korrosionsschäden. Beschädigungen der Seile sind auch beim Transport der Gurtwickel, bei der Montage oder im Betrieb durch aufschlagendes Fördergut möglich. Die Stahlseile im Gurt können dabei beschädigt oder durchtrennt werden, ohne daß die Schäden äußerlich am Gurt erkennbar sein müssen. Die Abbildungen 3 und 4 zeigen Röntgenaufnahmen von Seil-

bruchfehlern, bei denen nur geringe Lücken im Seil sichtbar werden.

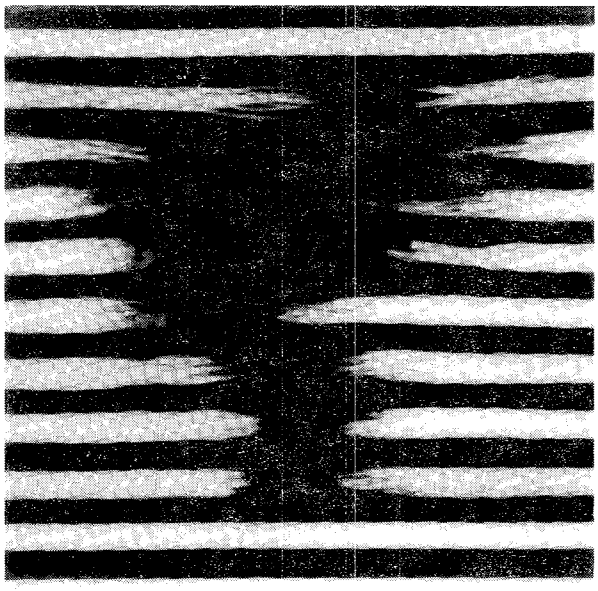


Abb. 1. Röntgenbild eines Korrosionsschadens

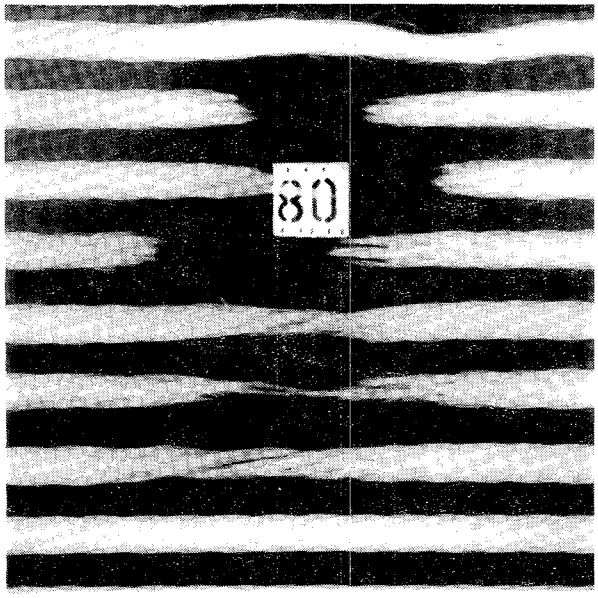


Abb. 2. Röntgenbild eines Korrosionsschadens

Korrosionsschäden und Brüche in den Seilen sind nur zum Teil an Beschädigungen der Gummideckschicht zu erkennen. Das genaue Ausmaß des Schadens ist auch bei äußeren Schäden der Deckschichten nur schlecht abzuschätzen. Die Korrosions- oder Bruchschäden führen zu Tragkrafteinbußen, die in extremen Fällen zu einem Riß des Fördergurtes führen können. Bei Per-

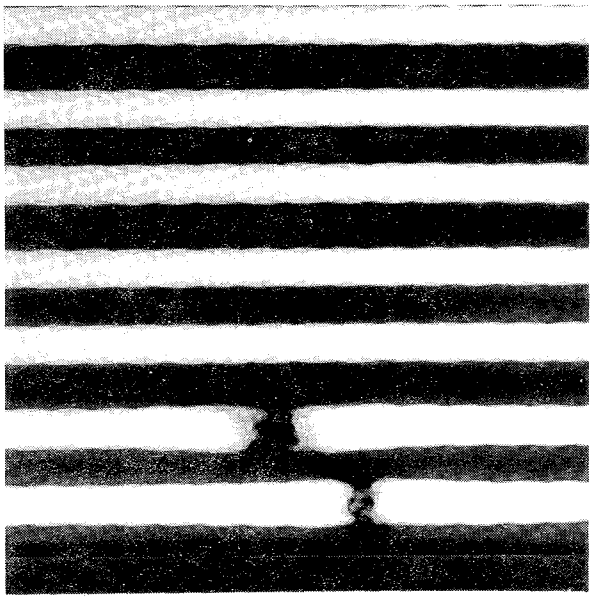


Abb. 3. Röntgenbild eines Seilbruches

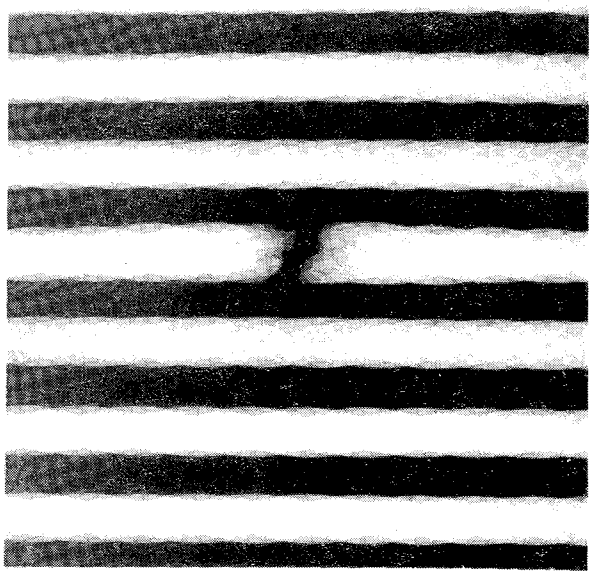


Abb. 4. Röntgenbild eines Seilbruches

sonenfahrung und auch zur Vermeidung von Betriebsstörungen und größeren Folgeschäden müssen gravierende Schäden, die ein sicherheitliches Risiko darstellen, vermieden werden. Die Überwachung der Fördergurtanlagen muß daher so gestaltet werden, daß ein Gurtriß auf jeden Fall ver-

mieden wird und durch Reparaturmaßnahmen sicherheitliche und auch wirtschaftliche Risiken so gering wie möglich gehalten werden.

Die Versuchsgrubengesellschaft mbH hat sich in den letzten Jahren mit Meßmöglichkeiten zur Kontrolle der Seilschäden beschäftigt und im Rahmen eines Untersuchungs- und Entwicklungsvorhabens des Landes Nordrhein-Westfalen ein Meßgerät bis zur Betriebsreife entwickelt. Das Meßgerät und damit durchgeführte Messungen am Mustergurt der Versuchsgrube Tremonia und an Fördergurtanlagen unter Tage werden im folgenden beschrieben.

MUSTERGURT DER VERSUCHSGRUBE TREMONIA

Verschiedene Meßverfahren und Geräte und die Auswirkungen von Meßgeräteänderungen konnten mit vertretbarem Aufwand nicht mit laufenden Untertagemessungen beurteilt werden. Zum einen waren Untertagemessungen zu zeitaufwendig und teuer, zum anderen lag eine Vielfalt von verschiedenen Fehlergrößen und -formen vor, die nur in relativ wenigen Fällen auch unter Tage geröntgt wurden. Daher wurde auf der Versuchsgrube Tremonia eine Kleinbandanlage (Abb. 5) mit 15 m

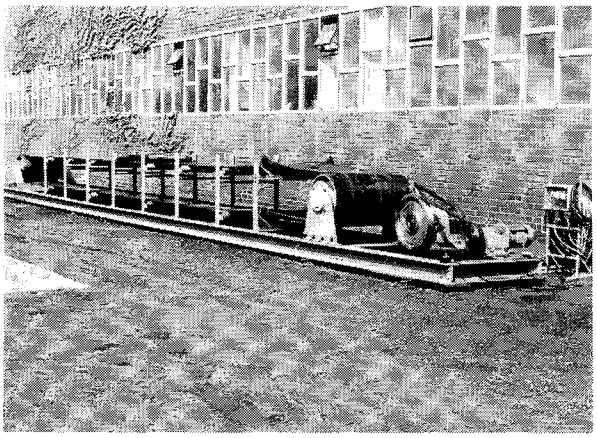


Abb. 5. Mustergurtanlage

Achsabstand gebaut, bei der die Stahlseile mit definierten Fehlern versehen wurden. Die Fehler (Abb. 6) reichen von einem durchtrennten Seil,

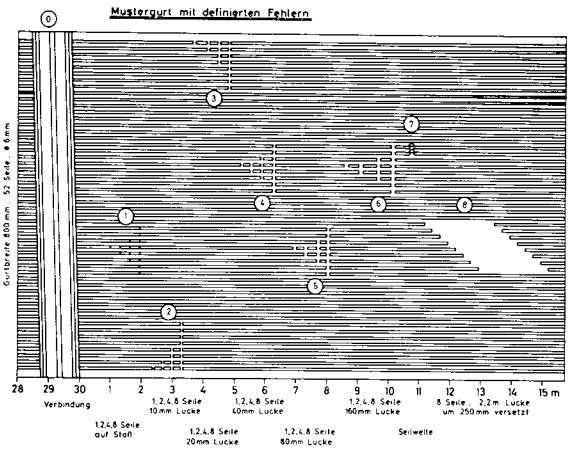


Abb. 6a. Fehlermuster des Mustergurtes (1. Hälfte)

bei dem die Seilenden voreinanderliegen (s. Fehler 1), bis zu großflächigen Fehlern (Fehler 8) und

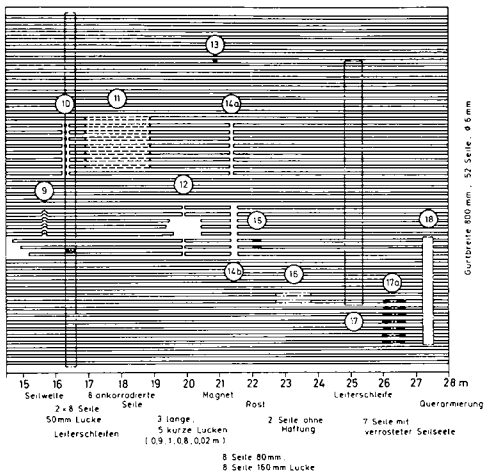


Abb. 6b. Fehlermuster des Mustergurtes (2. Hälfte)

verschiedenen Fehlerkombinationen. Dieser Mustergurt bildet ein Meßnormal, an dem verschiedene Meßverfahren verglichen, an dem die Auswirkungen von Meßgeräteverbesserungen beurteilt werden können und an dem die ordnungsgemäße Funktionsfähigkeit eines Meßgerätes überprüft werden kann.

MICRO-EPSILON-GURTPRÜFSYSTEM

An das Meßsystem wurden folgende An-

forderungen gestellt:

- Schneller Überblick über den Schadenszustand der Seile in Stahlseilfördergurten über die gesamte Gurtlänge
- Erkennen sowohl großflächiger Korrosionsschäden als auch von Seilbrüchen mit nur kleinen Seillücken, um die Festigkeitseinbuße des Gurtes sicher beurteilen zu können
- Geeignete Konstruktion für den Untertageeinsatz
- Schlagwettergeschützte Geräteausführung
- Eignung für eine automatische Dauerüberwachung.

Ein Gerät zur Messung der Seilschäden wurde von der Versuchsgrube Tremonia zusammen mit der Firma Micro-Epsilon-Meßtechnik, Ortenburg, entwickelt. Das Gurtprüfsystem besteht aus mehreren Sendespulen in einem Balken über dem Gurt. Die mit hoher Frequenz abgestrahlten elektromagnetischen Felder werden von Empfangsspulen in einem zweiten Balken unter dem Gurt empfangen. Abb. 7 zeigt den prinzipiellen Aufbau. Das Meßprinzip ba-

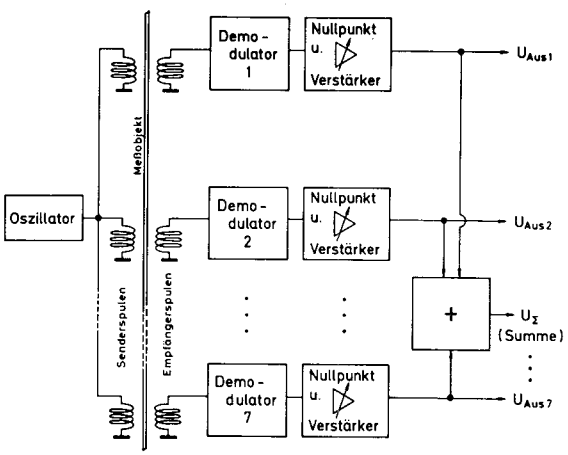


Abb. 7. Blockschaltbild des Micro-Epsilon-Gurtprüfsystems

siert auf der Abschwächung des elektrischen Feldes durch elektrisch leitende Materialien; das sind im Falle der Fördergurte die im Gummi eingebetteten Seile. Die Stahlseile wirken dabei als Abschirmung zwischen Sende- und Empfangsbalken. Seillücken, durch Korrosion oder Bruch der Seile verursacht, lassen mehr Sendeenergie durch, die proportional zur fehlenden Stahlmasse zur Anzeige gebracht wird.

Die gesamte Elektronik ist in den Sende- und Empfangsbalken (Abb. 8) integriert. In der vorliegenden Version ist das Meßgerät für Stahlseilfördergurte bis 1600 mm Breite geeignet und liefert Meßsignale für 7

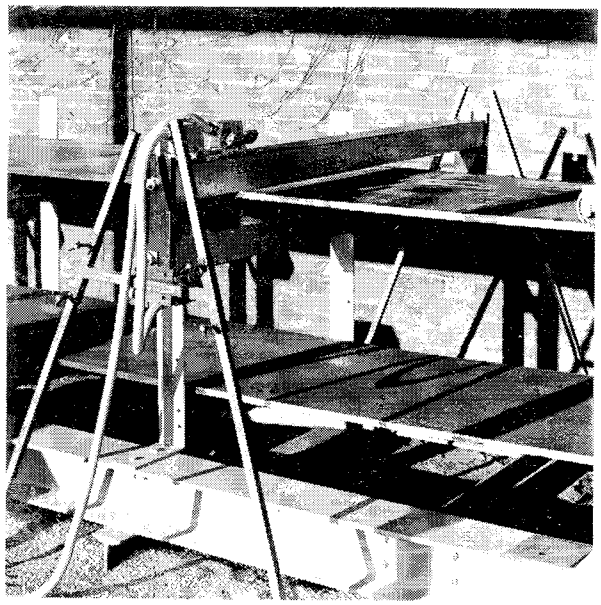


Abb. 8. Micro-Epsilon-Gurtprüfsystem Typ SN 1002

Einzelkanäle von je 240 mm Breite sowie ein Summensignal für die gesamte Gurtbreite. Um Seilbrüche mit sehr kleinen Lücken in den Seilen deutlicher hervorzuheben, werden die Ausgangssignale parallel über die im Meßbalken integrierten "Rißdetektoren" geführt, eine spezielle Filter-/Verstärkerkombination, die ebenfalls analoge Ausgangssignale liefert. In einem gemeinsamen Ausgangsstecker stehen damit 16 Analogmeßsignale zur Verfügung.

ERPROBUNGSMESSUNGEN

Die labormäßige Erprobung wurde mit Fördergurtstücken und mit Muster-

brettern durchgeführt, bei denen Seilstücke so in Rillen eingelegt wurden, daß sie dem normgerechten Aufbau von Stahlseilfördergurten entsprachen. Untersucht wurde damit die Linearität des Gurtprüfsystems, die Art der Kalibrierung, der Einfluß von Montageungenauigkeiten sowie Anforderungen an Nullpunkts- und Verstärkereinstellungen.

Als Ergebnis dieser Erprobungsmessungen zeigte sich, daß alle Gurtqualitäten mit einer Verstärkereinstellung gemessen werden können. Die Nullpunkte werden vor einer Messung im Labor kontrolliert und brauchen für die Messung nicht mehr nachgestellt zu werden. Die Linearität des Systems erwies sich als gut (Abb. 9).

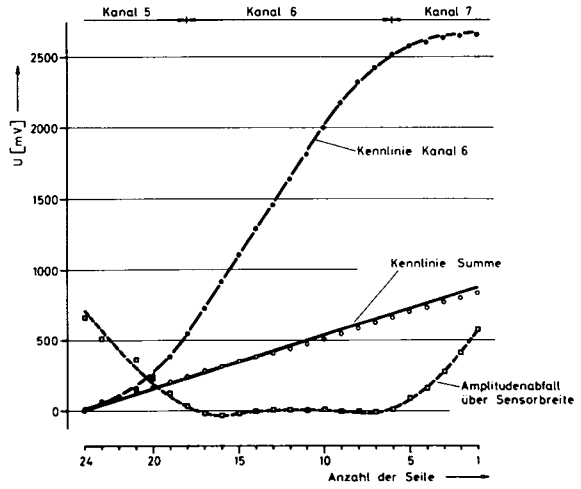


Abb. 9. Kalibrierkennlinien des Micro-Epsilon-Gurtprüfsystems

Die Sensorspulen sind so angeordnet, daß sich in der Summenspur durch günstige Überlappung im Randbereich der Einzelkanäle eine lineare Kennlinie ergibt. Bei der Bestimmung der Kennlinie durch Vergrößerung von Seillücken im Gurt (Abb. 10) zeigte sich, daß beim Mustergurt die volle Amplitude bei ca. 100 mm Seillücke erreicht wird, an anderen Probestücken zum Teil noch eher. Unter ca. 50 mm Seillücke beginnt ein starker Amplitudenabfall. Bei Seillücken, bei denen die Seile voreinanderstoßen, ist

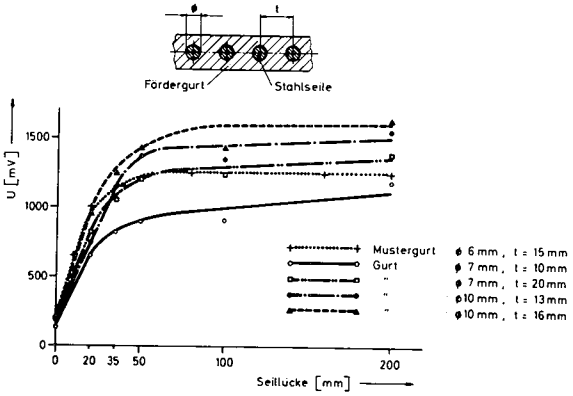


Abb. 10. Kalibrierkennlinien durch Vergrößern von Lücken in 8 Seilen

noch etwa ein Zehntel der vollen Amplitude vorhanden. Dabei entspricht die Impulsweite im Diagramm der Meßbalkenbreite von 100 mm (Abb. 11),

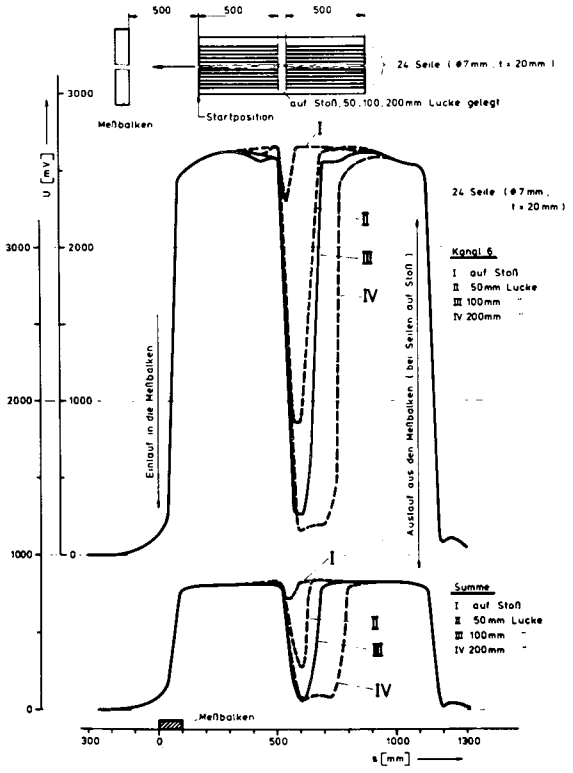


Abb. 11. Durchlaufen von Seillücken durch das Gurtprüfsystem

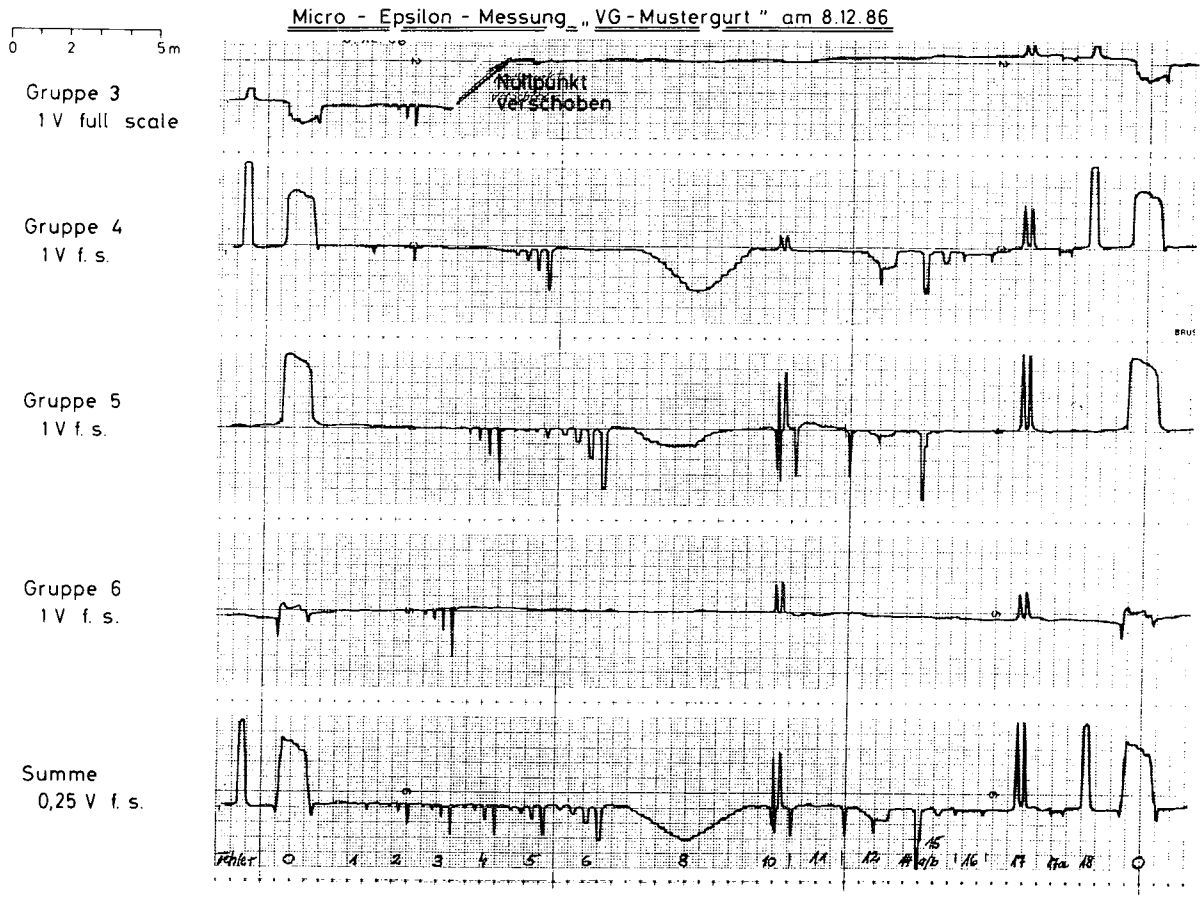


Abb. 12. Meßdiagramm des Mustergurtes, Gruppen- und Summenkanal

sie vergrößert sich entsprechend der Größe der Seillücke. Auch in Abb. 11 ist abzulesen, daß ab ca. 100 mm Seillücke die volle Amplitude im Diagramm erreicht wird.

Die in Abbildung 12 dargestellten Ergebnisse von Messungen am Mustergurt der Versuchsgrube Tremonia zeigen Ausschläge proportional zur Stahlmasse. Dabei entspricht die Nullinie der Stahlmasse des intakten Gurtes. Ausschläge nach oben stellen mehr Masse dar; zum Beispiel in der Gurtverbindung (Fehler 0), in der die Seile überlappend eingelegt sind. Ausschläge nach unten bedeuten Stahlmasseverlust, besonders deutlich beim Fehler 8 zu erkennen. Alle Fehler des Mustergurtes sind im Diagramm aufzufinden. Die Schreibereinstellung wurde so gewählt, daß die großen Schäden noch amplitudengetreu dargestellt werden, kleine Fehler aber auch noch zu sehen sind. Dieser Kompromiß bei der Schreibereinstellung führt dazu, daß kleinste Fehler im Normaldiagramm kaum noch hervortreten. Durch eine spezielle Filter- und Verstärkerkombination, die als Zusatzelektronik parallel zu den Ausgangssignalen integriert wurde, werden sehr kleine Fehler deutlich hervorgehoben. Abb. 13 zeigt die Ausgangssignale dieser als Rißdetektor bezeichneten Zusatzelektronik. In dieser Darstellung ist auch der kleinste Fehler (Nr. 1, Bruch eines Seiles ohne Seillücke) noch zu erkennen. Bei größeren Schäden ergeben sich Übersteuerungen; diese Fehler sind besser im Diagramm nach Abb. 12 zu beurteilen.

Bei den Erprobungsmessungen unter Tage in den letzten 2 Jahren wurden hauptsächlich kleine Schäden mit geringen Seillücken gemessen. Abb. 14

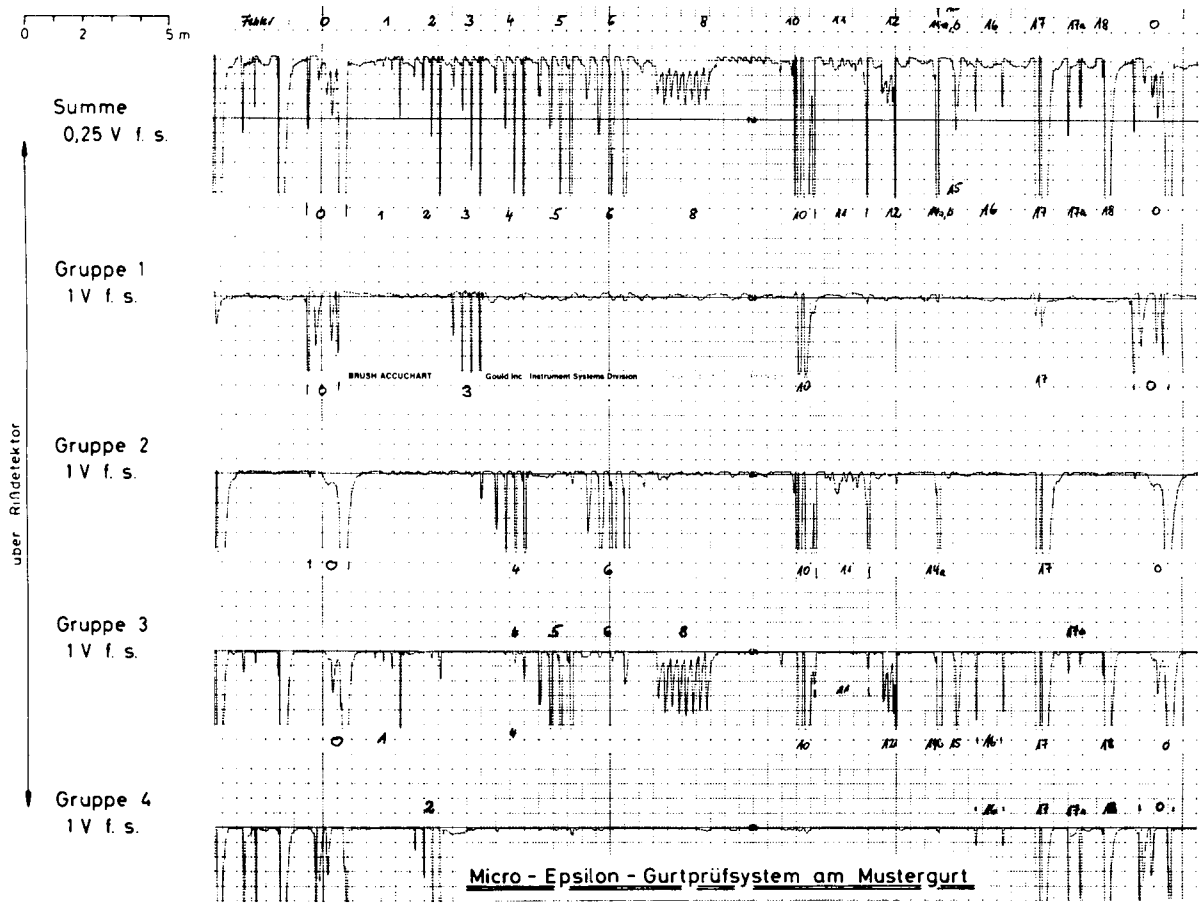


Abb. 13. Meßdiagramm des Mustergurtes, Ausgangssignale des Rißdetektors

zeigt ein Diagramm, das Ausschnitte aus einer Untertagemessung zeigt. Im Summensignal und verteilt in den Einzelkanälen sind mehrere Signalspitzen zu erkennen, die Brüchen in 1 bis 2 Seilen entsprechen. Fehler Nr. 40 stellt einen Schaden dar, bei dem ein Seil auf 2,5 m Länge fehlt.

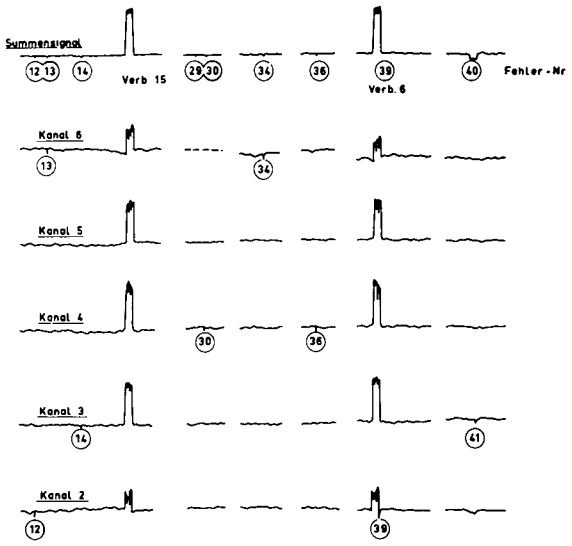


Abb. 14. Meßdiagramm einer Untertagemessung (Ausschnitte)

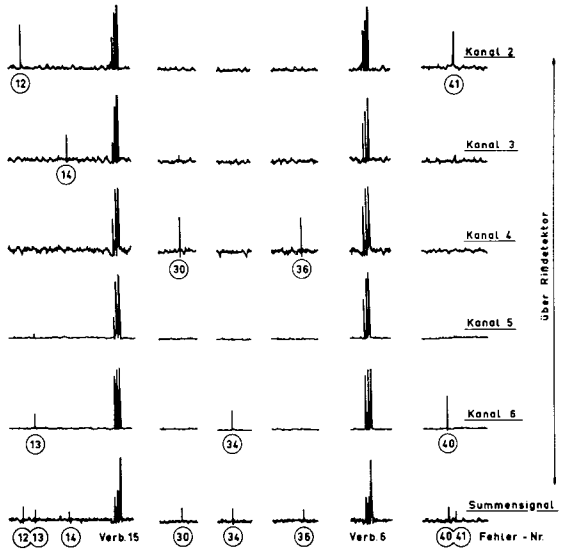


Abb. 15. Meßdiagramm einer Untertagemessung (Ausschnitte)

In Abb. 15 sind die Signale der Abb. 14 als Ausgangssignale der Rißdetektorschaltung dargestellt. Darin treten die Fehler als deutliche Ausschläge hervor.

FEHLERSUCHE AM FÖRDERGURT UNTER TAGE

Die Meßdiagramme liefern einen Überblick über Schäden in den Seilen der Stahlseilfördergurte. Die Lage der Fehler in den oft kilometerlangen Fördergurten kann in Längsrichtung aus dem Maßstab der Darstellung zur tatsächlichen Gurtlänge (zum Beispiel 1:1000) und quer zum Gurt durch die Aufteilung in Einzelkanäle bestimmt werden. Unter Tage wird eine Schadensstelle ausgehend von der nächsten Gurtverbindung mit einem Maßband eingemessen. Durch Ungenauigkeiten bei der Ablesung im Diagramm und beim Einmessen des Schadens mit dem Maßband bedingt, kann die Lage eines Fehlers nur ungefähr eingemessen werden und auch in Querrichtung wegen der Meßspurbreite von 260 mm nur ungenau bestimmt werden. Die genaue Fehlerlage ist dann erkennbar, wenn die Deckplatte im eingemessenen Bereich beschädigt ist. Ist aber kein Deckplattenschaden vorhanden oder ist ein Deckplattenschaden vorhanden, der aber nicht mit dem Schaden im Seil zusammenhängt, dann erfolgt das genaue Einmessen des Schadens mit anderen Hilfsmitteln. Kleine Brüche konnten mit einem Handgerät der Firma Micro-Epsilon-Meßtechnik zentimetergenau eingemessen werden. Eine weitere Möglichkeit zur genauen Schadenslokalisierung besteht in einer Magnetisierung der Stahlseile. An Bruchstellen in den Seilen bilden sich magnetische Nord- und Südpole aus, deren Magnetfelder mit einem Restfeldmagnetometer gemessen werden können.

Zur genauen Schadensbetrachtung kann entweder die Fehlerstelle geöffnet werden oder durch Röntgen ein Bild des Schadens aufgenommen werden. Mit einem tragbaren Röntgengerät (Abb. 16), das "Inspector" genannt wird, hat die Versuchsgrube Tremonia mehrere Fehlerstellen dokumentiert. Die Abb. 17 und 18 zeigen eine Auswahl von Seilschäden. Die Genauigkeit der Lokalisierung der Schäden wird durch diese Bilder dokumentiert. Die Schäden waren nur zum Teil auch äußer-

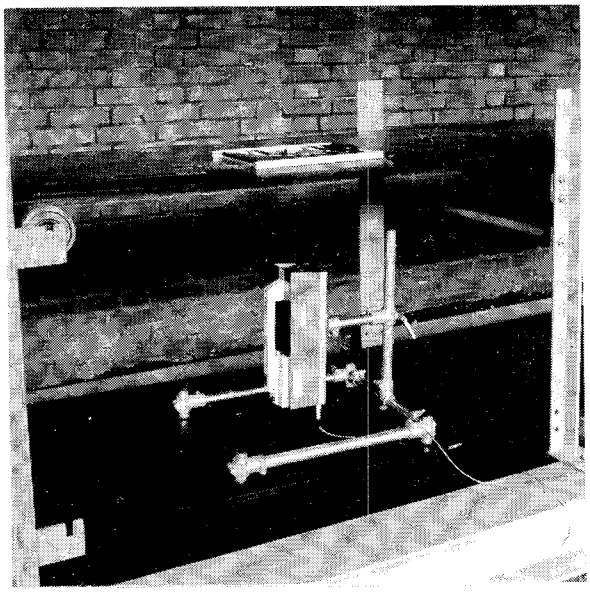


Abb. 16. Röntgen-Inspector am Mustergurt

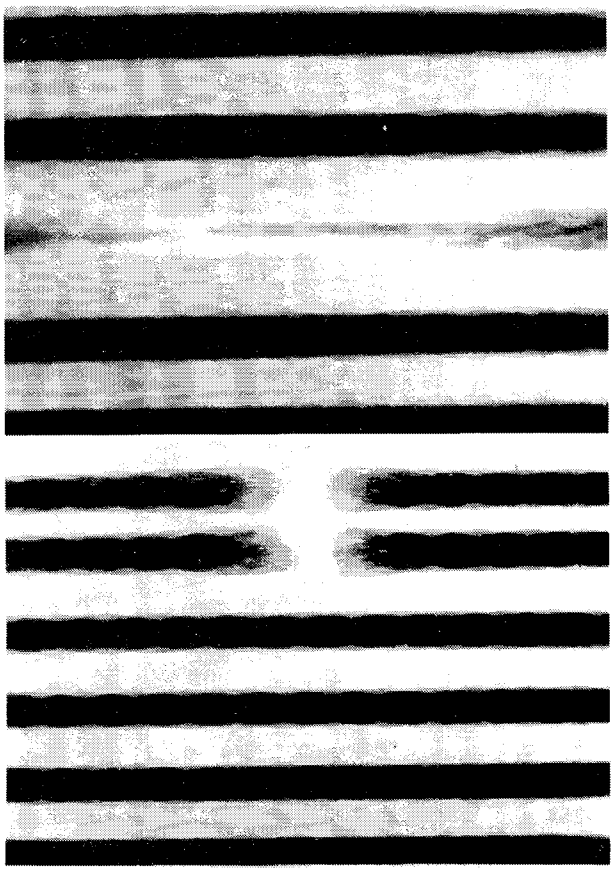


Abb. 17. Fehler an einem Fördergurt unter Tage

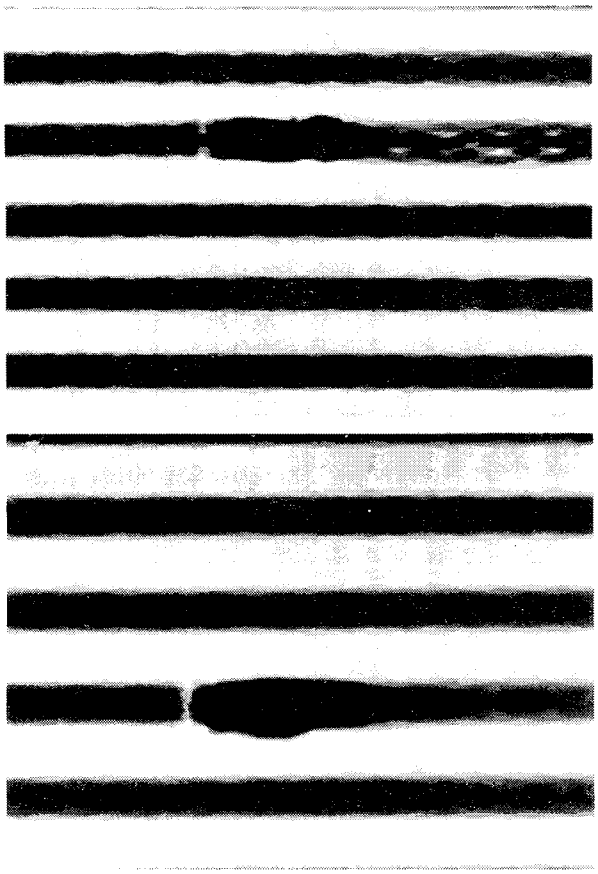


Abb. 18. Fehler an einem Fördergurt unter Tage

lich durch Deckplattenschäden auffindbar. Sie wurden so genau eingemessen, daß die Röntgenbilder mit einer normalen 9x13-cm-Sofortbildkassette aufgenommen werden konnten.

DISKUSSION DER ERGEBNISSE

Mit den Röntgenbildern wird belegt, daß die Kontrolle von Stahlseilfördergurten auf Korrosion und Brüche in den Seilen mit dem Micro-Epsilon-Gurtprüfsystem zu genauen Ergebnissen auch bei sehr kleinen Schäden führt. Neben der reinen Fehleranalyse an Hand des Meßdiagramms ist auch die Lokalisierung der Schäden an den meist mehrere Kilometer langen Fördergurten unter Tage erprobt worden. Der Aufbau des Gurtprüfsystems auf zwei Ständern ist einfach durchzuführen.

Das Meßsystem ist für den Einsatz im Steinkohlenbergbau unter Tage eigensicher konzipiert. Die Konformitätsbescheinigung liegt aber noch nicht vor, so daß zur Zeit nur Messungen mit Ausnahmegenehmigung möglich sind. Die Meßdaten sollen nach der Zulassung in einem ebenfalls eigensicheren Meßdatenspeicher aufgenommen und über Tage analysiert werden. Die weitere Entwicklung befaßt sich mit Möglichkeiten der Dauerüberwachung der Stahlseilfördergurte.

DEVELOPPEMENT ET ESSAIS D'APPAREILS DE MESURE POUR LA DETECTION DE CORROSION ET DE FRACTURES SUR LES COURROIES DE CONVOYEURS EN ACIER

Langebracke, F.

Les pannes et les défauts des systèmes de convoyeurs à courroies, qui sont d'importants équipements de transport, doivent être détectés dès que possible pour assurer une maintenance et des réparations adéquates, surtout s'il n'existe pas d'équipements auxiliaires. Les courroies d'acier des convoyeurs peuvent se fracturer et se rouiller, ce qui, dans des cas extrêmes, peut provoquer leur rupture. Etant donné que ces équipements sont également utilisés pour le transport des personnes, il est nécessaire de les vérifier de manière à empêcher à tout prix la rupture de leurs courroies.

Dans le cadre d'un projet de recherche de Northrhine-Westphalia, "Versuchsgrubengesellschaft" a développé et testé, en collaboration avec "Micro Epsilon Messtechnik", un appareil de vérification permettant de déceler les fractures et la corrosion des courroies. Cet appareil est maintenant opérationnel. Notre rapport décrit le développement des appareils de mesure, ainsi que les essais effectués sur une courroie-échantillon de "Versuchsgrube" et sur différentes courroies de convoyeurs souterrains. Des diagrammes de mesures ont permis d'analyser et de repérer les défauts des courroies; dans certains cas, ces défauts ont été enregistrés par un appareil portatif à rayons-X, aux fins de documentation.

РАЗРАБОТКА И ИСПЫТАНИЕ ИЗМЕРИТЕЛЬНЫХ УСТРОЙСТВ ДЛЯ ПРОВЕРКИ ТРАНСПОРТЕРНЫХ ЛЕНТ СО СТАЛЬНЫМ ТРОСОМ В СВЯЗИ С ПРОБЛЕМОЙ КОРРОЗИИ И ТРЕЩИН В ТРОСАХ

Ф. Лангенбраке

Конвейерные установки являются основным средством транспортировки, поэтому возможные поломки и дефекты должны быть определены как можно раньше с целью организации соответствующего обслуживания и ремонта, особенно в тех случаях, когда не имеется запасных конвейеров. Трещины и коррозия, которые могут появиться в стальных тросах транспортерных лент, могут в экстремальных случаях привести к их разрыву. В тех случаях, когда конвейерные транспортеры используются для перемещения людей, они должны проверяться таким образом, чтобы полностью исключить возможность разрыва ленты.

В соответствии с исследовательским проектом, Северо-Рейнской компанией экспериментальных шахт совместно с компанией "Микро Эпсилон Местекник" была разработана и испытана система для проверки транспортерных лент с целью определения мест трещин и коррозии в тросах и их пригодности к дальнейшей эксплуатации. В докладе описывается расположение измерительных устройств, а также измерения в процессе испытаний ленты из экспериментальной шахты и других транспортерных лент из различных подземных шахт. Используя измерительные диаграммы. были проанализированы дефекты, обнаруженные на конвейерных лентах, и в некоторых случаях с целью документирования они регистрировались с помощью портативного рентгеновского аппарата.

输送机钢索皮带钢索锈蚀和断裂检测仪器的发展和试验

F．郎盖布拉开，煤矿研究所 mbH，多特蒙德

皮带输送机是主要运输工具，必须及早觉察它停顿或损坏的可能性，以便维护及修理，尤其是在没有双线生产的情况下，这个问题就更加严重．输送机皮带里的钢索如果有断裂或锈蚀现象，在极端的情形下可以引起输送机皮带的破裂．

诺尔士兰——卫士发利亚煤矿研究所与"迈可罗依不锡隆测试公司"合作发展了一套断裂和锈蚀的测定装置，试验后可以正式应用．本文叙述这套测定装置的布设和测定的手续，用了一条 "煤矿研究" 的试验皮带和几条地下矿用的皮带．瑕疵是根据测量图沿着皮带加以分析，有些情形下用一个袖珍式X－光设备记录下来．

THE BRAKING OF MINING LOCOMOTIVES

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ABSTRACT

The paper deals with the inherent difficulties of operating steel-tyred adhesion locomotives with unbraked rolling stock on the steep inclines encountered in U.K. mines. The retardation produced when a train (locomotive and rolling stock) is braked depends primarily on the brake effort developed by the train, its mass, and the inclination of the track. With the exception of locomotives fitted with track-reactive emergency brakes, adhesion locomotives transmit their service and emergency brake efforts to the track by the combined effect of their weight and the coefficient of friction between their wheels and the rails. The paper outlines the factors affecting coefficient of adhesion and its relationship to locomotive brake effort. The current specification and assessment of locomotive braking performance is reviewed and recommendations made for new criteria.

INTRODUCTION

This paper is confined to the braking of steel-tyred adhesion locomotives which are the most common type used in U.K. coal mines. U.K. Regulations (Coal and Other Mines [Locomotives] Regulations 1956) permit them to operate with unbraked rolling stock on inclines up to and including 6.7% (1:15). By way of comparison the steepest incline on the U.K. mainline surface railway is 2.7% (1:37), and in most cases braking is provided throughout the whole of a train. Rubber-tyred, induced-adhesion captive-rail, and rack-driven locomotives are used on steep inclines, but do not rely on the frictional force that can be developed between steel tyres and steel rails.

The force that can be developed by a locomotive or vehicle to retard its motion is termed the brake effort, and this is commonly expressed as a percentage or fraction of its weight and called the brake ratio. The brake effort should be optimised so that maximum retardation can be achieved without loss of adhesion, i.e. without skidding. This optimisation becomes more critical as the inclination of the track increases and locomotives operate with unbraked rolling stock, because the brake effort is then limited by the frictional force which can be maintained between a locomotive's wheels and the rails.

An 'overbraked' locomotive is potentially just as dangerous as an 'underbraked' one as excessive braking can result in the locomotive's wheels ceasing to rotate and the locomotive sliding along the rails. The brake effort will then reduce substantially to the dynamic frictional force between the locked wheels and the rails, rather than being a function of the static frictional force. Unless control can be regained the skidding train may accelerate down a steep incline.

The paper discusses the adhesion of mining locomotives and proposes design criteria for assessing whether the brake effort is likely to cause loss of adhesion, and for matching the achievable brake effort with the locomotive's duty.

BRAKING SYSTEMS

Locomotives used in coal mines must be of a type approved by the Health and Safety Executive (HSE) and are presently tested to the requirements published in Testing Memorandum 12 (Health and Safety Executive 1977). This requires a locomotive to be provided with three braking systems, i.e. service, emergency and parking. The systems may use common components, but the failure of any one component must not prevent the emergency brakes from stopping the locomotive safely. At least one of the braking systems must be applied by either direct mechanical action or springs, the latter normally being 'held-off' by fluid pressure.

The primary system is a variable control service brake and customarily uses compressed air to bring brake blocks into contact with the treads of the wheels. Some recently introduced designs use compressed air to operate water-cooled, liquid-immersed, multiplate brake units installed either on the transmission or wheels.

The emergency brake is provided as a precaution against failure of the service brake and utilises either a secondary air supply or springs to apply the wheel brakes. It can be applied by the driver and, in some cases, automatically in the event of overspeed or malfunction which is likely to affect a locomotive's safe operation. It is neither controllable nor, in most cases, can it be released quickly. It is therefore imperative that the emergency brake effort should not be sufficient to cause loss of adhesion.

Some recently introduced locomotives use track-reactive brakes, which also serve as parking brakes, instead of emergency wheel brakes. Track-reactive brakes are applied by the release of fluid pressure, the brake effort being developed by transferring a substantial proportion of the weight of a locomotive from its wheels to brake pads which contact the top of the rails.

LOCOMOTIVE ADHESION

Coefficient of adhesion is a measure of the ability of a locomotive to utilize the static coefficient of friction between its wheels and the rails. In this paper the coefficient of adhesion is defined as the locomotive brake ratio just before one or all of its wheels cease to rotate. Theoretically, the maximum coefficient of adhesion is the coefficient of static friction between a locomotive's wheels and the rails. In practice this cannot be achieved because a small amount of relative motion ('creepage') is always present at the wheels of a locomotive during traction or braking. If 'creepage' increases to 'slippage' the wheels rotate at a peripheral speed which is significantly less than the linear speed of the locomotive. 'Slippage' can occur without the brakes being applied when the braking capability of a locomotive's power unit exceeds the limiting adhesion between the wheels and the rails. 'Slippage' will occur if the taper on the treads of fixed wheelsets is insufficient to compensate for the unequal distances travelled by the inner and outer wheels on curved track. Application of the brakes whilst 'slippage' is occurring will cause the wheels to lock and adhesion to be lost completely.

The highest coefficient of adhesion is achieved when the relative motion between each wheel and rail is restricted to 'creepage'. In practice this does not occur, principally because the full weight of a moving locomotive is not supported at all times by equal reac-

tions between its braked wheels and the rails. Theoretically it is best to have all the wheels coupled together but this may be precluded by other design requirements.

The principal parameters affecting the coefficient of adhesion of a locomotive are:

a) the coefficient of friction present between the wheels and the rails;
b) the coupling arrangement of the wheels;
c) the force in the drawbar coupler;
d) the position of the centre of gravity of the locomotive;
e) the height of the drawbar coupler;
f) the resilience and damping between the couplers of the train,
g) the curvature of the track;
h) variations in the cross-levelling of the track;
i) the suspension characteristics of the locomotive;
j) the speed of the locomotive.
k) acceleration and retardation of the locomotive.

Variations in these parameters make it possible for a locomotive with uncoupled wheels to develop better coefficients of adhesion than one with coupled wheels.

BRAKING REQUIREMENTS

Testing Memorandum 12 (TM12) requires a steel-tyred adhesion locomotive to develop a service brake effort of 16% of the maximum gross weight of the locomotive, i.e. a brake ratio of 16%. It also requires the emergency brake effort to be not less than half that required from the service brake. The parking brake must be capable of holding the locomotive stationary under the most onerous operating condition.

The required brake ratios imply that a steel-tyred locomotive can achieve a coefficient of adhesion of 0.16, but that in some operating conditions it may fall to 0.08. The lowest coefficient of adhesion that is likely to be achieved in practice is a principal consideration in deciding the emergency brake effort.

The minimum brake ratios required by TM12 were specified with the intention of providing sufficient brake effort to control trains on the permitted inclines without introducing an unacceptable risk of skidding. However, the service brake ratio of 16% required for mining locomotives demands coefficients of adhesion that are considerably higher than those expected by London Underground (0.11) and by European surface railways (0.052 to 0.096). Locomotive drivers have to exercise considerable skill to control unbraked rolling stock on the steep inclines encountered in some coal mines.

The brake ratios specified in TM12 were based upon recommendations made by Crook (Crook 1947), but the introduction of new designs of mining locomotive mean that these have to be interpreted broadly, rather than applied strictly, because of the changes in the technology of locomotives and their braking systems. HSE is to publish a Topic Report on underground locomotive haulage which will include guidance on system design. The Research and Laboratory Services Division (RLSD) of HSE have measured the coefficients of adhesion of a range of mining locomotives to assist HSE's Inspectorate of Mines and Quarries in providing technical data for this report.

ADHESION MEASUREMENTS

Most of RLSD's tests to measure coefficients of adhesion were conducted on surface test tracks. The locomotives were braked on both dry and wet rails, and each test was repeated at least five times. A tachometer wheel apparatus, which ran on one of the rails, was attached to the locomotive and produced electrical pulse signals. Each pulse was a measure of a distance travelled by the train and the time between individual pulses enabled the instantaneous velocity of the train to be determined. Rotation of the braked

wheels of the locomotive was monitored by one or more optical transducers. The signals obtained from the tachometer and wheel transducer(s) during each test of the service brake were recorded on multi-channel magnetic tape for subsequent computer processing. Velocity-time graphs were produced from which loss of adhesion was identified by a sudden decrease in the slope of a graph and a reduction in the pulse frequency from the wheel transducers.

Twelve locomotives were tested and approximately 150 measurements were made. The coefficients of adhesion obtained are summarized in Fig 1. The values should be used for guidance and not as design data, but it is the authors' opinion that some general conclusions may be drawn from the limited tests.

1. Coefficients of adhesion are variable, even for locomotives having the same wheel coupling arrangement.
2. Coefficients of adhesion developed on wet rails are usually less than those developed on dry rails.
3. The brake ratios specified in Testing Memorandum 12 are appropriate for fixed-frame locomotives with braked coupled wheels.
4. The initial investigation of fixed-frame locomotives with braked uncoupled wheels supports the opinion held by many engineers in the mining industry that this type of locomotive develops lower coefficients of adhesion than those with braked coupled wheels.

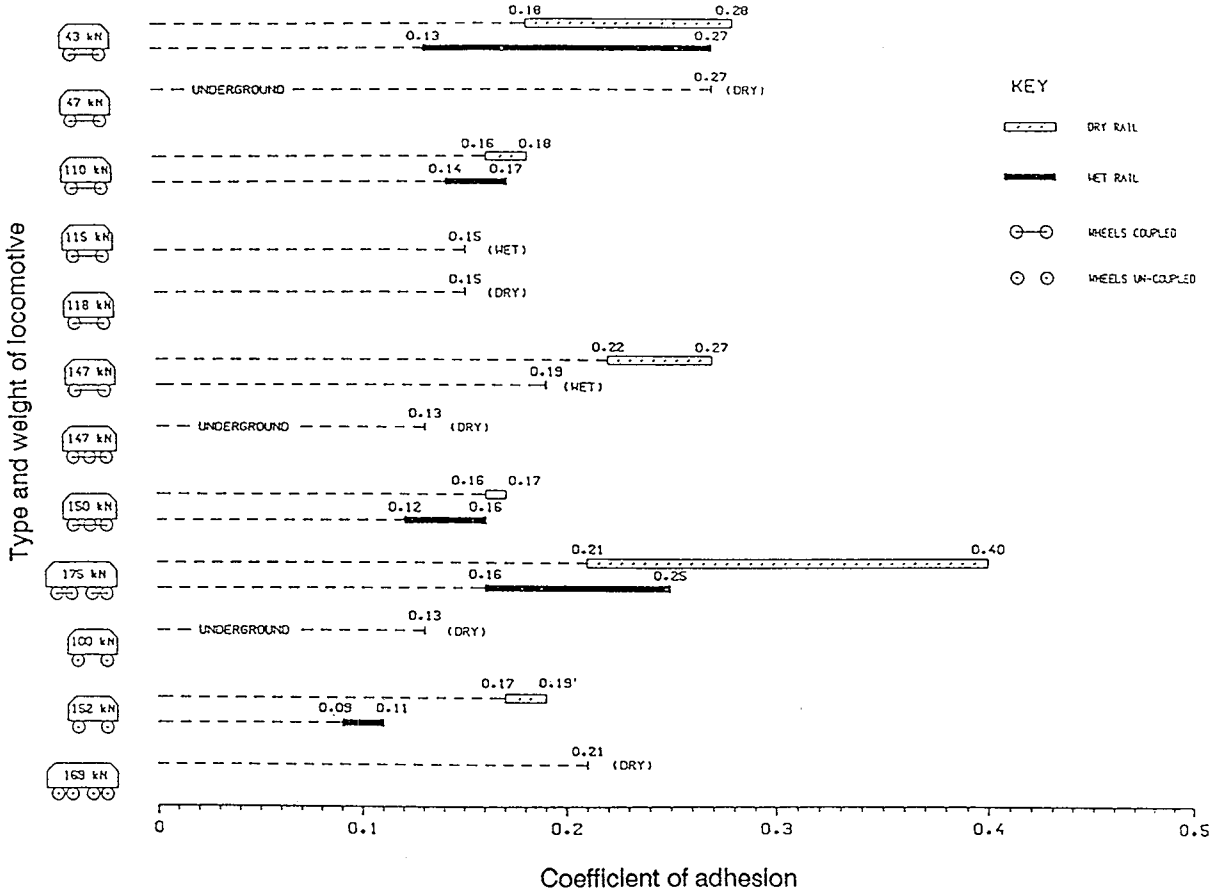


Fig. 1 - Coefficients of adhesion of mining locomotives

5. Bogie-frame locomotives with braked uncoupled wheels can develop higher coefficients of adhesion than fixed-frame locomotives with braked uncoupled wheels.
6. Bogie-frame locomotives with braked wheels which are coupled within the bogies can develop coefficients of adhesion which are similar to or better than than those obtained from fixed-frame locomotives with braked coupled wheels.

BRAKE TESTING

The customary method of testing the braking performance of a mining locomotive has been to measure the distance taken for the locomotive to brake a train of known mass to a standstill from a known speed on a particular length of track. A deterioration in the braking performance is indicated when the stopping distance exceeeds a maximum specified distance. Misleading values of retardation may be obtained due to inaccuracies in the measurement of speed and distance and uncertainty about the time taken for the brakes to be fully applied. This latter parameter is especially critical when tests are undertaken with insufficient unbraked load at low speeds.
Intrinsically safe equipment is now available which enables retardation, and therefore brake effort, of a train to be measured more directly. One method uses a portable SIMRET retardometer, an instrument containing an accelerometer and microprocessor, to provide a direct reading of the mean value of the absolute retardation. Another method uses instrumentation installed either on the locomotive or on a test vehicle to obtain a velocity-time graph, the slope of the graph being the retardation relative to the track.
Measurement of the time taken for full brake effort to be developed requires additional instrumentation which is not normally used during braking tests.
Therefore, irrespective of whether a retardation is measured directly as a mean value, or obtained from a velocity-time graph, some variability is likely to be introduced because of the time between brake application and the development of full brake effort. Despite this difficulty the authors consider that the brake efforts of mining locomotives should be calculated from measurements of retardation rather than stopping distance.

BRAKING CALCULATIONS

Two 'standard' formulae are used within the mining industry to calculate the mass of rolling stock which may be braked by a locomotive. Both formulae include numerical 'constants' to make the conversions required by the units used, and to make empirical allowances for the effect of rotating parts and the time taken for brake effort to be developed.
British Coal Corporation's Codes and Rules CR/3 (National Coal Board 1986) provides a formula for calculating the maximum allowable stopping distance in locomotive brake tests. The variables used are speed, mass of train, mass of locomotive, resistance of rolling stock, and inclination of the track. Implicit allowances appear to be made for the effect of rotating parts and for a delay of approximately 2 sec to allow for a driver's reaction time and time to develop the brake effort. Brake ratio is included as a constant, which is changed to suit the type of locomotive; i.e. a brake ratio of 16% for steel-tyred coupled wheels, and a brake ratio of 13.5% for steel-tyred uncoupled wheels.
It has been common practice to transpose this formula to calculate the mass of unbraked rolling stock that may be attached to a locomotive. It is the authors' opinion that the use of a formula based on a specified stopping distance (usually 60 m), low speeds, and as-

sumed brake effort and brake delay time, can lead to trains having very low retardations.

Crook provided a formula in his paper for calculating the unbraked rolling stock that may be attached to a locomotive. This formula expresses the unbraked load in terms of retardation and brake ratio, rather than stopping distance and speed, and allowance is made for the effect of rotating parts. Crook recommended a minimum retardation of 0.2 mph/sec (0.91%g) and a brake ratio of 16%. Braking distance, rather then stopping distance, is subsequently calculated.

RLSD has recently developed the formula shown in Equation 1 for calculating the unbraked rolling stock that may be attached to a locomotive.

$$Xb = \frac{Kl - Rt}{\frac{f}{c \times g} - Rt + Gs} \quad \text{-1 - - - - (1)}$$

where,

c is the fraction of the total brake effort producing linear retardation (assumed to be 0.94 by Crook),

f is the retardation of the train,

g is the gravitational constant ,

Gs is the sine of the angle of the incline,

Kl is the brake ratio of the locomotive,

expressed as a fraction (e.g. 0.16),

Rt is the resistance ratio of the rolling

stock (customarily assumed to be 0.003),

Xb is the ratio of the mass of the rolling stock to that of the locomotive.

In common with Crook's formula the unbraked load is calculated in terms of brake ratio and retardation rather than distance and speed, but the effect of rotating parts is included as a variable. The RLSD formula is intended to be used with SI units, but it does not contain conversion constants and may be used with any system of consistent units.

NEW BRAKING CRITERIA

Investigation of incidents involving the loss of control of locomotives have shown that the brake efforts were sometimes either insufficient to retard the trains, or insufficient to retard them at an acceptable value. These low retardations have caused some drivers to assume that a train was not retarding and they have abandoned their locomotives. The authors therefore consider that a train should never retard at less than 2%g, i.e. the minimum retardation of a train should be approximately twice the value suggested by Crook.

It is proposed that the brake effort needed to retard a train at 2%g in the most onerous operational circumstance should be termed the Operational Brake Effort and the brake effort that a locomotive is designed to provide should be termed the Design Brake Effort. The Design Brake Effort should be measured in periodic brake tests, required at weekly intervals by U.K. Regulations.

Service Braking

The Design Brake Effort of a service brake may fall because of one or more of the following reasons:

a) a reduction in the system operating pressure;

b) a reduction in the coefficient of friction at the braking surfaces;

c) excessive wear of the brake mechanism;

d) brakes are out-of-adjustment.

To overcome these possible deficiencies the Operational Brake Effort should not exceed 80% of the Design Brake Effort of the service brake. This would introduce a safety margin into service braking by making an allowance for a reduction in brake effort which may occur between weekly brake tests.

However, even though service braking has variable control, the Design Brake

Effort should not cause a locomotive to lose adhesion because of 'overbraking'. Table 1 shows Design Brake Ratios for service braking which are considered to be appropriate for steel-tyred locomotives, together with the corresponding Operational Brake Ratios.

Table 1: Suggested service brake ratios for locomotives

Locomotive	Service brake ratio type	
	Design	Operational
steel-tyred uncoupled wheels	13%-18%	10%-14%
steel-tyred coupled wheels	16%-20%	13%-16%

Emergency Braking

Application of the emergency brake will result in a reduced brake effort if the emergency brake ratio is less than the service brake ratio. The Design Brake Effort of the emergency brake must not be less than the Operational Brake Effort required to retard a train at 2%g. However, application of the emergency brake must not cause loss of adhesion. The proposed service brake ratios of Table 1 limit the maximum Operational Brake Ratio of a locomotive to 16%. The coefficients of adhesion shown in Fig 1 provide some guidance to the Design Brake Ratios which are appropriate to emergency wheel braking. Tests made by RLSD have shown that locomotives fitted with track-reactive emergency brakes and copper brake pads can develop brake ratios in excess of 18% on both dry and wet rails.

Unbraked Rolling Stock

Fig 2 shows the result of using RLSD's formula (Equation 1) to calculate the unbraked rolling stock (Xb) that would be retarded at 2%g on inclines between 1.0% (1:100) and 6.7% (1:15) by Operational Brake Ratios ranging from 9%

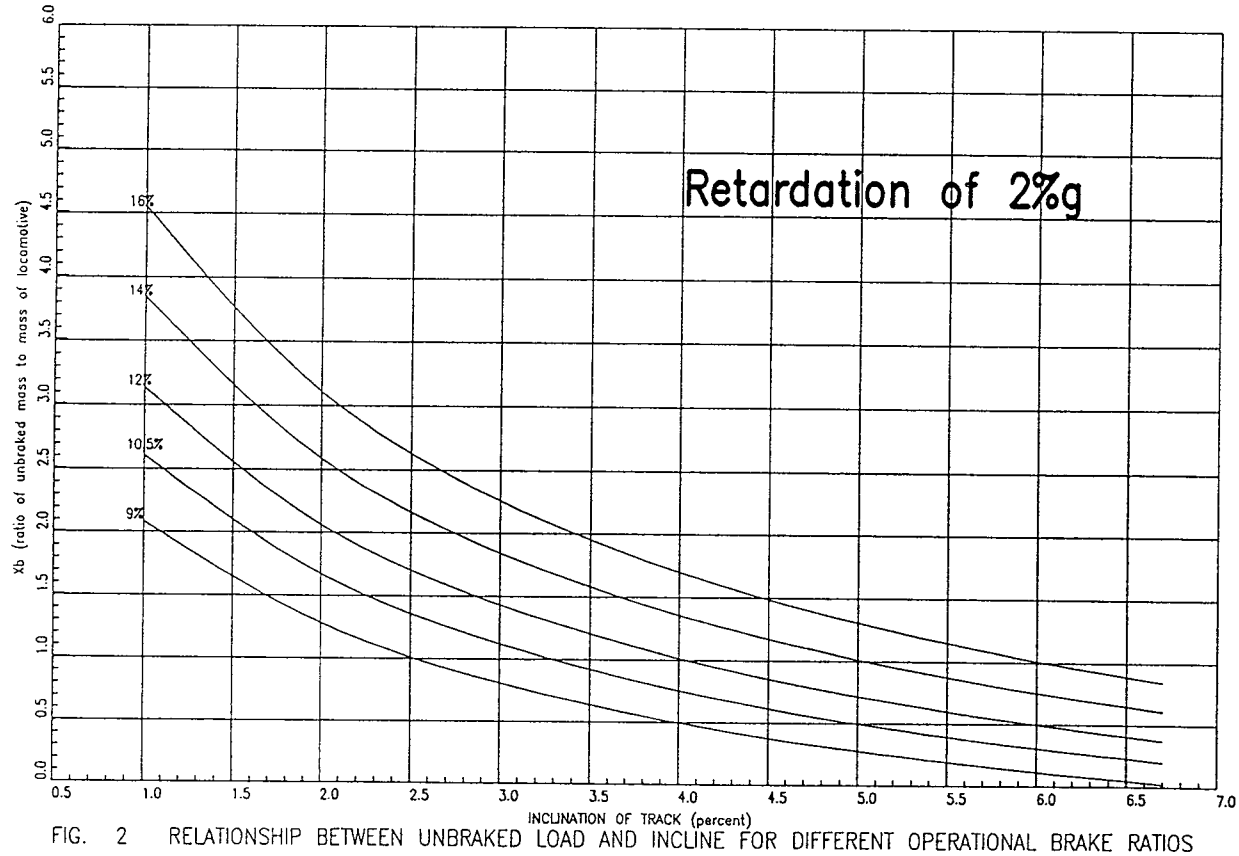


FIG. 2 RELATIONSHIP BETWEEN UNBRAKED LOAD AND INCLINE FOR DIFFERENT OPERATIONAL BRAKE RATIOS

to 16%. A value of 0.94 has been assumed for the rotational factor (c) of the train and of 0.003 for the resistance of the rolling stock (Rt). An Operational Brake Ratio of 16% requires the locomotive to have service and emergency brake ratios of 20% and at least 16% respectively. It can be seen that for these operating conditions the mass of the unbraked rolling stock should not exceed 85% of that of the locomotive, i.e. the Xb ratio is 0.85. This is a severe operational restriction and consideration should be given to the use of track reactive brakes when operating on inclines steeper than 3.3% (1:30).

RECOMMENDATIONS

1. The mass of unbraked rolling stock that may be braked by a locomotive should be calculated on the basis of the train retarding at not less than 2%g.
2. The brake effort required to retard a train at not less than 2%g in the most onerous operating condition should be termed the Operational Brake Effort.
3. The brake effort that a locomotive can provide should be termed the Design Brake Effort and should be determined from the retardations measured during instrumented tests on a train of suitable mass.
4. The Design Brake Effort should be appropriate to the type of locomotive and braking system, consideration being given to the possibility of the wheels losing adhesion with the rails.
5. The Operational Brake Effort should not exceed 80% of the Design Brake Effort of the service brake.
6. The Design Brake Effort of the emergency brake be at least equal to the Operational Brake Effort. The Design Brake Effort of an emergency brake acting at the wheels must not cause them to lose adhesion.
7. Locomotives operating on inclines of 3.3% (1:30) or steeper should have reliable emergency brakes, such as track-reactive brakes.

ACKNOWLEDGEMENTS

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FREINAGE DES LOCOMOTIVES MINIERES

D. Waterhouse

Lorsqu'un train freine, l'accélération négative dépend de l'effort de freinage déployé par ce train, de sa masse totale et de l'inclinaison de la voie ferrée. La condition de base de l'ingénierie ferroviaire est de fournir un effort de freinage suffisant pour maximiser l'accélération négative d'un train sans que celui-ci perde son adhérence et dérape. Cette condition est encore plus importante dans les mines, en raison de leurs pentes raides et de l'utilisation de matériel roulant non freiné. Le présent rapport examine des essais effectués sur 12 locomotives minières différentes; toutes déploient leurs efforts de freinage par leur poids et par le coefficient de friction entre leurs roues à bandages en acier et les rails. On a mesuré les efforts de freinage normal et d'urgence et on a évalué les effets des montages de type châssis et d'assemblage de roues ainsi que l'état des rails, selon le Memorandum 12 d'Essais de HSE. Nous donnons des suggestions sur les efforts de freinage exigés par rapport aux poids des locomotives testées.

DIE BREMSUNG VON GRUBENLOKOMOTIVEN.

D. Waterhouse

Bei der Bremsung eines Zuges hängt sein Bremsweg von seiner Bremsleistung, seiner Gesamtmasse und der Neigung der Gleise ab. Eine grundsätzliche Anforderung beim Lokomotivenbau besteht darin, die Verzögerung zu maximieren, ohne die Haftung zwischen Schiene und Rad zu verlieren und damit ein Rutschen hervorzurufen. Im Bergbau ist diese Anforderung besonders wichtig wegen der starken Neigungen und der Benutzung von ungebremsten Wagen. Der Vortrag berichtet über Versuche an 12 verschiedenen Grubenlokomotiven, die ihre Bremsleistungen aus ihrem Gewicht und dem Beiwert aus der Reibung zwischen Schiene und Rad entwickeln. Die Leistungen der Fahr- und Notbremsen wurden gemessen, der Einfluß des Chassis -[3]Typen und der Radkupplung, sowie der Zustand der Schienen in Berücksichtigung des HSE Testing Memorandum 12 bewertet. Der Vortrag gibt Vorschläge für die nötige Bremsleistung im Hinblick auf das Gewicht der Lokomotiven.

ТОРМОЖЕНИЕ ШАХТНЫХ ЛОКОМОТИВОВ

Д. Вотерхауз

При торможении состава его замедление зависит от развиваемого тормозного усилия, общей массы и уклона пути. Основным требованием при проектировании является обеспечение оптимального тормозного усилия с целью достижения максимального замедления состава без потери сцепления и скольжения. Это требование особенно важно в шахтах в связи с наличием уклонов и частичным использованием бестормозного подвижного состава. В докладе рассматриваются результаты испытаний двенадцати различных шахтных локомотивов, у которых развиваемое тормозное усилие зависит от собственного веса и трения между стальными колесами и рельсами. Были измерены усилия, развиваемые эксплуатационными и аварийными тормозами, а также были оценены влияние типа шасси и расположения сцепок и проверено соответствие состояния рельсов испытательной инструкции #12. Были предложены необходимые в соответствии с весом величины тормозных усилий для испытанных локомотивов.

矿用机车制动机的研究

D．华既浩斯

当机车制动时，制动作用与机车产生的制动力，机车的总质量，及轨道的倾角有关．铁道工程的基本要求是要有足够的制动作用，使机车能尽快减速，不致失去轨道的粘着力而滑行．这种要求在矿里特别需要，因为倾角很大，并且车辆无制动设备．本文讨论１２种不同的矿用机车，所有机车的制动作用都从它们的重量和钢轮及轨道之间的摩擦系数所产生．操作闸和紧急闸的制动作用都作了测定．底架种类，车轮偶联装置和轨道状态等的影响系根据卫生及安全署第１２号公文评定．关于这些试验过的机车我们对根据机车重量所需要的制动作用做了建议．

THE SAFETY OF MINE-SHAFT WIRE-ROPES: COOPERATIVE WORK BY CANADA AND THE USA

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ABSTRACT

Concern with safety in mines is a constant preoccupation of mining authorities the world over. It is also one of the many subjects dealt with in the ongoing cooperative research work undertaken, for many years, by United States and Federal/Provincial Canadian organizations. This report covers one aspect: nondestructive testing by electro-magnetic (EM) examination of mine-shaft wire-ropes. It is an example of the benefits that cooperative work can achieve.

The authors of this report examine the data contained in the Ontario Ministry of Labour's uniquely extensive rope-test data-bank; they augment these by information obtained from industrial and laboratory sources, both in Canada and the USA. As a result they conclude that, while EM testing has greatly increased the safety of mine-shaft hoisting, work to further improve the accuracy of this process - especially in certain corrosion and rope-construction related areas - is both necessary and urgent.

INTRODUCTION

In situ non-destructive testing of mine-shaft wire-ropes, with EM instruments, has been practiced internationally for many decades. Over the years a range of much improved instruments was developed by scientists in a number of countries including Britain, West Germany, France, Belgium, Switzerland, Poland, Czechoslovakia, the USA and Canada. It is generally acknowledged that the safety of mine-shaft hoisting has increased several fold since the use of EM rope testers has become widespread. Unfortunately, however, wire-rope failures, involving serious material damage if not loss of life, still occur. This happens even in situations where regulatory testing requirements have been followed to the letter and where the failed ropes had been previously declared fit for further use.

We examine some of the factors that are likely to be at the root of this serious problem. In particular:

(1) We summarize an analysis of a uniquely informative data-bank, compiled during the past three decades or so, by the Wire Rope Testing Laboratory of the Ontario Ministry of Labour (OML) in Canada, as well as of data obtained from other Canadian and US sources. Our objective is (a) to quantify just how accurately EM instrument based, non-destructive test (NDT) estimates of wire-rope strength-losses match those established by comparative destructive testing (DT), and (b) to compare the accuracy levels attained with different commercial variants of these EM testers;

(2) We summarize a comparison of the regulatory requirements established by a number of mining authorities. Our objective is to examine (a) the very wide diversity of technical and regulatory demands placed upon the users of EM instruments; (b) whether, in fact, all of these requirements can be met at the present state of EM instrument development, and (c) the regulatory requirements dealing with the training of the instrument operators.

BACKGROUND

While we are cognizant of the significant amount of literature dealing with the general subject area of non-destructive testing with EM instruments, at this point we would mention only one publication. It (Rice and Jentgen 1983) deals with extensive data of the kind we examine. In fact, it is an earlier example of cooperation between the USA and Canada, insofar as this paper is also based on the OML data bank, as it existed in 1983. One of the findings of this report was the conspicuous absence of the EM predicted strength from among several independent variables (with strength loss as the dependent variable) thus reflecting the initially poor fidelity of the early EM testers, and the vicissitudes of a newly emerging technology.

While this reference examined a large portion of the report data (some 1,670 out of a total of more than 5,500) we chose to severely reduce even that number. First, we limited our analysis to strength-loss data alone, ignoring all other available data (such as rope-extension values, visual corrosion and lubrication ratings, and wire torsional test results). Second, even of these data we used only those DT/NDT data-pairs where it is reasonably certain that they refer to identical rope lengths, or where identity is not of primary importance.

BENCHMARK PARAMETERS

The mine-shaft wire-rope retirement criteria - as mandated by the Mine Safety and Health Administration (MSHA) in the USA (Office 1988) and by most provincial regulatory authorities in Canada (e.g., Ontario 1987) - involve, inter alia, rope-strength (rather than metallic-area) losses. Consequently, one major focus of our paper is to quantify the accuracy with which various EM testers can predict a mine-shaft wire-rope's loss of breaking strength, and thus by extension, the safety and economy with which these ropes are currently being retired. The benchmark parameters used by us for the present analysis are, therefore, a reflection of these regulations as well as of Ontario's supplementary requirements (Geller et al 1988). It is for this reason that we:

(a) consider strength losses of 10 pct for hoist ropes, 15 pct for tail ropes and 25 pct for guide ropes to be of critical importance. As an extension, our analysis is centered on hoist ropes with "true" strength losses of at least 10 pct. True strength loss is measured by destructive testing.

(b) accept the Ontario regulatory requirement that mine-shaft ropes be examined in situ throughout their entire working length, by a competent person using an approved EM testing device.

(c) calculated the loss of breaking strength as a percentage of the original Certificate of Test strength value (and not as a percentage of some "best piece" found in a current rope).

(d) report an EM instrument estimated strength loss value, as established on an in situ rope with a confidence level of 95 pct, as being "acceptable" if it is within four percentage points of the true rope strength.

DATA ANALYSIS

Data analysis on the basis of estimated loss of breaking strength -

(1) Statistical analysis

After restricting our data-choice, 475 data-pairs were retained. Of these, 49 pct were obtained with locked-coil ropes, 8 pct with non-rotating rope constructions, and 43 pct with stranded ropes. Three Canadian EM instruments were used: the Rotesco AC for 61 pct of the data, the Rotescograph for 18 pct, and the Magnograph for 21 pct. The AC tester measures loss of metallic area (LMA) only, while the other two instruments produce both local fault (LF) and LMA traces. Figures 1, 2 and 3 show the error distribution curves for stranded ropes. A summary of all results is listed in Table 1. "Estimating Error" is defined as the loss of strength as determined by a destructive test minus the loss of strength as measured by a non-destructive test.

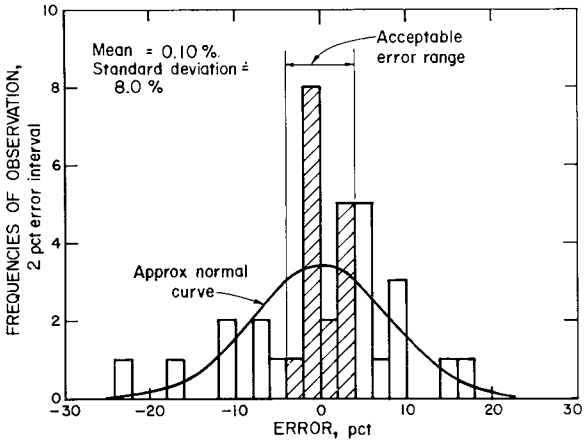


Figure 2. Error percent versus frequencies of observations for the Rotescograph instrument used on stranded ropes.

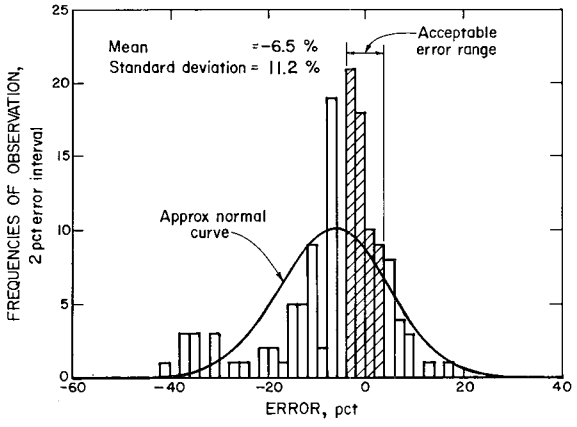


Figure 1. Error percent versus frequencies of observations for the Rotesco AC instrument used on stranded ropes.

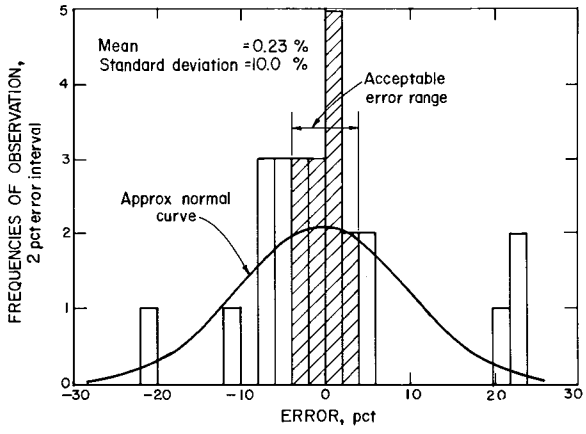


Figure 3. Error percent versus frequencies of observations for the Magnograph instrument used on stranded ropes.

Table 1: Analysis of Test Data -- A Summary

EM instrument and results	Rope construction		
	Locked-coil	Non-rotating	Stranded
Rotesco AC --			
Total No. of DT/NDT data-pairs	113	31	144
Average of all errors, pct	-0.72	-16.0	-6.5
Standard deviation of errors, pct	8.1	16.9	11.2
"acceptable" data*, pct of total	37	33	38
Rotescograph --			
Total No. of DT/NDT data-pairs	51	1	34
Average of all errors, pct	1.1	--	0.10
Standard deviation of errors, pct	4.0	--	8.0
"acceptable" data*, pct of total	65	0	56
Magnograph --			
Total No. of DT/NDT data-pairs	69	6	26
Average of all errors, pct	0.83	-3.3	0.23
Standard deviation of errors, pct	4.8	6.9	10.0
"acceptable" data*, pct of total	64	83	50

*: pct of total EM data that are within the Ontario "permissible" $\pm$ 4 pct error range; the "NDT = 0" values are not considered in this table, but are discussed below.

It will be noted that in case of the locked-coil and stranded ropes an average of 36 pct of all Rotesco AC measurements are within the Ontario "permissible" $\pm$4 pct accuracy range. For the Rotescograph, this figure is 61 pct, and for the Magnograph, it is 57 pct. Unfortunately, many more data pairs were available for the AC instrument than for the other two. This is particularly so in case of the non-rotating rope constructions.

The curves clearly indicate that Ontario's Performance Requirement is far from being satisfied. On the basis of the analyzed data (Table 1) and of standard statistical tables, predictions of accuracy at a 95 pct confidence level are given in Table 2.

(2) Special case -- rope failures

Table 3 shows the EM estimates recorded in tests obtained shortly before failure, on failed ropes and on companion ropes that sustained an equal amount of loading under identical conditions.

The foregoing are examples of the unfortunate results that have arisen as a consequence of the uncertainty that is inherent in the current inspection cum evaluation systems.

(3) Special case -- Estimate of loss by NDT = 0

In case of the locked-coil ropes in particular, but elsewhere as well, we noted a number of test data that indicate no estimated strength losses (i.e., NDT = 0); yet they are coupled with extensive service histories, and often even with increased DT values. In these cases, the true picture was only obtained by examining other test results as well, such as the rope's loss of extension, increase in rotation resistance of the wires, and when possible, hardening of the wires. We think that the rotation resistance, in particular, should always be included in the laboratory tests of rope samples, since it is a relatively simple torsion test, yet one that yields valuable information on the cold working and fatigue condition of the rope.

In view of all the above noted uncertainties inherent in the determination of strength-loss values on the basis of EM measurements, we decided to examine the following questions:

(a) if retirement criteria based of the loss of breaking strength cannot be met with presently

TABLE 2: Percent of Future Tests Predicted to be Acceptable

EM instrument used	Rope construction		
	Locked-coil	Non-rotating	Stranded
Rotesco AC	28-46	18-51	30-46
Rotescograph	50-78	--	39-73
Magnograph	52-73	40-99	30-70

TABLE 3: Rope Failures With Broken Strands

Rope Type	EM-Estimated Percent of Strength Loss		
	Before Failure	After Failure	Companion Rope
Non-rotating --			
25 mm, 18 x 7	5.5	--	--
32 mm, 34 x 7	--	11.0	5.0
31 mm, 34 x 7	--	--	16.0
44 mm, 34 x 7	14.0	--	--
	27.0	--	--
44 mm, 34 x 7	10.0	--	--
Stranded --			
57 mm, 6 x 7	--	--	4.5
54 mm, 6 x 7	--	--	12.0
58 mm, 6 x 7	--	--	2.5
58 mm, 6 x 7	--	--	4.0
44 mm, 6 x 27	--	--	2.5

available instrumentation, could an LMA-based requirement better satisfy them;

(b) how prevalent is the regulatory requirement of having to retire mine-shaft ropes, inter alia, on the basis of EM-determined strength loss estimates;

(c) can such regulatory requirements be reasonably well satisfied with presently available EM testers.

Our answers to the foregoing questions follow hereunder.

(a) A 1978 SMRE report (Commission 1978) provided an excellent insight into the capabilities of a number of EM testers to detect LF in various rope constructions. LMA is the other chart-trace displayed by many EM instruments and might, perhaps, be considered to be a more natural choice than loss of breaking strength, upon which to base regulatory requirements. Very little has, however, been published about how the "true" and EM-estimated LMA data compare in case of operational mine-shaft ropes, perhaps because it is much more difficult to ascertain a rope's "true" LMA than it is to determine its strength loss. True LMA is determined by dismantling the rope and measuring the diameter of each wire. We list hereunder some relevant data obtained by the US Bureau of Mines (BOM) and during our mine-site activities.

Laboratory Tests

The Rotesco AC Instrument

Metallic area change measurements were made at the BOM using a moving, spliced loop of 19 mm diameter, 6 x 25 FC test rope with a total metallic area of 1.311 cm^2. Wires, 760 mm long, in diameters of 1.2 and 2.0 mm were added to the rope. These wires represented metallic area increases of 0.82 and 2.47 pct respectively. Results were obtained for 10 Hz and 30 Hz driving frequency settings. Recommended frequency is 10 HZ for rope speeds of 0 to 1 m/s and 30 Hz for rope speeds of 0 to 2 m/s.

(1) at 10 Hz, and a rope speed of 0.4 m/s, the instrument recorded some 61 pct and 56 pct of the actual metallic area changes. On increasing the rope speed to 1 m/s, some 95 pct and 65 pct of the same actual area changes were recorded.

(2) at 30 Hz, the corresponding metallic area measurements were: 44 pct and 52 pct at a rope speed of 0.4 m/s; 67 pct and 51 pct at a rope speed of 1.0 m/s; and 71 pct and 47 pct at a rope speed of 2.0 m/s.

The Rotescograph Instrument

Other results with the same test rope as was used with the Rotesco AC tester, include the following:

(1) for the 0.82 pct actual area change and 0.4 m/s rope speed, a 6.4 mm added length was the shortest depicted on the LF trace; however, the shortest wire that the instrument could resolve into a constant LMA trace was 254 mm.

In the case of the 2.47 pct actual area change and 0.4 m/s rope speed, the shortest length recorded on the LF trace was 4.8 mm. The shortest wire length that produced a constant LMA trace was again 254 mm.

(2) metallic area changes recorded at a rope speed of 0.4 m/s were 73 pct and 61 pct of the actual metallic area for the 760 mm lengths of added wire.

Metallic area changes recorded at a rope speed of 1.0 m/s were 61 pct and 57 pct for the 760 mm lengths of added wire.

(3) LF trace results were the same at rope speeds of 0.4 m/s, 1.0 m/s and 2.0 m/s.

(4) our tests indicate that at rope speeds of 0.4, 1.0 and 2.0 m/s, a 50 mm long area change of either 0.79 pct or 1.09 pct is the shortest that can be detected.

The Magnograph Instrument

(1) the original designers of this instrument claimed that the instrument can detect a loss of metallic area as small as 0.5 pct.

(2) elsewhere it has been said (Haller 1984) that an accuracy of about ± 2 pct of the rope's cross sectional area should be achievable with present day EM instruments.

(3) our tests indicate that at a rope speed of 0.4, 1.0 and 2.0 m/s, a 200 mm long area change of either 0.79 pct or 1.09 pct is the shortest that can be detected.

(4) the BOM also did a comprensive examination of a 50 mm, 6 x 25 IWRC wire rope, 260 m long, that had been run on the bending fatigue machine until many broken wires appeared in the middle section. This machine is designed to deteriorate a rope to varying degrees along its length. Distances were carefully marked along the rope and, based on LMA traces produced by a Magnograph service representative, sections were selected for testing. The rope diameter was measured, under load, on each section, then 17 samples were cut from the rope. Figure 4 shows the metallic area losses calculated from the measured diameters compared to the losses obtained from the Magnograph LMA traces. As can be seen, the agreement between them is fairly good on the first half of the rope and very good on the last half. The conclusion to be drawn is that the instrument measured area loss as well as the more laborious diameter measurement method.

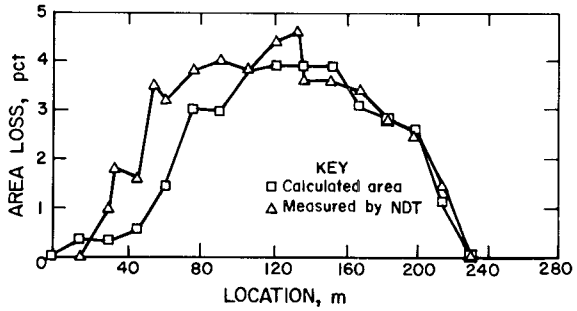


Figure 4. Area losses from Magnograph charts and diameter measurements.

To measure loss of breaking strength, tensile tests were run on the samples, yielding the data shown in Table 4.

The last two samples, taken from the drum, were used as the basis for comparison of the data. It can be seen that the lowest values occur in the 120-150 m section. Breaking strain is reduced by 65 pct and the torque constant is down by 11 percent, indicating increased hardness of the rope. Torque K is the slope of the line relating torque to load. The loss of strength of the rope is shown in Figure 5.

The results indicate that the rope goes through three phases of degradation: strength reduction due to metal area loss, strength increase due to cold working and, finally, serious strength loss due to the fatigue failure of the wires. Degradation is symmetrical about the middle. In the case of this rope, the 120-150 m zone had so many broken wires that the LF trace was unintelligible. Normally, of course, a rope would be retired long before that many wires were broken. It would appear from these data, as well as from the NDT = 0 special case referred to earlier, that measurement of metal hardness would be a valuable supplement in assesing the condition of wire rope.

Table 4: Tensile Test Data on Bending Fatigue Samples

Distance m	Breaking Load kg	Breaking Strain pct,m/m	Torque K g-m/kg
33.5- 38.7	170.68	2.30	4.49
50.3- 55.5	176.62	2.15	4.23
57.9- 63.1	174.76	2.06	4.24
74.7- 79.9	188.40	2.23	3.96
91.5- 96.6	189.61	2.30	3.98
108.2-113.4	175.07	1.84	4.04
115.9-121.0	171.35	1.73	3.99
124.4-129.6	165.46	1.65	4.01
131.4-136.3	159.61	1.52	4.02
139.0-143.9	167.09	1.78	4.05
146.3-151.5	152.78	1.43	4.04
160.1-165.2	185.84	2.18	4.03
176.8-182.0	186.41	2.16	4.03
196.6-201.8	177.28	1.85	4.11
221.0-226.2	176.35	2.32	4.32
236.3-241.5	187.52	4.43	4.52
253.0-258.2	187.66	4.19	4.45

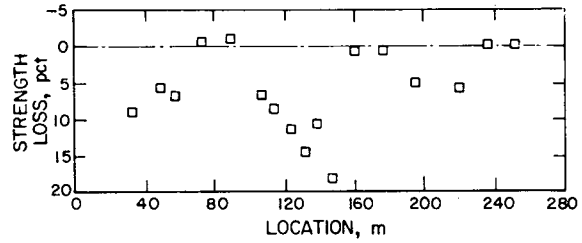


Figure 5. Percent loss of breaking strength versus position along the rope.

The LMA 250 Instrument

Although the US-designed LMA series instrument was not used for any of the test runs recorded in the OML's data-bank, a limited amount of data were available to us, primarily on the basis of tests in the US. As an example, we noted that with this instrument a 25 mm long (in some cases, even 13 mm long) area change could be recorded - whether the metallic area change was 0.79 pct or 1.09 pct of the total rope metal area - at both the 0.4 and 2.0 m/s rope speeds.

Field Tests

In one particular case, in situ test results showed that the LMA estimates obtained with a Magnograph and an LMA 250 tester on the same operational mine-shaft ropes (one a 32 mm locked-coil, the other a 48 mm non-rotating construction), differed by more than 100 pct. Moreover, the greater estimate was not obtained consistently with either of these instruments. Another example involved field-tests on a 25 mm, 6 x 26 wire-rope (Poffenroth and Sheridan 1986). In this case, the in situ loss of metallic area data were obtained with a Rotesco AC tester. On comparing these with the actual LMA values, it was found that the actual area losses were some 100 pct greater than had been estimated from the EM traces.

On reviewing these data we cannot, therefore, say with any degree of certainty that the EM instrument's LMA-trace-based rope retirement criteria are any more reliable than the ones based on loss of breaking strength. We shall re-examine this question as soon as the results of a Canadian research project, currently underway, become available. We also suggest that much additional information could be obtained in this context, at no extra cost, if the regulatory authorities would specify (1) that all EM test reports are to contain separate sections on the observed LF, LMA, and loss-of-breaking-strength values, and (2) that the exact location of the rope samples, submitted for routine regulatory destructive testing, be precisely identified.

(b) We reviewed a number of rope retirement criteria (Geller et al 1988) and found an extraordinarily wide range of regulatory requirements. These include the situation in Britain, with no mandatory EM testing at all, as well as those in many Canadian provinces, where EM testing with an approved instrument by competent operators is the rule. On balance we find that at present, few European but many North American mining acts include EM instrument-based loss of breaking strength determination among their mine-shaft rope retirement criteria. However, we also note that in some Canadian Provinces new regulations are now being considered that will be based on NDT-determined LMA and/or calculation based on reduction of diameter.

As to the regulation that EM instruments must be "approved": we are well aware that this requirement is not customary outside of Canada. As an example:

(1) In one of West Germany's most heavily industrialized area, Nordrhein-Westfalen, EM instrumentation is designed in-house

by a group of experts, from an industry cum state-supported independent rope research organization, whose decisions are acceptable to the mining authorities.

(2) In Hungary too, the mining authorities accept the expertise of a selected group of people. In this case it is a university faculty, that decides on the type of EM instrumentation to be used and can recommend that others be acknowledged as qualified operators.

(3) US authorities leave the choice of EM instrument to the users.

(4) In Britain, EM testing itself is not mandated.

(5) With the Swiss Federal Laboratories for Materials Testing and Research (EMPA) the practice is to only use an EM tester as a means for locating all rope defects. Emphasis is then placed upon subsequent visual examination of any major defects thus detected.

As for requiring operator competency in regulations, we believe that the EM tester's mechanical operation is not unduly difficult to learn. However, evaluation of the instrument charts cannot, at this time, be based on purely mechanical operations and thus requires some skill. The crux of the problem is to be thoroughly knowledgeable about the history of the rope in question, and of that of its predecessors. To illustrate: in Canada there are at least two major mining organizations who own and operate their own EM testers (two Rotesco AC and a Rotescograph, respectively). Results are correspondingly favorable. As an example: of the 34 data-pairs listed in Table 1 for stranded ropes and the Rotescograph instrument, 23 (68 pct) were obtained by one of these mining companies. Of these 78 pct were acceptable in the sense used elsewhere in this paper. Even of the 9 cases with a true loss of breaking strength of 10 pct or more (ranging from strength losses of 10 pct to 31 pct), only two displayed an error in excess of 4 pct, and even these two erred on the safe side, i.e., they were overestimates rather than underestimates of the true loss of breaking strength.

(c) We are aware of the serious difficulties in predicting strength loss on the basis of LMA and LF data, which are the only measurements currently recorded by the EM instrument charts. We think that these instruments can be strong contributors in the assessment of wire rope condition, provided the LF and LMA recordings are accurate and sufficiently well defined. But, supplemental data such as hardness or rotation resistence of the wires must also be considered. We prefer not to answer this question in further detail at this point. We shall, however, re-examine it as soon as additional data are available as a result of the current Canadian work.

CONCLUSIONS

In summary we note that the benchmark requirement of being able to estimate the breaking strength loss of mine-shaft wire-ropes with an accuracy of ± 4 pct at a level of confidence of $\overline{95}$ pct, has not been achieved with any of the commercially available instruments we analyzed.

The number of seriously flawed EM test estimates has also been noted, and the number of rope failures that occurred in spite of routine testing. Even in cases where no rope failures occurred, the actual breaking strength losses in the retired ropes greatly exceed - on a number of occasions - the ones allowed for by regulations.

We also note that estimates of metallic-area losses made by current equipment are not as reliable as would be desirable. However, we do

not necessarily believe that this goal cannot be attained in the future. We hope to be able to obtain additional significant data as a result of the Canadian project.

We believe that EM test reports should always contain a clear indication of the LF, of the LMA, as well as of the strength-loss values that have been estimated from the chart traces. Another suggestion is that the exact location on the rope of the samples which must be tested destructively on a routine regulatory basis, be stated. This way, the store of comparative NDT/DT data-pairs could be increased at no extra cost. Of course this store of valuable information will also have to be analyzed and published at regular intervals. Finally we suggest that the rotation resistance of the wires be tested as often as possible.

While there is room for further improvement, and despite the above noted shortcomings, we maintain that the safety of mine-shaft hoisting has increased many times since the worldwide acceptance of regular testing with EM instruments. As mentioned, the Rotesco AC instrument was the first tester that found widespread use in the North American mining industry. Thereby it contributed substantially to this increase in mine-hoisting safety. Evidence that the subsequent generation of testers reduced uncertainty in evaluating the fitness of ropes for continued service was illustrated in Table 1.

We are of the opinion that close national and international cooperation is most desirable, as has been amply proven by the Canada-USA joint project in which we are engaged. The extensive data bank of the OML's Rope Testing Laboratory has supplied us with a wealth of invaluable information. We would be grateful to receive similar information from others who might be reading this report with interest.

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SECURITE DES CABLES METALLIQUES DES PUITS DE MINES - PROJET EN COLLABORATION ENTRE LE CANADA ET LES ETATS-UNIS

L.B. Geller, Grant L. Anderson, A.J. Miscoe, E.W.Mitchell

La sécurité minière est une préoccupation constante des organismes miniers du monde entier. Depuis de nombreuses années, elle constitue également l'un des nombreux sujets des travaux de recherches menés en collaboration par les organisations fédérales des Etats-Unis et par les organisations provinciales du Canada. Notre rapport étudie l'un de ces aspects: l'examen électro-magnétique (EM) des câbles métaliques des puits de mines. Il prouve le succès des travaux effectués en collaboration.

Les auteurs du présent rapport examinent les informations de l'importante et exceptionnelle banque de données du Ministère du Travail de l'Ontario sur les essais de câbles; ils les complètent par des informations de sources industrielles et de laboratoires aux Etats-Unis et au Canada. Ils en concluent que, si les essais EM ont permis d'augmenter considérablement la sécurité du hissage dans les puits de mines, il est à la fois nécessaire et urgent de poursuivre les travaux visant à améliorer encore la fiabilité de ce processus, spécialement dans certains domaines relatifs à la corrosion et à la fabrication des câbles.

SICHERHEIT VON FÖRDERSEILEN - GEMEINSCHAFTSARBEIT ZWISCHEN KANADA UND DEN VEREINIGTEN STAATEN.

L. B. Keller, Grant L. Anderson, A. J. Miscoe, E. W. Mitchell

Eine der Hauptaufgaben der Bergbaubehörden weltweit ist die Sicherheit in der Bergbauindustrie. Seit Jahren arbeiten kanadische und US-amerikanische Organisationen auf Bundes- und Provinzebene an gemeinsamen Forschungsvorhaben auf dem Gebiet der Sicherheit im Bergbau. Der Vortrag behandelt ein Thema: das der elektro-magnetischen (EM) Untersuchungen an Förderseilen. An diesem Beispiel wird der Erfolg von Gemeinschaftsarbeit illustriert.

Die Autoren des Berichtes untersuchen die Daten der ungewöhnlich reichhaltigen Datenbank des Ontario Ministry of Labour über Seiltests; sie ergänzen diese Daten durch Informationen aus sowohl der kanadischen als auch der US-amerikanischen Industrie und Forschung. Die Ergebnisse beweisen, daß die EM-Testmethode die Sicherheit der Förderseile sehr erhöht hat, daß jedoch weitere Arbeiten zur Verbesserung der Genauigkeit des Prozesses - insbesondere in Zusammenhang mit gewissen Korrosions- und Seilkonstruktionsproblemen - dringend notwendig sind.

БЕЗОПАСНОСТЬ СТАЛЬНЫХ ТРОСОВ, ИСПОЛЬЗУЕМЫХ В ШАХТНЫХ СТВОЛАХ: СОТРУДНИЧЕСТВО МЕЖДУ КАНАДОЙ И США

Л.Б. Геллер, А.Дж. Мискоэ, Грант Л. Андерсон и Е.В. Митчелл

Безопасность работы в шахтах служит предметом постоянной заботы специалистов по горному делу во всем мире. Это также явилось темой совместных исследований, проводившихся в течение многих лет канадскими и американскими общенациональными и штатными организациями. Настоящий доклад рассматривает только один аспект: электромагнитную проверку стальных тросов в шахтных стволах. Этот проект представляет собой пример плодотворного сотрудничества. Авторы рассматривают уникальные по широте данные, полученные при испытаниях стальных канатов, которые хранятся в Министерстве Труда Онтарио. Эти материалы были дополнены за счет информации, полученной из производственных и лабораторных источников как Канады, так и США. На основе этих данных авторы заключают, что хотя электромагнитные испытания значительно повышают безопасность грузоподъемных устройств в шахтных стволах, необходимо не откладывая продолжать работу по дальнейшему усовершенствованию точности этого процесса, особенно в областях борьбы с коррозией и конструирования канатного оборудования.

矿井用钢丝绳的安全问题——加拿大与美国的合作工作

L．B．盖勒，格兰特 L．安德生
A．J．密士可，及 E．W．密契儿

矿里的安全问题是全世界矿业当局常常全神贯注的事，这也是多年来加拿大和美国联邦及省级单位合作发展的多项方案之一．本文专门讨论一个课题：矿井用钢丝绳的电磁试验法．这是这种合作发展有成果的一个例子．

本文作者们查阅了安大略省劳工部搜集的有关钢丝绳试验非常详尽的数据库，然后用加拿大和美国工业界与实验室所得到的资料加以补充．从这里所得到的结论是：电磁试验对矿井提升的安全大为改进，但这种作业精确度的提高——特别是在锈蚀和与钢绳结构有关的场合之下——仍然是非常必要和急迫的事情．

A NEW GEOMECHANICAL CONCEPT FOR THE INTENSIVE USE OF DEPOSITS IN POTASH MINING OF THE GDR

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ABSTRACT

As in other potash producing countries, great efforts are being made in the GDR toward the intensive exploitational potash deposits. Within the bounds of a complex geological, geophysical and geomechanical test program initiated for this purpose, the Institut fur Bergbausicherheit (Institute of Safety-in-Mines) is conducting fundamental research on the following subjects:

- Evaluation of the natural stress state of the deposit;
- Determination of the stress-strain behaviour of the solid rock in the laboratory;
- Development of computational models for the determination of the stress-strain state in the rock mass.

All these fundamental tests are aimed at establishing the basis for new mining variations allowing as complete recovery of the deposit as possible, while conforming with the requirements of safety, which under the existing geological-tectonic and hydrological conditions is very difficult in some areas.

The paper presents the results of in-situ tests to determine the stress state and the results of laboratory tests on the stress-strain state of the salt rock studied; the potential of new mining variations derived from these results is discussed.

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Manuscript was not received in time to be included in Proceedings.

NOUVEAU CONCEPT GEOMECANIQUE DE L'UTILISATION INTENSIVE DES GISEMENTS DANS L'EXTRACTION DU POTASSE EN RDA

Dr. U. Gross, Dr. W. Menzel, Dr. K. Thoma

Ainsi que les autres pays producteurs de potasse, la RDA a fait des efforts importants ces dernières années en ce qui concerne l'utilisation intensive des gisements. Dans le cadre d'un programme complexe d'essais géologiques, géophysiques et géomécaniques institué dans ce but, l'Institut de la Sécurité Minière (Institut für Bergbausicherheit) effectue des recherches fondamentales sur les sujets secondaires suivants:

- Evaluation de l'état de contrainte naturel des gisements;
- Détermination en laboratoire du comportement contrainte-déformation des roches solides;
- Développement de modèles de calculs pour établir l'état de contrainte-déformation de la masse rocheuse.

Tous les essais fondamentaux servent à établir les bases de nouvelles variantes minières permettant d'utiliser les gisements en minimisant les pertes, ceci conformément aux exigences des mesures de sécurité pour les conditions géotectoniques et hydrologiques, parfois très défavorables, en existence.

Ce rapport présente les résultats d'essais effectués in situ pour déterminer l'état de contrainte et d'essais effectués en laboratoire sur l'état de contrainte-déformation des roches de sel étudiées; il examine également les possibilités de nouvelles variantes minières qui en découlent.

EIN NEUES GEOMECHANISCHES KONZEPT FÜR DIE INTENSIVE NUTZUNG VON KALIVORKOMMEN IN DER DEUTSCHEN DEMOKRATISCHEN REPUBLIK.

Dr. U. Groß, Dr. W. Menzel, Dr. K. Thoma

In der Deutschen Demokratischen Republik, wie in anderen Kali-produzierenden Ländern auch, werden seit einigen Jahren große Anstrengungen zur intensiven Nutzung der Vorkommen unternommen. Innerhalb eines komplexen, geophysikalischen und geomechanischen Untersuchungsprogrammes führt das Institut für Bergbausicherheit fundamentale Forschungsarbeiten in den folgenden Untergebieten aus:

- Bewertung der natürlichen Spannung im Vorkommen;
- Bestimmung des Spannungs - Dehnungsverhalten von massivem Gestein im Labor;
- Entwicklung eines Rechenmodells zur Bestimmung des Spannungs-Dehnungs-Zustandes im Gestein.

Alle fundamentalen Untersuchungen zielen darauf ab, trotz der teilweise schwierigen geologisch-tektonischen und hydrolytischen Bedingungen eine Basis für neue Abbauvarianten zur möglichst verlustfreien Nutzung des Vorkommens bei Einhaltung aller Sicherheitsforderungen zu finden.

Der Vortrag stellt die Ergebnisse von Untertageversuchen zur Bestimmung des Belastungszustandes und Ergebnisse von Laborversuchen des Spannungs-Dehnungs-Zustandes in dem geprüften Steinsalz vor, und legt die daraus entstehenden Möglichkeiten für neue Abbauvarianten im Einzelnen dar.

НОВАЯ ГЕОМЕХАНИЧЕСКАЯ КОНЦЕПЦИЯ ИНТЕНСИВНОЙ РАЗРАБОТКИ МЕСТОРОЖДЕНИЙ ПРИ ДОБЫЧЕ ПОТАША В ГДР

Доктор Э.У. Гросс, доктор Э.В. Мензель, доктор Э.К. Тома

Большие усилия, направленные на интенсивную разработку месторождений, были предприняты в последние годы как в ГДР, так и в других странах, добывающих поташ. В рамках комплексной геологической, геофизической и геомеханической экспериментальной программы, начатой с этой целью, Институт безопасности горных работ проводит следующие научные исследования:

- Оценка природного стресса на месторождении;
- Определение поведения коренных пород при растяжении в лабораторных условиях;
- Разработка расчетных моделей для определения стресса в толще пород.

Все научные эксперименты направлены на создание основы для новых методов добычи, позволяющих разрабатывать месторождения с минимальными потерями и в соответствии с требованиями техники безопасности в существующих, иногда неблагоприятных, геолого-тектонических и гидрологических условиях. В докладе излагаются результаты экспериментов по определению стресса в исследуемой каменной соли, которые были проведены как в полевых, так и в лабораторных условиях. Детально обсуждаются возможности применения новых методов разработки месторождений, основанные на результатах этих испытаний.

民主德国钾碱矿充分利用沉积矿床的一个地质力学新概念

U. 葛罗斯 工程博士，W. 孟彻尔工程博士，K. 透玛 工程博士

近年来，民主德国亦象其他钾碱生产国一样，非常注意充分利用沉积矿床。局限于这种开采方法的地质的，地球物理的和地质力学的试验计划的深度和广度，矿业安全研究院展开了一项包括如下范畴的基础研究：

-- 估计沉积矿内的自然应力状态；
-- 在试验室内测定固体岩石的应力应变关系；
-- 建立计算模型来计算岩石体内的应力应变状态。

所有这些基础试验都是为了在尽量充分利用沉积矿的前题下，建立新的开采变量的基础。这同时还要顾及到在一些很不利的地质结构和水文条件下能安全操作。

本文提供了现场试验应力状态所得结果，和试验室里测得的岩盐应力－应变状态，并详细说明了由此求出新矿冶变量的可能性。

PROTECTING CONVEYANCES FROM OVERRUNNING THE TERMINALS DUE TO A DOWN-GOING EXCESS PAYLOAD

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ABSTRACT

From analyzing the nine marked overrunnings that occurred with Koepe type hoisting equipments in German Coal Mines in the past seven years it was concluded, that a suitable monitoring device coupled to an electric winding engine would have prevented those occurrences, if constructed to the following principle:

An electrical signal, that is proportional to the actual payload and to the direction it is moved, may be derived from the electrical current of the engine and the differentiated voltage signal of the tachometer. Hence a trigger voltage may be set to activate an alarm in case of any down-going excess payload specified.

The action to be taken in case of an alarm will depend on the type of hoisting equipment in question, as outlined in the paper. Several installations of that kind have already been realized and have been tested to work the intended way.

OUTLINE OF THE PROBLEM

A safe and continuous operation of hoisting equipment requires the conveyances to stop at the terminals within close limits after each run. The situation if they do not stop the intended way, but pass the terminals, is termed an overrunning.

In case of a moderate overrunning the conveyances may stop some distance behind the terminals without being damaged. In case of an excessive overrunning the conveyances enter the cone-shaped end portions of the guides or even hit the bumping beam with the likely consequence of the conveyances and the shaft installation being more or less severely damaged and of down-time required for repair. In addition an overrunning may endanger riding men on the cage or men staying nearby at the terminals.

On an average basis of the past years about one case of an overrunning associated with material damage, but without men being injured, had to be stated with Koepe type hoisting equipments in German Coal Mines. Nine cases of an overrunning, that occurred in the years of 1981 to 1987 were investigated for their reasons by experts of WBK-Seilprüfstelle.

All these nine cases of an overrunning happened due to winding a down-going payload and under the following circumstances, Table 1:

Table 1: Circumstances under which the overrunnings occurred

Cases A: down-going payload intended:

Case A1:	down-going payload 43,0 t	instead of	27,0 t	(+60%),	material handling
Case A2:	down-going payload 13,5 t	instead of	8,4 t	(+60%),	material handling
Case A3:	down-going payload 11,0 t	instead of	7,6 t	(+45%),	material handling
Case A4:	down-going payload 9,0 t	instead of	7,0 t	(+30%),	material handling

Cases B: down-going payload not intended

Case B1:	skip down-going still loaded,	switching between automatic and manual control
Case B2:	skip down-going still loaded,	switching between automatic and manual control
Case B3:	skip down-going still loaded,	re-loading switched off
Case B4:	skip down-going still loaded,	start in wrong direktion in manual mode
Case B5:	skip down-going still loaded,	automatic mode but safety brake blocked

In conclusion two typical situations become obvious:

- an intended but excessive down going payload during material handling,
- a non-intended down-going payload in a skip during product winding.

POSSIBLE SOLUTION OF THE PROBLEM

As a result of thorough considerations an electronic device to supplement the safety circuit and the control of the electrical winding engine was conceived that would be capable to prevent overrunnings of both kinds in future: A device that identifies the amount of a down-going payload and activates an alarm signal, if the down-going payload exceeds a limit value particularly specified for the hoisting equipment in question: For a cage winding installation, also used to supply material underground, this limit value would be set equal to the allowable maximum payload, for a skip winding installation, exclusively used for hoisting coal to surface, this limit value would be set to much smaller payload, however.

Accordingly the provisions in case of an alarm signal may be defined in a way depending on the type of hoisting equipment in question:

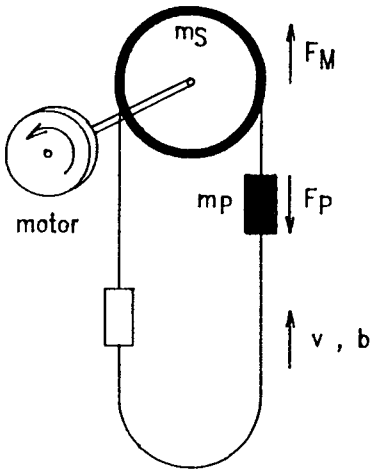
As a skip winding installation normally operates automatically with an up-going payload only, any down-going payload being identified may be excluded by a stop of the drive immediately after start, which then requires clearing of the situation by manual control. For other installations that operate automatically with up-going as well as down-going payload an alarm must be activated only in case of an down-going excess payload with the effect of a distinctly limited winding speed or a stop. Consequently, in manual mode of operation an acoustical warning and a speed limitation have to be provided in the situation mentioned.

THEORETICAL CONSIDERATIONS

Referring to the scheme and the notations in Figure 1 the principle of the monitoring device may be theoretically considered the following way, where all masses, forces and kinetic quantities have been related to the linearly moving rope:

The driving force of the engine F_E has to produce the acceleration a of both all the (balanced) masses m_S of the system and of the (one-sided) payload m_P and further has to compensate the weight of the payload $\pm F_P = \pm m_P \cdot g$, where the sign distinguishes whether the payload is placed in the up-going or in the down-going compartment:

$$F_E = \pm F_P + [m_S + m_P] \cdot a \ .$$



F_E = driving force of the engine
F_P = weight of the payload
m_S = equivalent mass of the system
m_P = mass equal to the payload
v = velocity
a = acceleration
g = 9.81 m/s^2

Figure 1: Scheme of a Koepe type hoisting equipment and illustration of the notations used.

In terms of the payload F_P the formula is obtained:

$$\pm F_P = F_E - [m_S + m_P] \cdot a \ .$$

If the payload must not exceed a specified limit value $F_L = m_L \cdot g$, the formula may be written:

$$\pm F_P = F_E - [m_S + m_L + (F_P - F_L)/g] \cdot a.$$

Accordingly the formula to determine the payload F_P may be simplified to

$$\pm F_P = F_E - [m_S + m_L] \cdot a \ ,$$

that is, by accepting an error of $(F_P - F_L) \cdot a/g$. It becomes zero, however, if the payload F_P equals the limit value F_L or if the acceleration is a = 0 , but in all remaining situations with usual accelerations a < 1.5 m/s2 is less than 15 percent.

In particular, the above formula shows a possibility of how to realize a continuous monitoring of the actual payload even in the period of acceleration as it occurs at and shortly after start.

PRACTICAL REALIZATION

Practically, such a monitoring device may be realized by electronic means conforming to the following principle:

The driving force of the engine F_E is proportional to the electrical current and may be represented in terms of a voltage U_M. The acceleration a is determined by differentiating the electrical velocity signal U_V delivered by the tachometer to obtain $U_a = d(U_V)/dt$. The masses to be accelerated, or the product $[m_S + m_L] \cdot a$ respectively, may allowed for by the setting x_S of a potentiometer in order to adjust $x_S \cdot U_a$ to a value specific of the installation in question. In addition, to define the sign of F_P it has to be known what direction the drive is moving, and this information may be gained from the sign of the velocity signal as $SIG(U_V)$.

A down-going payload ($-F_P$), represented by a voltage $U_P < 0$, that exceeds the specified limit value ($-F_L$), electrically set by a voltage $U_L < 0$, can then be identified in an unique manner from the condition $U_P - U_L < 0$ or:

$$[UM - x_S \cdot U_a] \cdot SIG(U_V) - U_L < 0 .$$

RELIABILITY REQUIREMENTS

Reliability requirements for such a kind of monitoring devices may be characterized by the following figures:

The fore-mentioned empirical number of about one overrunning per year is related to roughly 200 hoisting equipments, for which an annual total number of windings may be estimated to range in the order of 10^7. This means, the suggested type of monitoring device shall be capable to identify non-regular modes of operation that occur with a rather low probability only, or in other terms with reference to some particular installation, may happen only about once in a hundred years period.

Under normal operating conditions the monitoring device hasn't any effect at all. Therefore its malfunctioning would probably remain undetected for a long time, unless the defect would result in an alarm signal. Consequently, a design observing reliability aspects and function test to be regularly made in short intervals are essential.

CONCLUDING REMARKS

Meanwhile several of the described kind of monitoring devices have been delivered and attached to Koepe type hoisting equipments. Investigations performed for their commissioning by experts of WBK-Seilprüfstelle confirmed that they work the intended way:

After start, when having reached some initial speed necessary in order to differentiate the velocity signal and to determine its sign, any down-going excess payload will be definitely identified. So the drive may be stopped or limited in speed well before the high regular operating speed has been reached.

The described kind of monitoring device therefore provides a further means to assure the safety and availability of hoisting equipment.

REFERENCES

/1/ Analyzed cases of an overrunning. Internal Paper of WBK-Seilprüfstelle. Unpublished.

/2/ Submitted specification of monitoring devises supplied by AAB, AEG and Siemens. Unpublished.

VERHINDERN EINES ÜBERTREIBENS DER FÖRDERMITTEL AUFGRUND EINER ÜBERGROSSEN EINHÄNGELAST

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ÜBERSICHT

Aus einer Analyse der neun markanten Fällen eines übertreibens, die bei Koepe-Förderanlagen im deutschen Steinkohlenbergbau in den zurückliegenden sieben Jahren zu verzeichnen waren, wurde gefolgert, daß eine geeignete überwachungseinrichtung an einer elektrischen Fördermaschine solche Vorfälle hätte vermeiden können, wenn sie nach folgendem Prinzip aufgebaut ist:

Ein elektrisches Signal, das der Größe und Förderrichtung der jeweiligen überlast proportional ist, kann aus dem Strom der Fördermaschine und aus der differenzierten Ausgangsspannung des Tachogenerators gewonnen werden. über einen eingestellten Spannungs-Grenzwert kann somit ein Signal auslöst werden, wenn eine größere als die elektrisch vorgegebene überlast einhängend bewegt wird.

Welche Auswirkungen eine Signalgabe haben soll, richtet sich nach der Art der infrage stehenden Förderanlage, wie nachstehend dargelegt wird. Mehrere Einrichtungen der beschriebenen Art sind bereits an Anlagen eingebaut und untersucht worden mit dem Ergebnis, daß sie in der beabsichtigten Weise arbeiten.

PROBLEMSTELLUNG

Der sichere und ungestörte Betrieb einer Schachtförderanlage verlangt, daß die Fördermittel zum Ende eines jeden Treibens an den Endanschlägen innerhalb enger Bündigkeitsgrenzen zum Stillstand kommen. Geschieht dies nicht in der beabsichtigten Weise, sondern bewegen sich die Fördermittel über die Endanschläge hinaus, so spricht man von einem übertreiben.

Bei geringfügigem übertreiben kommen die Fördermittel ohne Schaden kurz hinter den Endanschlägen zum Stehen. Beim Einfahren in die Spurlattenverdickung und insbesondere beim Anfahren an den Prellträger kommt es u.U. zu erheblichen Schäden an den Fördermitteln und der Schachtkonstruktion und dadurch bedingten Förderstillstand zur Reparatur. Gegebenenfalls besteht auch eine Gefährdung von Menschen, wenn sie sich auf dem Korb befinden oder im Umfeld der Anschläge aufhalten.

Im Mittel der letzten Jahre war bei Koepe-Förderanlagen im deutschen Steinkohlenbergbau etwa jährlich ein übertreiben mit Sachschaden zu verzeichnen, jedoch ohne daß Menschen dabei zu Schaden kamen. Neun derartige Vorfälle, die sich in den Jahren 1981 bis 1987 ereigneten, wurden hinsichtlich ihrer Ursachen von den Sachverständigen der WBK-Seilprüfstelle untersucht.

Alle diese neun Fälle eines übertreibens ereigneten sich bei Förderzügen mit einer einhängend bewegten überlast und zwar unter folgenden Umständen, Tabelle 1:

Tabelle 1: Umstände unter denen sich die übertreiben ereigneten

Fälle A: der Einhängezug war gewollt

Fall A1:	einhängende überlast 43,0 t	statt	27,0 t	(+60%),	Materialtransport
Fall A2:	einhängende überlast 13,5 t	statt	8,4 t	(+60%),	Materialtransport
Fall A3:	einhängende überlast 11,0 t	statt	7,6 t	(+45%),	Materialtransport
Fall A4:	einhängende überlast 9,0 t	statt	7,0 t	(+30%),	Materialtransport

Fälle B: der Einhängezug war ungewollt

Fall B1:	Volles Gefäß einhängend	Umschaltung zwischen Automatik- auf Handbetrieb
Fall B2:	Volles Gefäß einhängend	Umschaltung zwischen Automatik- auf Handbetrieb
Fall B3:	Volles Gefäß einhängend	Abschaltung der Beladung
Fall B4:	Volles Gefäß einhängend	Anfahrt in falsche Fahrtrichtung bei Handbetrieb
Fall B5:	Volles Gefäß einhängend	Automatikbetrieb bei blockierter Sicherheitsbremse

Als Folgerung zeichnen sich zwei typische Situationen ab:

- gewollte, aber zu hohe Einhängelast beim Materialtransport (überbeladung)

- ungewollte Einhängelast bei Gefäßförderung (volles Gefäß einhängend)

VORGESCHLAGENE PROBLEMLÖSUNG

Eingehende überlegungen führten zu dem Ergebnis, daß eine zusätzliche elektronische Einrichtung an der Fördermaschine geeignet wäre, derartige Vorkommnisse in Zukunft zu vermeiden: Eine Einrichtung, die die Größe einer einhängenden überlast erkennt und ein Signal gibt, wenn die einhängend bewegte überlast größer ist als ein anlagenspezifisch vorgegebener Grenzwert: Bei Gestellförderanlagen für den Materialtransport in die Grube wäre dieser Grenzwert gleich der betriebsüblichen überlast, bei Gefäßforderanlagen, die planmäßig nur aufgehende überlasten fördern, wäre dieser Grenzwert auf eine wesentlich kleinere überlast einzustellen.

Dementsprechend können auch die Maßnahmen bei ansprechendem überwachungssignal abhängig von der Art der betrachteten Förderanlage festgelegt werden:

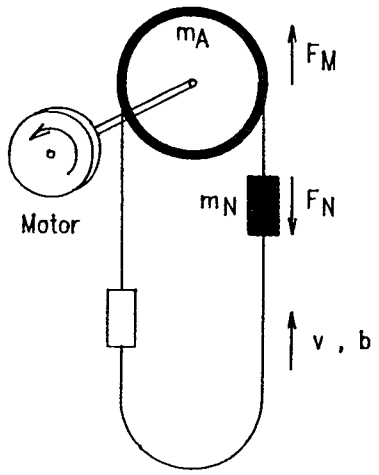
Da bei einer automatisch betriebenen Gefäßförderanlage Lasten normalerweise nur aufwärts gefördert werden, kann jede als einhängend erkannte überlast dadurch ausgeschlossen werden, daß der Antrieb unmittelbar nach dem Anfahren stillgesetzt wird, was sodann eine Behebung der Situation im Handbetrieb erfordert. Bei anderen Förderanlagen, die Lasten automatisch sowohl aufwärts wie abwärts bewegen, darf das überwachungssignal erst bei einer zu großen Einhängelast ansprechen und dann eine deutlich verminderte Geschwindigkeit oder ein Stillsetzen erzwingen. Sinngemäß sind unter den genannten Umständen bei Handbetrieb ein akustisches Warnsignal und eine Geschwindigkeitsbegrenzung vorzusehen.

THEORETISCHE ÜBERLEGUNGEN

Anhand des Schemabildes einer Koepe-Förderanlage und mit den Bezeichnungen nach Bild 1 lassen sich folgende Überlegungen anstellen, wobei alle Massen, Kräfte und Bewegungsgrößen auf das sich translatorisch bewegende Seil bezogen sind:

Die Antriebskraft des Motors F_M muß sowohl die Anlage mit all ihren (kräftemäßig ausgeglichenen) Massen m_A wie auch die (einseitige) Überlast mit ihrer Masse m_N um den Wert a beschleunigen und zudem das Gewicht der Überlast $\pm F_N = \pm m_N \cdot g$ kompensieren, wobei durch das Vorzeichen unterschieden wird, ob sich die Überlast im aufgehenden oder einhängenden Trum befindet:

$$F_M = \pm F_N + [m_A + m_N] \cdot a$$



F_M = Antriebskraft des Motors
F_N = Gewicht der Nutzlast
m_A = äquivalente Masse der Anlage
m_N = Masse gleich der Nutzlast
v = Geschwindigkeit
a = Beschleunigung
g = 9.81 m/s^2

Bild 1: Schemabild einer Koepe-Förderanlage und Darstellung der verwendeten Bezeichnungen.

Für die Überlast F_N ergibt sich daraus die Formel:

$$\pm F_N = F_M - [m_A + m_N] \cdot a$$

Soll nun die Überlast einen vorgegebenen Grenzwert $F_G = m_G \cdot g$ nicht übersteigen, so kann geschrieben werden:

$$\pm F_N = F_M - [m_A + m_G + (F_N - F_G)/g] \cdot a$$

Damit läßt sich die Formel zur Bestimmung der Überlast F_N vereinfachen auf

$$\pm F_N = F_M - [m_A + m_G] \cdot a \ ,$$

d.h. unter Hinnahme eines Fehlers $(F_N - F_G) \cdot a/g$, der jedoch bei dem vorgegebenen Grenzwert F_G der Überlast F_N wie auch bei einer Beschleunigung a = 0 zu null wird, und im übrigen bei den üblichen Anfahrbeschleunigungen $a < 1{,}5$ m/s^2 kleiner als 15 % bleibt.

Inbesondere zeigt die vorstehende Formel aber die Möglichkeit auf, wie eine kontinuierliche Überwachung der tatsächlichen Überlast geschehen kann, und zwar selbst in einer Beschleunigungsphase, wie sie beim und kurz nach dem Anfahren vorliegt.

TECHNISCHE AUSFÜHRUNG

Technisch ist eine solche Einhängelast-Überwachungseinrichtung durch eine elektronische Schaltung nach folgendem Prinzip zu verwirklichen:

Die Antriebskraft des Motors F_M ist bei konstanter Felderregung dem Motorstrom proportional und durch einen Spannungswert U_M darstellbar. Die Beschleunigung a ergibt sich als Ableitung der Geschwindigkeit durch Differenzieren des vom Tachogenerator gelieferten Signals U_V als $U_a = d(U_V)/dt$. Die zu beschleunigenden Massen, bzw. das Produkt $[m_A + m_G] \cdot a$, sind über ein Potentiometer mit x_A anlagenspezifisch auf $x_A \cdot U_a$ einstellbar. Um das Vorzeichen von F_N zu erkennen, muß zusätzlich noch die Drehrichtung des Antriebs bekannt sein, die sich aus dem Vorzeichen des Geschwindigkeitssignals als $SIG(U_V)$ gewinnen läßt.

Eine einhängend bewegte Überlast ($-F_N$), dargestellt durch einen Spannungswert $U_N < 0$, die den elektrisch als $U_G < 0$ vorgegebenen Grenzwert F_G übersteigt, ist dann eindeutig zu erkennen aus der Bedingung $U_N - U_G < 0$ oder:

$$[U_M - x_A \cdot U_a] \cdot SIG(U_v) - U_G < 0 .$$

ZUVERLÄSSIGKEITSANFORDERUNGEN

Die Zuverlässigkeitsanforderungen an solche Einhängelast-überwachungseinrichtungen lassen sich zahlenmäßig wie folgt kennzeichnen:

Der vorgenannte Erfahrungswert von etwa einem übertreiben pro Jahr bezieht sich auf eine Gesamtheit von grob 200 Förderanlagen, für die man eine jährliche Gesamtzügezahl in der Großenordnung von 10^7 veranschlagen kann. Das heißt, die vorgeschlagene Einhängelast-überwachungseinrichtung soll unerwünschte Betriebsbedingungen erkennen, die mit einer sehr geringen Wahrscheinlichkeit auftreten, oder anders ausgedrückt, bei einer bestimmten Förderanlage nur etwa einmal in einhundert Jahren zu erwarten sind.

Unter normalen Betriebsbedingungen ist die Einhängelast-überwachungseinrichtung ohne jeden Einfluß. Insofern würde ihr Funktionsausfall möglicherweise für lange Zeit unerkannt bleiben, außer der auftretende Fehler würde zur Abgabe eines Signales führen. Ein Auslegung nach Zuverlässigkeitsgesichtspunkten und regelmäßige Funktionsprüfungen sind infolgedessen unverzichtbar.

PRAKTISCHE FOLGERUNGEN

Mittlerweile wurden eine Anzahl solcher Einhängelast-überwachungseinrichtungen geliefert und an Koepe-Förderanlagen installiert. Von Sachverständigen der WBK-Seilprüfstelle bei ihrer Inbetriebnahme durchgeführte Untersuchungen bestätigten, daß sie in der gewünschten Weise funktionieren:

Nach Erreichen einer geringen, verfahrensbedingten Anfangsgeschwindigkeit, die das Differenzieren und die Vorzeichenermittlung des Geschwindigkeitssignals gestattet, wird eine unerwünschte Einhängelast sicher erkannt. Der Antrieb kann dann stillgesetzt oder in der Geschwindigkeit begrenzt werden, deutlich bevor die übliche, hohe Fördergeschwindigkeit erreicht ist.

Die beschriebene Einhängelast-überwachungseinrichtung trägt mithin dazu bei, die Sicherheit und die Verfügbarkeit von Schachtfördereinrichtungen zu gewährleisten.

SCHRIFTTUM

/1/ Analysierte Fälle eines übertreibens. Internes Papier der WBK-Seilprüfstelle. Unveröffentlicht.

/2/ Funktionsbeschreibungen für gelieferte Geräteausführungen der Firmen AAB, AEG und Siemens. Unveröffentlicht.

PROTECTION CONTRE LE RENVERSEMENT DES CONVOYEURS DU A DES CHARGES DESCENDANTES EXCESSIVES

E. Haibach

Sur la base de l'analyse de huit renversements de convoyeurs qui se sont produits ces six dernières années dans des mines allemandes, nous avons conclu qu'on aurait pu éviter ces renversements par l'emploi d'un dispositif de surveillance adéquat couplé au moteur électrique.

Ce type de dispositif de surveillance a été conçu à WBK-Seilprufstelle; il comporte un signal de commande fonctionnant sur le courant électrique du moteur et un signal de tension différencielle du tachymètre, proportionnel à la charge utile et à sa direction. Une tension de déclenchement actionne l'alarme pour n'importe quelle charge descendante spécifiée.

La procédure à suivre en cas d'alarme dépend de l'installation de hissage spécifique. Dans le cas d'une installation à skip, qui fonctionne en général avec une charge montante uniquement, on exclut toute charge utile descendante par un arrêt automatique de la commande tout de suite après le démarrage. En ce qui concerne les installations fonctionnant avec des charges utiles montantes et descendantes, l'alarme n'est actionnée que lorsque la charge descendante est excessive, ce qui restreint de manière substantielle la vitesse de hissage ou provoque la sonnerie d'une alarme acoustique manuelle.

Plusieurs installations ont déjà été mises en place et fonctionnent comme prévu.

ПРЕДОТВРАЩЕНИЕ ПРЕВЫШЕНИЯ СКОРОСТИ ПЕРЕГРУЖЕННЫМИ ТРАНСПОРТНЫМИ СРЕДСТВАМИ ВО ВРЕМЯ ДВИЖЕНИЯ ВНИЗ ПО УКЛОНУ

Е. Хайбах

Анализ восьми случаев аварий, связанных с движением накатом, происшедших на шахтах ФРГ, показывает, что аварии могли бы быть предотвращены, если бы электромотор был соединен со специальным контрольным устройством. Такое устройство было изобретено на ВВК-Сайлпруфштелле на основе токового контрольного сигнала от электромотора и дифференциального сигнала напряжения тахометра, пропорционального полезной нагрузке и ее направлению. Реле может задействовать аварийный сигнал в зависимости от заранее установленной нагрузки. Выбор противоаварийных мер в случае срабатывания аварийного сигнала зависит от особенностей установки данного оборудования. В случае скипового подъемника, используемого только для подъема грузов, подъемник автоматически останавливается сразу после старта, если груз начинает опускаться. Для устройств, работающих как для подъема, так и для спуска грузов, аварийный сигнал включается только в случае перегрузки при движении вниз, при этом автоматически резко уменьшается скорость лебедки или включается звуковой сигнал. Несколько устройств уже успешно работают на шахтах ФРГ.

保护由于向下输送超载而发生的输送机故障

E。海巴克

根据在德国发生的八次输送机故障分析结果，这些故障均可用连在电机上的监测装置来避免。这种监测装置是在ＷＢＫ－钢索试验所发展出来的。它是由马达里的电流所产生的控制信号和速度计里的微分电压信号来操作。速度计与净载量及其方向成比例。

向下净载量达到某一数值时触发电压便启动警铃。警铃启动后所应采取的步骤则因卷扬装置而异。倾卸设备一般只有向上载荷，开机后可由自动止动装置排除任何向下载荷。在容许向上及向下载荷的设备里，如果向下载荷超量过多则触动警铃，使卷扬机速度急剧减低，或用人工音响警告器。这些设备已在几处安装，而且效果良好。

FURTHER PROGRESS ON RESEARCH INTO LIGHTNING-RELATED INCIDENTS IN SHALLOW SOUTH AFRICAN COAL MINES

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ABSTRACT

Lightning causes premature detetonation of electrical detonators and the ignition of methane explosions in shallow collieries. These problems have been extensively investigated by the South African coal industry. The results and recommendations have been published elsewhere.

This paper reports on two measurement exercises which were instituted in order to obtain in-depth knowledge on the mechanisms through which lightning couples into the working area of a mine, and details the magnitude and statistical occurrence of surges underground caused by lightning.

The studies have resulted in the design of a simple measurement device which can be used to evaluate the risk in a particular mine and a method, which is still being developed, which will be used to evaluate the risk in a particular mine, based on the geological structure and earth resistivity analyses of the mine.

INTRODUCTION

The South African Coal Mining industry has experienced a considerable number of underground incidents or disturbances which were frequently related to the passage of lightning storms on the surface. These incidents included electrical shocks or visible sparking from underground equipment, premature detonation of explosives, and methane explosions. These incidents were particularly prevalent in shallow collieries and were of considerable concern to the various mining groups concerned. A number of wide-ranging investigations were intitated by certain of these companies, and these culminated in 1978 in the formation of a co-operative subcommittee of the Explosions Hazards Advisory Committee of the South African Coal Mining Research Controlling Council - with a view to carrying out an in-depth study of the problem and to formulating appropriate safety measures. Representation on the Lightning and Stray Current Sub-Committee was broadly based - including the participation (under the Government Mining Engineer) of the mining and explosives manufacturing industries - as well as electricity supply, railway and research organisations.

This work has been discussed in a previous paper (Geldenhuys et al 1985) and this paper reports on further results of this research and some of the progress made since 1985.

Studies were conducted in order to quantify the occurrence of lightning surges underground and to gain a better understanding of the mechanisms through which lightning couples into the strata. This paper reports specifically on the results of these measurement exercises. Two in-depth measurement studies were conducted:

- A study in which lightning surges were measured through sophisticated

oscillographic recording equipment which is capable of recording the waveshape of lightning currents induced in a typical underground electrical detonator circuit (Geldenhuys 1987a), correlated to lightning activity on the surface. The statistical sample in this study is very small but gives in-depth information on the coupling process.

- A study in which lightning surges were recorded by means of a simple fuse-based device called the Fuse Surge Detector (FSD). FSDs were installed in fifty dummy electrical circuits in five mines. This study provides a much larger statistical sample (Geldenhuys 1987b).

The work is further reinforced by a theoretical study of the phenomena involved (Zeh 1987).

MECHANISMS THROUGH WHICH LIGHTNING COUPLES INTO THE UNDERGROUND WORKINGS

It has been postulated (Berger 1977, Geldenhuys 1985) that two different mechanisms are capable of coupling lightning surges into the underground working areas. The two mechanisms are illustrated in Figure 1.

Mechanism 1: Lightning strikes the surface above the working area and the surge propagates through the strata. This mechanism is the most important one due to the fact that safety measures are not very effective in controlling its dangerous effects.

Mechanism 2: A strike to the structures at the shaft entrance occurs and is conducted into the working area via the metallic structures (pipes, conveyor belts and electric cables). The effects of this mechanism can be be effectively controlled by adequate earthing of the headgear at the shaft entrance.

ACCURATE MEASUREMENT OF SURGES

A project was launched by the Coal Mining Research Controlling Council to study the waveshape and magnitude of lightning surges underground. It was felt that this would lead to a better understanding of the coupling mechanisms, which, in turn, would enable the Council to devise improved safety measures. A measuring station was then erected to carry out a surge-recording programme.

Figure 2 shows a schematical diagram of the measurement station. This consists of an underground point where a dummy

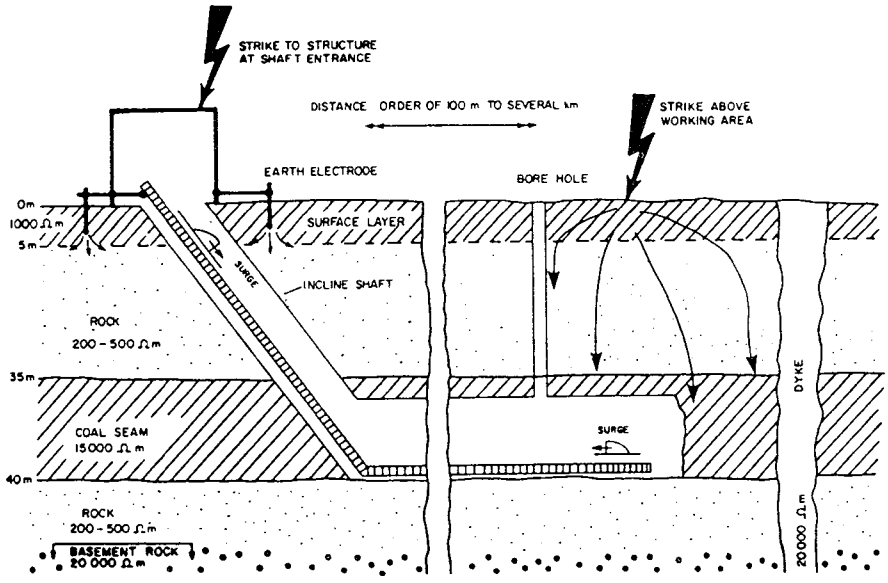


Figure 1: A simplified situation depicting how lightning surges may enter collieries.

electrical detonator circuit is located. Both voltage and current could be measured in this study, although Figure 2 shows the latest current measurement configuration. The underground instrument boxes are battery powered and "float" electrically, which enables measurements to be made without a remote earth connection interfering in the normal phenomena in such a detonator circuit. The signal is converted into an optical signal and relayed via non-conducting optic-fibre cables to the surface station.

The surface station contains digital oscilllographic recording equipment to record the underground signals with a time resolution of 100 ns per point. The optical signals are converted to electrical signals at the surface station. The lightning activity in the area of the station is recorded by means of a lightning flash counter, as well as with a TV time-lapse recorder with an all-sky (180°) recording ability. The lightning activity on the surface is correlated to the occurrence of surges underground by means of an event recorder. An example of such a recording is given in Figure 3. The top trace shows the surges that occurred and the bottom two traces show the occurrence of

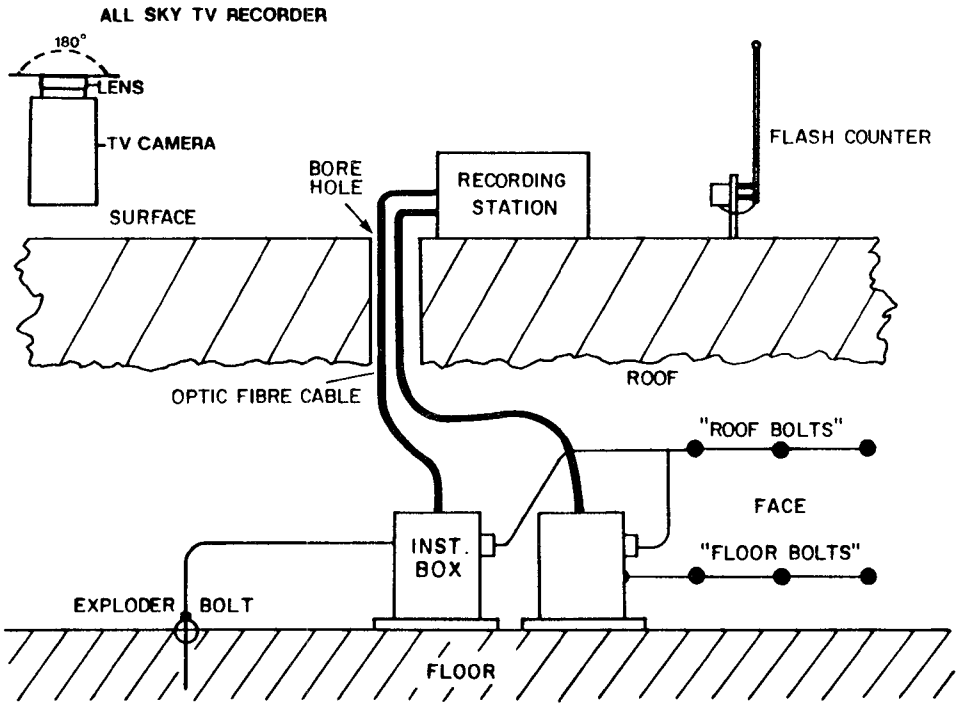


Figure 2: A schematic diagram of the underground measuring installation and the recording station on the surface.

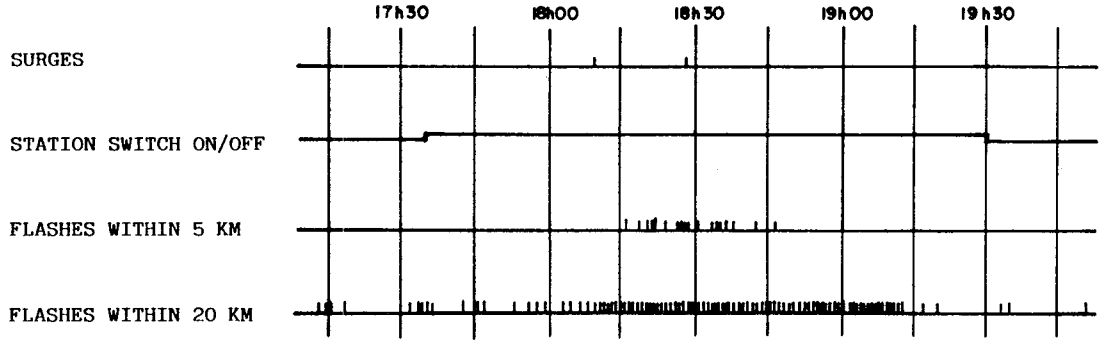


Figure 3: A typical event recorder registration.

lightning within a radius of 5 km and 20 km from the station. Figure 4 shows a typical current record. The waveshape of the current measured is very similar to that of lightning current, as observed in direct measurements (see Anderson 1980).

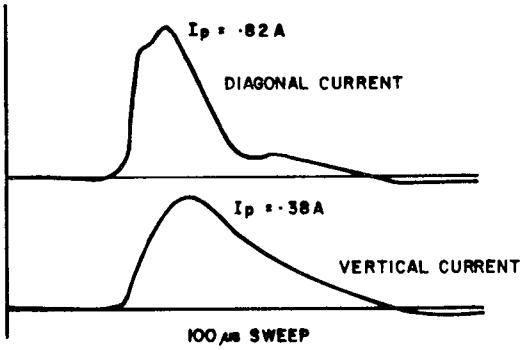


Figure 4: An example of a current pulse oscillogram recorded during the storm depicted in Figure 3.

The results of earlier records of the accurate measurement programme were discussed elsewhere (Geldenhuys 1987a) and only the main conclusions are summarised here:

- Surges due to Mechanism 1 were observed with measurements between the conveyor structure and a coal face, and also where measurements were made in the coal face within a typical (isolated) electrical detonator circuit.

- As can be expected, surges due to Mechanism 2 were observed only where the measurements were made electrically relative to the conveyor structure. These surges were quite frequent but their amplitude was limited due to effective earthing of the gear at the shaft entrance.

The frequency of surges recorded in the typical electrical detonator circuit were analysed and plotted in Figure 8 where they are compared to both the simple theoretical model as well as to the results from the fuse surge detector (FSD) programme.

A SIMPLE LIGHTNING SURGE DETECTOR (THE FUSE SURGE DETECTOR)

It was realised that if the occurrence of lightning could be monitored in a mine, this information could assist mine managers in providing cost-effective and appropriate safety measures to avoid lightning-related accidents. Also, the information obtained from these measurements will advance the general understanding of the problem of lightning stray currents underground. A device to monitor the occurrence of lightning underground had to be both cheap and easy to use in order to make it generally acceptable to the coal-mining industry. A device was designed which complies with the above objectives and is described below:

The FSD makes use of fuse wire of different diameters, each of which fuse at a given level. These levels can be compared directly to the level at which the electrical detonators (generally used in South African industries) will ignite. Figure 5 shows the PC board layout of the FSD.

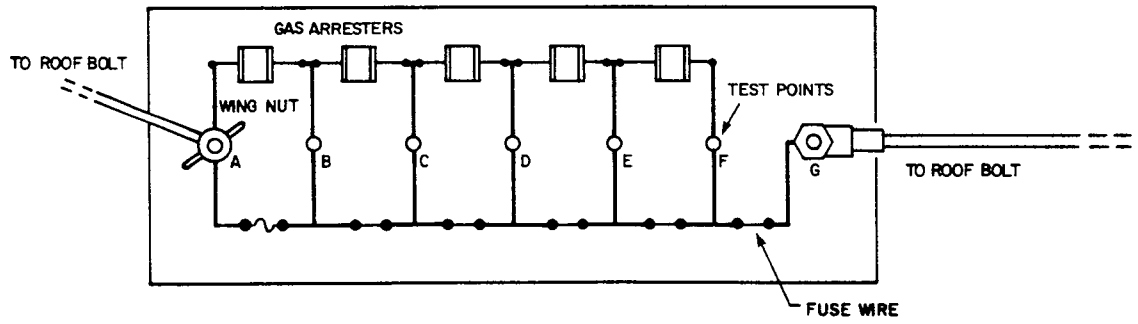


Figure 5: Drawing of the physical layout of the fuse surge detector (FSD). The FSD comprises six fusable wires in series, with gas arresters in parallel to five of the six fuses. (The electronic circuit is not shown.)

Table 1: Fuse current rating of the different fuses in the FSD

Fuse No.	Fuse type	Fuse current (1.2/50 μs current impulse)
A-B	IC aluminium track	± 3 A peak
B-C	Φ = 25 μm nichrome wire	± 7 A peak
C-D	Φ = 30 μm nichrome wire	± 10 A peak
D-E	Φ = 36 μm nichrome wire	± 20 A peak
E-F	Φ = 61 μm nichrome wire	± 40 A peak
F-G	Φ = 91 μm nichrome wire	± 90 A peak

The FSD consists of a series of six fuses with gas arresters (spark gaps) in parallel to five of the six fuses. The fuses are referenced to a 1.2/50 μs waveshape current impulse with sensitivies as shown in Table 1.

The FSD is connected in series with any circuit where lightning current is to be detected. If a lightning surge exceeding the equivalent of, say 25 A (1.2/50 μs), is passed through the FSD, it will fuse the four fuse wires between A and E. As soon as a fuse wire breaks, the path for the current flow disappears - this problem is solved by the gas arresters - these will flash over into a short circuit and will allow the current to continue to flow.

The use of FSDs was further simplified by the addition of a on-board battery-driven electronic circuit. This circuit causes an LED to flicker when a fuse is blown. This makes continuity testing unnecessary. The LED will flicker for a period of at least one month having a shelf life of five years.

In order to prevent corrosion and for mechanical strength, the FSD is encapsulated into an epoxy resin with stainless steel connection points for connection and continuity testing.

THE INSTALLATION OF FSDs IN COLLIERIES

FSDs were installed in a number of mines in South Africa in 1986. Areas which were were regarded as being particularly exposed were selected in each mine and two fuses were rigged up in each area, as shown in Figure 6.

The arrangement of the FSD at the first face shown is aimed at determining effects occurring at the face, whereas the FSD arrangement at the second face shown will supply information on the interaction between the face and the exploder point.

The results obtained from these measurements, to date, are summarised in Table 2 below and are also plotted in Figure 8 against the theoretical model's prediction.

Table 2: Results obtained by means of the FSD programme
(The total number of fuses x flash density over the programme period was equal to 676 fuses X flashes/km^2)

Fuse Rating	3A	7A	10A	20A	40A	90A
Cumulative number blown (1.2/50 μs)	17	8	5	3	3	3
Normalised to 1 flash . face . km^{-2}	.025	.012	.0074	.0044	.0044	.0444

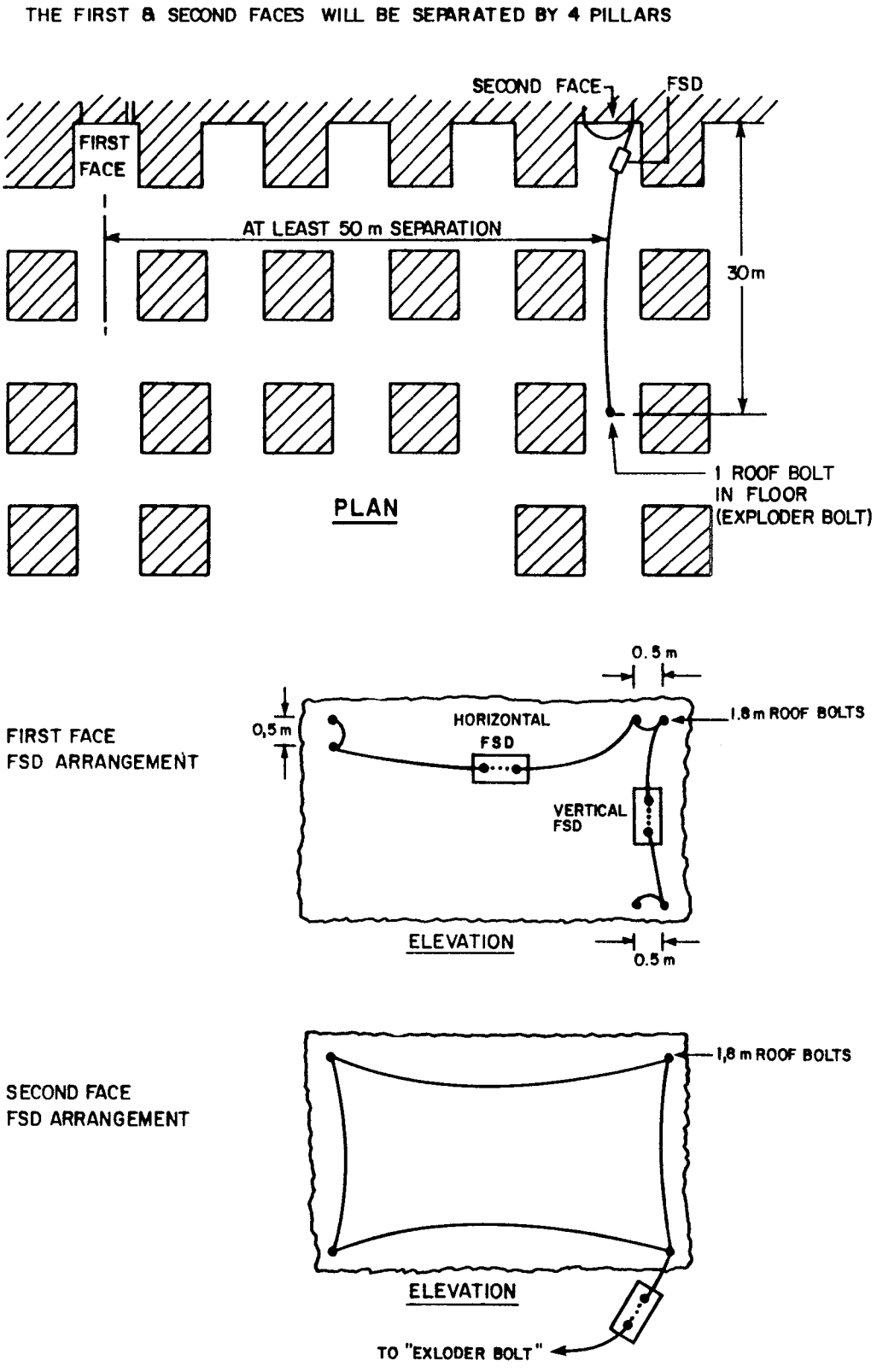


Figure 6: The circuit configurations implemented at five areas in each of the five mines.

A SIMPLE THEORETICAL MODEL OF SURGE FREQUENCY

It has already been pointed out in the introduction that conduction through the strata is regarded as the most important way in which lightning couples into the working area (Mechanism 1). This situation was modelled in the most simple way possible, as shown in Figure 7.

A more detailed description of this model is given by Geldenhuys et al (1987b). The results of the model are given in Figure 8.

The model has two asymptotic trends which are of great importance:

- At relatively small current magnitudes, the frequency (N) is inversely proportioned to the current (I) amplitude (N α 1/I) and is independent of the depth of the coal seam.

- The current at a given depth has a maximum value. Figure 8 shows a model predicting for both 20 m and 80 m and it can be seen that 400 A and 100 A are unlikely to be exceeded at the respective depths.

COMPARISON BETWEEN THE MODEL AND THE MEASUREMENT PROGRAMMES

The results obtained at the Accurate Measurement Station as well as from the Fuse Surge Detector Programme are plotted in Figure 8.

Both the measured results strongly support the (N α 1/I) inverse proportion relationship between the frequency and surge magnitude. The fact that the measured results and the model does not exactly coincide is to be expected. Some of the simplifying assumptions of the model are somewhat crude and this difference is also to be expected. Nevertheless the relative close match is encouraging.

CONCLUSIONS

The studies carried out since 1985 have confirmed the belief that

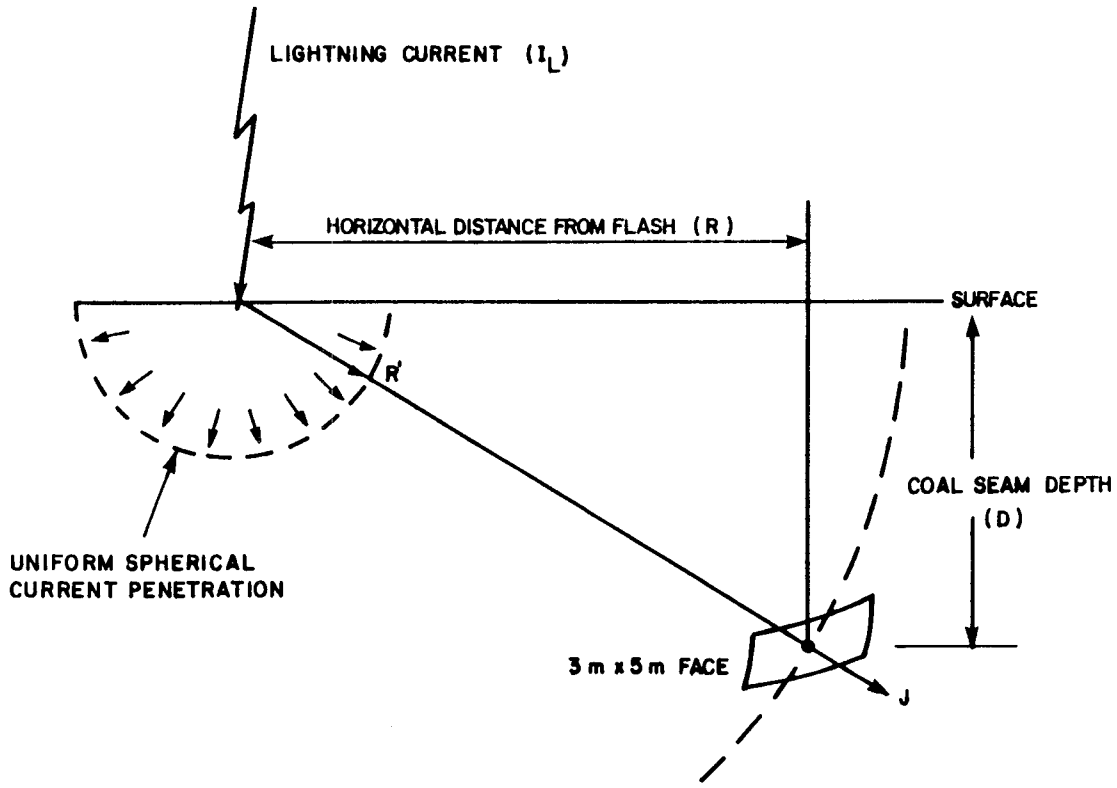


Figure 7: The model and its parameters.

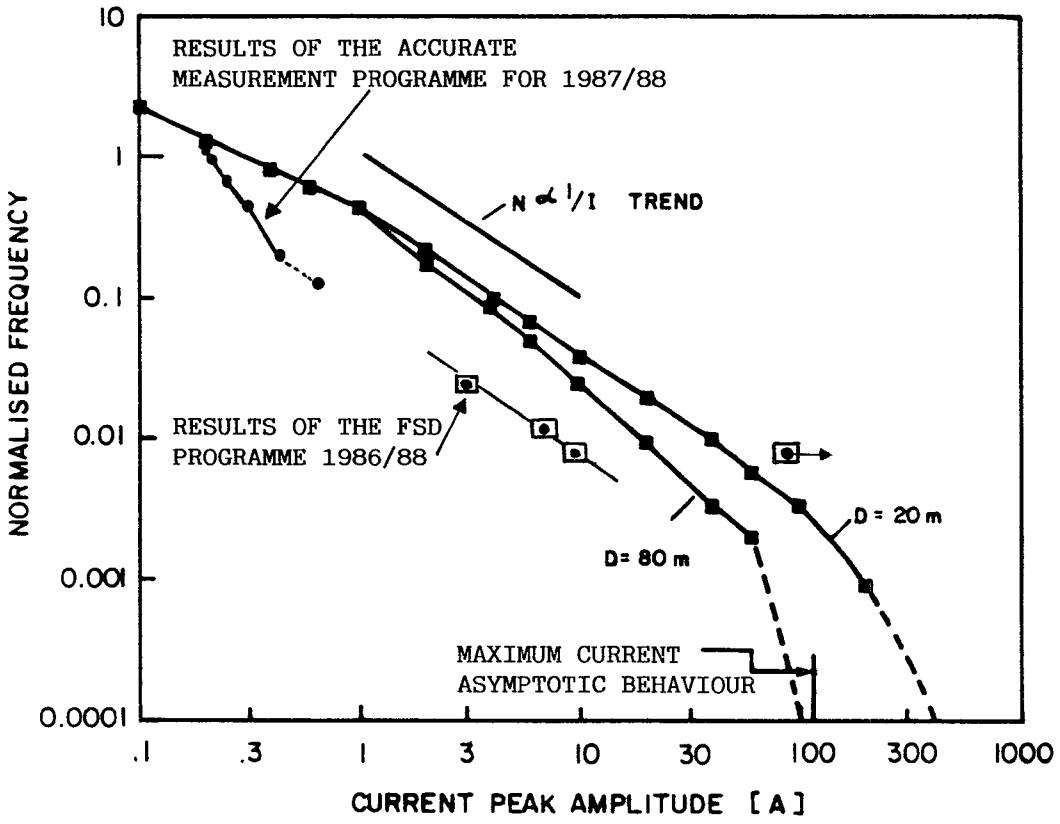


Figure 8: The frequency of currents obtained through a Monte Carlo simulation for faces at 20 m and 80 m depth respectively. The results are for a study with Ng = 10 000 normalised to Ng = 1 and one face.

Mechanism 1 is the most important way in which lightning enters mines (provided that normal safety measures are applied.)

A cheap method was developed, namely the Fuse Surge Detector, to evaluate the exposure of a particular mine to lightning. Mine managers can use this information in order to decide on appropriate safety measures to be taken.

A method of analysing the lightning risk to a mine is under development as a result of these studies. The geological structure of the mine strata and the resistivity of the strata will be analysed in order to determine the risk factor.

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DERNIERS PROGRES CONCERNANT LES INCIDENTS CAUSES PAR LA FOUDRE DANS LES MINES DE CHARBON SUD-AFRICAINES PEU PROFONDES

H.J. Geldenhuys, K.A. Zeh, J.M. Kinghorn

Les recherches destinées à minimiser les risques causés par la foudre dans les chantiers souterrains ont été présentées à la 21ème Conférence Internationale des Instituts de Recherches sur la Sécurité Minière qui eut lieu à Sydney en octobre 1985. Le présent rapport fait état des progrès survenus depuis cette date.

Nous y décrivons un dispositif simple, peu onéreux et très efficace, qui a été mis au point pour évaluer la fréquence des poussées de la foudre sous le sol. Le relevé de ces fréquences permet de juger des risques auxquels une mine est exposée. Sur la base de ces informations, un directeur de mine peut décider s'il est ou non nécessaire de prendre de coûteuses mesures de précautions. Nous présentons également les résultats obtenus jusqu'à ce jour par ce dispositif.

Nous proposons, décrivons et examinons aussi la théorie d'une méthode possible à suivre dans une mine exposée au risque. Cette théorie prend en considération les caractéristiques de la foudre, de la structure géologique et des strates d'une mine. Nous donnons un exemple d'une telle évaluation.

WEITERE FORTSCHRITTE IN DER ERFORSCHNG VON BLITZSCHLAGBEDINGTEN VORFÄLLEN IN TAGESNAHEN GRUBEN IN SÜDAFRIKA.

H. J. Geldenhuys, K. A. Zeh, J. M. Kinghorn

Über die Forschungsarbeiten auf dem Gebiet der Minimierung der Blitzschlag-Gefahr wurde bereits während der Konferenz in Sydney im Jahre 1985 berichtet. Weitere Fortschritte werden in diesem Vortrag vorgestellt.

Ein einfaches, billiges und sehr effektives Gerät wurde zur Bewertung der Häufigkeit von blitzbedingten Stromstößen entwickelt und wird beschrieben. Von der aufgezeichneten Häufigkeit kann ein Risikofaktor für die Grube abgeleitet werden. Aufgrund dieser Information kann die Grubenleitung vernünftige Entscheidungen über die Notwendigkeit von kostspieligen Gegenmaßnahmen treffen. Die bisher mit dem Gerät gesammelten Erfahrungen werden beschrieben.

Die Theorie für eine mögliche Methode für eine gefährdete Grube wird vorgeschlagen, beschrieben und diskutiert. Diese Theorie berücksichtigt die Eigenschaften der Blitze, die geologische Struktur und die Schichtung in der Grube. Es wird ein Beispiel für eine solche Bewertung gegeben.

ПОСЛЕДНИЕ ДОСТИЖЕНИЯ В ОБЛАСТИ МИНИМИЗАЦИИ ОПАСНОСТИ, СВЯЗАННОЙ С УДАРАМИ МОЛНИЙ, В НЕГЛУБОКИХ УГОЛЬНЫХ ШАХТАХ ЮЖНОЙ АФРИКИ

Х.Дж. Галденхайя, К.А. Зе и Дж.М. Кингхорн

Результаты исследований по минимизации опасности связанной с ударами молний под землей были доложены на 21-ой международной конференции в Сиднее в октябре 1985 года. В настоящем докладе представлены новые результаты, полученные с того времени.

Описывается простое, дешевое и очень эффективное приспособление, которое было сконструировано с целью оценки частоты случаев удара молний под землей. Риск, которому подвергается каждая шахта, может быть выведен из этой частоты. На основании этой информации директор шахты сможет решить, нужно ли предпринимать дорогостоящие меры предосторожности. Описываются результаты, полученные с помощью этого приспособления.

В дополнение, описывается теория предлагаемого метода определения степени риска, которому подвергается отдельно взятая шахта. Эта теория учитывает характеристики молний, геологическую структуру и стратиграфию пластов в шахте. Приводится пример применения этой теории.

南非浅煤矿由于闪电引起的事故的研究续论

H. J. 盖尔顿海士, K. A. 陈, J. M. 金霍尔恩

在1985年10月悉尼召开的第二十一届矿业安全研究所国际会议上我们曾报告过对减少由于闪电引起的地下矿井危害的研究。本文是继那以后的研究成果。

本文叙述一个我们设计和制造的简单，廉价但又极有效的测量装置。此装置可以用来测定地下矿井闪电冲击的发生次数。矿井的危险性便可由测定的次数来衡量。根据其提供的资料，矿长可对是否有必要购置昂贵的防护设施作出明智的决策。本文报导了这种设备至今为止的使用效果。

此外本文还提出和讨论了一个发现矿井是否处于危险方法的理论。这理论考虑到了闪电的特征，地质结构与煤矿的地层性质，并举例说明了该理论的应用。

ESTIMATION OF SAFETY FACTORS ATTRIBUTABLE TO ELECTRODE MATERIALS IN THE SPARK TEST APPARATUS

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Bureau of Mines, Pittsburgh Research Center, USA

ABSTRACT

Underwriters Laboratories UL 913, a national intrinsic safety standard, and U.S. Mine Safety and Health Administration (MSHA) testing procedures specify that tungsten and cadmium electrodes be used in the spark test apparatus when conducting intrinsic safety tests. However, tungsten and cadmium surfaces are rarely found in mining environments; therefore, sparking seldom occurs between these materials. Data that define the relative probability of spark ignition between more common materials may estimate the actual explosion hazard more precisely for realistic mining situations. The Bureau of Mines Pittsburgh Research Center is estimating the probability of spark ignition for common electrode materials such as aluminum, brass, copper, lead, tin, zinc, and stainless and cold rolled steels in the standard spark test apparatus. By comparing currents that give the same probability of spark ignition at a given voltage or inductance, a safety factor may be associated with each material with cadmium assigned a safety factor of 1.00. Results to date are summarized.

INTRODUCTION

Methane gas is a natural by-product of coal mining. Unless diluted by adequate ventilation, methane may accumulate to an explosive level and pose a potentially disastrous problem. Two things are necessary for a gas explosion: the presence of an explosive fuel-air mixture and an ignition source. MSHA's ventilation requirements are intended to minimize the occurrence of explosive methane concentrations. It also has design criteria for electrical/electronic devices used in mining. Intrinsic safety design criteria, found in CFR 30[1], Part 18.68-69, are rather vague, and, in practice, UL 913[2], with some modifications, is used as the de facto standard.

MSHA's intrinsic safety approvals process has only one purpose: to determine if a particular circuit design, after two worst case failures, will ignite a methane-air atmosphere. The test now used to evaluate intrinsic safety gives a go/no-go indication that tends to ignore the probabilistic nature of electrical spark ignition. For example, a circuit under test (CUT) whose probability of igniting an 8.3% $\pm$ 0.3% methane-air mixture on any given spark is 1 in 10,000, will pass two consecutive 1600-spark

[1]U.S. Code of Federal Regulations, Title 30--Mineral Resources, Part 18, Section 18.68-69. July 1, 1984, pp. 111-113.

[2]Underwriter's Laboratories Standard 913, "Intrinsically Safe Apparatus and Associated Apparatus for Use In Class I, II, and III, Division 1, Hazardous Locations," 3rd Ed., February 18, 1980, p. 33, Underwriter's Laboratories, Inc., 333 Pfingsten Road, Northbrook, IL 60062.

intrinsic safety tests about 73% of the time. If its probability of ignition is 1 in 1,000, it will still pass two consecutive tests about 4% of the time. Clearly these levels of probability of ignition are not acceptably low. Fortunately, MSHA assesses a "safety factor" in its tests to compensate for component tolerances, slight gas mixture variations, atmospheric conditions, and other factors that may give erroneous test results.

One of the factors that significantly affects the ignition point of gases is the electrode materials causing sparking. Magison recognizes that different electrode materials give rise to substantial variations in ignition capability. He cites prior studies[3] that quantify this phenomena on the basis of ignition currents and voltages causing spark ignition in intrinsically safe circuits. The project described in this paper estimates, through an extensive experimental program, the probability of electrical spark ignition in simple resistor and inductor test circuits as a function of the materials comprising the spark electrodes. Knowledge of the probability of ignition allows MSHA and the designer to assess the effect that materials of construction have on ignition capability and to estimate the degree of accepted risk in each case. Through an understanding of spark ignition probabilities, we may be able to improve both our design and approvals processes.

EXPERIMENTAL PROCEDURE

The resistor circuit, as described in UL 913, is shown in figure 1. Test voltages of 5 to 50 Vdc would represent the most likely voltage range encountered in mine electronic

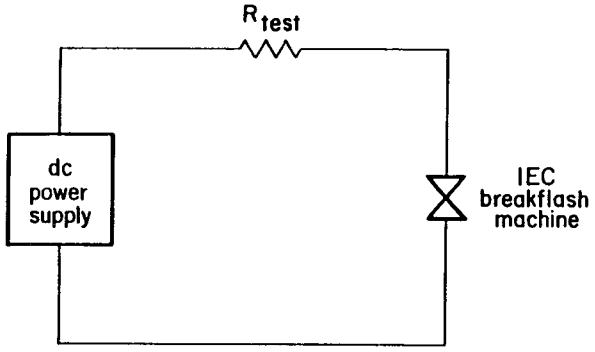


FIGURE 1. - dc Resistor Test Circuit

devices. However, the increased currents necessary to conduct testing below 20 Vdc could give rise to hot wire ignition. This effectively precludes testing at voltages below 20 Vdc. The inductor circuit, also described in UL 913, is shown in figure 2.

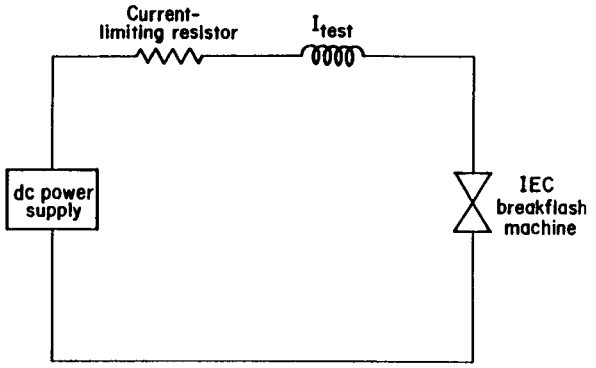


FIGURE 2. - dc Inductor Test Circuit

Inductances of 1, 10, 100, and 600 mH were tested for an open circuit voltage of 24 Vdc. For these voltages and inductances, various short circuit currents were tested using various disk electrode materials.

The test gas, delivered from the gas mixing system to the explosion chamber at a rate of 500 cc/min, consisted of 8.3% $\pm$ 0.3% methane, 20.0% oxygen, and the balance nitrogen. The spark test apparatus verification, detailed in UL 913, was conducted on a daily basis to

[3]Electrical Instruments in Hazardous Locations, E. C. Magison, 3rd Ed., Instrument Society of America, 67 Alexander Dr., P. O. Box 12277, Research Triangle Park, NC 27709, USA, 1978, pp. 86-89.

verify the integrity of this gas. To negate propagation of an ignition back into the gas mixing system, a flame arrester was installed on the inlet of the explosion chamber.

The electrode holder contained four round tungsten wire electrodes of .20 mm diameter and an unsupported length of 11 mm. At a speed of 80 rpm, the electrode holder counter-rotated against the disk electrode to create the sparks. The sparks and resulting ignitions were then electronically counted. In the event of an ignition, a pressure switch mounted on the explosion chamber automatically stopped the electrode drive spindles. The explosion chamber was then purged for 3-1/2 minutes to remove the residual gas before the drive spindles automatically restarted.

Because new disks are less likely to cause an ignition, each disk electrode required a 20-minute break-in period in a nonenergized circuit. This disk was then used for the duration of the resistor (or inductor) testing. The only exceptions proved to be tin and lead resistor circuit testing; their softness and their high ignition currents caused these materials to undergo an unusually high amount of pitting and wear. Therefore, two of each disk were used during testing. The tungsten wire electrodes were annealed prior to use. However, they were still susceptible to wear, splintering, and breaking. Therefore, they were replaced when visibly worn or after 4,500 sparks, whichever occurred first. If two or more wire electrodes failed simultaneously, testing was halted for electrode replacement.

The target probability of ignition, p_i, was somewhat arbitrarily chosen as 10^{-3}. This value does have its advantages in that for a confidence level of 98%, it was fairly easy to generate enough data because of the reasonable sample size. Also, past research[4] has shown that data may be extrapolated down into the area where the probability of ignition is 10^{-7}. In any event, the intent was to gather data points in the ranges slightly above and below 10^{-3} and then develop a predictor equation from the results. To calculate the sample size, the following equation was used:

$$S = (Z^2 * p_i * q) / d^2$$

where p_i is the target probability of ignition, q is the probability of not getting an ignition, $1 - p_i$, d is the acceptable probability of ignition error, 5×10^{-4}, and for a confidence level of 98%, the confidence coefficient, Z, equals 2.36. This indicates a sample size of approximately 22,000 sparks. Since the target p_i was 10^{-3}, 22 ignitions may be expected. When either 22,000 sparks or 22 ignitions occurred, a data set was considered complete and testing halted.

RESULTS & DISCUSSION

Resistor and inductor circuit test results are summarized in Tables 1-11. From these results, the plots of figures 3-9 can be constructed. These plots represent the logarithmically transformed current and probability data and have the form:

$$p_i = BI^A$$

where p_i is the probability of ignition, I is the ignition current in mA, A is the slope of the fitted line, and B is simply a multiplier. Previous Bureau research generated the cadmium curves shown in figures 3-9.

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[4]Cawley, J. C. Probability of Resistor Spark Ignition Caused by Very Low Currents, Bureau of Mines RI 9114, 1987, 11 pp.

TABLE 1. - I vs p_i for Aluminum

Current(mA)	Probability
30 V	
1803	3.18×10^{-4}
1873	9.13×10^{-4}
1973	8.64×10^{-4}
2100	1.01×10^{-3}
2520	9.75×10^{-3}
40 V	
950	4.54×10^{-5}
1080	1.36×10^{-4}
1200	2.11×10^{-3}
1360	5.94×10^{-3}
50 V	
847	1.36×10^{-4}
892	4.55×10^{-4}
980	1.11×10^{-3}
1108	1.54×10^{-3}

TABLE 2. - I vs p_i for Brass

Current (mA)	Probability
30 V	
1240	2.67×10^{-4}
1285	4.03×10^{-4}
1473	2.00×10^{-3}
40 V	
712	4.54×10^{-5}
820	6.77×10^{-4}
907	1.13×10^{-3}
1110	1.09×10^{-3}
50 V	
561	2.66×10^{-4}
604	4.55×10^{-4}
710	1.23×10^{-3}
910	2.01×10^{-2}

TABLE 3. - I vs p_i for CR Steel

Current(mA)	Probability
30 V	
3330	8.94×10^{-5}
3450	1.32×10^{-4}
3550	5.87×10^{-4}
3780	1.84×10^{-3}
40 V	
1785	8.59×10^{-5}
1800	1.79×10^{-4}
1877	1.80×10^{-4}
1900	2.69×10^{-4}
1966	5.00×10^{-4}
2000	1.11×10^{-3}
50 V	
1200	9.02×10^{-5}
1300	1.04×10^{-3}
1400	2.22×10^{-3}
1500	8.42×10^{-3}

TABLE 4. - I vs p_i for Copper

Current(mA)	Probability
30 V	
2770	9.09×10^{-5}
2830	7.72×10^{-4}
2900	1.16×10^{-3}
3000	1.38×10^{-3}
40 V	
1580	1.36×10^{-4}
1640	6.36×10^{-4}
1709	2.31×10^{-3}
2000	2.36×10^{-3}
2200	6.50×10^{-3}
50 V	
1110	1.82×10^{-4}
1191	5.45×10^{-4}
1305	9.65×10^{-4}
1462	7.83×10^{-3}

TABLE 5. - I vs p_i for Lead

Current (mA)	Probability
30 V	
1317	2.73×10^{-4}
1445	5.12×10^{-4}
1710	5.45×10^{-4}
2000	2.03×10^{-3}
40 V	
755	6.61×10^{-5}
798	1.71×10^{-4}
906	2.73×10^{-4}
1200	4.09×10^{-4}
50 V	
496	9.09×10^{-5}
562	2.34×10^{-4}
596	1.37×10^{-3}
632	1.06×10^{-3}
686	1.09×10^{-2}

TABLE 6. - I vs p_i for S Steel

Current(mA)	Probability
30 V	
2640	2.73×10^{-4}
2820	5.12×10^{-4}
2950	5.45×10^{-4}
3410	2.03×10^{-3}
3700	3.70×10^{-3}
3860	5.09×10^{-3}
40 V	
1696	6.61×10^{-5}
1800	1.71×10^{-4}
1900	2.73×10^{-4}
2050	4.09×10^{-4}
2200	7.23×10^{-4}
2500	9.20×10^{-3}
50 V	
1250	1.05×10^{-4}
1350	1.34×10^{-4}
1460	1.39×10^{-3}
1598	3.52×10^{-3}

TABLE 7. - I vs p_i for Tin

Current(mA)	Probability
30 V	
1912	4.50×10^{-4}
2110	1.81×10^{-4}
2230	3.63×10^{-4}
2450	5.73×10^{-4}
2660	7.32×10^{-4}
2860	1.28×10^{-3}
40 V	
1400	1.87×10^{-4}
1620	4.73×10^{-4}
1700	1.11×10^{-3}
1792	2.43×10^{-3}
50 V	
1006	3.18×10^{-4}
1050	6.96×10^{-4}
1106	3.13×10^{-3}
1200	8.98×10^{-3}

TABLE 8. - I vs p_i for Aluminum

Current(mA)	Probability
1 mH	
900	1.25×10^{-4}
1000	1.36×10^{-4}
1100	4.02×10^{-4}
1140	2.29×10^{-3}
1224	4.09×10^{-3}
1300	1.15×10^{-3}
10 mH	
520	2.23×10^{-4}
535	2.72×10^{-4}
550	1.30×10^{-3}
575	1.59×10^{-3}
100 mH	
160	2.53×10^{-4}
168	1.36×10^{-4}
171	1.26×10^{-3}
175	5.29×10^{-3}
200	9.09×10^{-2}
600 mH	
101	2.62×10^{-4}
105	6.91×10^{-4}
110	2.79×10^{-3}
115	8.91×10^{-3}

TABLE 9. - I vs p_i for Brass

Current(mA)	Probability
1 mH	
900	4.55×10^{-4}
950	1.18×10^{-3}
1001	1.83×10^{-3}
1306	1.61×10^{-2}
10 mH	
425	1.31×10^{-4}
433	6.73×10^{-4}
440	1.32×10^{-3}
450	1.20×10^{-3}
460	3.44×10^{-2}
100 mH	
137	1.82×10^{-4}
145	1.85×10^{-3}
150	3.21×10^{-3}
200	1.33×10^{-1}
600 mH	
95	9.08×10^{-5}
100	1.38×10^{-3}
105	4.63×10^{-3}
120	1.09×10^{-1}

TABLE 10. - I vs p_i for Lead

Current(mA)	Probability
1 mH	
825	6.55×10^{-4}
850	4.91×10^{-4}
875	1.01×10^{-3}
900	2.88×10^{-3}
925	1.26×10^{-3}
10 mH	
412	2.66×10^{-4}
425	6.42×10^{-4}
450	2.77×10^{-3}
500	1.88×10^{-2}
100 mH	
130	1.64×10^{-4}
133	4.08×10^{-4}
137	1.20×10^{-3}
140	2.23×10^{-3}
600 mH	
80	1.35×10^{-4}
83	5.45×10^{-4}
87	1.44×10^{-3}
90	3.82×10^{-3}

TABLE 11. - I vs p_i for Tin

Current(mA)	Probability
1 mH	
1120	1.60×10^{-4}
1180	1.29×10^{-4}
1233	3.17×10^{-4}
1290	1.14×10^{-3}
1356	6.36×10^{-3}
10 mH	
582	2.45×10^{-4}
600	7.47×10^{-4}
607	9.33×10^{-4}
627	7.33×10^{-3}
100 mH	
170	4.04×10^{-4}
176	4.65×10^{-4}
183.5	4.01×10^{-3}
190	1.80×10^{-2}
600 mH	
111	2.96×10^{-4}
112.5	9.94×10^{-4}
115	1.01×10^{-3}
130	6.36×10^{-2}

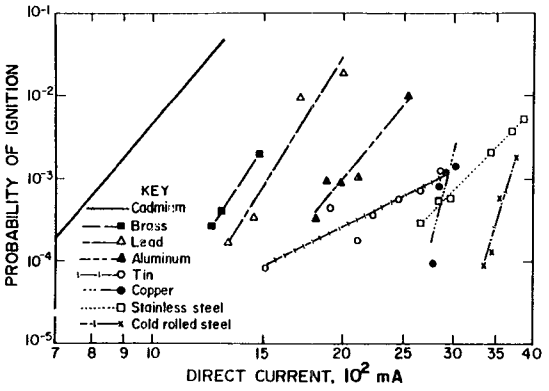


FIGURE 3. - Resistor Circuit Results 30 Volts dc

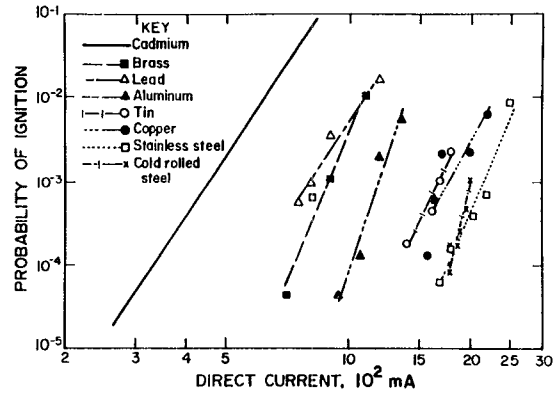


FIGURE 4. - Resistor Circuit Results 40 Volts dc

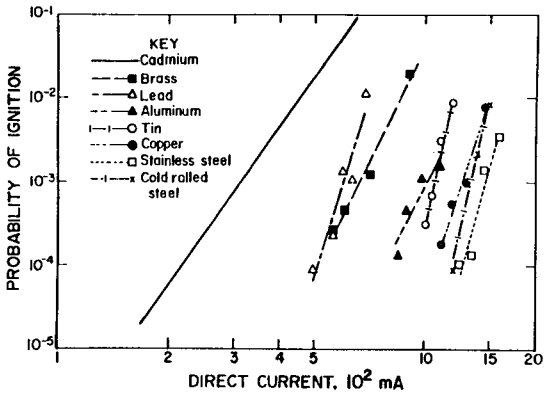


FIGURE 5. - Resistor Circuit Results 50 Volts dc

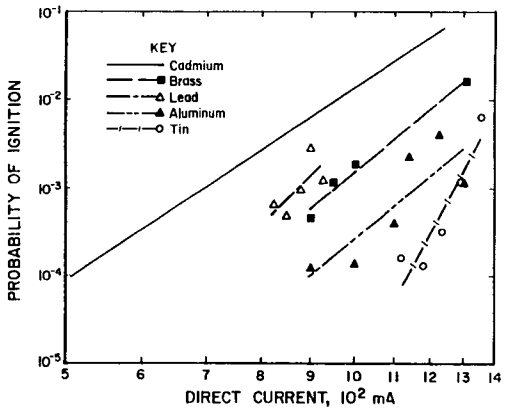


FIGURE 6. - Inductor Circuit Results 1 mH

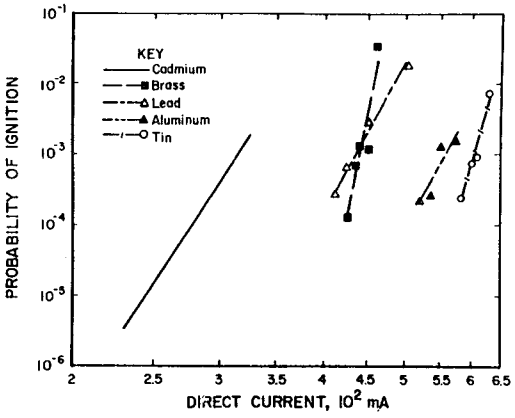


FIGURE 7. - Inductor Circuit Results 10 mH

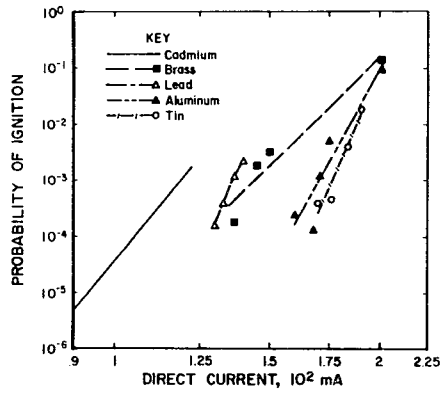


FIGURE 8. - Inductor Circuit Results 100 mH

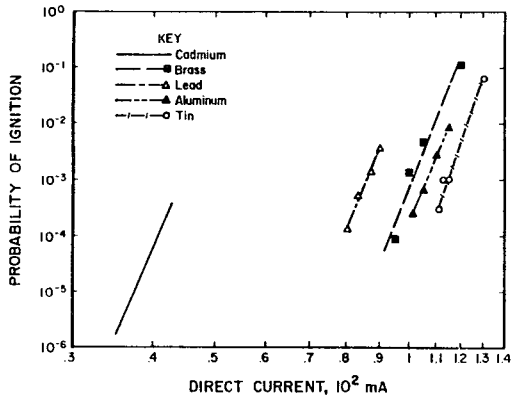


FIGURE 9. - Inductor Circuit Results 600 mH

Tables 12 and 13 contain values for the exponent A and coefficient B for the various disk electrode materials as well as the value I_e, the ignition current corresponding to a probability of 10^{-3}. Also listed is the safety factor for each material.

The safety factor is defined as the ratio of the ignition currents for a given disk electrode material and for a cadmium disk electrode that causes each to exhibit a probability of spark ignition of 10^{-3} under the same test conditions. For the

TABLE 12. - Safety factors for various common materials. Resistor circuits. Tungsten wire electrode, disk material as shown.

Disk Material	Voltage (Vdc)	A	B	I_e(mA)	Safety Factor
Cadmium	30	9.063	3.159×10^{-30}	839	1.00
Brass	30	11.705	1.649×10^{-40}	1388	1.65
Lead	30	12.321	6.237×10^{-43}	1520	1.81
Aluminum	30	9.197	4.375×10^{-34}	1999	2.38
Tin	30	3.798	7.812×10^{-17}	2828	3.37
Copper	30	30.503	1.857×10^{-109}	2926	3.49
S Steel	30	7.712	1.116×10^{-30}	3125	3.73
CR Steel	30	25.282	7.114×10^{-94}	3679	4.39
Cadmium	40	7.266	4.997×10^{-23}	453	1.00
Lead	40	7.306	6.256×10^{-25}	798	1.76
Brass	40	11.818	1.322×10^{-38}	894	1.97
Aluminum	40	14.573	1.545×10^{-48}	1189	2.63
Tin	40	10.071	3.287×10^{-36}	1680	3.71
Copper	40	9.028	4.852×10^{-33}	1767	3.90
CR Steel	40	18.429	1.169×10^{-64}	2024	4.47
S Steel	40	11.562	2.936×10^{-42}	2150	4.75
Cadmium	50	6.217	2.969×10^{-19}	314	1.00
Lead	50	14.388	1.079×10^{-43}	599	1.91
Brass	50	8.897	7.854×10^{-29}	663	2.11
Aluminum	50	8.435	4.281×10^{-29}	1017	3.24
Tin	50	19.465	1.226×10^{-62}	1063	3.39
Copper	50	13.033	3.627×10^{-44}	1267	4.04
CR Steel	50	19.416	2.051×10^{-64}	1336	4.25
S Steel	50	15.786	1.007×10^{-53}	1470	4.68

TABLE 13. - Safety factors for various common materials. Inductor circuits. Tungsten wire electrode, disk material as shown.

Disk Material	Inductance (mH)	A	B	I_e(mA)	Safety Factor
Cadmium	1	6.189	2.410×10^{-21}	702	1.00
Lead	1	10.792	1.845×10^{-35}	872	1.24
Brass	1	9.036	1.211×10^{-30}	953	1.36
Aluminum	1	9.084	1.497×10^{-31}	1157	1.65
Tin	1	20.136	3.129×10^{-66}	1271	1.81
Cadmium	10	18.082	6.687×10^{-49}	315	1.00
Lead	10	21.753	4.146×10^{-61}	434	1.38
Brass	10	59.951	3.342×10^{-167}	439	1.39
Aluminum	10	21.886	7.820×10^{-64}	558	1.77
Tin	10	45.153	2.880×10^{-129}	603	1.91
Cadmium	100	19.310	9.253×10^{-44}	118	1.00
Lead	100	35.404	2.493×10^{-79}	137	1.16
Brass	100	15.652	1.593×10^{-37}	144	1.22
Aluminum	100	29.077	1.243×10^{-68}	171	1.45
Tin	100	36.299	2.766×10^{-85}	177	1.50
Cadmium	600	27.416	7.825×10^{-49}	44	1.00
Lead	600	27.346	1.420×10^{-56}	86	1.96
Brass	600	28.730	2.692×10^{-61}	101	2.30
Aluminum	600	27.495	1.978×10^{-59}	106	2.41
Tin	600	32.120	7.965×10^{-70}	114	2.59

safety factors shown, cadmium was used as a reference material and assigned a safety factor of 1.00.

The effects of electrode materials on the probability of ignition may be estimated by

$$(P_1/P_2) = (I_1/I_2)^A$$

where P_1, P_2 are the probabilities of ignition at I_1 and I_2, respectively. This allows a circuit designer or approval authority to estimate the safety of a given circuit. As an example, test results indicate that for a 24 Vdc, 100 mH circuit, a current of 171 mA yields a probability of ignition of 10^{-3} for an aluminum disk electrode. To design for a probability of 10^{-6}, then

$$\left(\frac{10^{-3}}{10^{-6}}\right) = \left(\frac{171}{I_2}\right)^{29.077}$$

or the design current, I_2 is 135 mH. To design for a specific safety factor, such as 1.5

$$\left(\frac{10^{-3}}{P_2}\right) = \left(\frac{171}{\frac{171}{1.5}}\right)^{29.077}$$

or the design probability of ignition, P_2, is 7.58×10^{-9}. This information allows the user to design circuits for a given level of safety if a specific material is involved. The exponent A is both material and test condition specific. By testing to establish a current at a known probability of ignition, the designer may easily calculate other probabilities and/or currents associated with varying degrees of safety.

The boiling point of each disk material has been plotted versus its safety factor and is shown in

figures 10 and 11. The boiling point, BP, in degrees C, may be obtained from numerous reference publications. From this information, the safety factor of a material may be crudely estimated if its boiling point is known.

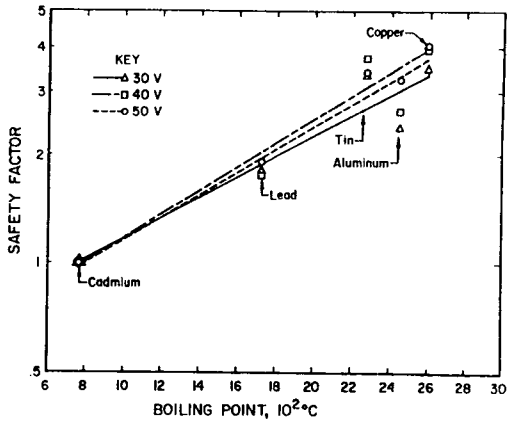


FIGURE 10. - Resistor Circuit Tests Safety Factor vs Boiling Point

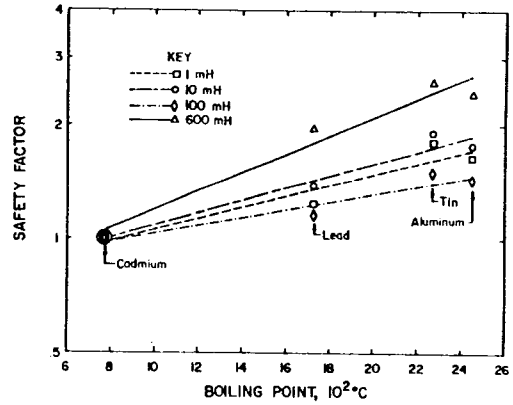


FIGURE 11. - Inductor Circuit Tests Safety Factor vs Boiling Point

Future plans are to complete inductor circuit testing for copper, stainless steel, and cold rolled steel disk electrodes. Thereafter, testing will be initiated using capacitor circuits. Possible wire electrode materials to be tested include copper, stainless steel and cold drawn steel. When completed, this work will provide a statistical basis for comparison of the ignition hazards arising from various commonly used materials.

SUMMARY AND CONCLUSIONS

The influence of disk electrode material on the probability of ignition has been quantified in this report. Spark ignition safety factors vary widely with cadmium requiring the lowest ignition current for both resistor and inductor circuits. If electrical apparatus can be designed to preclude spark ignition by cadmium electrodes, significantly higher currents may be permitted without sacrificing safety.

EVALUATION DES FACTEURS DE SECURITE ATTRIBUES AUX MATERIAUX D'ELECTRODES DANS LE DISPOSITIF D'ESSAI D'ALLUMAGE

J.S. Peterson et J.C. Cawley

Les procédures d'essais d'Underwriters Laboratories UL 913 -- organisme qui établit les normes américaines de sécurité intrinsèque -- et de l'Administration de la Sécurité et de la Santé Minières des Etats-Unis (U.S. Mine Safety and Health Administration -- MSHA) préconisent l'usage d'électrodes de tungstène et de cadmium dans les dispositifs d'essais d'allumage par étincelles lors d'essais sur la sécurité intrinsèque. Toutefois, étant donné que l'on trouve rarement des surfaces de tungstène et de cadmium dans l'environnement minier, il y a peu de chances qu'un allumage par étincelles se produise entre ces matériaux. Les données qui définissent la probabilité relative d'un allumage par étincelles entre des matériaux plus courants permettent d'évaluer de manière plus précise les risques effectifs d'explosions dans des mines réelles. Le Centre de Recherches du Bureau des Mines de Pittsburgh évalue la probabilité d'allumage par étincelles de matériaux d'électrodes courants, tels que l'aluminium, le laiton, le cuivre, le plomb, l'étain, le zinc, les aciers inoxydables et laminés à froid, dans un dispositif standard d'essais d'allumage par étincelles. La comparaison des courants qui donnent la même probabilité d'allumage par étincelles à une tension ou inductance données permet d'établir un facteur de sécurité relatif à tout matériau contenant du cadmium avec un facteur de sécurité de 1.00. Nous examinons les résultats obtenus jusqu'à ce jour.

ABSCHÄTZUNG DER SICHERHEITSFAKTOREN IM ZUSAMMENHANG MIT DEM ELEKTRODEN-MATERIAL IM FUNKENTEST-APPARAT

J. S. Peterson, J. C. Cawley

Underwriters Laboratories UL 913, eine im Lande gültige Eigensicherheits-Norm und die Testverfahren der U.S. Mine Safety and Health Administration (MSHA) schreiben die Verwendung von Wolfram- und Kadmiumelektroden bei Funkentests von eigensicheren Anlagen vor. Da allerdings Wolfram- und Kadmiumoberflächen nur sehr selten im Bergbauumfeld vorkommen, kommt es auch nur sehr selten zu einer Funkenbildung zwischen diesen beiden Materialien. Daten, welche die relative Wahrscheinlichkeit einer Funkenentzündung zwischen häufiger vorkommenden Materialien definieren, ließen eine realistischere Abschätzung der Explosionsgefahr im Grubenbetrieb zu. Das Bureau of Mines Research Center bestimmt die Wahrscheinlichkeit einer Funkenentzündung für häufiger auftretende Materialien, wie z. B. Aluminium, Messing, Kupfer, Blei, Zinn, Zink und rostfreien sowie kalt gezogenen Stahl in dem standard Funkentest-Apparat. Werden die Ströme, welche die gleiche Wahrscheinlichkeit der Funkenentzündung bei einer vorgegebenen Spannung oder Induktivität ergeben, miteinander verglichen, so kann für jedes Material ein Sicherheitsfaktor vergeben werden, ausgehend davon, daß Kadmium den Faktor 1,00 erhält. Die bis heute vorliegenden Resultate werden zusammengefaßt.

ОЦЕНКА ЭЛЕКТРОДНЫХ МАТЕРИАЛОВ В ИСКРОИСПЫТАТЕЛЬНОМ АППАРАТЕ С ТОЧКИ ЗРЕНИЯ ТЕХНИКИ БЕЗОПАСНОСТИ

Д.С. Петерсон и Д.С. Коули

Стандарт UL 913, установленный организацией "Андеррайтерс Лаборатори", стал национальным внутренним стандартом безопасности. Управление по технике безопасности в шахтах США в своих правилах для испытаний указывает, что вольфрамовые и кадмиевые электроды должны использоваться в искровых испытательных аппаратах при проведении испытаний на безопасность. Однако, вольфрамовые и кадмиевые поверхности редко встречаются в условиях шахт, поэтому искрение между этими материалами является редким явлением. На основе данных, определяющих возможность искровых вспышек между более типичными материалами, можно точнее подсчитать действительную взрывоопасность в реальных шахтных условиях. Питсбургский исследовательский центр Горного Бюро США проводит работы по определению вероятности искровых вспышек для таких распространенных электродных материалов, как алюминий, латунь, медь, свинец, олово, цинк, нержавеющая сталь и холоднопрокатная сталь в стандартных искровых испытательных аппаратах. Сравнивая токи, дающие одинаковую вероятность искровых вспышек при заданном напряжении и индуктивности, может быть определен фактор безопасности каждого материала по сравнению с кадмием, фактор безопасности которого принят за 1.00. Подводятся итоги результатов, полученных до настоящего времени.

火花试验设备中关于电极材料安全系数的估计

J。S。彼得生 及 J。C。考雷

保险实验室UL913，一项国家的固有安全标准，及美国矿业安全卫生署（MSHA）试验规划规定举行固有安全试验必须用钨及铬电极。但矿场很少发现有钨和铬的表面，所以很少有火花是由这些金属所引起的。用比较常见的物质间所引起火花的可能性的数据估计实际煤矿中爆炸的危险性恐怕远较准确。矿业局匹兹堡研究所用铝，黄铜，铜，铅，锡，锌，不锈钢和冷扎钢这些常用的电极材料在标准的火花试验设备里作了火花引发可能性的估计。在一定的电压或电感值来比较引起同样火花引发可能性的电流，我们便可的到每一种材料的安全系数。铬的安全系数指定为1。00以作标准。结果在本文中概括叙述。

Session XIV

NEW TECHNOLOGIES, II

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TECHNOLOGICAL EVOLUTION: A NECESSITY TO IMPROVE SAFETY LEVELS IN SPANISH QUARRY OPERATIONS

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ABSTRACT

Quarry mining has always been a constant activity in the industrialized world to supply the needs for buildings materials. In most cases, quarries are exploited by small to medium-size companies, hence those companies suffer from the drawbacks and limitations implied by their size. The paper analyzes the activity of quarry mining within the Spanish mining industry and establishes a comparison with the equivalent from other countries of the EEC. The work includes an analysis of safety problems, their statistics and causes. As a conclussion, the authors suggest certain technical actions to be taken both by private companies and the goverment in order to improve quarry's output and safety levels.

1 INTRODUCTION

Quarrying stone was man's first mining operation, in order to produce tools during the prehistoric period. Since these ancient times quarry workings have supplied society with basic products for housing, infrastructure, etc. Common generic principles have been maintained in these thousands of years of quarry working: surface operations with little or no burden, simple working systems, very little or no subsequent processing of the product, generally small production units, which frequent employ few people. Current examples of the oldest systems can be seen in quarries producing granite cobblestone.

This mining subsector of quarries - with very similar features in the Mediterranean countries - faces twin requirements from modern society:

- Improved working conditions; and
- less environmental impact.

The only way that these twin objectives can be met is by technology improvements and the technification of working and processing systems which must, furthermore, adequately respond to market demand for higher quality products.

This article will analyse the features of this subsector of spanish mining and its safety problems, in order to identify action which may be taken to advance quarrying technology.

2 DEFINITION AND SIGNIFICANCE OF THE QUARRYING SUBSECTOR

Mining is a traditional spanish industry producing ptas. 360 billion every year. This is equivalent to a little more than 1% of G.D.P., employing 80,000

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people (2% of the industrial workforce) and moving an annual investment of over ptas. 65 billion (16.5% of the value of saleable output). This gives a ratio of ptas. 0.9 million per worker.

The work is carried out in more than 3,500 companies, i.e an average production unit of 23 employees.

This activity is divided into four main subsectors, with the following specific weightings (tables nº 1 and nº 2).

Table nº 1. Breakdown of mining activity into subsectors (1986)

	Energy mining	Metal mining	Non-metallic mining	Quarry products	TOTAL
Output value in Ptas. million	205.600	53.000	45.100	55.400	359.100
% of Total	57%	15%	13%	15%	100
Nº of workers	52.100	7.800	6.400	14.400	80.700
% del Total	64%	10%	8%	18%	100
Annual investment in Ptas. million	33.500	12.600	7.800	3.400	57.300
Workings	250	60	260	2.950	3.520
% of Total	7%	2%	7%	84%	100

Tabla nº 2. Size of mine workings (1986)

Nº Workers	1-49	50-99	100-499	>500
Energy minerals	62,9%	12,0%	18,3%	6,8%
Metal ores	50,0%	17,9%	25,0%	7,1%
Non-metallic ores	92,2%	4,0%	2,3%	1,5%
Quarry products	99,2%	0,7%	0,1%	-
TOTAL MINING SECTOR	95,3%	2,0%	1,9%	0,7%

The quarry subsector defines the average operation as follows:

- Number of workers = 5 (4.6)
- Annual output value = ptas 18.8 million ($0.165 million)
- Annual investment = ptas 1.15 million ($0.036 million)
- Value of output per worker = ptas 4.1 million ($0.036 million)
- Investment per worker = ptas 0.25 million ($0.002 million)

However, if we analyse trends in production, employment and the number of workings in recent years, it can be seen that quarrying is the only subsector that is maintained. Specifically in employment, with an annual loss of some 2,000 jobs for mining overall, it is the only subsector still generating employment (Tables nº 3, 4 and 5).

Table nº 3. Value of output

Years	Energy mining	Metal mining	Non-metallic mining	Quarry products	TOTAL	
1984	172.100	73.200	40.500	42.200	320.000	-
1985	194.400	69.700	43.900	48.000	356.000	36.000
1986	205.200	53.000	45.100	55.400	358.700	2.700

In Pta. million

Table nº 4. Employment trend

Years	Energy mining	Metal mining	Non-metallic mining	Quarry products	TOTAL	
1984	53.788	9.783	7.621	14.067	85.259	-2404
1985	53.258	8.805	6.753	14.335	83.151	-2108
1986	52.166	7.767	6.414	14.416	80.763	-2388

Employment at each year end

Table nº 5. Trend in the number of workings

Years	Energy mining	Metal mining	Non-metallic mining	Quarry products	TOTAL	
1984	258	78	301	3.034	3.671	
1985	262	71	301	2.981	3.615	- 56
1986	253	56	256	2.983	3.508	- 107

The characteristics of the subsector approach those of our neighbouring countries, as can be seen in the data given in Table nº 6 below:

Table nº 6. Quarrying data for several EEC countries

	France	Italy	W. Germany	U.K.	Belgium	Spain
Nº of quarries	7.232	5.899	3.243	3.164	710	3.034
Employment	25.120	27.080	139.732	32.157	12.474	14.067
Nº of workers/ quarry	3,5	4,6	43,1	10,2	35,2	4,6
Date of information	1984	1983	1984	1984	1983	1984

This means that quarries are usually run small and medium sized businesses, not only in Spain, but throughout Europe. This business solution has its disadvantages, but it undoubtedly has its advantages and it could be argued that this is the industrial response to the supply of raw materials such as limestone, dolomite, sand, gypsum, clay, slate, marble, granite, etc.

3 SAFETY APPRAISAL

A fundamental aspect when evaluating mining safety are the accident statistics. Analysis of frequency index trends over time and comparison with date from other countries and sectors is essential for this purpose and for the promotion of the adoption of specific measures or the advisability of researching into certain aspects of accidents.

Unfortunately, this analysis is incomplete for spanish quarries, due to the lack of sufficiently realiable statistics. This problem arises from different accident classification criteria combined with date limitations.

Data for mortal accidents are the most reliable. Therefore, on the basis of statistics from the Directorate General of Mines at the Ministry of Industry and Energy, summarised table nº 7 has been drawn up showing the number of mortal victims in mining for 1980-86.

With this information and the data relating to hours worked for each subsector, table nº 8 was prepared, giving the mortal accident frequency index (nº deaths/1,000,000 hours worked).

Table nº 7. Mortal victims for the period 1980-86

	1980	1981	1982	1983	1984	1985	1986
COAL	32	44	41	41	42	55	29
METAL AND NON-METALLIC MINING	9	10	14	12	8	15	2
QUARRIES	10	15	18	9	7	5	7
TOTAL	51	73	62	57	75	75	38

Tabla nº 8. Mortal accident frequency index

	1980	1981	1982	1983	1984	1985	1986
COAL	0,40	0,53	0,47	0,48	0,52	0,68	0,37
METAL AND NON-METALLIC MINING	0,23	0,27	0,41	0,41	0,27	0,55	0,08
QUARRIES	0,37	0,59	0,74	0,40	0,32	0,22	0,30

Despite data variability over the period in question, the conclusion cannot be drawn that significant statistical differences exist in the mortal accident frequency indices of each subsector. It is still surprising to note that subsectors like quarries which, objectively speaking, pose fewer technical difficulties for the application of preventative measures, reach accident indices comparable to the coal subsector, where there are many additional safety factors in comparison with the former.

Unfortunately, there are currently no European statistics providing EEC information to enable these data to be compared with other neighbouring countries. Nevertheless, from information obtained at the International Symposium on Mining and Quarry Safety and Hygiene organised by the EC Commission, it may be deduced that the European average frequency index lies between 0.18 and 0.25. This means that indices in our quarries normally exceed this figure.

Without a detailed breakdown of accident statistics it is difficult to objectively analyse the causes; however, valid conclussions may be drawn from the mortal accident reports which, to a certain extent, provide an idea of the risk situation. From studies carried out by the Higher Council at the Ministry of Industry and Energy it can be deduced that the main causes of accidents, in order of importance, are:

* Transport
* Landslides
* Machinery
* Personal falls, and
* Explosives.

Transport is the most important factor, for the following principal reasons:

* Machinery tipping over
* Falls down slagheaps or embankments
* Being run over

These causes coincide with statistics from other EEC countries, and are generally of the same order of importance.

Over the last few years, and as part of the Mining Safety Campaign, over 200 quarries were visited working different products and in different spanish regions. The work has not as yet been completed and official information does still not exist; however, the recommendations given in the partial reports concentrate on the following points, given in order of importance:

* Work platforms and road
* Access roads and signalling
* Bench height
* Face gradient
* Walls
* Lack of specialist personnel
* Blasting

Once again this confirms the coincidence of risk situations with the concentration of the causes of accidents.

4 CONCLUSSIONS AND ACTION PROPOSED

There are various factors that determine the situation described in the previous section: firstly, structural

factors associated Government, business structure, etc.; sencondly, technical factors such as work rationalisation, mining legislation, training, etc. Properly attending to these factors may well lead of improved safety levels in this subsector over the medium term.

The business structure requires a twin approach: one for small and medium businesses, and another considering the mining aspect. Undoubtedly these actions will take place through the business associations which also defend common interests, which should guarantee the following safety aspects:

- An adeaquate technical level at the engineering level via specific design projects for the working, transport, processing and restoration. It is not profitable to carry out this operation at an individual level.

- Technical monitoring of these projects be a team of specialists to define operating conditions. This follow-up should reach certain mines down to the level of minng watchman.

- A search for companies specialising in mining operations. An ideal example is boring, charging and blasting, which is not profitable for quarries with between one and six blasting operations per year.

- Proper technical drafting of accident information, events, statistics, etc. by a team of specialists.

The Goverment should act in the following area:

- Drafting of mining safety legislation adequate for this type of working, providing the subsector with solutions to its problems, e.g. supervision of the workings, authorisation of standard projects common to many quarries.

- Follow-up of the mining and restoration plans, with a methodology different to that of the working plan.

- The requirement of reliable reports on accidents, in order to prepare adequate statistics.

- The creation of professional training programmes at both secondary and tertiary levels, adapted to the sector (which is a generator of employment).

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EVOLUTION TECHNOLOGIQUE: NECESSITE D'AMELIORER LES NIVEAUX DE SECURITE DANS L'EXPLOITATION DES CARRIERES EN ESPAGNE

C. Martinez Diaz et J. Obis Sanchez

L'exploitation des carrières se poursuit constamment pour satisfaire les besoins en matériaux de construction du monde industrialisé. Dans la plupart des cas, ce sont des petites et moyennes entreprises qui exploitent les carrières; ces entreprises souffrent donc des inconvénients et des contraintes causés par leur taille. Notre rapport examine l'exploitation des carrières dans le cadre de l'industrie minière espagnole; il établit également une comparaison avec les autres pays de la CEE. Il effectue une analyse des problèmes de sécurité, qui comporte leurs statistiques et leurs causes. Pour conclure, les auteurs suggèrent certaines mesures techniques à prendre, tant par les compagnies privées que par les organismes gouvernementaux, pour augmenter le rendement des carrières et améliorer leurs niveaux de sécurité.

TECHNOLOGISCHE EVOLUTION: EINE NOTWENDIGKEIT ZUR VERBESSERUNG DES SICHERHEITSNIVEAUS IN SPANISCHEN STEINBRUCHBETRIEBEN

C. Martinez Diaz, J. Obis Sanchez

Schon immer werden in der industrialisierten Welt Steinbrüche betrieben, um den Bedarf an Baumaterial zu decken. In den meisten Fällen werden Steinbrüche von kleinen Gesellschaften ausgebeutet, d.h. sie leiden auch unter den Nachteilen und Einschränkungen, die ihre geringe Größe mit sich bringt. Der Vortrag analysiert die Aktivitäten der Steinbruchbetriebe in Spanien und vergleicht diese mit den entsprechenden Betrieben in der Europäischen Gemeinschaft. Die Arbeit beinhaltet eine Analyse der Sicherheitsprobleme, ihrer Statistik und Ursachen. Als Folgerung schlagen die Autoren dem privaten Unternehmer und der Regierung gewisse technische Maßnahmen vor, durch die sowohl die Leistung als auch die Sicherheit der Steinbrüche verbessert werden könnte.

ТЕХНОЛОГИЧЕСКАЯ ЭВОЛЮЦИЯ: НЕОБХОДИМОСТЬ УЛУЧШЕНИЯ ТЕХНИКИ БЕЗОПАСНОСТИ В ИСПАНСКИХ КАРЬЕРНЫХ ВЫРАБОТКАХ

К. Мартинез Диаз и Х. Обис Санчез

В индустриально-развитых странах мира постоянно проводятся горные работы в открытых карьерах, чтобы обеспечить потребности в строительных материалах. В основном карьерные разработки производятся небольшими и средними по размеру компаниями, средства и возможности которых ограничены. В докладе анализируется место карьерной добычи в испанской горной индустрии и проводится сравнение с аналогичными предприятиями в других странах Европейского Экономического Сообщества. Доклад обсуждает проблемы техники безопасности, анализирует вероятность их появления и причины. В заключение, авторы вносят определенные предложения по увеличению добычи и повышению уровня безопасности, которые могут быть приняты как частными компаниями, так и правительствами.

技术进化：改进西班牙石矿作业安全性的必要

C。马丁奈士 狄亚士 及 J。奥别士 删锴士

石矿开采为工业国家供给建筑材料的常恒工业。大多数的石矿开采是由中小型公司经营，故有很多由于这些公司的规模所产生的缺点和限制。本文分析西班牙石矿工业的作业并与ＥＥＣ其他国家这类工业作了比较。文中叙述了安全问题及它的统计数字和原因。作者在结论里提出了私人公司和政府应采取的技术措施以增加石矿开采的产量及安全性。

UNDERGROUND COAL MINE AUTOMATION IN AUSTRALIA

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ABSTRACT

The Australian coal industry has recognised that automation of underground operations will assist the industry to continue playing a major role in providing energy on the domestic front while also maintaining its outstanding position in world coal trade. The Australian Government, through the National Energy Research, Development and Demonstration Program, has provided support for progressive introduction of remote control and automation into underground coal mining.

The advanced coal mining technology program has been developed with the active participation and support of coal mining companies, unions and research organisations. The program is addressing, through a number of separate projects, new machine developments, roof bolting, coal clearance and machine guidance in relation to mine development roadway drivage. Major advances with the potential for increased productivity and, simultaneously, enhanced working environment and safety have already been achieved.

The integration of individual components, viz. cutting, roof bolting and coal clearance, into a total system present the greatest challenge in the immediate future.

INTRODUCTION

A major proportion of coal mined underground in Australia today is won by continuous miner machines. Whereas a single machine is theoretically capable of mining over 4000 tonnes of coal per shift, actual achievement on average is below 300 tonnes per shift. Such a low utilisation factor, for a machine costing over $1 million, is caused by forced interruptions to coal production as the roof is being supported or coal moved from the face.

Moreover, the Australian coal industry has since the late 1970s and through the 1980s continued to increasingly adopt retreat longwall mining as an underground extraction method. Whereas some impressive production statistics and records have been achieved, one of the key factors restricting long term record extraction rates has been the need to improve the rate and efficiency of roadway drivage to keep pace with extraction.

To address these issues, the Australian Government through the National Energy Research, Development and Demonstration (NERD&D) Program administered by the Department of Primary Industries and Energy, initiated a series of advanced underground coal mining technology projects in 1985. The projects are largely aimed at increased underground productivity and profitability and simultaneously improving the work environment by developing and demonstrating the following:

1. new heading development machines
2. remote control and automation of roof supporting rigs that could operate concurrently with the coal cutting machine
3. continuous clearance of coal from the face
4. remote control and automation for guidance of cutting machine
5. automatic machine monitoring and communications.

The program has been developed with the active participation and support of coal mining companies, the Australian coal and shale Employees' Federation, and research organisations. With progressive remote control and automation, the miners will gradually be distanced from the immediate coal face zone, away from outburst danger, coal dust and more likely locations of roof failure. Figure 1 illustrates a typical development layout together with the current position of equipment and operators, and the likely changes to these as progressive automation is introduced.

HEADING DEVELOPMENT MACHINES

Under the NERD&D Program, support has been provided for three machines working on the principle of a low cutting rate (3.5 tonnes per minute) but having the ability for continuous cutting and simultaneous roof bolting. They include the following:

1. Dosco in-seam miner (ISM)
2. Kemcol Beaver
3. Maintel.

The Dosco ISM project was supported in the late 1970s to demonstrate single entry mine development at the Ellalong Colliery of the Newcastle Wallsend Coal Co. Pty. Ltd. The project was successful and demonstrated the substantial savings, in terms of reduced costs and time, of single entry mine development. The company later purchased a larger and newer model, the ISM 3000, which has been operational at Ellalong for some time. The fixed width and height of this machine are its major drawbacks compared with the new

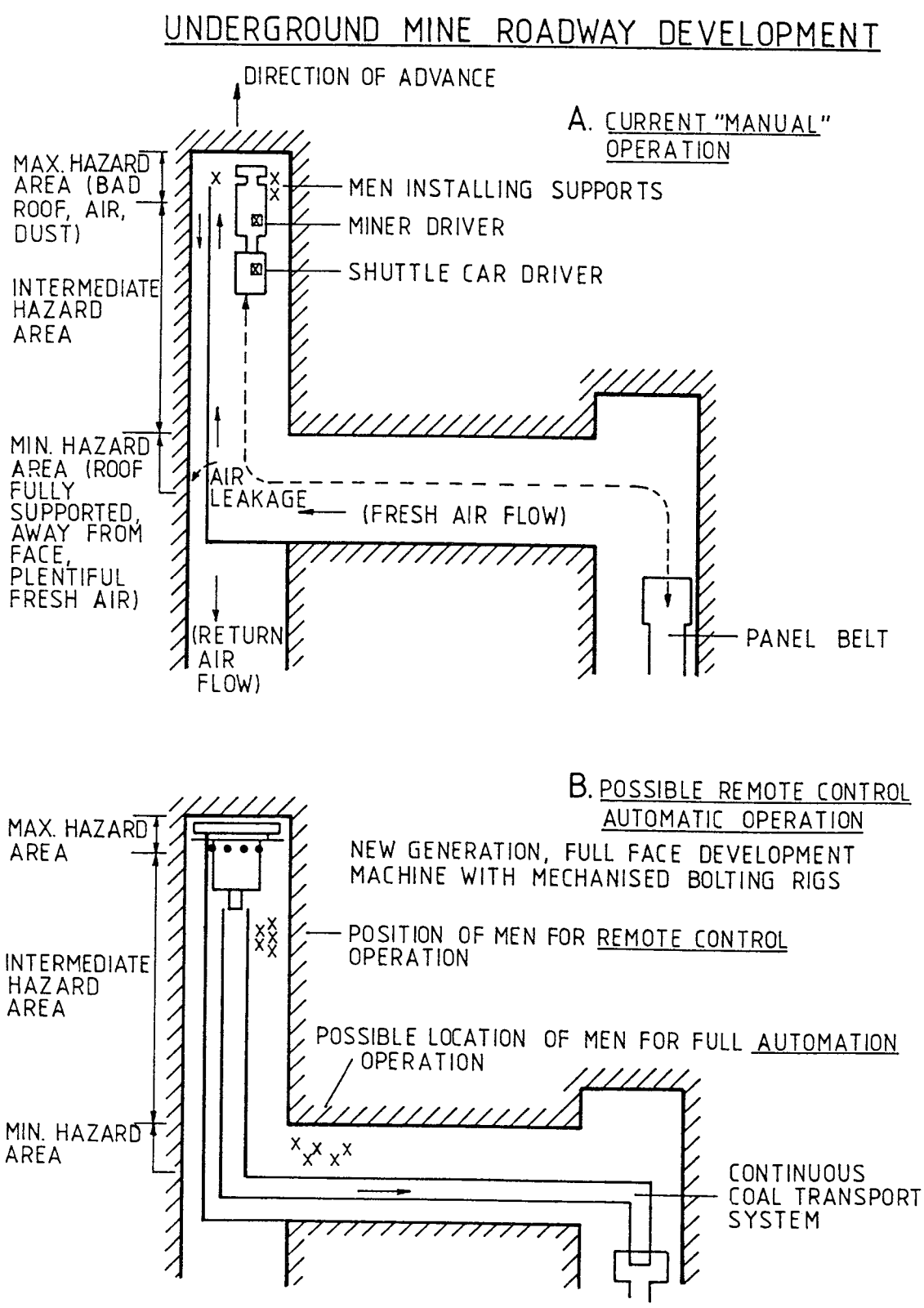

UNDERGROUND MINE ROADWAY DEVELOPMENT
DIRECTION OF ADVANCE
A. CURRENT "MANUAL" OPERATION
MAX. HAZARD AREA (BAD ROOF, AIR, DUST)
INTERMEDIATE HAZARD AREA
MIN. HAZARD AREA (ROOF FULLY SUPPORTED, AWAY FROM FACE, PLENTIFUL FRESH AIR)
MEN INSTALLING SUPPORTS
MINER DRIVER
SHUTTLE CAR DRIVER
AIR LEAKAGE
(FRESH AIR FLOW)
(RETURN AIR FLOW)
PANEL BELT
B. POSSIBLE REMOTE CONTROL AUTOMATIC OPERATION
NEW GENERATION, FULL FACE DEVELOPMENT MACHINE WITH MECHANISED BOLTING RIGS
MAX. HAZARD AREA
INTERMEDIATE HAZARD AREA
MIN. HAZARD AREA
POSITION OF MEN FOR REMOTE CONTROL OPERATION
POSSIBLE LOCATION OF MEN FOR FULL AUTOMATION OPERATION
CONTINUOUS COAL TRANSPORT SYSTEM

Figure 1

generation of machines being developed. Its greatest advantage is its potential for close face proximity roof support.

The Kemcol Beaver roadway heading machine is, unlike the Dosco project, an Australian development by Kembla Coal and Coke Pty. Ltd. (KCC). When operational, the machine is positioned in the centre of the roadway on caterpillar tracks. Four roof-bolting masts at the front of the main frame allow roof support activity to proceed, while the cutter head progressively sumps forward. With this continuous operation, the Beaver mines the full width and height of the roadway. The machine has been tested successfully at the Huntley Colliery of the Electricity Commission of New South Wales and at KCC's West Cliff mine. The operating principles and performance potential of the machine have therefore been established, and development of the next version of the machine is underway.

The Maintel heading development machine has the potential to provide an "assembly line concept" to roadway development through continuous mining, continuous roof control and continuous haulage. The machine is of the skid type with the advantage that hydraulic props are used to stabilise the machine against the ribs, and these are located towards the floor. The actual cut produces a concave wall for greater strength, resulting in a far safer machine in mines with poor roof conditions. Manufacture of the machine is scheduled to commence in early 1989, and a prototype trial is planned for late 1989.

These machines will need to compete against the Joy Sump Shearer under private sector development in Australia and the Eickhoff ESA machine. The Joy Sump Shearer will be designed to allow for a range of width, height and profile selections. The Eickhoff machine is commercially available and operating in the Federal Republic of Germany but has yet to be demonstrated for heading development. All these machines are ideally suited to continuous haulage due to their lower but continuous cutting rates and are also prime candidates for machine guidance system application and automation.

The high power, high cutting rate (10-15 tonnes per minute) machines also have a place in the Australian coal industry, but they must be stopped for bolting to take place. Development of machines of this type, allowing bolting close to the face, has been an extension of conventional continuous miner designs and has been mainly carried out by the private sector. Two of the significant machines are the Paurat E230 and the Joy 12CM20.

The Paurat has been trialled at the Cordeaux Colliery of BHP and is regarded as a prototype machine with scope for significant improvements. The Joy machine is currently in operation in several collieries. The Paurat has an angled cutter head which scrolls the coal inwards to the conveyor and the Joy has an East-West conveyor discharging into the central conveyor. The machines are also characterised by bolting rigs just behind the cutting head which allows bolting close to the face.

ROOF SUPPORT

Under the NERD&D Program, support has been provided for two projects in the field of roof bolter automation.

Clarence Colliery Pty. Ltd. has co-operated with Atlas Copco Pty. Ltd. to redesign Atlas Copco's remotely operated roof bolting equipment for the metalliferrous mining industry. The existing drilling rig has been modified to make it suitable for side mounting

on conventional continuous miners and has been successfully trialled. Automation of the rig is currently being undertaken by the Australian Coal Industry Research Laboratories Ltd. (ACIRL).

The Atlas Copco rig is however far too large for use on platform-type machines. The Newcastle Wallsend Coal Co. Pty. Ltd. has been developing an automated rig suitable for platform-type miners. The unit is extremely compact, enabling up to seven rigs to fit across the width of a full face heading machine. Trials are planned for 1989.

Besides the work funded under the NERD&D Program, the University of Wollongong is addressing integrated drilling and bolting. The project has conducted extensive tests on bit design and tubular steel sections. The major difficulties relating to the method of resin injection and the economics of the system are yet to be addressed.

COAL CLEARANCE

Under the NERD&D Program, two projects have been supported.

A roof-mounted version of the Joy flexible conveyor train (FCT) was successfully demonstrated at the Box Flat No. 9 colliery in Queensland in conjunction with a Jeffrey continuous miner and a Joy breaker car. The FCT was shown to be an extremely reliable unit and the total system incorporating the FCT appeared to offer the potential of 50-100% improvement in productivity. There are now a number of roof-mounted FCTs installed in Australian coal mines, and a floor-mounted unit is planned to be introduced into a mine in the near future.

In another project, the floor mounted, mobile Klockner-Becorit continuous conveyor system was initially trialled at the Cordeaux colliery of BHP and shown to be capable of at least 50% improvement in average roadway advance rates. Capricorn Coal Management Pty. Ltd. has just installed a similar system at its German Creek Central Colliery.

MACHINE GUIDANCE

All projects in this category have been funded under the NERD&D Program. They represent a wide range of technologies and also a considerable variation in time-frame in terms of their applicability.

In the short time-frame, Newcom Collieries Pty. Ltd., together with ACIRL, has been investigating the application of the laser-based Zedminer guidance system for continuous miners. The first prototype of this system is operational at Myuna Colliery and provides information for the machine driver. The next stage of the project is to provide automatic track control to steer the miner using the guidance system and this will be implemented in 1989. The laser system is ideally suited to full automation for horizontal guidance but has limited application for vertical guidance due to the undulating nature of most coal seams. The system has been demonstrated to have the potential for a 10% productivity increase, in addition to greater roadway stability and reduction in the amount of outbye roadway maintenance work.

ACIRL and the Commonwealth Scientific and Industrial Research Organization (CSIRO) have been investigating two different seam-following vertical guidance systems, both based on the assumption that top coal is left - a situation which is fairly common in Australian mines for development roadways to provide greater roof control. The ACIRL project, involving a prototype sonic system

which requires a pulse to be transmitted to the roof and a receiver to monitor the return signal, demonstrated that the system has potential in certain conditions where there is sufficient contrast between coal and roof and little delamination within the immediate roof. The CSIRO work on acoustic monitoring of noise generated by the machine cutter head using hand held microphones is more promising. The monitoring can be continuous and preliminary results have shown definite variations in ampitude as the cutter head approaches the roof or floor rock.

On the long term horizon is the ACIRL project on feasibility of inertial guidance to evaluate the componentry developed for the aerospace and defence industries. Inertial guidance has the potential to provide horizontal guidance for any mobile equipment, together with the basis for a mine-wide vehicle location information system.

MACHINE MONITORING AND COMMUNICATIONS

Most of the long term activity in this area is being supported under the NERD&D Program.

ACIRL is currently involved in a project to clearly identify the needs of the industry in terms of pit-wide communications and monitoring systems. This follows up a key recommendation of a study commissioned under the NERD&D Program to review advanced electronic information and control system applications for coal mining.

Australian Optical Fibres Research Pty. Ltd. has done some innovative work on development of network branching techniques which, together with fibre optic transducers for equipment monitoring, has considerable potential for application in the coal mine environment.

THE FUTURE

Progress in the various components of the underground coal mine automation program over the last three years has been variable. The greatest priority is the successful demonstration of the new generation heading development machines. In 1989, the more promising machines are due to undergo trials in Australia.

It has always been accepted that automation is a long term objective and it will be achieved in progressive steps through remote control of individual components initially. Provided one or more of the new machines is successfully trialled in 1989, the stage will be set in 1990 for progressive integration of components.

It is very likely that initial integration will include continuous coal cutting and clearance in 1990, followed by automatic roof bolting while cutting. Horizontal and vertical guidance systems will be integrated later.

Crucial to the realisation of the benefits of underground coal mine automation will be a sound monitoring and communications system.

The NERD&D Program will continue to promote the remote control and automation aspects of underground coal mine technology as the long term benefits for the industry are very real. Economic and safety benefits of such technologies will be continually assessed to ensure that the research program is justified.

AUTOMATISATION DES MINES DE CHARBON SOUTERRAINES EN AUSTRALIE

N.S. Bharucha et B.K. Hebblewhite

L'industrie minière australienne du charbon a reconnu que l'automatisation des exploitations souterraines aidera l'industrie à conserver un rôle important dans l'approvisionnement énergétique domestique tout en maintenant son excellente position dans le marché mondial du charbon. Le Programme National de Recherche, Développement et Démonstration Energétique a fourni son soutien pour introduire progressivement la télécommande et l'automatisation dans l'exploitation du charbon souterrain.

Le programme de technologie avancée de l'exploitation de charbon a pu être développé grâce à la participation active et au soutien des sociétés d'exploitation du charbon, des syndicats et des organismes de recherches. Ce programme entreprend, par un certain nombre de projets séparés, le développement et le guidage de nouvelles machines, le boulonnage des toits et le déblaiement du charbon. D'importants progrès ont déjà été accomplis en ce qui concerne l'augmentation de la productivité et l'amélioration de l'environnement et de la sécurité du travail.

L'introduction d'éléments individuels par havage, boulonnage des toits et déblaiement du charbon dans tout un système constitue la tâche la plus importante de l'avenir immédiat.

DIE AUTOMATISIERUNG VON UNTERTAGE-KOHLEGRUBEN IN AUSTRALIEN.

N. S. Bharucha, B. K. Hebblewhite

Die australische Kohleindustrie hat erkannt, daß die Industrie nur durch Automatisierung des Untertagebetriebes weiterhin eine Hauptrolle in der Energieversorgung im eigenen Land spielen und auch seine bisher leitende Rolle auf dem Weltmarkt beibehalten kann. Das National Energy Research, Development and Demonstration Programm unterstützt die schrittweise Einführung von Automatisierungen und Fernsteuerungen im Untertagebetrieb.

Das Programm zur Kohlegewinnung mit Hilfe fortschrittlicher Technologien wurde unter aktiver Beteiligung und Unterstützung von Kohlegesellschaften, Gewerkschaften und Forschungsorganisationen entwickelt. Dieses Programm zielt mit mehreren Einzelvorhaben auf die Entwicklung neuer Maschinen, Maschinen-Führungen, Ankertechniken und Kohleräumtechniken ab. Große Fortschritte in der Steigerung der Produktivität bei gleichzeitiger Erhöhung der Sicherheit und Verbesserung der Arbeitsbedingungen sind bereits erreicht worden.

Die Integration von individuellen Komponenten in ein Gesamtsystem, wie z. B. : die schneidende Gewinnung, Ankern und Räummethoden, stellt die größte Herausforderung für die nächste Zukunft dar.

АВТОМАТИЗАЦИЯ ПОДЗЕМНЫХ УГОЛЬНЫХ ШАХТ В АВСТРАЛИИ

Н.С. Бхаруша, Б.К. Хебблуайт

Автоматизация подземных операций в угольных шахтах Австралии способствует осуществлению основной функции горной промышленности, а именно, обеспечению страны источником энергии. В то же время, Австралия занимает выдающееся место на мировом угольном рынке. Программа исследования и развития национальных энергетических ресурсов создала благоприятную обстановку для внедрения прогрессивных методов дистанционного управления и автоматизации в подземных угольных шахтах. Программа современной технологии добычи угля была разработана при активном участии и поддержке угольных компаний, профсоюзов и исследовательских организаций. Программа включает несколько отдельных проектов, направленных на разработку нового оборудования, новых систем управления и методов крепления кровли и откатки угля. Уже достигнуты значительные успехи в увеличении производительности труда при одновременном улучшении условий работы и техники безопасности. Соединение таких компонентов, как нарезание угля, крепление кровли и откатка угля в единую комплексную систему представляет собой трудную задачу, которую необходимо разрешить в ближайшем будущем.

澳大利亚地下煤矿的自动化

N. S. 巴鲁恰　　B. K. 赫波怀特

澳大利亚煤矿工业认清了地下煤矿操作的自动化将会使得煤矿工业在国内能源供应上占很重要的位置，在世界煤炭贸易上也维持崇高的地位．国家的能源研究，发展和示范方案曾对地下煤矿遥控和自动化的积极引进给予支持．

本文所述的高度的煤矿技术方案的发展曾得到煤矿公司，工会和研究机构的大力参与和支持．该方案分成几个项目，研究新式机器的发展，机器制导，坑顶锚杆和煤炭储运．这样使产量的增加，工作环境和安全的改进都取得了大幅度的进展．

将个别的分件如开采，坑顶锚杆和煤炭储运合而为一个整体的系统是为最近的将来急切的需要．

THE INTRODUCTION OF SYSTEMATIC SAFETY ASSESSMENT TECHNIQUES TO THE AUSTRALIAN COAL INDUSTRY

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ABSTRACT

A grant from the Federal Government to the Australian Coal Association is allowing new and novel investigation techniques to be trialled in operating mines.

Systematic Safety Assessment (SSA) is a generic term encompassing a large number of investigative techniques such as MORT (Management oversight and risk tree), HAZOP (Hazard and Operability), FMEA (Failure mode and effect), Fault Tree etc. Such techniques originated in petrochemical, nuclear and space industries and are rarely used in the Australian coal mining industry. The introduction of SSA techniques is put forward as a vital step towards cost efficient risk management at mine sites. Voluntary introduction of such technology is also seen as a method of preventing additional legislation in an industry which is already heavily regulated by State and Federal authorities.

To demonstrate the benefits of selective use of appropriate techniques, three collieries are being used for pilot studies. In each colliery a different operation or event is analysed with the appropriate techniques. The first such operation is a Methane drainage system from coal face to exhaust stack.

The Australian coal industry, being safety conscious, is extremely supportive. The paper presents the results of one site study and details the benefits of the project in general. The applicability of the techniques to a wider range of coal mining activities is discussed.

WHAT IS SYSTEM SAFETY AND WHAT DO WE DO WITH IT?

Systematic safety assessment (SSA) is a logical and organised methodology for evaluating the potential hazards or interactions of a system, installation or device. Properly applied, it ensures that all hazard sources and possible interactions are taken into consideration. Some techniques within the general framework of SSA give a qualitative assessment, while others provide a quantitative result. Some of the more basic techniques could be applied by staff at mines after having received some training, but others, particularly those giving a qualitative result, would require substantial technical input from outside and quite likely some modifications to adapt them into an extractive industry such as mining. Some of the techniques are listed in Table 1 (Tenniswood, 1988).

The origins of SSA lie in the Nuclear, Chemical and Aerospace industries where the cost of unwanted incidents is high.

A. LOGIC ORIENTATED SYSTEM/ TECHNIQUE	B. METHOD	C. WHERE USED	D. EXPERTISE REQUIRED	E. SCOPE OF ANALYSIS	F. DIFFICULTY IN USING	G. GENERAL COMMENTS
1.FAULT TREE ANALYSIS	Identify undesirable event and contributing elements that precipitate. Trace backwards using network paths through Boolean logic gates.	Applicable to systems of all kinds, requires undesired events to be forseeable, and elements to quantifiable.	Prior knowledge of logic gates, Boolean algebra helpful. Study necessary (8-40 hours).	Regarded as one of the more thorough techniques giving identification of undesirable events and contributing factors. Focus on significant losses.	Time consuming but not difficult to use when technique understood.	Useful, particularly in association with HAZOP.
2.FAILURE MODE & EFFECT ANALYSIS (FMEA)	Examine system element by element. Identify modes in which each element can fail. Determine effect upon system of each failure mode.	Universally applicable to systems, subsystems, components, and interfaces.	Working knowledge of Boolean algebra helpful. Requires study of some days to weeks plus experience to master.	Thoroughness depends on identifying : 1. Failure modes 2. Paths 3. Interaction to other failures.	Not difficult once learnt, very time consuming.	Use only when the risk is high or not easily identified otherwise.
3.MANAGEMENT OVERSIGHT & RISK TREE ANALYSIS (MORT)	Apply a pre-designed systematised logic tree to the identification of total system risks, in the physical equipment and procedures used.	Use comparatively to describe all phases of a programme/system. Valuable in accident/ incident investigation to uncover system weakness.	Basis learnt in 8-40 hours with practical experience. Logic charts available.	Limited only by accuracy of model against reality and depth of exploration.	Tedious and time consuming.	Used primarily with accident investigation at this stage.
4.HAZARD AND OPERABILITY ANALYSIS (HAZOP)	Used within process flow operations by a team of persons familiar with process, but with differing responsibilities. Using guide words each step in process is examined in depth in a series of meetings to look at different consequences of various scenarious, re design of element is then undertaken.	Process flow operations, particularly new processes, but also retrospectively.	Expertise in systems and training in process. Multidisciplined Team Essential	Very exhaustive process.	Very demanding on concentration, usually only conducted over 2-3 hours, 2-3 days per week.	As process is very resource costly, should only be used in high risk/high loss situations.

TABLE 1
SOME SYSTEM SAFETY TECHNIQUES
Tenniswood, 1988

From 1980 to 1986 the USDOE sponsored several projects with two major contractors to investigate and apply selected SSA techniques (Briscoe, 1982; Daling and Geffen, 1983). This series of investigations does not appear to have resulted in widespread applications of the technology at operating mines. A review of the results of this prior work by the author and researchers in the United Kingdom has highlighted several factors which could contribute to a low level of acceptance :

- Many mines work on management by "fire fighting" processes.
- Some techniques are complex and difficult to use.
- Gathering together a large block of mine officials for several days of investigations is very difficult.

Despite the above comments, the basic concepts and principles including;

management system failure,
oversights and omissions by management, engineering or operatives,
unwanted energy flows,
human errors,
change of process or status,

make SSA a powerful, user friendly process, for identifying hazards prior to unwanted events, or even back analysis of such events.

SSA tools tend to be very enquiring and are biased towards the deficiencies in the managed system rather than the individual. By recognising that the managed system should allow for errors, omissions etc. of human operatives, then investigations tend to be more objective, less critical of persons, and therefore more useful.

The techniques should only be seen as an aid to decision making. They are particularly useful for deciding when sufficient safety protection is built in, whether particular designs will meet availability requirements, selection of the best option etc.

The use of SSA techniques on mine sites will tend to move the analysis of safety away from statistics on minor injuries and towards the analysis of causation factors, particularly in high loss incidents.

SITE STUDIES

A pilot program involving 3 collieries is being used to introduce the concepts of SSA. This is being implemented by staff of the Australian Coal Industry Research Lab (ACIRL) and selected consultants, experienced in particular SSA techniques. At each colliery ACIRL work with a different consultant to achieve 3 aims :

(a) Review the mining operation and make recommendations as to how SSA can be effectively used at that mine.

(b) Conduct a specific study, for the benefit of the mine, to demonstrate one or more of the investigative techniques within SSA.

(c) Leave the mine staff with an understanding of the range of techniques and how their application can profit their mine.

By adopting this process the project team are able to determine which investigative technique of SSA might be applicable, and then, with the help of site staff, employ the technique to examine or solve a practical site "problem".

Despite the word "Safety" in SSA, safety may not be the principal reason for performing SSA. For example, various techniques within SSA are being used to analyse the poor production performance of one mine which has introduced a new coal haulage system.

The site study concept just described has the added benefit of closely involving site staff in all stages of the project. This has several benefits.

(a) Site staff gain a good basic

understanding of the project aims.

(b) The project is used to investigate practical site "problems".

(c) Site staff learn how to use the simpler of the SSA techniques in day-to-day operations. Technology transfer is maximised.

(d) Complex techniques such as MORT (see Table 1) are quickly rejected as unworkable without specialist training and staff.

SITE A STUDY

This colliery is an underground longwall operation with an annual output of around 1.8 million tonnes from a workforce of just over 350. This mine is reasonably typical of Australian longwalls in this particular geographical area. Gas drainage is practised as part of the normal mine operation.

The initial phase of the study sought to identify if and how the techniques of SSA could be applied at Site A. This was undertaken through a mixture of formal and informal meetings, basically aimed at deciding how the colliery staff currently handle tasks such as :

. Accident and job safety investigation.
. Delegation of authority.
. Evaluation and introduction of new or modified techniques and/or equipment.
. Day-to-day versus long term planning.
. Recording and "costing" of delays and accidents.
. Operator training.

The initial finding of the study team were that the concepts of SSA could be applied to assist the safe opertion of this mine. Several recommendations were made :

(a) All mine losses, must be accurately costed and the root causes identified.

(b) Job Safety Analysis should concentrate on new work methods or infrequently performed, complex tasks.

(c) Engineering groups should more fully and systematically investigate new or modified equipment from the viewpoints of operator and environment risks.

(d) A hazard analysis system should be established to prioritize hazards in terms of dollar risk/annum. This would include :-

. Establishing the energy source.
. Defining loss scenarios.
. Examining controls and their effectiveness.
. Determining the maximum $ risk.
. Determining event frequency and risk of maximum loss.
. Calculating a Hazard Index of some form.

(e) The entire coal industry should develop a uniform, System Safety based, incident investigation process, to critically examine significant incidents, for managed system and engineering causal factors.

(f) The entire coal industry should establish System Safety methods and capability for the review of major new equipment, procedures or systems.

HAZOP STUDY

Following the initial investigation at Site A a study case in the form of the mine's gas drainage system was identified. This mine operation was of particular interest because :

(a) It involved technology outside the realms of traditional mining engineering.

(b) It introduced new risks to both underground and surface operations.

(c) The surface pumping facilities were professionally designed along

typical petrochemical plant lines, whilst the underground system had more-or-less evolved from experience of other mines.

(d) The entire system is somewhat complex with interacting components.

(e) The underground plant is in a moderately hostile environment.

(f) The consequences of an accident could be severe (eg. explosion, fire).

The HAZOP process (see Table 1 and Figure 1) was chosen to examine the system as it is ideally suited to process or flow systems. HAZOP's work on three basic assumtions.

(a) Problems occur when there is a deviation from what is expected.

(b) Most problems are not recognised until too late because the system is complex with many interactions.

(c) Collectively, operators, staff and site engineers, possess the skills to identify most potential deviations, decide if the current safety measures are adequate, and initially review alternative ideas.

The HAZOP at Site A was performed by a team of seven comprising mine staff and external experts in SSA techniques. The methodology consists of examining segments of flow lines using a deviation word model. For example, within a particular pipe length, the guide words more of, none of, less of, part of, more than, other than, are applied as shown in Figure 1.

By following the HAZOP process given in Figure 1 it is obvious that the results are primarily qualitative. Other techniques would then be applied as required to quantify particular risks identified.

From the Site A study 16 specific recommendations were made and 17 questions left to be answered. The HAZOP revealed 7 deviations which could result in hazardous situations developing. Mainly these were underground where the level of protective and monitoring equipment was negligible. A typical output from the HAZOP is given in Table 2. In this case the portion of the plant being examined is the flexible hose from the borehole standpipe to the header pipe (actually there are multiple such hoses).

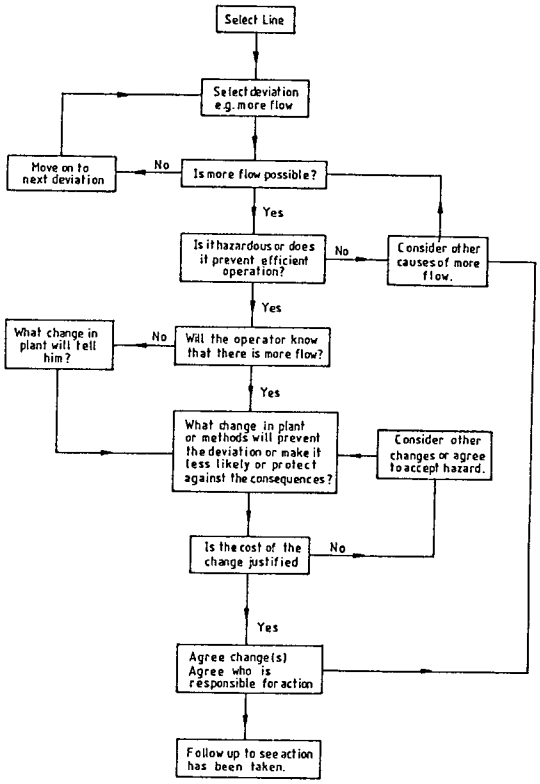


FIGURE 1: HAZOP procedure

At Site A the major problems have been defined, however, risk quantification remains to be performed. Such a process typically provides management with hazard index in terms of dollar risk/annum, and is arrived at by considering the following.

DESIGN INTENTION : L2 This line consisted of all the flexible hoses attached to the boreholes and the valves to the first header

GUIDWORD/ DEVIATION	POSSIBLE CAUSES	POSSIBLE CONSEQUENCES	PROTECTIVE EQUIPMENT ON P & I.D.	ACTIONS	QUESTION/RECOMMENDATION
NO FLOW	Uneven piping of L3 causing water to block lines in low elevation sections.	If there is a blockage in L3 and the gas pressure is high it may cause leaks in L2 into the roadways. If there is a block in L3 the vacuum will build up downstream and trip the plant. Uneven loads on piping can lead to rupture and release of materials into accessways.	Currently installing new system for automatic water draining.	Check valves and lines upstream. Ensure lines are installed level.	
LESS FLOW	See LESS FLOW L1 Breakage of L3 via roof collapse, structural failure collision with vehicles, etc.	see c: production delay f: leak, inflow oxygen fl: inefficiency k: uneven lines Not achieving intended purpose of plant to reduce hazardous gases underground. If L3 breaks it will put methane and water into the roadway an explosion may result.	No contingency plans for a pipe break underground which co-ordinates all actions to be taken.	Install gas monitors in drainage roadways. Install L3 only in return airways. Check pipe clearances for vehicles. Develop contingency plans for pipe break scenarios	What is the likelihood of failure of L37 Estimate 1 in 2-50 years. What can be done to protect L3? Should L3 be installed in return airways? If L3 failed is an explosion likely in the roadways?
PART OF HIGH WATER	High water levels in L3. Failure of drainage systems. High water content in seam.	Water in lines reduces flow and causes additional stresses including water hammer.	Manual draining procedure at change of shift. Listen for high water levels in pipe.		
MORE OXYGEN	Broken standpipe in borehole.	Not significant if small.		Installation of underground sensing system.	
MAINTENANCE HAZARDS	Valves left open which admit air to the system.	Oxygen enters system and a flammable mixture may fill piping network. Possible explosion. (detonation)	Oxygen sensing system above ground (plant may detect oxygen too late). Flame arrestor installed in line above ground. Tagging system in operation	Investigate possibility of installing flame arrestor or flame trap underground	Can a detonation occur if the piping network is filled with a flammable mixture? Should a flame arrestor be installed underground?
MORE PRESSURE	High pressure venting system inoperative. Valve VG-24-G-3 inadvertantly closed.			Check high pressure venting system for faults and use preventative maintenance.	

TABLE 2

Extract from HAZOP Study

- What is the likelihood of the deviation occurring (i.e. frequency)?
- What protection and controls can be employed?
- What is the maximum direct loss scenario ($ and description)?
- What are the indirect costs for the maximum loss scenario?
- What is the risk of the maximum loss occuring?

Hazard indices are particularly useful in that they provide a reasonably clear prioritizing of the identified hazards.
Therefore the areas for priority spending can be identified. This minimizes the emotive issues often associated with determining the recipients of a limited budget allocation.

SITE B

At Site B various SSA techniques have been used to analyse the safety risks and production problems associated with the introduction of a new continuous haulage system.

At the time of writing, only an analysis of the production problems had been performed. This analysis was based on a HAZOP study of the coal production system followed by FMEA study of particular plant. From extensive downtime and production data supplied by the site, a prioritizing of suggested changes was undertaken. This not only examined discrete changes but implications on the entire system. In considering changes, the proposed cost was increased by various factors corresponding to the risk associated with the change (ie. the liklihood that spending the money will result in the total predicted benefit).

One of the possible changes involves an alteration to the support system for pillar extraction and this will be examined in detail for the more traditional safety implications.

SITE C

The implications of computers in control was chosen for study at Site C. At this site (a longwall mine) three different underground monitoring systems had been installed, and another one imminent with limited interconnection.

The initial studies at the site concentrated on the technicalities of system problems whilst working within the general framework given in Figure 2.

This study had only just begun at the time of writing.

ASPECT	METHOD
IDENTIFICATION OF RISKS	. Asset register and evaluations ($) . Loss history . Surveys . Perceived threats
ANALYSIS OF RISKS	. Classification : . Business interruption . Asset loss . Income loss . Liabilities . Personnel
MEASUREMENT OF RISKS	. Quantification : . Frequency . Severity . Predictability (probabilities and trends
REVIEW OF CONTROLS	. Match perceived threats to controls in place and required . Reliability of controls . Specification of controls
ASSESSMENT OF CONSEQUENCES OF LOSSES	. Calculation : . Annual loss expectancies (ALEs) for each risk . Cost - benefit of control measures
ACTION PLAN	. Review risks and ALE's . Prioritize suggested changes . Instigate regular reviews

FIGURE 2
Investigation Sequence - Site C

OTHER MINING USES OF SSA

The techniques given in Table 1 are but a few of those developed to assist other industries. Some of the

immediate applications suggested for Australian mining are:-

HAZOP - Analysis of coal production (flow)
- Review of computer controlled systems

FMEA - Analysis of mining equipment

Hazard Analysis - Mine accidents

Contingency Analysis - Mine accidents

Fault tree - Back analysis of operational delays

The above are a few specific examples, however the general principles need to be introduced into the mine environment. One of the major difficulties found has not been in applying the techniques to specific examples, but rather to convince mine officials they can use the techniques to improve their management process, and eventually productivity.

CONCLUSIONS

Techniques such as those described in this paper derive their power from their methodical procedures and team approach. It is these latter two points which separate System Safety Assessment techniques from those practised today.

SSA techniques tend to move the emphasis away from minor incident prevention and statistics gathering, towards more significant, high risk incidents. Whilst these techniques are proven in other industires they need significant adaption to work in an extractive industry such as coal mining.

The Australian coal industry has the technology and skills to effectively introduce selected techniques to improve their risk management, incident review and ultimately, productivity.

ACKNOWLEDGMENTS

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INTRODUCTION DE TECHNIQUES SYSTEMATIQUES D'EVALUATION DE LA SECURITE DANS L'INDUSTRIE AUSTRALIENNE DU CHARBON SOUTERRAIN

T. O'Beirne

Une subvention du Gouvernement Fédéral permet à l'Association Australienne du Charbon d'essayer de nouvelles et innovatives techniques d'investigation dans les mines en exploitation.

L'évaluation systématique de la sécurité est un terme générique qui comprend de nombreuses techniques d'études telles que MORT (surveillance de l'exploitation et échelle des risques), HAZOP (risque et capacité de fonctionnement), le mode et l'effet de rupture, l'échelle des failles, etc... De telles techniques, nées des industries pétrochimique, nucléaire et spatiale, sont rarement utilisées dans l'industrie minière.

Pour démontrer les avantages de l'utilisation sélective de techniques appropriées, trois houillères font l'objet d'études pilotes. Dans chacune d'entre elles, on analyse une opération ou un évènement différent au moyen de techniques appropriées. La première de ces opérations est un système d'évacuation de méthane du front de taille vers la colonne d'évacuation.

L'Industrie Australienne du Charbon, concernée par la sécurité, soutient pleinement ce programme dont les avantages se font déjà sentir et qui font l'objet de relevés.

Ce rapport présente les résultats d'une étude de site et examine les avantages globaux du projet ainsi que les méthodes d'application de ces techniques à toutes les activités d'exploitation du charbon.

DIE EINFÜHRUNG VON TECHNIKEN ZUR SYSTEMATISCHEN SICHERHEITSBEWERTUNG IN DEM AUSTRALISCHEN UNTERTAGE-BERGBAU.

T. O'Beirne

Ein Zuschuß von der Regierung an die Australian Coal Association ermöglicht das Testen von neuen und ungewöhnlichen Forschungstechniken in aktiven Gruben.

Systematische Sicherheitsbewertung (Systematic Safety Assessment) ist eine allgemein gebräuchliche Bezeichnung, die eine Vielzahl von Untersuchungs-Techniken wie MORT (management oversight and risk tree), HAZOP (hazard and operability), "failure mode and effect", "fault tree", etc... umfaßt. Diese Techniken aus der Petrochemie-, der Atom- und Raumfahrtindustrie werden selten in der Kohleindustrie angewandt.

Um die Vorteile der selektiven Anwendung der jeweils einschlägigen Technik nachzuweisen, werden drei verschiedene Gruben für Pilotprojekte benutzt. In jeder Grube wird ein spezieller Betrieb oder Vorfall mit der entsprechenden Technik analysiert. Der erste auf diese Weise untersuchte Betrieb ist eine Methanabsaugung aus dem Kohlestreb in einen ausziehenden Wetterschacht.

Da die australische Kohleindustrie sehr auf Sicherheit achtet, ist sie äußerst hilfsbereit, was bereits deutlich fühlbar und auch schon dokumentiert ist.

Der Vortrag beschreibt die Untersuchungsergebnisse aus einer Grube und die Vorteile des Gesamtvorhabens im einzelnen. Die Anwendbarkeit der Techniken in allen Phasen der Kohleindustrie wird diskutiert.

ПРИМЕНЕНИЕ МЕТОДОВ СИСТЕМАТИЧЕСКОЙ ОЦЕНКИ БЕЗОПАСНОСТИ В АВСТРАЛИЙСКОЙ ПОДЗЕМНОЙ УГОЛЬНОЙ ПРОМЫШЛЕННОСТИ

Т. О'Бирне

Субсидия федерального правительства, предоставленная Австралийской угольной ассоциации, позволяет испытывать новые исследовательские методы в действующих шахтах. "Систематическая оценка безопасности" является общим термином, охватывающим большое количество исследовательских методов, таких, как оценка административных ошибок и степени риска; угроза опасности и условия продолжения горных работ; условия, при которых происходит обвал, и его последствия; последовательность ошибок и т.д. Такие методы сначала применялись в нефтехимической, ядерной и космической промышленностях. Они исключительно редко применяются в горной промышленности. Чтобы продемонстрировать выгоды избирательного применения некоторых методов, были проведены пробные испытания на трех угольных шахтах. Различные операции были проанализированы соответствующими методами в каждой шахте. Первая такая операция - система высасывания метана из угольного забоя в отработанную массу. Австралийская горная индустрия, будучи озабоченной сохранением безопасности, поддерживает эти исследования и с готовностью принимает и документирует их положительные результаты. Настоящий доклад представляет результаты одного исследования в действующей шахте и рассматривает общие выгоды от этого проекта. Обсуждается также приложимость этих методов к другим областям горной промышленности.

对澳大利亚地下煤炭工业系统性安全审定技术的引进

联邦政府拨了一笔经费给澳大利亚煤炭协会，以使煤矿的操作使用新颖的研究技术。

系统性安全审定是一个笼统的名称，它包括很多的研究技术，例如：MORT（管理监督和冒险支节），HAZOP（危险和操作性），故障方式和效果，症点支节等。这些研究技术发源于石油化学工程，核能和航天工业，而很少应用在煤矿工业上。

为了示范这类技术应用的利益，我们选了三个煤矿来做试验性的研究。每一煤矿里使用某种技术分析一种操作或项目。其中第一项是操作为从煤炭工作面到抽风机烟囱的甲烷排除设备。

澳大利亚煤炭工业因为对安全特别注意，非常积极支持，所以得益很大，多有记载。

本文讲到一处实地试验及研究结果，所得益处详有叙述。文中也讨论到这些技术应用在整个煤矿操作系统的可行性。

A PERSONAL INTEGRATED LIGHTING, GAS DETECTION AND LOCATION SYSTEM

J. L. Fuentes-Cantillana and J. R. Medina
Aitemin and Suministros Adaro, Spain

ABSTRACT

Suministros Adaro in cooperation with AITEMIN has developed a new concept in cap lamps. It is based on Ni-Cd batterries, is lighter than conventional units and completely maintenance-free. The ergonomic design is complemented with a gas measurement system connected in the cable between the battery and the headpiece, providing indication of dangerous gas concentrations by making the lamp flash. A signal transmitter for miner location is currently under development.

1 INTRODUCTION

Spanish underground mining had a high comparative accident index in 1987. There were a total of 39 deaths in that year, of which 31 were in coal mines. A comparison of these figures with those of the EEC-10 in 1985 may be seen in Table I:

Country	Year	Thousands of hours worked underground	Deaths	
			Total	Frequency rate
Spain	1987	45.239	31	0,69
EEC-10	1985 (1)	356.660	107	0,3

Table I. Comparison between mortal accident rates in underground coal mining for Spain and the EEC-10.

(1): Lastest figures available for the EEC as a whole.

The reasons for this situation are to be found in the different conditions of the coalfields, as many vertical seams are worked in Spain. Approximately 60% of coal production comes from seams with gradients of over 35º, with 40% form seams over 60º. Mechanization of these seams is difficult and other methods are used, such as inverted stepped face, longwall face with shearer or sublevel caving. In particular, stepped face and longwall face methods, together with related development work, is not as safe as working with caving methods, and are responsible for most of the accidents. These are mainly caused by coal falls and asphysia from gases (with 20 and 13 deaths in 1987, respectively), meaning that both causes combined were responsible for 74% of deaths in that year.

In the light of these figures, the importance of developing safety measures and equipment which reduces risks deriving from these causes is obvious. These measures vary in nature, ranging from work scheduling to personal protection equipment and including ventilation techniques, supports, etc. The system specifically being developed by Suminstros Adaro, S.A. (a safety equipment and cap lamp manufacturer) in collaboration with AITEMIN (Mining Equipment Technology Research Association) aims to enable implementation of two types of measures:

- Personal warning in the presence of dangerous gases or the lack of oxygen.

- Location of personnel buried under a coal fall.

2 SYSTEM REQUIREMENTS

In order to fulfill the above objectives, the system to be developed must meet the following requirements:

- Be an individual unit, so that each miner is protected at any time and in any situation.

- Operate continuously, in order to be independent of being turned on/off by human intervention.

- Be study, but light and ergonomic, in order to be accepted by miners.

- Be reliable in operation, to avoid false alarms and thus lose credibility with users.

- Be cheap, allowing as many miners as possible to be provided with the system.

In the light of these requirements, and after analysing all of the options available, a system associated with the cap lamp was selected, as this is part of the underground miner's essential and irreplaceable equipment. The unit would operate from the lamp's own battery, providing indications by making the lamp flash, making the warning noticeable at any time -independently of the back ground noise and work being carried out by the miner. In this respect it must be taken into account that the equipment is mainly directed at coal face and headings workers, where ambient lighting conditions do not usually mask the light of each individual's cap lamp.

As adding a system similar to that described caused problems to a conventional lamp, because of power consumption, additional volume, etc. a new lamp had to be designed which was both self-sufficient and capable of the modular addition of each system component.

3 DESCRIPTION OF THE SYSTEM

Construction

The system consists of four modules (see figure 1). These are respectively:

- Battery
- Gas detection unit
- Headpiece
- Location signal transmitter

The first three modules form a unit as the gas detetion section is connected in the cable joining up the battery and the headpiece (see figure 2). This unit has been developed and is now at the final testing and certification stage.

The location signal transmitter module is, in principle, independent from the rest and will include its own internal battery. This unit is currently at the initial development stage of basic experimentation.

The features of each module are described below:

Battery

The battery consists of hermetic Ni-Cd alkaline cells, ensuring fully-sealed (IP 67) maintenance-free construction, while offering a better weight/capacity ratio compared to lead-acid units. A nominal voltage of 3.6 V is supplied with a minimum capacity of 14 Ah, meeting the requirements of European Standard EN 50.033.

The casing is in injection-moulded plastic. It has an ergonomic design which adapts to the user's waist. The unit is manufactured from materials guaranteeing that there is no risk of ignition from static electricity.

The battery unit weighs approximately 1,7 kg, which is some 300 g less than conventional lead-acid batteries.

Headpiece

The headpiece is the most conventional part of unit and is also made in injection-moulded plastic. The main bulb has the following specification:

Type	: halogen
Power rating	: 3.6 W
Flux	: 48 Lm
Nominal cons'n	: 1 A

As the battery can supply 1.5 A as a maximum, in accordance with EN 50.033, 0,5 A are available to supply the gas detection unit. This limitation combined with the low battery voltage imposed severe conditions on the design of the gas detection unit.

Gas detection unit

This unit is shaped like a compact module which is small in size (118 x 76 x 43 mm) fitted in the connecting cable between the battery and the headpiece.

This module fits to the user's chest or side with a

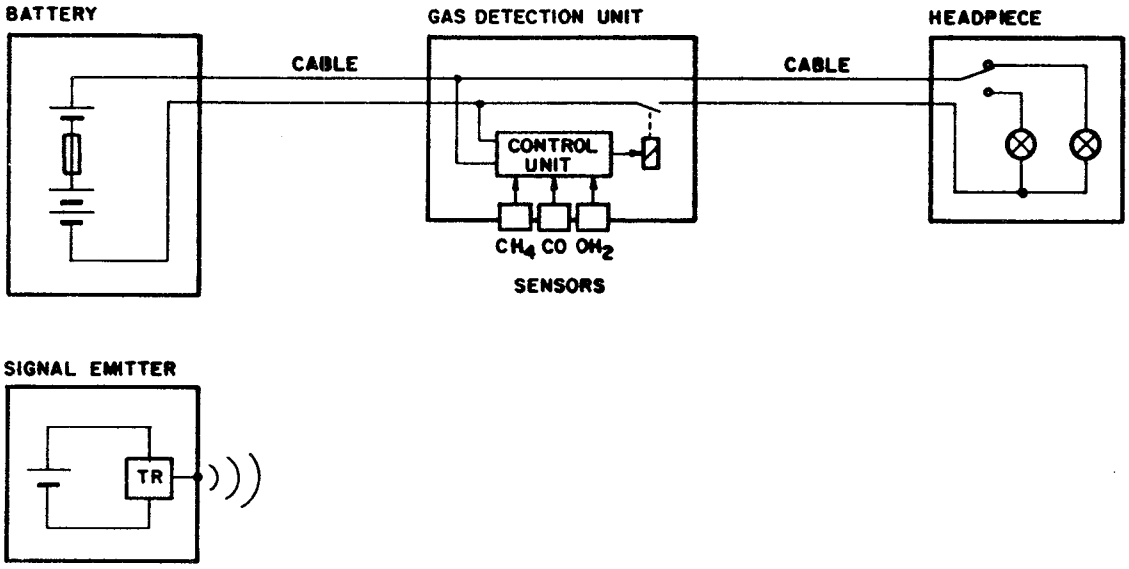


FIG. 1–SCHEMATIC OF SYSTEM ARRANGEMENT.

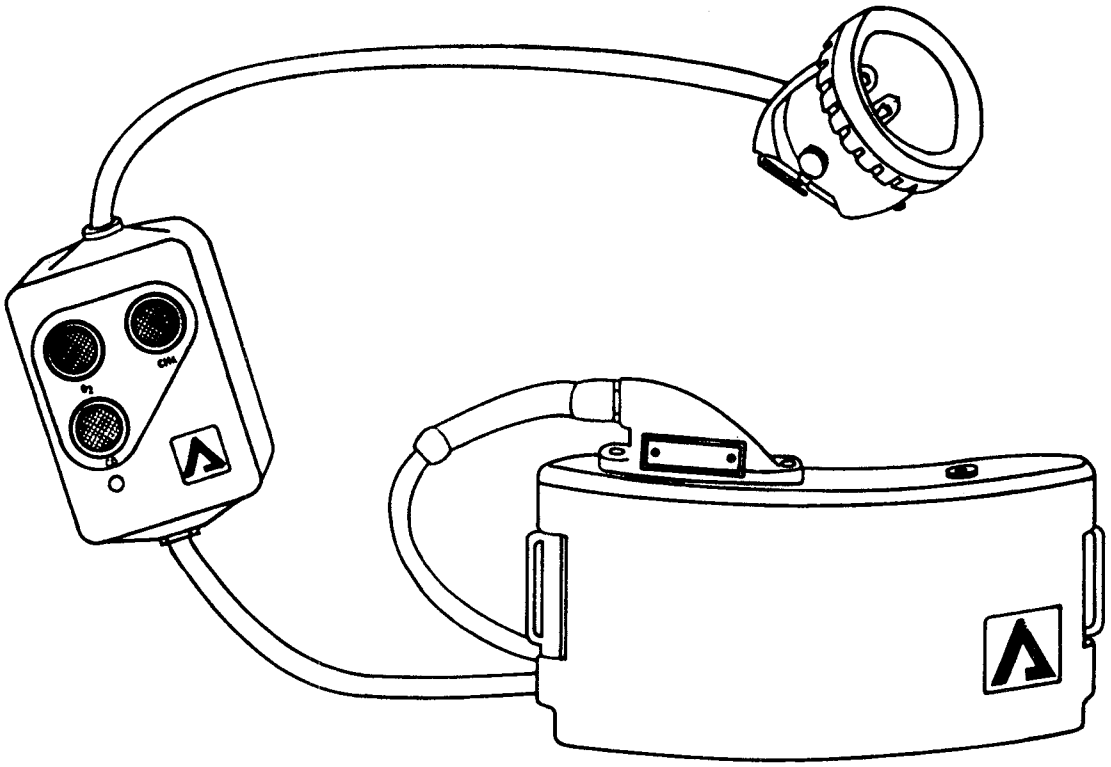


FIG. 2 - SET OF CAPLAMP AND GAS DETECTION UNIT.

clip. This arrangement avoids extra weight on the head and ensures a modular structure, so that the gas detection unit can be adapted to other cap lamps, provided that they have a suitable battery.

The module itself contains three gas sensors (see figure 2) for methane, oxygen and carbon monoxide, together with the related power supply, amplifier and control circuits. Il also includes a relay for switching the headpiece power supply.

This unit operates as follows: the gas sensors continuously measure ambient gas concentrations. When any of these exceed (or fall below, in the case of oxygen) preset limits, the unit responds by energising a relay which makes the headpiece lamp blink. In order to identify the type of gas generating the alarm, the switching frequency is different in each case. The fastest rate is for lack of oxygen, intermediate rate for the presence of carbon monoxide and the slowest rate for the presence of methane. The blinking continues until all alarm situations have disappeared.

The bulb flashing service cycle was selected so as to be easily perceptible, but without dangerously reducing visibility.

The alarm points for each gas are independently adjustable in each lamp, within the measuring ranges. The main specifications of each sensor used are given below:

a) Methane:
   Type: catalytic combustion (pellistors)
   Alarm adjustment range: 0.1 - 5% CH_4
   Reponse time: typically 2 seconds

b) Oxygen:

   Type: electrochemical cell
   Alarm adjustment range: 0 - 21% O_2
   Reponse time: typically 15 seconds
   Average life: 66 months

c) Carbon monoxide:

   Type: electrochemical cell
   Alarm adjustment range: 0 - 100 ppm CO
   Reponse time: typically 30 seconds
   Average life: 6 months

The gas detection unit is an intrinsically safe piece of equipment to EEx ia in accordance with EN 50.020, allowing it to be used in any part and location into the mine.

Location signal detector

This unit is not connected to the main battery, but forms an independent module hanging from the same belt as the battery, but without being connected to it. The main reason for this is the life that this module must have. This life should exceed the nine hours stipulated for cap lamps, as it fulfills a different objective.

The function of this unit is to transmit a signal which can be detected by an external receiver, enabling the position of the person carrying the transmitter to be identified.

This equipment is under development and two types of applications may be considered; this may lead to two different units:

a) Personnel detection in dangerous or forbidden places

This may be applied to belt transport of personnel, to closed or badly ventilated parts of the mine, etc and is based on the existence of fixed receivers which respond by halting the machinery or by emitting a warning signal when the presence of personnel is detected outside safe or permited areas.

In this case the signal transmitted is directly propagated through the air and the range should not be too large. Good results have already been obtained in several countries with personal radio-frequency transmitter units working on the UHF band.

b) Location of personnel buried in landslides

When coal falls from the coalface in vertical sections, miners may be pushed to areas away from their initial positions. For this reason it is very difficult to properly organise rescue operations, which are necessarily slow as the working environment consists of a non-consolidated mass. Overall it takes a long time to locate buried miners, reducing the possibility of finding them alive. Even when there is no hope left, the delay in finding the bodies creates a tense human and social situation, not only in the mine affected but also in nearby areas and towns, for which recent experience unfortunately exists.

This type of phenomena are highly specific to coal workings in vertical seams using non-mechanised methods. They are only avoidable by proper workface planning and strict safety conditions governing supports, fill, staff training, etc.

However, the availability of a means of orientating rescue work after the accident may even save lives.

For this type of use, the transmitters should operate on very low frequencies, which are those that are best transmitted through rock masses. However, this places severe limitations on power requirements and aerial size. Research is being carried out into the behaviour of this type of wave and ultrasound waves through loose materials. This study is still at the initial development stage.

4 CONCLUSIONS

In order to avoid accidents caused by dangerous gases or asphyxia to the extent possible (of great importance in mining vertical coal seams) a personalised modular system has been developed to warn miners of dangerous situations resulting from the presence of dangerous gases and the lack of oxygen. This system has been incorporated into the cap lamp, which has been redesigned. The unit is light, ergonomic and reliable

An enhanced technique is being researched for the development of personalised signal transmitter units for the detection of personnel in dangerous or phohibited areas or in landslide zones.

UN SYSTEME PERSONNEL INTEGRE POUR LA FOUDRE, LA DETECTION DE GAS ET LA LOCALISATION

José R. Medina, José L. Fuentes

Suministros ADARO, S.A. développe actuellement un nouveau concept de lampe frontale, en collaboration avec AITEMIN. Fonctionnant avec des piles Ni-Cd, ces lampes sont plus légères que celles utilisées habituellement, n'exigent pas de maintenance et éclairent mieux. Leur conception est ergonomique et on peut les combiner à un appareil de détection de gaz de méthane, d'oxygène et d'oxyde de carbone, alimenté par la même pile, qui donne l'alarme en cas de concentrations dangereuses de gaz.

Un émetteur de signaux peut également être couplé au système, pour permettre la localisation des personnes prises au piège.

Ce rapport examine les caractéristiques de cet équipement et explique les raisons rendant le développement d'un tel système utile pour la sécurité dans les chantiers souterrains.

EIN INTEGRIERTES, PERSÖNLICHES SYSTEM DER BELEUCHTUNG, GASAUFSPÜRUNG UND LOKALISIERUNG.

Jos. R. Medina, Jos L. Fuentes

Suministros Adaro, S. A. hat in Zusammenarbeit mit AITEMIN ein neues Konzept für Grubenlampen entwickelt. Die mit Ni-Cd Batterien ausgestatteten Lampen sind leichter, wartungsfrei und bieten bessere Ausleuchtung. Sie sind ergonomisch konstruiert und können mit einem Methan-, Sauerstoff- und Kohlenmonoxid-Spürgerät kombiniert werden, das von der gleichen Batterie gespeist wird und bei gefährlichen Gaskonzentrationen eine Alarmanzeige abgibt.

Außerdem kann ein Signalsender mit der Lampe verbunden werden, um die Auffindung eingeschlossener Bergleute zu erleichtern.

Der Vortrag beschreibt die Eigenschaften dieser Ausrüstungen und erklärt die Sicherheitsaspekte im Hinblick auf Untertagegruben, die zur Entwicklung des Systems führten.

КОМПЛЕКСНАЯ ПЕРСОНАЛЬНАЯ СИСТЕМА, ОБЪЕДИНЯЮЩАЯ ОСВЕЩЕНИЕ, ОБНАРУЖЕНИЕ ГАЗА И ОПРЕДЕЛЕНИЕ МЕСТОНАХОЖДЕНИЯ

Хозе Р. Медина и Хозе Л. Фуентес

Новая конструкция шахтерской лампы была создана компанией Суминистрос АДАРО в сотрудничестве с компанией Айтемин. Источник питания для этой лампы - никель-кадмиевая батарея. Эта лампа легче обычной, не требует обслуживания и обеспечивает лучшее качество освещения. Лампа спроектирована с учетом эргономики и может сочетаться с устройством для обнаружения газов (метана, кислорода, окиси углерода). Для питания этого устройства используется та же батарея, которая также обеспечивает и аварийную сигнализацию в случае появления опасных концентраций газа. Излучатель сигнала может быть подсоединен к этой системе. Он позволит установить местонахождение шахтера, оказавшегося в угрожающей ситуации. В докладе в общих чертах описывается вся система и объясняются различные факторы, относящиеся к технике безопасности подземных работ и обусловливающие создание подобной системы.

个人用的照明，气体侦察和定位合用装备

侯赛 R．麦迪纳，侯赛 L．福恩退恩 -- 堪的拉那

苏米镍士屈罗斯阿达罗 S．A．与 AITMIN 合作发明一种新原理的帽灯．利用镍 -- 镉电池使这种灯较为轻巧，无须保养，照明度较高．它是根据人机工程学的原理而设计的，与可以侦测甲烷，氧和一氧化炭的气体侦察器连在一起，同用一个电池．遇有危险气体含量较高时便会发出警告信号．

同时信号发生器也合在一起，使被困工人的位置可以传到外面．

本文叙述该装备的特点，并且说明为了地下矿安全问题，这种装备的发明很有价值．

TRENDS IN THE DEVELOPMENT OF GAS SENSORS FOR MINING USE

J. G. Firth and T. A. Jones
Health and Safety Executive, Occupational Medicine and Hygiene Laboratories, UK

ABSTRACT

Recent developments in various types of gas sensor are examined, the advantages and disadvantages of currently available sensors are summarised and possible directions of future developments identified. The trends in the utilisation of sensors and the possible impacts of these new developments on the mining industry are discussed.

INTRODUCTION

During the past 5 to 10 years, there has been an upsurge of interest in chemical sensors, including chemical sensors for gas measurement[1]. Sensors are the vital interface between automatic and robotic systems and their environment and this upsurge of interest stems from the realisation that the technology of available sensors of all types lags far behind that of computing and electronic circuitry. Nowhere is this more apparent than for chemical sensors. The increased interest has resulted in a number of sensors based on new technologies becoming commercially available and in addition to these, a significant number are at various stages of development in laboratories. In considering the effects of these developments on the mining industry, which has a good record of introducing new technology into its environmental monitoring an safety protocols, the whole range of sensors and sensor systems which might become available must be taken into account.

THE NEEDS OF THE MINING INDUSTRY

The basic needs of the mining industry for gas detection change only slowly. The main requirements are:-

(a) Flammable gas detection;

(b) Oxygen deficiency measurement;

(c) Carbon monoxide detection for personnel protection, especially in post-fire rescue conditions;

(d) The measurement of oxides of nitrogen following shotfiring operations;

(e) The early detection of onset of fires or heatings, either through selective detection of carbon monoxide or the non-selective detection of products of combustion.

(f) Detection of oxides of nitrogen resulting from diesel exhaust.

Humidity measurement has not received much attention in the industry, but as sensors become available, such measurement could form part of any system aimed at total atmosphere monitoring and the requirement for this type of measurement is likely to increase as working temperatures increase with deeper mining operations.

Major considerations which are peculiar to coalmining and can limit the application of sensors, stem from their electrical power consumption and its interaction with electrical safety requirements and the problems of sensor/sampling point location, which become more serious as the rates of face advance increase.

It should be borne in mind that the main reasons for measuring gas concentrations is to provide information on what is happening in the working environment. In addition to absolute measurements of concentration levels, the rates of change of concentrations, the location of higher concentrations of gases and the relative concentrations of gases, all give information relating to what is happening in the mining environment and can provide early warning of the onset of hazardous conditions. As mining processes become increasingly capital intensive and rates of extraction increase, the need for total atmosphere monitoring, using intelligent systems which will automatically control the working environment, or even control rates of production as well as providing alarms in hazardous situations, will become more pressing. Whether or not the sensors which will be available in the near to medium term future (1-5 years) will be capable of providing such systems, will be discussed in this paper.

SENSORS

The increased interest in sensors has resulted in a number of new principles being utilised in sensor technology. Sensors based on these principles are already in use in a number of industries, including mining. Any sensor is a compromise of a number of different variables. Thus, increased operating temperature of a sensor will often reduce the time of response to gas concentrations but will often increase zero drift. Thus, for each measuring principle used, the combination of variables affecting its performance is optimised for the particular application. Nevertheless, the basic principles being used in gas sensing fall into about 7 different categories.

1 Catalytic (calorimetric) sensors[2]

These are amongst the most widely used sensors; the well-known "pellistor" type based on the measurement of heat released in the catalytic oxidation of reducing gases. These devices will continue, for the foreseeable future, to be the principle method of detecting and measuring methane and other flammable gases at concentrations up to the lower explosive limit. They are, and will be, used in all types of instrumentation; fixed installations, including multiple sensor monitoring and alarm systems, portable monitors and alarms and spotreading instruments.

The advantages of these devices are well catalogued[2]; they are robust, very simple to use, cheap and they have the prime advantage of providing a measure of the hazard in that they give an indication of the explosibility of an atmosphere, irrespective of the gas or gases present[3]. However, the devices have a number of disadvantages. Their sensitivity is limited in that it is difficult to measure concentrations below 500 ppm so that their range of usefulness is limited to 500 ppm up to around the lower explosive limit. At higher concentrations, above the stoichiometric concentration of reducing gas in air, there is a problem of ambiguous indications in that the reading begins to fall as the oxygen concentration decreases. In addition, the power consumption is relatively high, which limits the numbers which can be operated using intrinsically safe power supplies. Finally, the devices can be adversely affected by gases containing silicon, phosphorus, lead, sulphur, etc., which either forms solid residues on the surface of the catalyst or are strongly adsorbed onto the surface. This disadvantage is important in many industrial applications and a great deal of research and development has gone into producing sensors which are resistant to such "catalyst poisons". However, in coalmining, where the primary gas being measured is methane, effective protection can be afforded by an

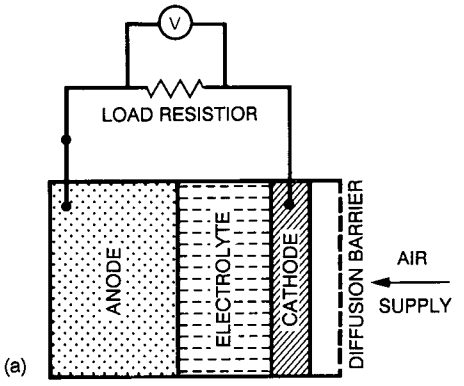
absorbant layer, such as carbon, which removes the poisoning gas before it reaches the sensor.

Longer term developments in catalytic sensors will probably use the micro-fabrication technique developed in the electronics industry to produce sensors with significantly lower power consumption. This may only be achieved at the cost of some deterioration in other aspects of performance but when such redesigned sensors are available, it will facilitate their increased use and also allow their incorporation into sensor arrays and total atmospheric monitoring systems.

2 LIQUID ELECTROCHEMICAL GAS SENSING CELLS[4]

These devices are based on the reactions of gases at the electrodes of electrolytic cells. At these electrodes the gases react to form ions in solution and these ions produce voltage or current changes in the cells which are measurable and related to the gas concentration. The most commonly used cells are amperometric ones shown schematically in Figure 1.

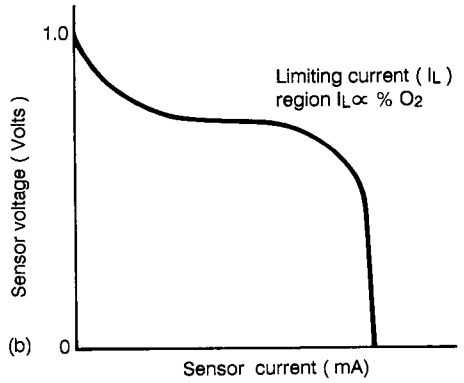
The gas supply to the active electrode is severely restricted by a diffusion barrier, either in the form of a permeable membrane or a capillary feed so that the limiting current is governed by the concentration of gas outside the cell. Devices are available for measuring oxygen deficiency, carbon monoxide, chlorine, nitrogen dioxide, nitric oxide and hydrogen sulphide. Oxygen deficiency is the first target for this type of device, closely followed by carbon monoxide. For oxygen deficiency measurement, these cells are superior to any other method and are likely to continue to be so. They also provide by far the best quantitative field measuring method for carbon monoxide in the range 1-100 parts per million. Outside this range there can be problems of lack of sufficient sensitivity and delayed recoveries. They are suitable for use in both personal and background monitors and alarms.



Anode reaction $2\,Pb + 4\,OH^- \rightarrow 2\,PbO + 2\,H_2O + 4\,e^-$

Cathode reaction $O_2 + 2H_2O + 4e^- \rightarrow 4\,OH^-$

Cell reaction $2\,Pb + O_2 \rightarrow 2\,PbO$



Principle of operation electrochemical cell Sensor (a) Current voltage curve (b)

Fig. 1

Development of electrochemical cells for other gases is at an advanced stage and some are commercially available. The ones of interest for mining are those for measuring NO or NO_2; unfortunately, there is at present a problem in detecting the low levels of interest, i.e. less than 5 ppm NO_2.

The advantages of this type of device are that they are stable, reproducible and highly sensitive and are therefore capable of providing a quantitative measure of some gases. They also require very little electrical power and are thus

suitable for use in all types of equipment, including personal monitors and alarms.

The diavantages are that the operating life of the cells is relatively short - about 1 year - and they are susceptible to interference, particularly from hydrogen and hydrogen sulphide. Finally, the cells are relatively expensive.

There will be continuing developments of amperometric cells, although the development effort on this type of sensor is small, compared to some other types. Increases in the lifetime of cells and their robustness can be expected. Selectivity will be further immproved through use of different electrode materials and the use of multiple electrodes. Because they are not compatible with micro-fabrication techniques, these types of devices are not likely to be incorporated into miniaturised sensor arrays but they are very likely to be used in multiple discrete sensor, total atmosphere monitoring systems.

There is considerable activity in the development of electrochemical systems where voltage changes are produced by the material to be measured. The principles used in ion - selective electrodes, have been combined with the technique of field-effect transistors, to produce a range of devices for use in biological monitoring applications, e.g. glucose in blood, etc. Little application of this technology to gas detection has yet taken place.

3 SOLID STATE ELECTROCHEMICAL DEVICES[5]

The best known devices in this category are those based on oxygen ion conduction in zirconia. A zirconia wall separates 2 volumes of gas, one of which contains a known oxygen partial pressure. The oxygen partial pressure of the unknown is obtained from the Nernst potential generated across the cell. These devices are primarily used for point measurement of the stoichiometry of the fuel supply to internal combustion engines and in the measurement of oxygen levels in the flues of oil and gas-fired boilers. The advantages of these devices is that they are very robust, eminently well suited for the above applications and being solid state, they lend themselves to micro-fabrication techniques. The disavantages are that they operate at high temperatures, greater than 600°C and would require very high electrical power levels for atmospheric monitoring applications. The principle to date is limited to oxygen measurement, which is adequately catered for in environmental monitoring by liquid electrochemical sensors. It is highly unlikely that the present generation of solid state electrolytic sensors will find use in the mining industry. Future developments in this field will come from the use of a range of different ionic conducting materials, including those in which proton conduction is prevalent. This could result in extending the range of gases which can be detected and, almost certainly, will reduce the operating temperature. Although no obvious applications of this type of sensor is apparent in the mining context, it is nevertheless a field in which much research and development work is going on and progress should be kept under careful observation.

4 SEMICONDUCTOR SENSORS

(a) Inorganic semiconductors (metal oxides[6])

When gases adsorb or reach with adsorbed ionic oxygen on the surfaces of semiconducting metal oxides, such as tin oxide, zinc oxide or ferric oxide, the surface electrical conductivity of the oxide is changed. Devices based on this effect have found numerous applications in a wide range of industry but despite intensive research and development effort, especially in Japan, have not as yet

realised their perceived potential. Their use has been largely confined to providing qualitative or at best, semi-quantitative indication of the presence of pollutants in an atmosphere. There have been numerous attempts in the mining industry to make use of these types of sensors and the most succesful application to date has been in providing an early indication of onset of heating or fire through the detection of unspecified products of heating or combustion, thus making use of their very high sensitivity and also making a virtue of the lack of selectivity. Even in this type of application, there are problems in interpreting the indications obtained and avoiding false alarms.

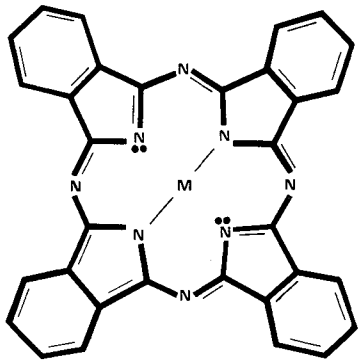
The advantages of this type of sensor are that they are very sensitive, concentrations as low as a few ppb of some gases can be detected, the sensor life is long, they are cheap and being semiconductors, they are compatible with micro-fabrication techniques.

The main disadvantages are their almost total lack of selectivity and absence of any correlation between the sensitivity to a gas and the hazard presented by it. The information they provide is therefore not quantitative. In some instances, there have been problems of long-term electrical instability in these devices but these have been largely overcome by improved materials technology. However poor reproducibility of behaviour between sensors remains a problem.

Future developments in this area will derive mainly from improved materials technology and fabrication techniques. Any remaining problems of electrical stability and those of sensor reproducibility, should be overcome. Different materials may provide more selective sensors, although a high degree of selectivity, i.e. sensitivity as one gas in a complex mixture of gases, is highly unlikely. The use of thick film and multi-layer devices could lead to devices capable of multiple gas resolution. The main uses are, however, likely to be in early fire detection, as part of multi-sensor systems and eventually in single chip sensor arrays.

(b) Organic Semiconductors[7]

As for inorganic semiconductors, gases adsorbing on the surfaces of organic materials, induce a change in the electrical conductivity of the solid. Organic materials have one particularly attractive feature; in principle, it should be possible to modify the electrical and chemical properties of the materials so that they react selectively with specific gases by substitution of selected functional groups onto the parent molecules.



Structure of metal phthalocyanine

Fig. 2

To date, investigations have been largely confined to metal phthalocyanines (Fig 2) and similar macromolecules. This is primarily because of the relatively high conductivity of these materials and their good thermal stability (stable in air to more than 400°C). They are p-type semiconductors and films deposited by vacuum sublimation have been shown to be highly sensitive to strongly electron accepting gases, such as nitrogen dioxide and chlorine, while being insensitive to electron donor gases, such as

hydrogen, carbon monoxide and hydrocarbons. Sensors based on this principle should prove useful in meeting the requirements for oxides of nitrogen monitoring in the mining environment.

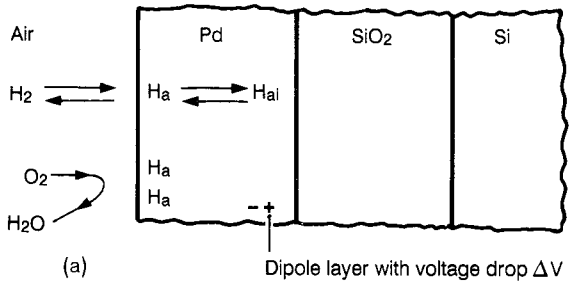
The advantages of these devices are their very high sensitivity (ppb NO_2), good stability, reproducibility and selectivity. They are also cheap, relatively easy to use and are compatible with micro-fabrication techniques.

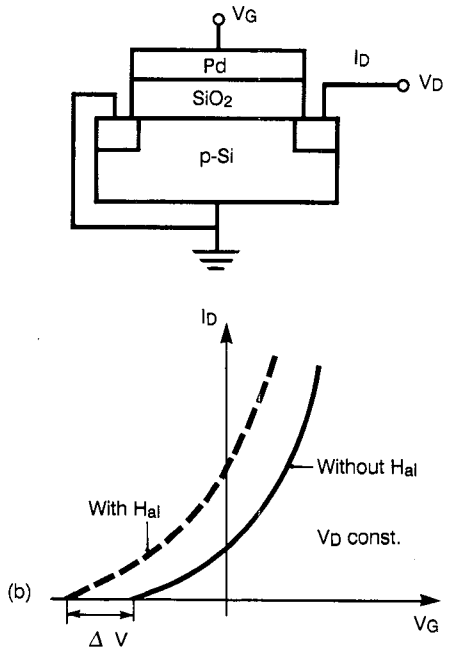
The disadvantages are that there is evidence of slow recovery from high concentrations of gas exposure so that there is a low upper limit to the range of use (less than 100 ppm) and hence they cannot be used for direct monitoring of exhaust gases from diesel engines. However, recent work on the structure of the phthalocyanines used in the devices, has indicated that these types of problems can be overcome[8]. The present generation of sensors require relatively high electrical power and are not suitable for personal monitoring, although they can be used in battery-powered spot-reading instruments. They are insensitive to nitric oxide and hence are not ideal for total oxides of nitrogen measurement. There is some evidence of water sensitivity but the changes in humidity usually encountered in normal use are not expected to constitute a serious problem.

Future developments will initially be in improved response and recovery times and an extension of the usable range to higher concentrations of nitrogen dioxide. It is possible that devices responding to nitric oxide and nitrogen dioxide will be developed to allow a mixture of NO_x to be measured. Such devices will however, be more likely to respond to other gases. The operating temperatures will probably be reduced, with a consequent reduction in power requirements. The use of organic materials for detecting a range of gases is likely in the medium term, and usefulness in the mining industry will depend on which gases the principle can be extended to cover.

5 CATALYTIC GATE FET DEVICES[9]

This type of device relies on the catalytic dissociation of hydrogen or hydrogen-containing molecules at a palladium, or other catalytic metal gate of a field effect transistor.





Principle of operation Catalytic Gate F.E.T. Chemical reactions on the palladium surface, hydrogen transport, and surface and interface adsorption of atomic hydrogen. (a) the effects of the hydrogen dipole layer at the interface on a Pd - MOS transistor (b) reference (8)

Fig. 3

The atomic hydrogen adsorbed on the metal surface diffuses through the thin palladium film to the

metal-insulator interface (Fig 3). When using other catalysts in which hydrogen does not dissolve, the gate is designed so that disociation ocurs at, or products can migrate along the surface to the metal-insulator interface. The hydrogen atoms at the interface are polarised, giving rise to a dipole layer, which corresponds to a voltage drop at the interface. This voltage is effectively added to the externally applied voltage, so that the characteristics of the device are shifted, the shift is proportional to the concentration of gas in the atmosphere.

The sensors are eminently suitable for detecting hydrogen with a high degree of selectivity and could be used as back-up for the electrochemical carbon monoxide sensors which are prone to interference from hydrogen. This problem is particularly serious under post-fire conditions.

These devices have received a great deal of attention, the main reason being that they are silicon-based and hence easily fabricated using techniques developed in the electronics device industry. The main advantages are that they are very sensitive to hydrogen and can be made sensitive to a limited number of other gases, such as hydrogen sulphide and ammonia and the more reactive hydrocarbons. They are cheap to make, take only a little power and are obviously compatible with micro-fabrication techniques.

The disadvantages of the devices are the long response and recovery times and the limited number of gases to which they can be made sensitive. Both these disadvantages, particularly the latter, are primarily due to the temperature limitation on silicon-based devices. However, the selectivity that this affords is a significant advantage in certain applications.

Future developments in this field will be concentrated on the use of different materials, possibly different semiconducting materials or differently doped silicon, and different insulating layer materials - particularly different gate materials. Any one, or all, of these, will change the chemical properties and the temperature range accessible. Together with improved device design, particularly gate construction, these will result in improved response times and extension to other gases, although it is doubtful whether this can be done while at the same time retaining selectivity. Sensors will become available with acceptable sensing characteristics for hydrogen sulphide, ammonia and the reactive hydrocarbons. Improved hydrogen sensitive devices will find uses in the mining industry as part of fire detection and post-fire condition atmospheric monitoring systems. They will form part of multiple sensor systems and sensor arrays. The hydrogen sulphide sensors may be used in a limited number of mines.

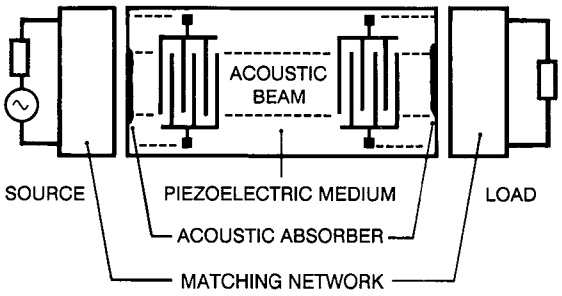
6 Piezo-electric and SAW Devices[10]

When gases or vapours adsorb on the surface of a Piezo-electrical crystal, the minute weight change is reflected in a change in the natural resonant frequency of the crystal. In recent years, the highly sensitive version of this method, the surface acoustic wave (SAW) technique has been developed in which a change of frequency in the surface layer is measured (Fig 4). This is clearly more sensitive to gas adsorption than measurement of the bulk crystal frequency. Since frequency can be measured with a high precision (about 1 in 10^8) the technique can provide a highly sensitive means of gas detection. The method is non-selective, since any gas that is adsorbed on the surface of the crystal will affect the frequency, but some selectivity can be obtained by coating the material with selectively adsorbant materials. However, such materials are at present rare and this type of sensor is currently little used in the

mining environment. Nevertheless, the development of selective adsorbant materials is receiving considerable attention, particularly for military applications.

The main advantages of this technique are its high sensitivity, robustness, ease of use and high compatibility with micro-fabrication techniques. The disadvantages are its non-selectivity, including high sensitivity to water vapour and high temperature sensitivity. Compensation for temperature effects using twin systems, are, however, well established.

Future developments will depend finally on the development of selectively absorbant coatings. This could ultimately lead to a number of sensors, each one sensitive to a particular gas or group of gases. Possible areas of interest in which developments could occur are detecting heavy hyrocarbons, i.e. aliphatic hydrocarbons above C_4 or C_5 and aromatic hydrocarbons and detecting highly adsorptive gases, such as nitrogen dioxide.



S. A. W. dual delay line oscillator

Fig. 4

A sensitive heavy hydrocarbon sensor would be useful in fire detection, a nitrogen dioxide sensor would provide an alternative to the electrochemical and organic semiconductor types. Other than this, it is difficult to envisage much wider use of SAW sensors in the mining industry at present. However, the level of effort being channelled into R & D in this area is high, mainly because of the possible use of this device in integrated sensors and sensor arrays and useful sensors for a wide range of gases could ensue. It is unlikely that this will be achieved in the medium term.

7 Humidity Sensors

Humidity sensing has received much attention in recent years[11]. This has been with a view to controlling air conditioning systems, greenhouse atmospheres and cooking processes. This means that improved sensors, based on a number of techniques, have become available. Most of these utilise the change in electrical conductivity produced in a material following the absorption of water. Although there is no need usually to monitor humidity in the mining industry, so many other sensors are affected by water vapour, that accurate humidity sensors to allow compensation for humidity effects, could be important. There is no doubt that such sensors would form an integral part of any multi-sensor or sensor array system.

8 Multi-Sensor Systems

As more reliable sensors have become available a number of instruments which utilise 2 or more sensors to provide an indication of specific gases, have appeared on the market. They are usually known as multi-gas instruments. Common combinations of sensors utilise 2 or 3 from a menu of catalytic sensors for flammable gases, electrochemical sensors for oxygen deficiency, electrochemical sensors for carbon monoxide and semiconductor sensors for general toxic gases or specifically for hydrogen sulphide. It is unusual to try to cover more than 3 gases, but such instruments covering in 1 package flammable gas detection, oxygen deficiency detection and carbon monoxide measurement, should prove extremely useful in the mining industry for rescue work in post-fire

conditions. These instruments are simply bringing together 3 separate instruments, there is no attempt to bring the data from all the sensors together and assess any implications of the combined information.

A number of investigators have, however, proposed and assembled systems of discrete sensors. Systems using different types of sensors and systems using similar sensors, operated under different operating conditions, are being used so that the combined information provides unambiguous indications of hazardous conditions. Even more complex systems are being used for providing profiles of the ambient atmosphere. In order to provide indications of hazards, both the hazard itself and the environment in which it occurs, must be defined. This allows a choice of sensors to be made on the basis of both the environment in which the measurements are to be made and of available sensors. An example of such a system proposed for the mining industry was aimed at early unambiguous indication of heating or fire[12]. Metal oxide sensors were used to provide an indication of the increase in the levels of products of combustion and fitted with carbon filters to provide an indication of carbon monoxide levels, whilst organic semiconductor sensors with and without oxidising (CrO_3) trap to give an indication of NO_x and NO_2 respectively. The latter respond to the fumes from diesel engines and shotfiring products, thus provide a means of identifying changes in the metal oxide sensors induced by these non-hazardous processes and eliminating the two main sources of ambiguity.

In the future, the increased availability of a wider range of improved sensors, will make such systems attractive and applicable to a wider range of hazards and conditions. One can already envisage adding electrochemical, carbon monoxide and oxygen sensors to the above system and together with catalytic sensors for flammable gases, building up a picture of the atmosphere at different locations. This will provide a means of intelligently assessing a wide range of hazards and hazardous conditions. Assemblies of such systems, each one with a microprocessor data handling facility feeding back to a central processor, would constitute intelligent systems whereby the atmosphere of a whole mine is monitored and the data interpreted so that alarms, shutdowns and other procedures can be implemented automatically.

9 Multi-Sensor Arrays

A number of sensor types lend themselves to micro-fabrication techniques. This provides the possibility of constructing multi-sensor systems, consisting of one or more of these types on a single chip and/or integrated with control and interogating circuitry. These will constitute complex sensor arrays, the data from which can be interpreted by microprocessor data handling and pattern recognition techniques, to provide qualitative or semi-quantitative indications of the atmospheric constituents. Such systems are at present confined to laboratory investigations[13] and are likely to remain so in the near future. The mechanisms associated with many of the sensor types need to be better understood and the technology has to be improved to provide reproducible single sensors before arrays become practically feasible. Data processing and deconvoluting techniques and pattern recognition techniques, which are suited to this type of array, have also to be developed. There is no doubt that in the longer term, it will be possible to replace systems of discrete sensors, by a more powerful system of sensor arrays. This will provide for the mining industry as for other industries, extremely powerful total atmospheric monitoring facilities.

Spectroscopic/Optical Techniques

No discussion of gas sensors in relation to any industry, would be complete, without consideration of spectroscopic or optical techniques. This is a field in which rapid changes are taking place through developments in optical fibres, in laser and luminescent device technology, in filter systems and in radiation detection systems. The effect of these developments can be considered in five main areas.

(1) Fibre Optic Based Sensors[14]

These utilise optical changes, either in the absorption properties or luminescence induced by gas adsorption on solids which can be monitored using optical fibre techniques. Both absorption changes and luminescence are relatively easy to monitor, using either double or single fibre systems. Although significant numbers of sensors are becoming available for monitoring specific ions in liquids, very few such systems have been developed for gas sensing. The development of gas sensors, depends very much on the development of suitable gas sensitive coatings. This is likely to be a long and difficult progression but the benefit in terms of selective sensors could be substantial.

(2) Conventional Infra-red Systems

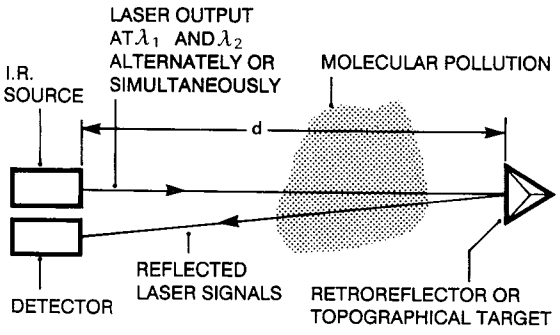
Improvements in filters and detectors and the use of electro-luminescent and solid state laser sources, as well as improvements in conventional ones, together with advances in instrument engineering, have resulted in significant reductions in size, complexity and fragility of non-dispersive infra-red analysers. Such instruments for hand-held spot-reading and monitoring are already close to being available. In the medium term, a range of instruments of interest to the mining industry will become available for measurement of methane, carbon monoxide, carbon dioxide and oxides of nitrogen and combinations of these gases.

(3) Distributed Infra-red Absorption Systems

The use of optical fibres to transmit radiation over considerable distances, provides a means of conveying radiation from a single source to a number of points at which infra-red absorption cells can be located for selected component analysis. Losses in the fibre can be compensated for by using radiation at two frequencies, only one of which corresponds to the absorption peak of the gas concerned. The attenuated radiation is also returned to the control point for interpretation, either via the same fibre by reflection or via a separate fibre link. A multi-point analysis system can therefore be assembled, which could replace, for example, the tube bundle[15] system currently used in British pits, with significant advantages in terms of obtaining data in real time. Hitherto, absorption systems have been limited to working on harmonics of the absorption wavelengths but fibres with high transmission well into the infra-red fingerprint region are becoming available.

(4) Open-Path Monitoring

A number of open-path infra-red absorption systems, both single-ended, i.e. source and detector at the same location and double-ended, i.e. source and detector on opposite sides of the area surveyed, have become or are becoming available.



The open path absorption scheme

Fig. 5

The single-ended systems which are most common, use a retro-reflector to reflect radiation back to the detector, or alternatively, rely on reflection from selected topographical objects (Fig 5). These systems provide an average concentration measurement over the path used; this has advantages, particularly in terms of overall area or site monitoring but of course can miss a small volume of explosive or toxic concentration. Compensation for changes in infra-red absorption due to factors other than concentrations of the gas to be measured, can be achieved by using twin beams of infra-red radiation, one at the absorption frequency and one slightly away from it.

Single-ended systems relying on back reflection from existing objects could prove extremely useful for checking for gas in inaccessible or dangerous locations, e.g. under unsupported roofs in the mining industry. Such a system has indeed been developed.

(5) Visualisation Techniques

A number of visualisation techniques are being developed but the one of interest to mining is probably the one more correctly termed the imaging technique, in which an extended source of infra-red radiation, or alternatively, background infra-red imaging devices, to provide an image of the distribution of gas in a given location. The systems use imaging devices, with sensitivities primarily in one of the absorption bands of the gas to be measured, or, alternatively, use of infra-red radiation within the same absorption band. Similar information can be obtained using a single-ended system, with a scanning source to irradiate the areas of interest and relying on background reflection to provide the input to the images. This is effectively extending the open path single beam system into 2 dimensions. Extension to 3 dimensions is possible but requires access from orthogonal directions and may not be particularly relevant and would certainly be difficult in the mining environment.

CONCLUDING REMARKS

In this paper we have attempted to provide an indication of the types of sensors which are becoming available and of the developments which are likely to take place in the future. An attempt has also been made to predict the possible ways these developments can benefit the mining industry. Such predictions are, of course, very subjective and could be completely invalidated by a single development in an unforeseen direction. Nevertheless, we see the development of multi-sensor systems, followed by multi-sensor array systems in combination with complex data processing and pattern recognition techniques, having increasing impact on the industry. These developments, together with developments in gas visualisation techniques, will be the two directions in which significant progress can be expected.

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ORIENTATIONS DU DEVELOPPEMENT DES DETECTEURS DE GAZ UTILISES DANS LES MINES

J.G. Firth, T.A. Jones

Ces dernières années, des détecteurs chimiques destinés à diverses applications dans de nombreux pays du monde ont été développés. Si la plupart de ces applications se situent hors de l'industrie minière, leurs exploitations technologiques peuvent potentiellement servir à la détection des gaz pour cette industrie, en particulier à la mesure du méthane, de l'oxyde d'azote, à la détection d'incendies miniers, à l'évaluation de l'insuffisance en oxygène, ainsi que, de plus en plus, à la surveillance générale de l'atmosphère minière.

Ce rapport examine le développement de techniques telles que les appareils d'ondes acoustiques de surface, les transistors d'effets de champs à sensibilité chimique, la surveillance en circuit ouvert et les techniques de visualisation, ainsi que les appareils semi-conducteurs, catalytiques, électrochimiques et à rayons infra-rouges, mieux connus. Il examine les avantages et les inconvénients de ces techniques de mesure des gaz et étudie les directions possibles des futurs développements.

TRENDS IN DER ENTWICKLUNG VON GASSENSOREN FÜR DEN UNTERTAGE-EINSATZ.

J. G. Firth, T. A. Jones

In den letzten Jahren schritt die Entwicklung von chemischen Sensoren für eine Vielzahl von Anwendungen immer weiter voran. Die meisten der Anwendungen liegen außerhalb des Bergbaus, doch sind die technologischen Entwicklungen potentiell für Gassensoren in Gruben anwendbar. Dazu gehören vor allem Anwendungen für Methangasmessung, Grubenbrandaufspürung, Sauerstoffmangel-Warngeräte, NOx Messungen und in zunehmendem Maße die Überwachung der gesamten Grubenwetter.

Der Vortrag beschreibt die Entwicklung der Technologie für akustische Warnsignalgeräte, chemisch empfindliche Feldeffekttransistoren, "open path" Überwachung und Anzeigetechniken, sowie die besser bekannten Halbleiter, Katalysatoren, elektro - chemischen und Infrarot-Geräte. Die Technologie wird auf Vor - und Nachteile der Anwendungen in der Grubengasmessung untersucht, und mögliche Wege für zukünftige Entwicklungen werden vorgeschlagen.

ОБЩЕЕ НАПРАВЛЕНИЕ В РАЗВИТИИ ГАЗОВЫХ ДАТЧИКОВ В ГОРНОЙ ПРОМЫШЛЕННОСТИ

Дж.Г. Фирт и Т.А. Джоунс

В последние годы во многих странах наблюдается непрекращающаяся работа по созданию химических датчиков для применения в различных областях. В основном, они не применяются в горной промышленности, но имеются потенциальные возможности их применения для обнаружения газа в шахтах. Областями применения являются измерения содержания метана, обнаружение пожаров в шахтах, измерения недостатка кислорода, измерения окислов азота и, наконец, наблюдение за общим состоянием воздуха в шахтах.

Настоящий доклад описывает в общих чертах результаты разработки приборов, замеряющих поверхностные акустические волны; химически чувствительных передатчиков, регистрирующих полевые эффекты; методов открытого наблюдения и воспроизведения зрительных образов, а также более известных приборов, основанных на полупроводниках, каталитических, электрохимических и инфракрасных измерениях. Рассматриваются сильные и слабые стороеы применения этих приборов и методов для измерения количества газа в шахтах, а также обсуждаются возможные направления для будущих исследований в этой области.

矿用气体探测器发展的趋势

J．G．弗尔士 及 T．A．琼斯

过去几年很多国家都在发展各种不同用途的化学探测器．大都的用途并不在煤矿方面，但技术上的发展很容易用在矿里的气体探测需要．这些主要是甲烷探测，矿火侦测，缺氧测量，氮氧化物测量，和渐渐需要的，整个矿内空气的侦测．

本文将叙述地面声波仪，化学敏感的实况晶体管，开路侦测和显影等技术，以及半导体，催化性，电化学，和红外线为根据的仪器．并将讨论它们对矿内气体测定时各种方法的强点和弱点，以及将来发展的方向．