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Remedial and Strata Replacement Techniques on Longwall Faces

A Report on the State of the Art

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

cm	centimeter	m^2	square meter
ft	foot	m^3	cubic meter
h	hour	min	minute
in	inch	mm	millimeter
kg	kilogram	pct	percent
kg/L	kilogram per liter	ppm	part per million
kN	kilonewton	s	second
kPa	kilopascal	t	metric ton
L	liter	t/h	metric ton per hour
lb	pound	yr	year
m	meter		

REMEDIAL AND STRATA REPLACEMENT TECHNIQUES ON LONGWALL FACES

A Report on the State of the Art

By Robert S. Dalzell[1] and Ernest A. Curth[2]

ABSTRACT

Following the introduction of roof shields to the U.S. longwall mining scene in 1975, the number of shield faces (faces supported by shields) has increased steadily, and the occurrence of accidents associated with ground control inadequacies at longwall operations has shown a downward trend. However, ground control problems in longwall mining still exist. New mining systems, however efficient, have run afoul of roof control. Shield faces have suffered extended stoppages because of roof cavities.

In this report, the Bureau of Mines addresses the problem of longwall roof control. The report presents a review of foreign remedial and strata replacement techniques for longwall faces and evaluates their potential for domestic application. The technology reviewed includes meshing, chemical rock stabilization, cribbing, strain-absorbent modules, inflatables, synthetic foams, and monolithic fills.

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INTRODUCTION

Table 1 shows the recent decrease in the number of accidents associated with longwall ground control. The advent of shields (figs. 1-2) to the U.S. longwall mining scene in the seventies and their predominance in the eighties probably accounts for much of the accident reduction. The development of roof shields in West Germany preceded and paralleled the rapid adoption of roof shields by U.S. miners. In 1982, 85 pct of the production from West German coal mines came from shield faces (1).[3]

Shields have several safety and productivity advantages over chocks. Shields provide--

- A sheltered working space requiring minimum cleanup work.

[3]Underlined numbers in parentheses refer to items in the list of references at the end of this report.

TABLE 1. - Incidence of longwall ground-control-related accidents, 1977-82

Year[1]	Total faces	Shield faces		Ground-control-related accidents[2]
		Number	pct	
1977 (2)	77	15	19	143
1978....	NC	NC	NC	59
1979 (3)	91	40	43	60
1980 (4)	105	57	54	89
1981....	NC	NC	NC	73
1982 (5)	112	93	83	87
1983 (6)	118	99	84	71

NC No census available.

[1]Underlined numbers in parentheses refer to censuses of longwall faces listed in the references at the end of the report.

[2]Source: Health and Safety Analysis Center, MSHA, U.S. Department of Labor.

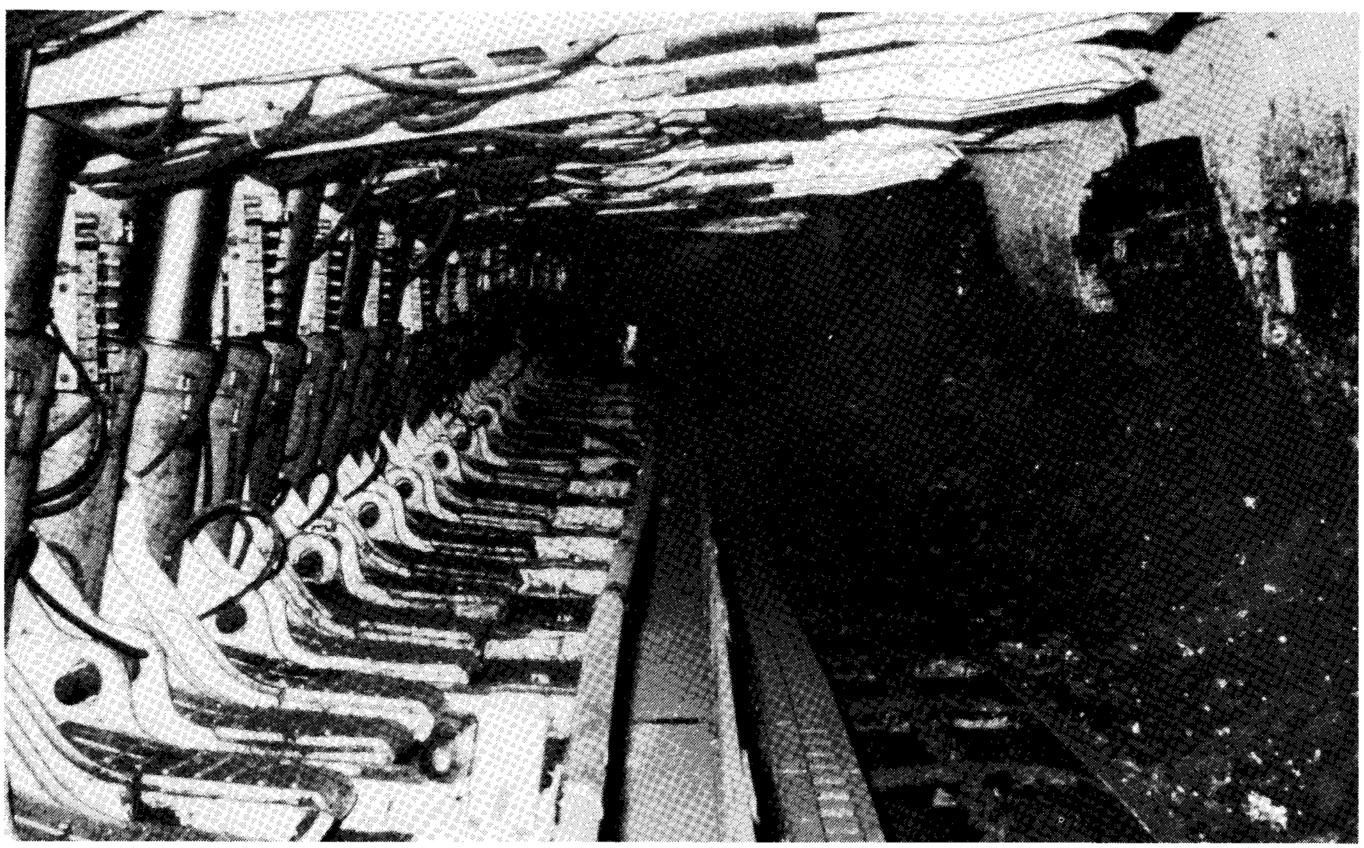

FIGURE 1. - Shield face with chainless haulage rack.

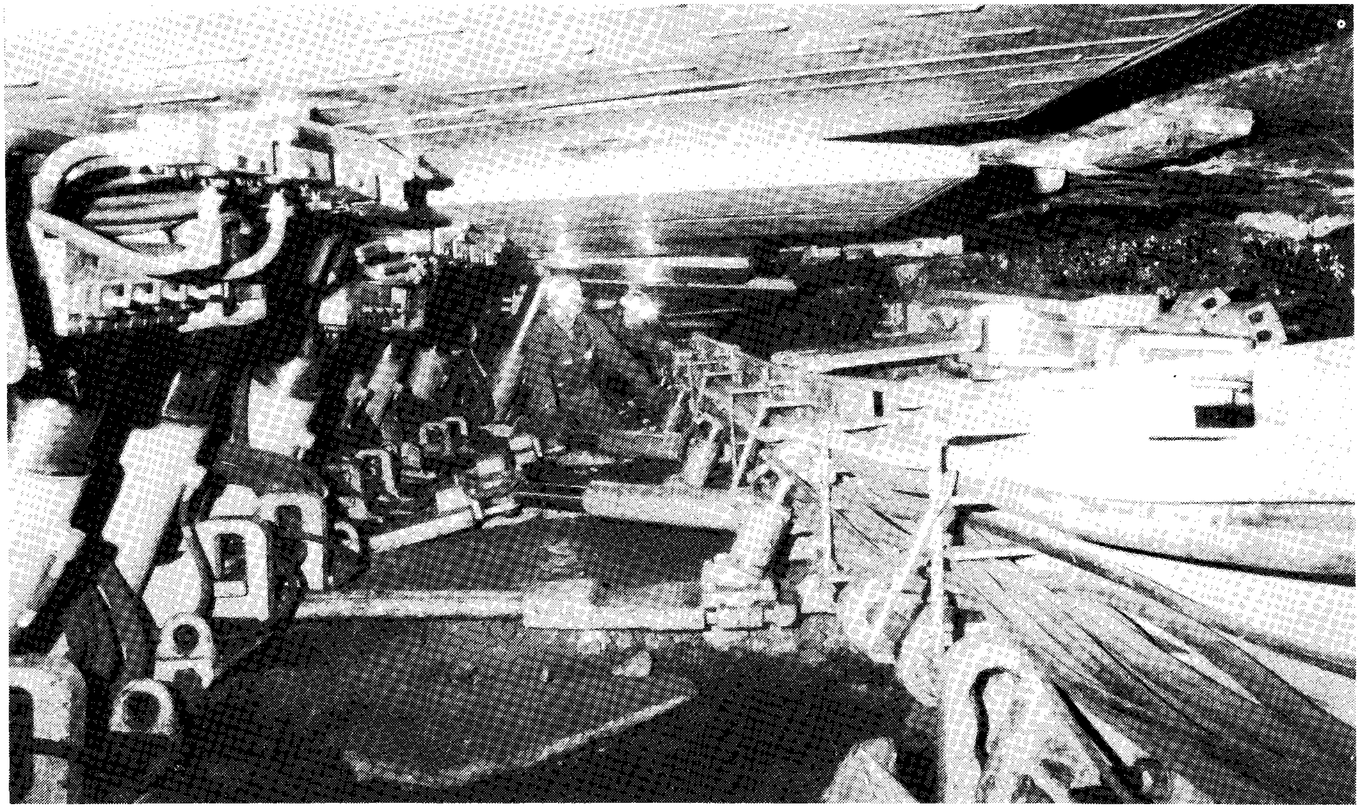

FIGURE 2. - Thin-seam shield face with plow.

- Structural stability that allows advancing without delay, even with brushing roof contact.

- Lemniscate gear that provides a narrow and nearly equal span of exposed roof ahead of the canopy over the entire vertical range.

- A less severe "trampel" effect. (West German miners use this expression to describe roof deterioration above powered roof supports caused by frequent setting and lowering.) Shields take fewer steps walking from gob to face, because shield canopies are shorter than chock canopies.

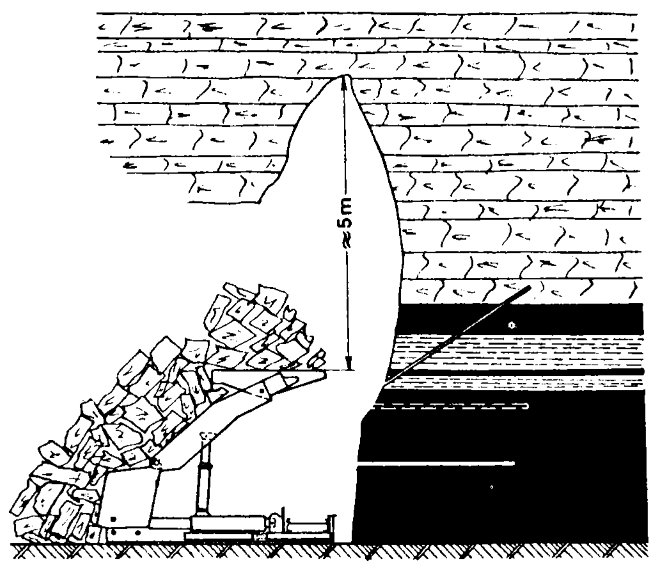

FIGURE 3. - A longwall face in a void.

However, a persisting problem is roof cavities above the roof supports. These cavities increase both in height and expanse, and the problem is exacerbated because the wide canopies of shields restrict access to the cavities for remedial work. Roof rock deterioration, if untreated, can lead to rock breaking forward in front of the canopies (fig. 3). Such deterioration must be prevented. Otherwise, long delays to restore roof control will impair productivity and induce safety hazards.

Results of studies on 132 longwall faces indicated the severity of the strata deterioration problem in West German mines. Cavities more than 50 cm high over more than 10 pct of the face length were observed on numerous longwalls and were found to be related to the support capacity and the span of exposed roof ahead of the canopies (7).

Great Britain's National Coal Board (NCB) was confronted with the same problem, uncontrolled cavities above and ahead of the shields, when it introduced shields to U.K. mines in recent years. Task forces appointed to research remedial techniques conducted operations studies on numerous faces (8).

The Bureau of Mines recognizes the need for remedial and strata replacement techniques for longwall faces in the United States and is reviewing foreign technology in order to evaluate its potential for domestic application. The parameters being considered are minimum exposure of workers to hazardous roof, effectiveness, speed, simplicity, and cost. The goals of the Bureau's research on remedial and strata replacement techniques are long range: to enhance safety, productivity, and resource recovery where extraction is vulnerable to ground hazards. Future research efforts will be directed to fitting remedial systems into the mining cycle, the development of chemical foam without harmful side effects to miners, and the technology of monolithic fills.

ACKNOWLEDGMENTS

The authors are indebted to Kirk McCabe, product manager, Mobay Chemical Corp., Pittsburgh, PA, and Derek Tuke, president, Micon Services, Inc., Pittsburgh, PA, for valuable information on chemical strata consolidation. The authors acknowledge permission granted by Gluckauf to use figures 3, 10-11, and 22; by Tinsley Wire (Sheffield) Ltd. to use figures 4-5; by the NCB, including the Mining Research and Development Establishment, to use figures 13-15, and 17-19; and by The Mining Engineer to use figures 20-21, and 23-24.

WIRE MESHING

In the early days of powered roof support, caved rock flushing in from the gob and debris of friable stratum dropping out between the canopies created a hazard in the working area and required labor-intensive cleaning of the operating space so the roof supports could be advanced. The "trampel" effect (roof deterioration above powered roof support) was, and continues to be, particularly harmful.

Prior to the advent of roof shields, European miners stretched wire mesh of high strength and adequate elasticity over the canopies to hold the fragile roof material and thus prevent the harmful ingress of debris into the travel space. A report on West German mining technology in 1971 described four faces where the roof space was covered with wire mesh. On one face, the wire mesh was paper backed to shelter the face area from dust originating from the gob during advance of the roof supports (9). In 1973, West German statistics on longwall operations indicated that 31 pct of a total of 159 active faces with powered supports required the use of wire mesh (10).

Two types of wire mesh were in use, ordinary wire netting of 50-mm mesh and 1.5- to 2.5-mm wire (16 to 12 SWG), which is readily available, and welded wire mesh with 40-mm mesh and 1.5-mm wire (16 SWG). Since it is welded at the knots, welded mesh is rather stiff, and therefore comes in sections only 1.5 m wide and 25 m long. Welded mesh weighs less than wire netting of comparable strength and suffers less sag because of its stiffness.

Threading wire mesh over the canopies and splicing the sections together is highly labor intensive. Special steel clips are used, and West German mining research has developed a plier-like tool, combined with a clip dispenser, to connect the sections together. To reduce the time required for splicing, 4-m-wide sections instead of the usual 1.5-m ones were fabricated and folded into 20-cm-diam rolls to facilitate transportation, storage, and handling on the face.

In Great Britain, the Mining Research and Development Establishment developed mechanical mesh laying to reduce the time-consuming, arduous, and hazardous manual effort; and a report from 1980 indicates that this technique was used in 17 collieries on 25 faces (11). Face manpower was reduced and performance greatly improved. The usual practice of leaving a substantial layer of roof coal to hold a fragile rock stratum could be abandoned, and instead the shearer cut up to the immediate roof, resulting in better resource recovery.

A 50- by 50-mm welded wire mesh with 1.5-mm wire (16 SWG) is recommended for mechanical mesh laying. A section (75 m long and 70 cm wide) is supplied in a 48-cm-diam roll that weighs 32 kg.

The latest version of a mechanical wire mesh applicator was developed at the Blidworth Mine in northern Nottinghamshire County, U.K. (12). It consists of a holder framework that accepts a roll of wire mesh (fig. 4). The framework, which is attached to a shearer cowl, presses

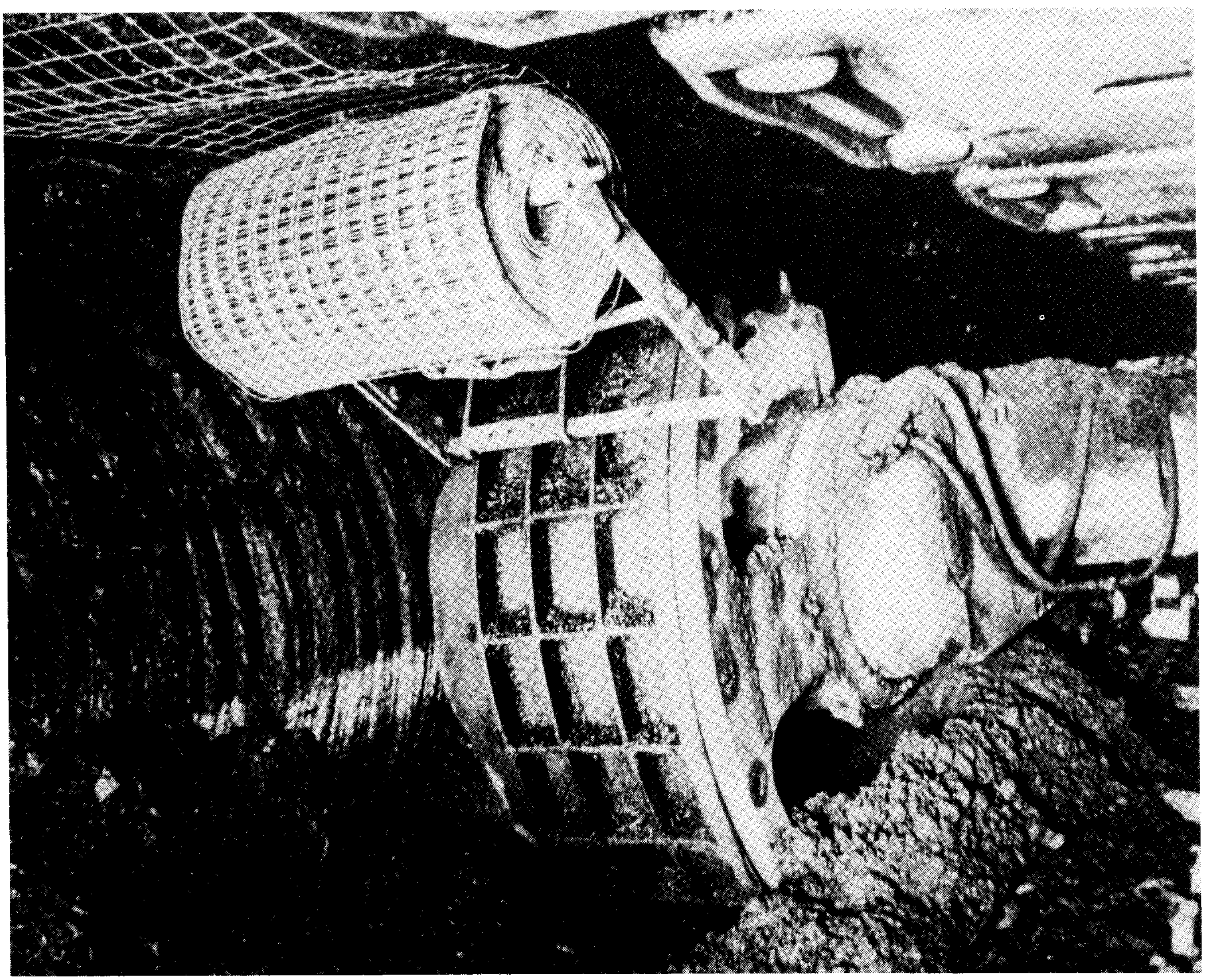

FIGURE 4. - Mechanical wire-mesh applicator.

the wire-mesh roll to the roof by spring-loaded arms so the roll can follow the outline of the roof. The spring-loaded arms also allow enough yield to keep damage from falling rock to a minimum. One end of the wire mesh is secured to the face-end roof support, and, as the shearer traverses the face, the wire mesh is pulled off the roll with very little sag. The roof supports can then be advanced under the mesh, and the shearer can cut without interference.

To replace a wire-mesh roll, the shearer operator raises the ranging drum that lets the cowl turn and drop downward; the face crew can then reload the mesh applicator. To resume mesh application, the operator maneuvers the cowl back into the operating position, pressing the wire-mesh roll against the roof. The face workers then clip the wire mesh from the new roll to the already-installed mesh. The mesh applicator can be easily detached from and remounted to the cowl at the face ends, when the cowls are reversed.

Handling the wire mesh and splicing the sections together still requires a great deal of manual effort. But the hazardous part of the work, namely, threading the wire mesh over the canopies with the miners exposed to unsupported roof, is eliminated by mechanical mesh laying. Also, the expense for manpower to remove debris from the supported area is saved.

Of course, roof shields are very effective in sheltering the working space from debris and dust. Their introduction to NCB mines in the late seventies eliminated the need for mesh laying on faces so supported except during salvage operations (shield recovery). Shield recovery poses an unusual problem, because part of the structure is under caved roof. Wire mesh, applied prior to face termination, forms a protective blanket under which recovery can take place unimpeded by debris from the caved roof rock.

In the United States, mesh laying was practiced on a longwall face in the York Canyon Coalbed in New Mexico in 1974. The coalbed was 2 m thick. A wide fault dissected the face and had to be blasted. The roof was stabilized with 1.8-m resin bolts, and wire mesh was stretched over the canopies of the four-leg 4,450-kN chocks to keep debris from dropping between the roof supports (fig. 5). Productivity under these conditions was very low and did not improve as long as the fault intersected the face.

At present, wire mesh is applied to U.S. longwall faces only during the salvage phase of operations that use shields. A protective mat is formed by threading imported wire mesh or domestic cyclone fence over the canopies, beginning 10 shears or 7.5 m from the terminal position of the face. Imported wire mesh is a semitwist 40- by 40-mm square mesh of 1.5-mm wire that is very flexible and easily handled in the confined longwall space; however, it is expensive and not always available. The domestic cyclone-fence-type netting is a 50- by 50-mm square mesh of 3- or 3.8-mm wire (13).

The wire mesh is laid by unrolling the sections either parallel to the face or in the direction of face advance (14) (figs. 6-7). The sections are spliced together with steel clips, leaving 15-cm overlaps. The wire-mesh mat is reinforced with 5- or 15-mm wire rope, which is stretched parallel to the face from gateroad to gateroad and anchored to roof bolts. One wire rope is installed for each shear at 0.75-m intervals. The recovery space is also secured with bridge boards and roof bolts (fig. 8). In thick seams, meshing the face prevents sloughing of the coal (15).

During the recovery phase, the salvage crew builds cribs in place of each removed shield while the wire mesh is held up by hydraulic props (fig. 9).

The following example illustrates how the quantity of wire mesh needed may be calculated: A blanket of wire mesh is to be installed on a 150-m face prior to shield recovery, taking in 10 strips (shears) or 7.5 m. Using 1.5-m-wide rolls of wire mesh, and leaving 15-cm

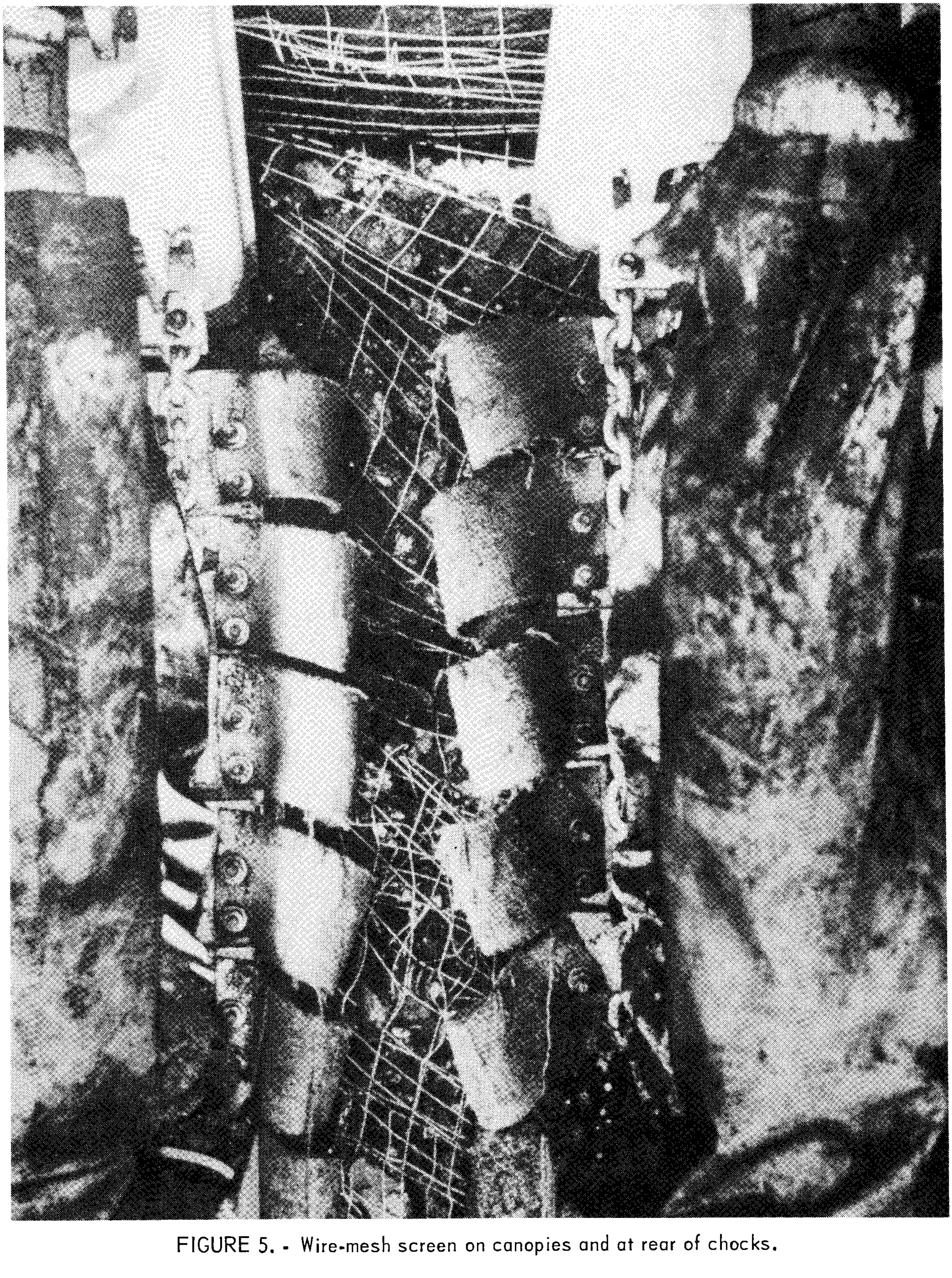

FIGURE 5. - Wire-mesh screen on canopies and at rear of chocks.

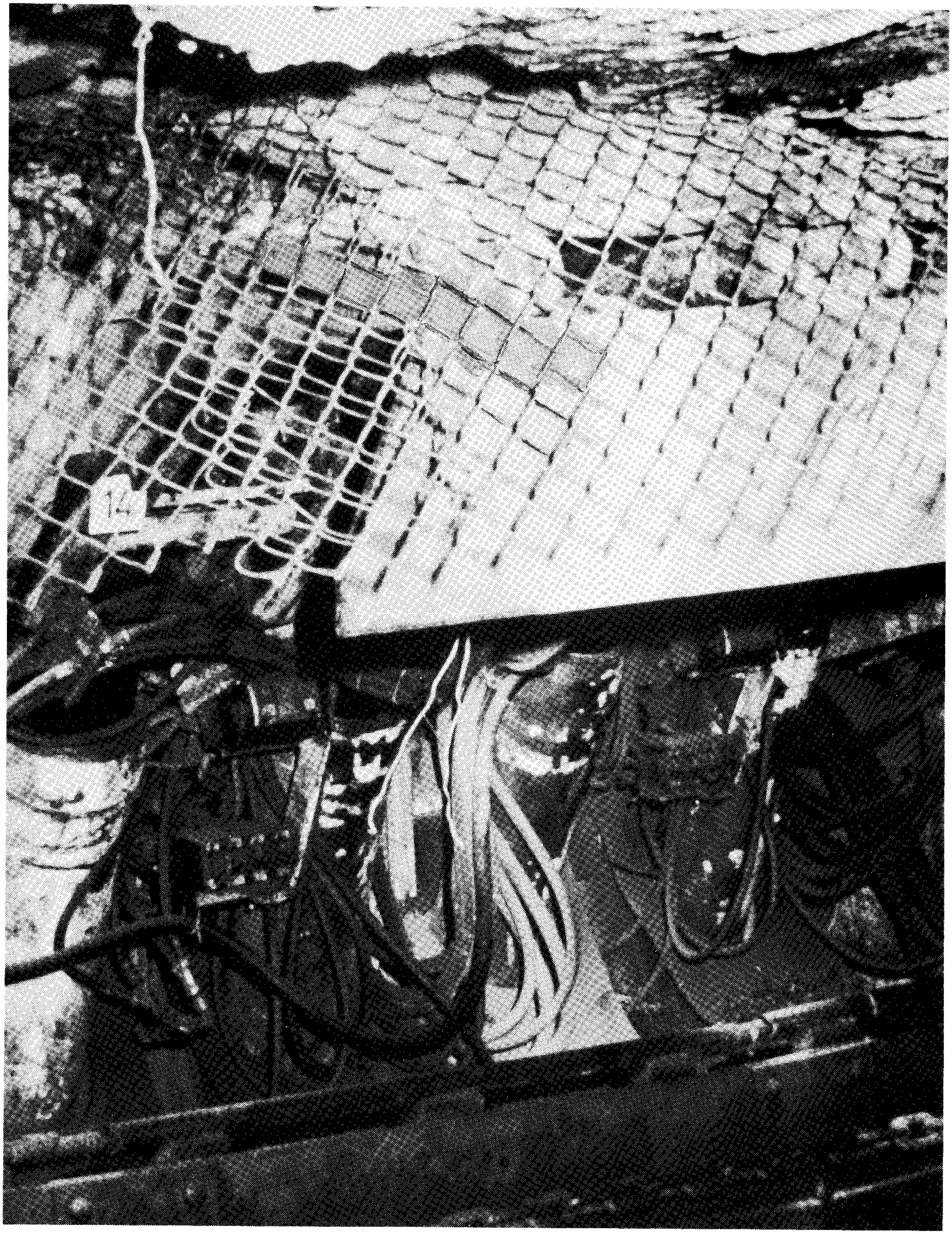

FIGURE 6. - Wire mesh laid on canopies.

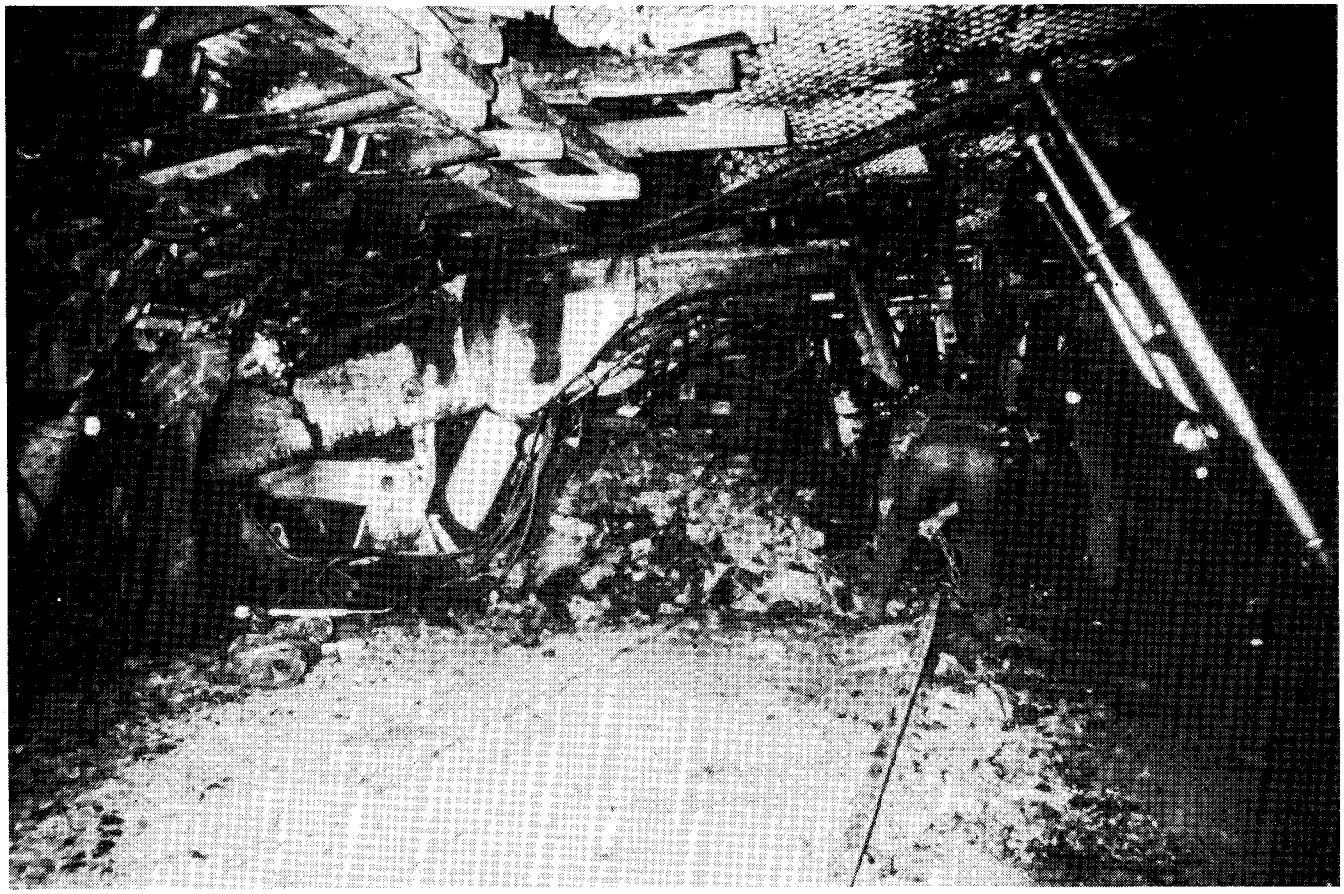

FIGURE 7. - Shield recovery under wire mesh.

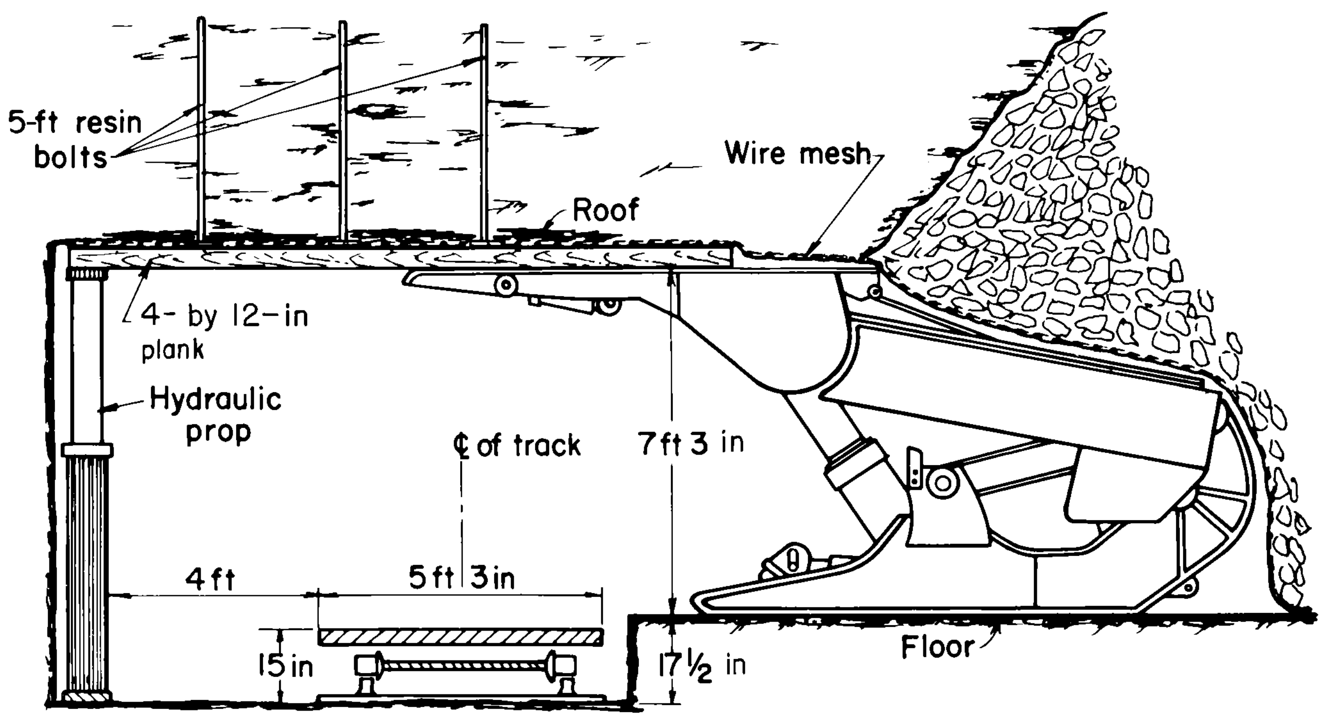

FIGURE 8. - Recovery area secured with wire mesh, bridge boards, and roof bolts.

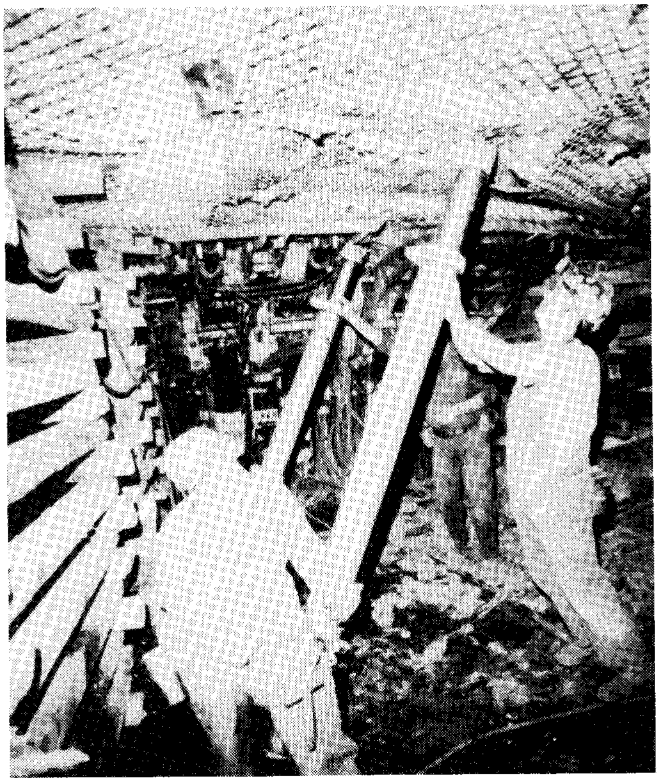

FIGURE 9. - Recovery area secured by cribs under wire mesh.

overlaps, six lengths of mesh 150 m long are needed to cover the roof area. The total quantity of wire mesh is

$$6 \times 1.5 \text{ m} \times 150 \text{ m} = 1{,}350 \text{ m}^2.$$

Domestic products of 50- by 50-mm mesh and 3- or 3.8-mm wire cost $\$3.45/\text{m}^2$ or $\$4.73/\text{m}^2$, for a total of $4,650 or $6,400, respectively. Clips, wire rope, and tools add 15 pct of the above to the material cost. However, material cost is negligible compared to the labor cost for installing the mesh. Hence, a more pliable grade of wire mesh, even though more expensive, could derive benefits because of easier application.

Compared with manual mesh application, mechanized mesh laying, as practiced in the United Kingdom, may reduce hazards and costs in the shield recovery process. A longwall face, supported by shields and mined by a double-ended ranging-drum shearer, has a potential for an average daily output of 3,120 t of coal (from three production shifts). Each 0.75-m strip on a 150-m face extracting 1.80 m of coal contains 260 t of coal, so that four strips are needed to fulfill the set shift production goal.

Ten strips without mesh blanketing could be mined in 2.5 shifts (10 ÷ 4). Manual mesh laying would reduce shift performance to an estimated 1.5 strips, meaning that 10 strips could be mined in 6.5 shifts. Thus, four additional shifts would be needed to accomplish the mesh laying, resulting in a revenue shortfall of $152,000 (4,160 t × $36.50), given the average coal output for four production shifts and the current cost of utility fuel. The opportunity loss for 1 day of stoppage is $7,000, with $8 million invested and a 20-pct return on investment, in addition to $6,000 for face labor. The cost for 1-1/3 days of work interruption (four shifts) would be as follows:

Revenue shortfall	$152,000
Opportunity loss	9,000
Labor	8,000
Total cost	169,000

This total cost does not include equipment-related interest and depreciation. If mechanization could shave off one-third of the mesh-application time, the monetary gain coupled with the hazard reduction would be sizable. All equipment needed for mechanical mesh laying can be easily fabricated in a mine shop.

CHEMICAL ROCK STABILIZATION

Under the effect of mining, coal and adjacent rock tend to break up in blocks along planes of weakness such as cleats, joints, crevices, cleavages, faults, and clay veins. Rock stabilization by chemical means can prevent such disintegration and keep the rock body intact. Polyurethane has been applied in West German mines since the early sixties to accomplish stabilization of fragmented strata (16).

Polyurethane has the following properties that make it suitable for rock stabilization:

- Low initial viscosity, allowing penetration into the smallest crevices;
- High expansion through foaming (allows it to fill all openings);
- Controlled rate of increasing viscosity after mixture (prevents the end product from flowing out of the stabilization space);
- Plasticity (so it follows rock movements); and
- Adherence to rock, coal, wood, cement, and steel.

Polyurethane is a two-component resin applied either in cartridge form or by injection. Cartridges contain the two reactive components separated by plastic tubing. They are 0.3 m long and are inserted into 50-mm-diam boreholes at a ratio of approximately 1.5 cartridges per meter of borehole. For instance, a borehole 3 m long will require 5 cartridges.

After the cartridges are pushed to the end of the hole, a hardwood dowell is inserted into the hole and rotated by the drill for 20 to 30 s to destroy the cartridges, mix the two components, and thus, to initiate the reaction. The borehole is then sealed. Cartridges are mostly used to stabilize coal on longwall faces in thick seams, and thus to counteract the tendency of such faces to hade forward, which can leave a wide span of unsupported roof between canopy tips and the face (fig. 10).

Polyurethane injection is a more general application, suited to the stabilization of fragmented roof rock (fig. 11). The two components of polyurethane, polyisocyanate and polyol, react after mixture at a ratio of 1:1 by volume. The polyisocyanate is a dark-colored fluid. Polyol is a clear fluid and is available in varieties according to starting, reaction, and hardening times, as shown in table 2.

Systems with accelerated reaction are used in rock strata where the process of deterioration has gone so far that the standard-type resin would flow out of the rock before it hardens. Examples of such critical areas might be face ends in retreat longwall mining where abutment stresses preceding the longwall face combine with those from the gob area of an adjacent mined-out panel.

Fast and standard systems are often used in conjunction. Large cleavages are closed quickly by the fast-reacting resin, and the final stabilization takes place after the standard resin is added. Water-resistant systems develop foam quickly to seal the strata against the influx of water.

The chemicals are supplied and transported in closed metal containers, either 19-L cans or 209-L drums that are color coded for identification. The resins are

TABLE 2. - Reactive characteristics of polyurethane systems

(Mixture ratio, 1:1 by volume)

Characteristic	Type of polyol		
	Standard	Fast	Water resistant
Starting time[1]...........s..	180- 360	65	45-65
Reaction time[2]...........s..	1,200-1,800	95	60-70
Hardening time.........min..	90- 120	15	15
Expansion factor[3]...........	5:3	3:2	8:1

[1]Beginning of mixture to start of foaming.
[2]Beginning of mixture to end of foaming.
[3]Foam-to-fluid ratio, by volume.

Source: Meyer (16, p. 832).

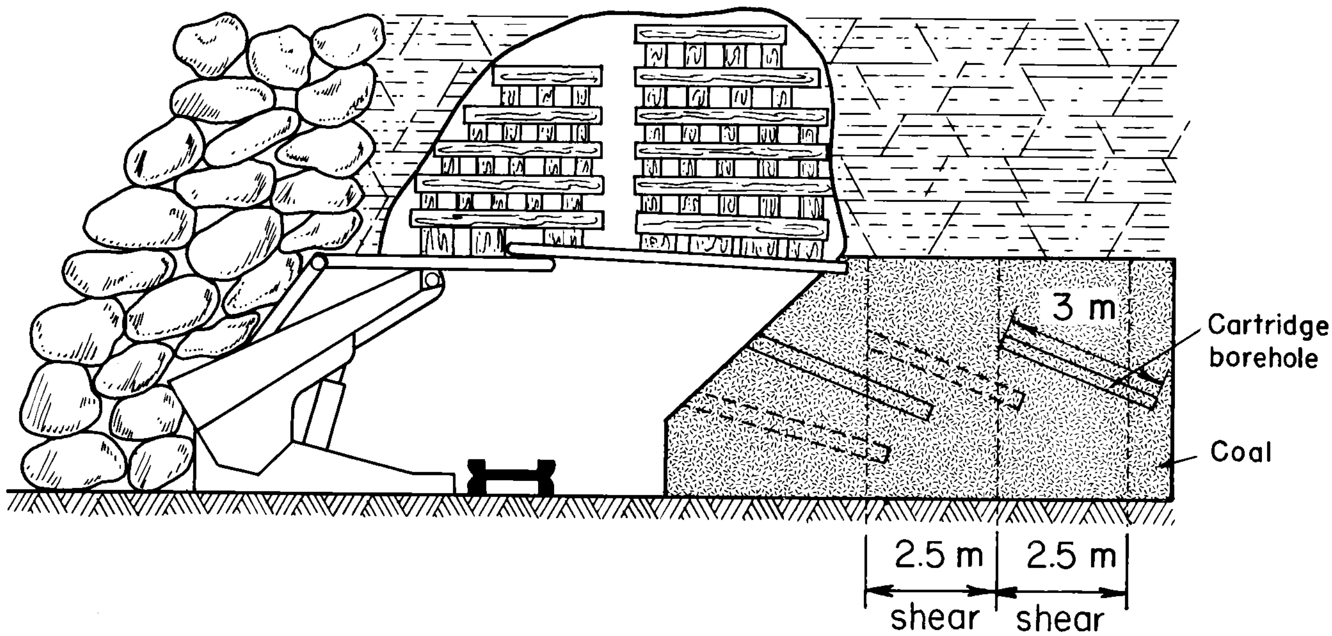

FIGURE 10. - Coal face stabilization using resin cartridges.

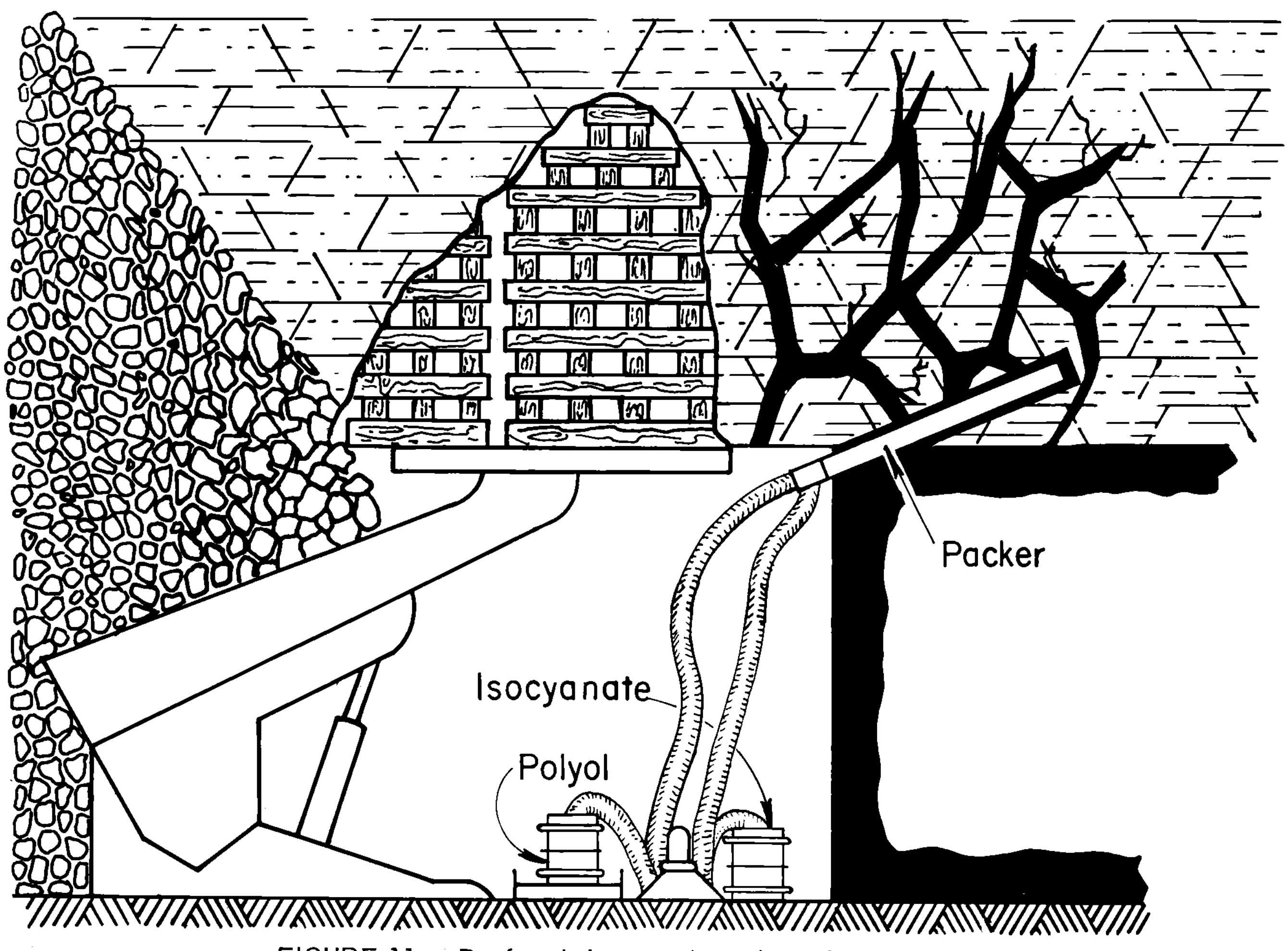

FIGURE 11. - Roof stabilization by polyurethane injection.

pumped separately from the drums by compressed air or hydraulically driven units and pushed through a labyrinth-type mixer immediately before they enter the borehole. The mixture is injected into the fragmented rock mass under a pressure approaching 14,500 kPa. Small pump units have been designed for direct use on the longwall face, and larger units are installed in the gateroads to pump the resins through hose lines to the scene of application. Effective telephone communication between the pump attendant in the gateroad and the injection operator on the face is essential.

Packers were developed to seal the polyurethane in the borehole. A packer consists of metal shells and a rubber hose section. The metal shells, which are spread against the borehole walls by a setting tool, hold the packer in the hole. The rubber hose section provides the sealing function. Packers are equipped with check valves to prevent resin from flowing out of the packer.

The volume of the infused resin expands 3 to 5 times as a result of foaming. The roof must be supported adequately to withstand downward movement of the newly consolidated rock mass subjected to injection and expansion pressures. The injection pumps stall at pressures exceeding 14,500 kPa, and this indicates that the borehole will not accept any more resin, because the material pressed into the crevices has set.

The array of injection holes is patterned according to site-specific considerations. As a rule, the length of boreholes should exceed the daily longwall advance by 0.5 to 1.0 m.

Since its introduction in the seventies, polyurethane has found many applications in U.S. deep mines for the stabilization of fragmented rock on faces, at face ends, during shield recovery, and in gateroads, often in faulted zones (17-18). The U.S. consumption of resins amounted to 544,000 kg in 1982, and manufacturers estimate the industry will use 1,360,000 kg in 1983. The estimate for European annual use of resins stands at 18 million kilograms (19).

The Mine Safety and Health Administration (MSHA) has formulated guidelines for the safe use of polyurethane in strata consolidation applied in cartridges or by injection (20). Spraying of polyurethane is prohibited. These guidelines encompass properties of chemicals, such as flashpoint, toxicity, and curing temperature; quality assurance; equipment standards; and operational procedures. Pumps must be provided with pressure relief (17,500 kPa). The burst pressure of fittings and hoses must exceed 70,000 kPa, and a packer of adequate holding power (52,500 kPa) must be used in the borehole. Recommended operational procedures concern underground storage of allowable quantities of the chemicals in closed metal containers, potable water for flushing, first aid supplies, protective clothing and goggles, telephone communication, responsible and knowledgeable supervision, and fire protection (because each component of polyurethane is combustible).

On the basis of the guidelines, MSHA district managers issue permits for polyurethane application for individual mine sites. State permits are also required. Pennsylvania does not allow the presence of any persons downwind from the scene of polyurethane injection.

Rock consolidation by polyurethane is expensive; therefore, to justify its application, strata monitoring by observation through a borescope and differential roof sag measurement should precede the infusion. A kilogram of polyurethane costs approximately $2.95. The weights of the components, polyisocyanate and polyol, are 1.20 and 1.0 kg/L, respectively. The rate of consumption is site specific, depending on the fissuration of the strata.

Figure 12 shows the roof stabilization procedure that was initiated in a recovery area next to a longwall main gate where a wedge of coal had to be left in place due to hazardous roof conditions.

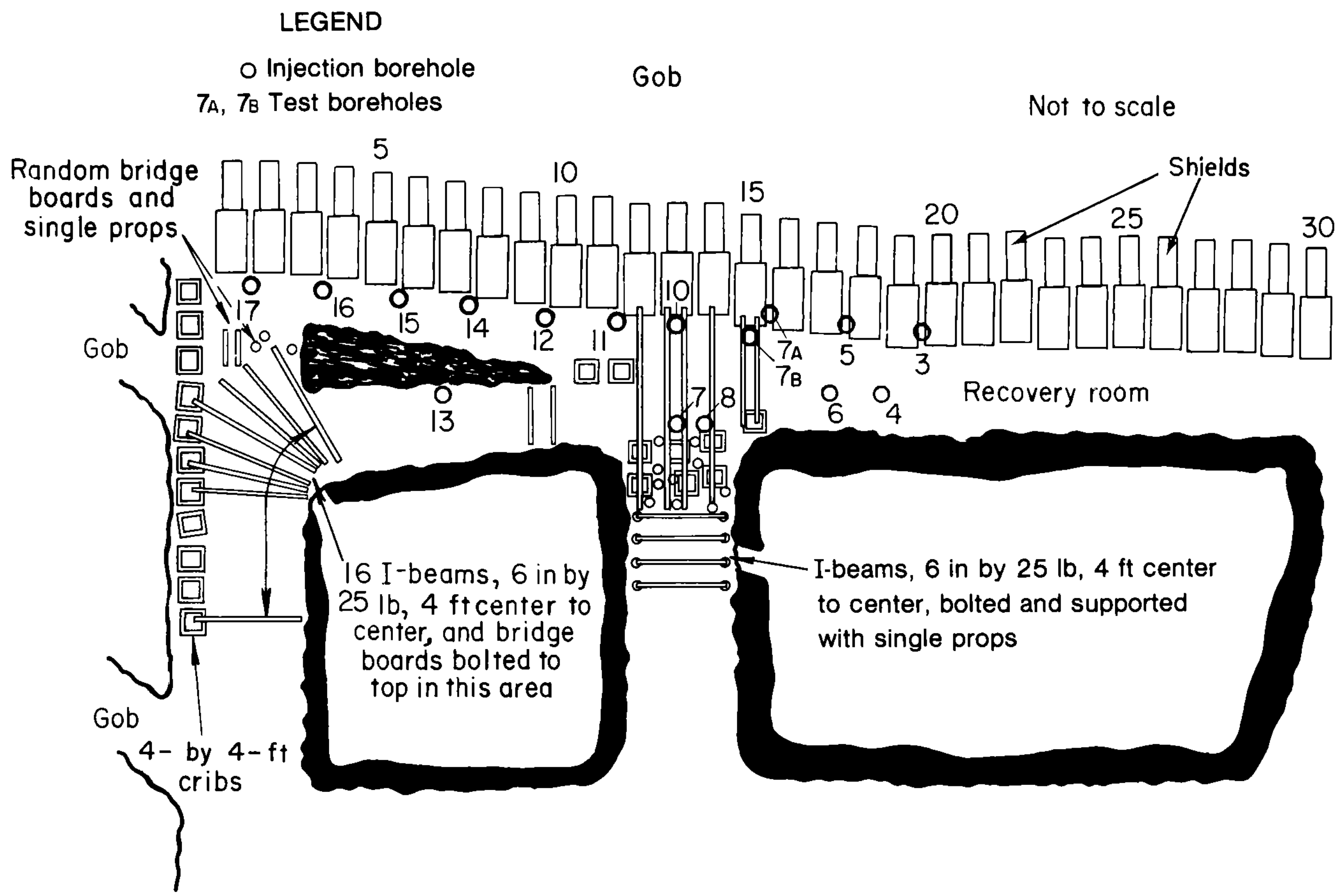

FIGURE 12. - Area stabilized by polyurethane injection during shield recovery.

The strata immediately on top of the shields was broken into small pieces. After 17 holes (including 2 test holes) were drilled and injected with polyurethane binder, the rock above the shields appeared to be laced with cracks filled with the binder, which consolidated the rock mass into a solid unit. The removal of the shields from under the stabilized zone did not pose any problems.

It took 6,100 kg of resin to consolidate the strata. The work, including mobilization, was performed in 10 days by a team consisting of an engineer and two craftsmen. The approximate cost was $30,000, broken down as follows:

Resin; 6,100 kg × $2.95/kg =	$18,000
Labor	10,000
Subsistence and airfare	2,000
Total	30,000

Polyurethane infusion during the terminal phase of longwall panel extraction is an alternative to blanketing friable roof strata above the shields with wire mesh, which is an arduous and labor-intensive task (21). Holes on 3-m centers are drilled in 2 rows, one row 6 m from the predetermined final position of the shield line and the other row into the roof of the recovery space. Each hole takes 140 to 320 kg of resin. Roof over recovery chutes that intersect the recovery room generated by the terminal position of the face can also be reinforced by polyurethane infusion. Hauling pans and roof supports through the chutes will accelerate the recovery procedure (22).

Such a salvage effort may consume 25,000 to 40,000 kg of polyurethane. However, if the duration of the recovery phase and labor costs can be significantly shaved by polyurethane infusion, this method may be more cost effective than the installation of wire mesh. The choice of a strata stabilization method is a site-specific consideration.

CONVENTIONAL CAVITY FILLING

Wherever roof deterioration has progressed to the degree that cavities appear above and in front of canopies, the practiced method of reestablishing roof contact is to build cribs on top of the canopies (fig. 10). During support advance, the cribbing is destroyed and must be rebuilt. The common method of supporting cavities in front of the canopies is to establish a platform by drilling holes into the face at roof level. Small-section H-beams, steel pipes, and rods are inserted into these holes. Their rear ends rest on the canopies (fig. 10). Above this platform timber chocks are built to the roof of the cavity. The practice of building and rebuilding cribbing is time consuming, hence costly, and involves exposure of workers to unsupported roof.

The NCB has reported that fatal and serious reportable accidents associated with cavity filling are on the increase in British mines, and it is a salient fact that people with a great deal of practical experience, notably a number of supervisors, have been victims of such accidents (23).

The number of miners injured in the United States while filling voids is unknown. The total number of longwall accidents reported during 1975-81 was 4,254. Of these, 675 were ground control related and 3 were fatal. Two of the fatal accidents were the result of roof cavities; the third was classified as a machinery accident, but was the direct consequence of a roof cavity. A jointed canopy jackknifed into this cavity. When the tip of the canopy dropped, it struck the portal bridge of a passing plow. The impact shifted the chock aside and caused fatal injuries to a miner who happened to be in the travelling space. It is believed that many accidents in which a roof cavity is a contributing factor may not be so identified because of the method of reporting accidents in effect.

STRAIN-ABSORBENT MODULES

The NCB has experimented with strain-absorbent modules (SAM's), which are wire-basket-type substitutes for wood cribbing used in the support of roof cavities. Straw bales or brush wood was used for filling roof cavities in the early days of coal mining, and the strain-absorbent feature of these materials inspired the development of SAM's. SAM's are rectangular, cubical, and triangular shapes, structured from welded wire-mesh panels (fig. 13). The panels are assembled and reinforced by helical steel binders. The assembled structure can be further strengthened by inserting inner panels.

Though filling cavities with SAM's does not eliminate exposure to unsupported roof, British miners claim that the use of SAM's reduces the exposure time considerably. SAM's can be made up on the surface and folded flat for transportation through the mine to the face. Adding a few helical binders is all that is needed at the place of application.

Ease of material handling and quick establishment of roof contact are the major advantages of SAM's versus hardwood cribbing. Hardwood cribbing of 600- by 100- by 100-mm blocks to fill a void of 600 by 600 by 600 mm weighs 65 kg, while a

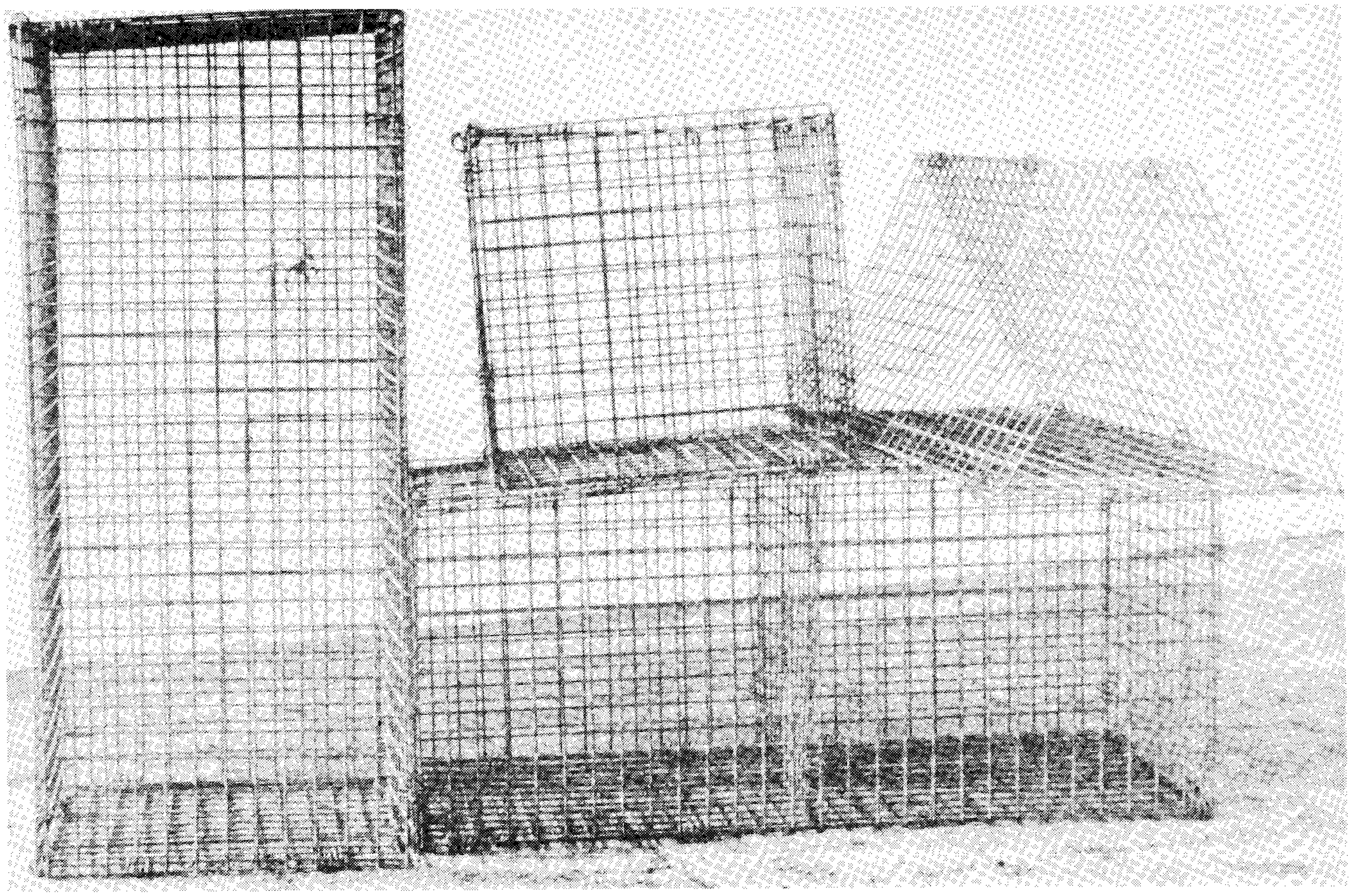

FIGURE 13. - Strain absorbent modules (SAM's).

respective SAM structure composed of six panels, each made of a 50- by 50-mm mesh of 6-SWG (5-mm) wire, weighs only 14 kg (8). The module size or sizes can be tailored to suit the space to be filled. In many instances, a single module can easily fill a cavity.

Though SAM's cannot sustain the same load as hardwood cribs, they are surprisingly strong. The NCB had load tests conducted to evaluate the strength of modules. In one of these tests, a 1,400- by 600- by 600-mm SAM composed of six 50- by 5-mm wire-mesh panels was reinforced in the middle of its interior space with a seventh panel of the same mesh. The SAM was weighted with incremental loads of 2.5 kN each (fig. 14). When a final load of 30 kN (12 weights) was attained, the module did not collapse, but suffered a deflection of only 41 mm. However, the test had to be discontinued because the stack of weights atop the module became unstable and threatened to tumble down (23). A SAM of the same dimensions and mesh, but without the inner support of a seventh mesh panel, could only sustain 17.5 kN before it collapsed.

As a result of these tests, the NCB concluded that--

- SAM's are capable of withstanding heavy loads and controlling strata in the top and sides of cavities.

- SAM's should be strengthened by clipping inner and end mesh panels to them with helical steel binders in sufficient numbers.

- SAM's should be shaped to conform to the cavities.

- The resistance to load increases if the SAM is filled with mine waste rock or foam.
- The powered roof supports beneath SAM's should be advanced with judicious caution.

INFLATABLES

An inflatable structure is another device that can be used to fill roof cavities. Research in the United Kingdom has determined that a pressure of only 7 kPa is adequate to prevent most roof falls, stabilize a roof cavity, and enable the roof supports to advance (8). Although 7 kPa may seem low for cavity stabilization, a study titled "Air-Inflatable Temporary Supports for Coal Mine Roof," conducted by Hauser Laboratories under Bureau contract H0210058, revealed that 90 pct of the roof falls in room-and-pillar mining could be prevented by a support pressure of 14 kPa, and a pressure of 7 kPa would reduce roof falls by 64 pct.

Similar information resulted from a Bureau analysis of U.S. coal mining fatality statistics during 1966-68. Median fall dimensions were computed to be 4.30 m long by 3 m wide by 0.3 m thick. Of the falls with fatal consequences, 75 pct were less than 0.6 m thick, and 90 pct were less than 1.2 m thick. This indicates that pressures of 7 to 28 kPa, distributed over the area, would be adequate to control the immediate roof strata.

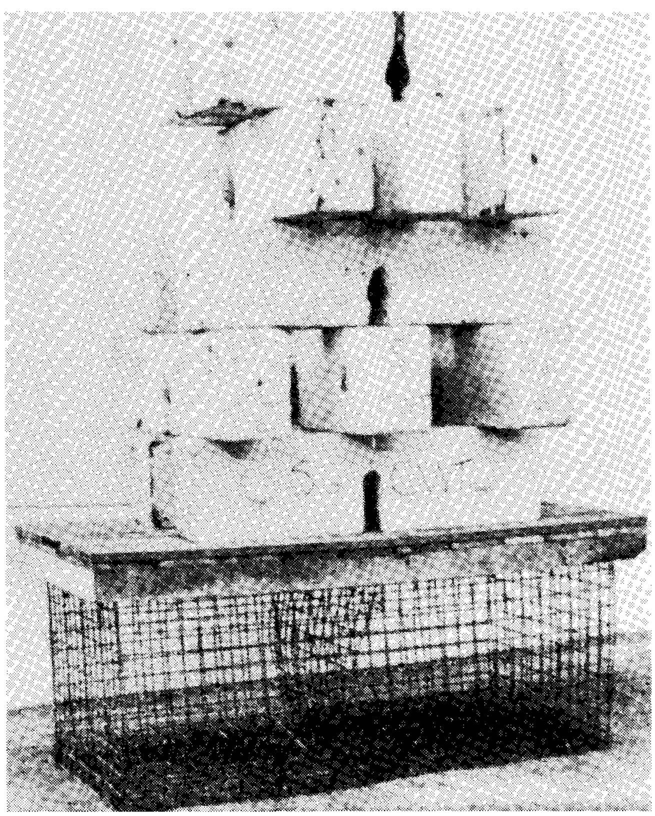

FIGURE 14. - SAM tested under 30-kN load.

The Essen Research Center for Roof Support and Rock Mechanics in West Germany determined that the height of 87 pct of measured roof cavities is less than 40 cm. The weight of a 40-cm-thick rock stratum can be supported by a load density of 9.6 kPa (1.4 psi). Therefore, if such a load density can be brought to bear against the roof strata, it may be kept intact most of the time (9). The support mechanism appears to consist of the combination of small pressures distributed over large areas, the inherent shear strength of the rock, and keying in of the stratum forming a voussoir arch.

To generate such small pressures, the NCB is experimenting with inflatable bags of various sizes. The typical shape of these inflatables can best be described as pillow shaped (fig. 15). Nylon fabric coated with either neoprene or polyvinyl chloride (PVC) is used in construction, and each inflatable is fitted with two safety valves and a nonreturn inlet valve.

An underground test was conducted in a British mine to determine if inflatables could control roof cavities as well as hardwood cribbing. A neoprene-coated nylon inflatable was used; it had uninflated dimensions of 4.6 by 0.9 m and weighed only 9 kg. It would take 900 kg of hardwood crib blocks (60 by 10 by 10 cm each) to fill a void the same size as the inflatable. Inflation took 7 min, using a low-pressure stowing machine. (In other tests, hand or foot pumps were used.)

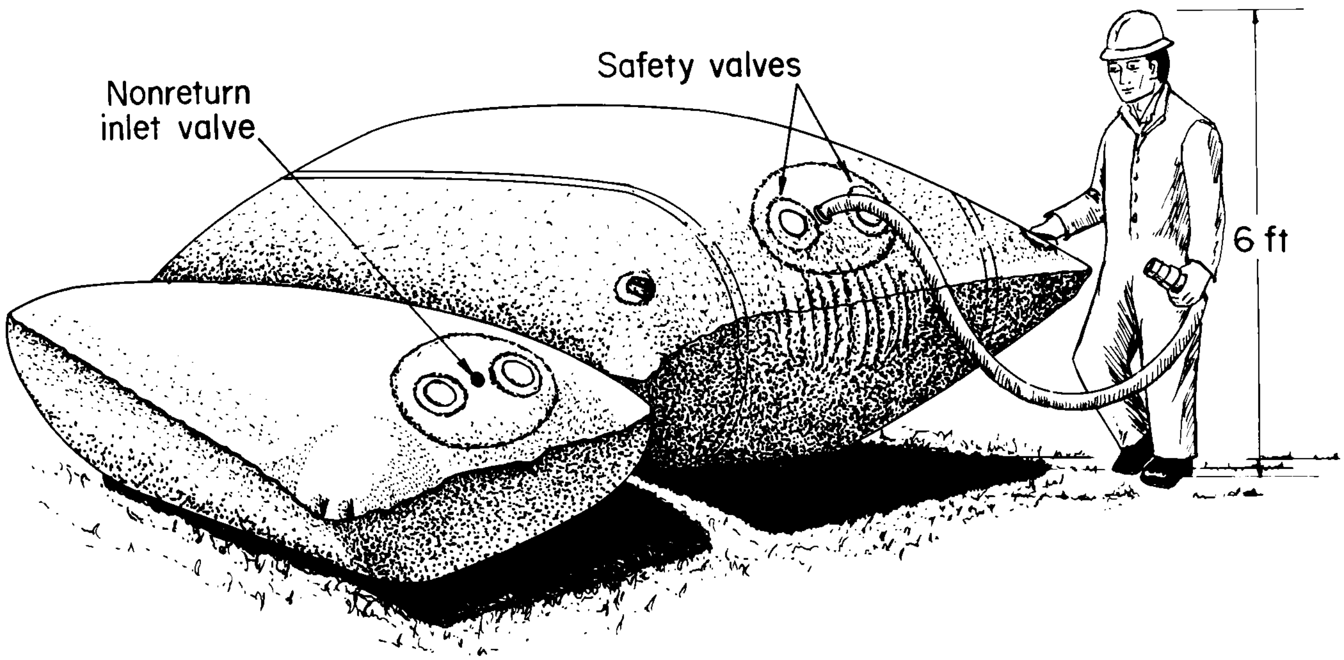

FIGURE 15. - Inflatables.

Inflatables may be vulnerable to tears and punctures that can make them unreliable. To compensate for loss of air pressure, an air regulator has been developed that admits enough air from the compressed-air source to maintain the pressure. During one test, an inflatable was punctured by a wire. A miner using a foot pump inflated the damaged device in less than 7 min to the desired pressure.

The British inflatables are designed to be deflated, retrieved, and reused, to reduce the cost with each use. However, the recovery of the inflatables is fraught with dangers and almost impossible on a shield-supported longwall. A cheaper substitute for neoprene coating may be Mylar[4] polyester film. Both neoprene- and Mylar-coated nylon have been tested by the Bureau of Mines for possible use in emergency inflatable stoppings in metal mines (24). Both materials were used to construct 3.7-m-diam spheres. The neoprene sphere weighed 50 kg and cost $3,500, and the Mylar sphere weighed 16 kg and cost $1,300.

[4]Reference to specific products does not imply endorsement by the Bureau of Mines.

Dunnage bags are a substitute that is considerably cheaper than neoprene- or Mylar-coated nylon. They are constructed from several plies of kraft paper that cover a polyethylene bladder. At 56 kPa, a dunnage bag is fully inflated and resembles the nylon inflatable (fig. 15). The transportation industry has used dunnage bags for years to stabilize loads up to 30,000 kg (25). Figure 16 shows a dunnage bag supporting a 1,270-kg vehicle. However, despite their high load capacity, dunnage bags have a low resistance to rips and tears. Also, the Bureau has reported that, when inflated, dunnage bags become very rigid and inelastic (24).

It is believed that at lower pressures dunnage bags may be flexible enough to conform to the interior of a void. Dunnage bags can expand only to 300 mm; if a larger void is to be filled, more than one bag must be used, or a noninflatable fill must be added. The cost of a dunnage bag with uninflated dimensions of 1.8 by 2.7 m is approximately $20.

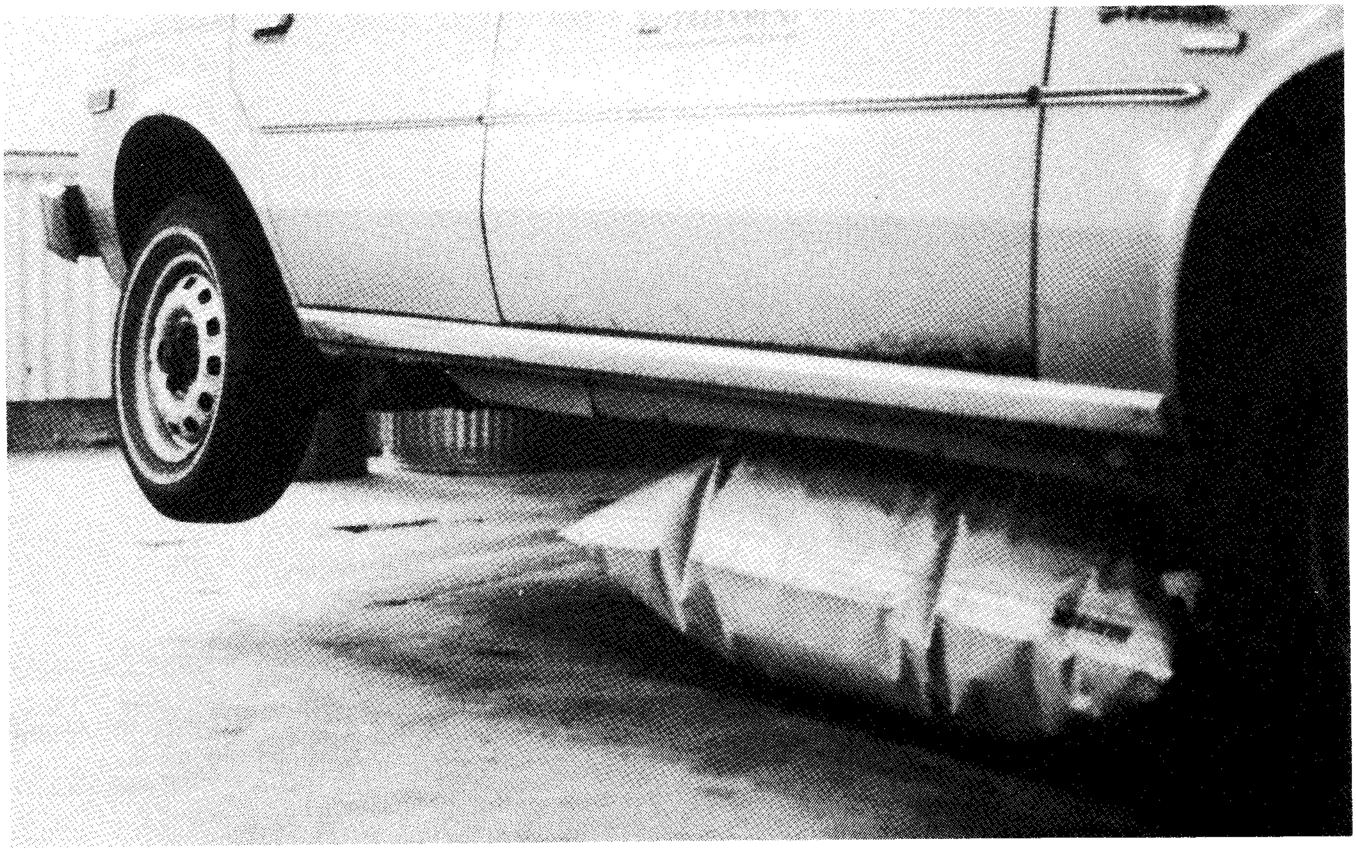

FIGURE 16. - Dunnage bag inflated to 12 kPa supporting a 1,270-kg vehicle.

CHEMICAL FOAM

West German miners have been using synthetic foams for the last 25 yr to provide airtight sealing of crib lines along longwall gobs and fill cavities on faces and along roadways. Chemically, these foams are urea-formaldehyde and are marketed under the trade name Isoschaum. The NCB has imported Isoschaum foam from West Germany, and its application on British longwall faces has been successful.

Isoschaum is produced from two chemicals, one a resinous solution and the other an acid foam agent. The chemicals react under air entrainment and generate a foam with a high expansion rate, 25:1. The two solutions, in correct dosage, are pumped into a mixing gun, where compressed air under 560 kPa pressure is added. The pumps can be powered by the oil-in-water emulsion of the face hydraulic system, and compressed air is supplied either from a main compressed-air network or a portable compressor (fig. 17). Miners can apply Isoschaum remotely through a lance, without exposure to unsupported ground. Due to its high expansion factor, the foam conforms to the shape of the void and quickly provides strong support of the stratum.

The procedure for treating cavities on a longwall face with Isoschaum involves two steps. First, a floor is built above the roof supports; then, the remaining empty space is filled with foam. The floor may consist of joints of ventilation tubing with the ends tied or pieces of brattice cloth sewn together. The floor material is placed into the voids above the canopies or their forepoling extensions and then injected with foam that hardens quickly (fig. 18). Any remaining empty space is filled with more foam (fig. 19). The foam exerts a

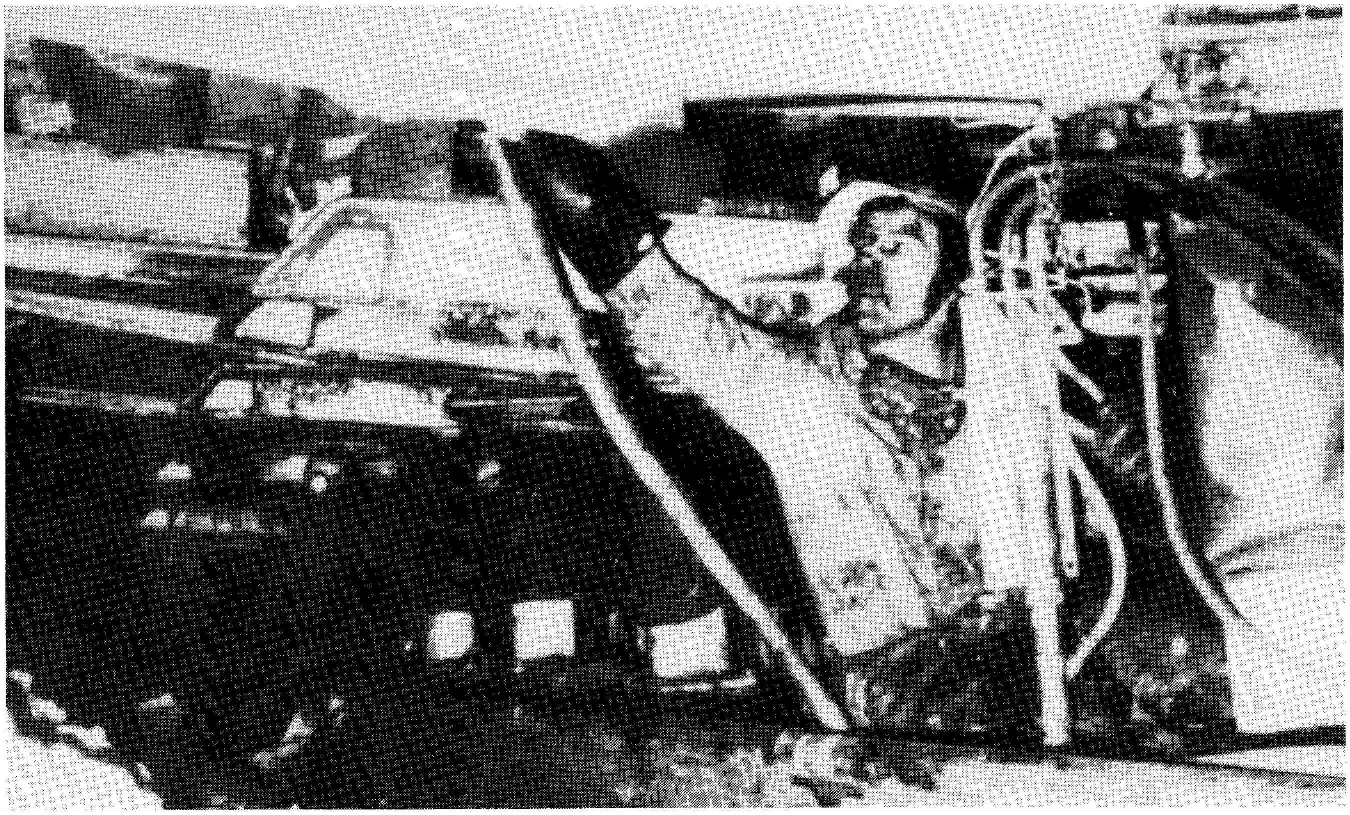

FIGURE 17. - Isoschaum foam application.

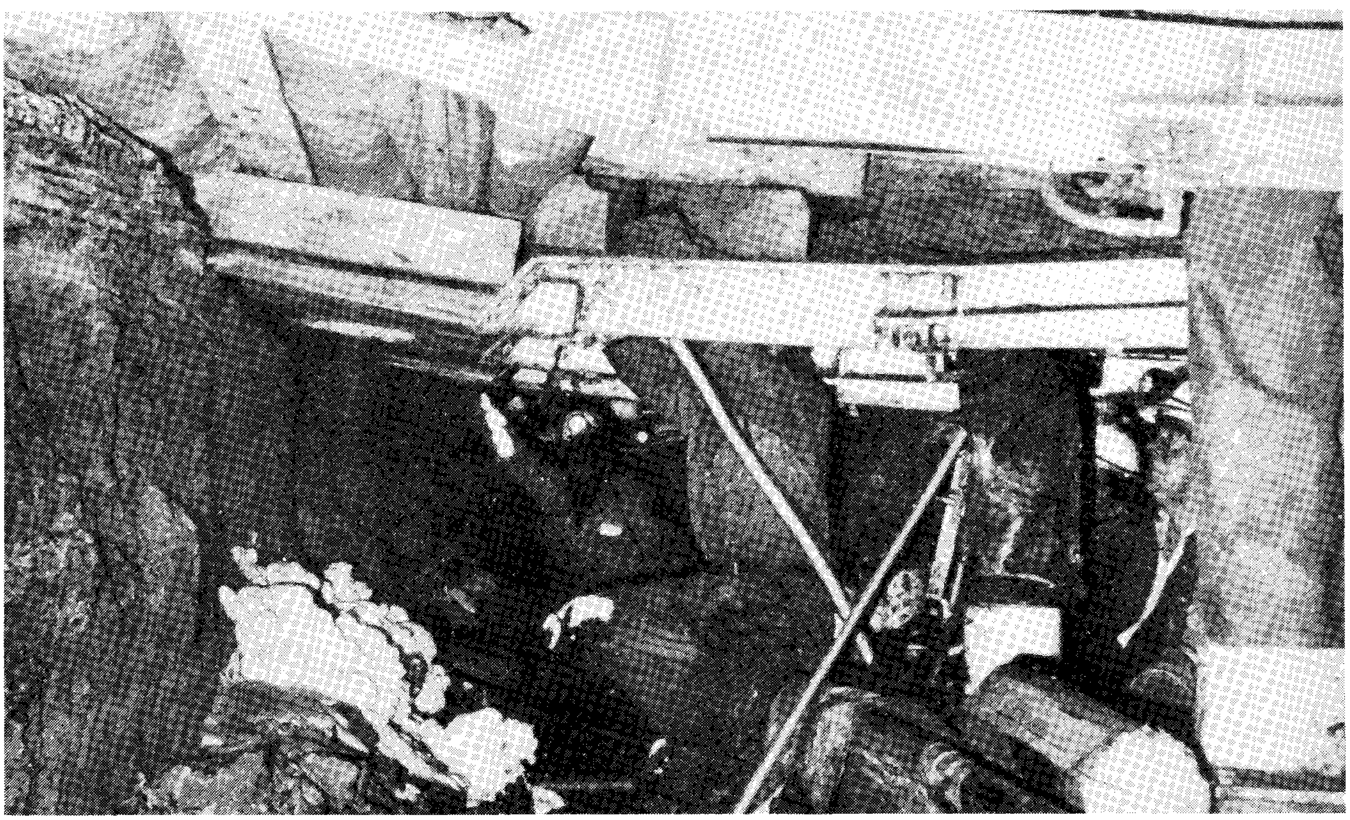

FIGURE 18. - Foam-filled vent tubes on base of forepoles and split bars.

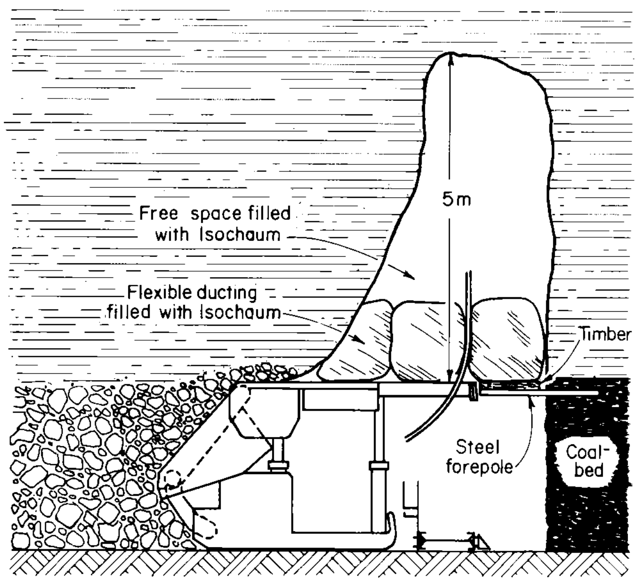

FIGURE 19. - Cavity filling with foam.

positive pressure on the surface of the cavity, stabilizes it, and prevents methane accumulation.

Filling cavities with Isoschaum may also mitigate the impact of additional roof falls. In one instance in a British mine, a cavity 7 m long and 3 m high had been filled with Isoschaum. After mining resumed, a second fall occurred in the same location. Large blocks of sandstone dropped out from the roof of the cavity, but were suspended in the foam without doing any harm (8).

In table 3, the relative properties and costs of various materials that could be used to fill a 10 m^3 cavity are compared. The table shows that synthetic (organic) foam costs 23 pct less than cribbing and 20 pct less than inflatables, but 13 pct more than SAM's.

Ease of application and a relatively low cost made Isoschaum the preferred material for filling cavities on British longwall faces (at one time), and the NCB was looking for domestic sources to supply the foam. However, for health reasons, the NCB reversed its position; it no longer favors Isoschaum for this application and has discontinued Isoschaum use in British coal mines.

TABLE 3. - Comparison of various materials for filling a 10-m^3 cavity

Material	Expansion factor[1]	Method of transport along face[2]	Full support to sides of cavity	Requirements for 10-m^3 cavity: Weight,[3] kg	Volume (to transport),[3] m^3	Relative exposure time[4]	Relative cost of materials[5]
Wood cribbing.	3:1	Conveyor and manual transport.	No	3,130	3.33	100	100
SAM's.......	12.5:1	...do.............	No	420	.79	10	68
Inflatables.	47:1	...do.............	No	35	.22	Nil	96
Organic foam	25:1	Hose lines........	Yes	450	.65	Nil	77
Inorganic foam.	8:1	...do.............	Yes	1,480	1.95	Nil	135
Packbind grout.[6]	2.5:1	...do.............	Yes	6,000	4.54	Nil	188

[1]Foam-to-fluid ratio, by volume.
[2]Important accident risk factor in handling material on coal face.
[3]Important risk factor in transportation on surface and underground.
[4]After completing base of girders, pipes, etc.; basis: exposure time for wood cribbing = 100.
[5]Basis: cost for wood cribbing = 100.
[6]Many of these data apply to placing pump packing and other grout forms.

Source: Lewis (23, p. 12).

Before the NCB reversed its position on Isoschaum, air samples were taken in British mines to evaluate the health aspect of Isoschaum application. During the hardening of urea-formaldehyde foam, quantities of formaldehyde are released. In some instances, while advancing roof supports under cavities filled with Isoschaum, formaldehyde concentrations of 1 to 2 ppm were detected. Even these low concentrations can cause watering and burning of the eyes and general upper respiratory irritation.

In consideration of studies performed by the Chemical Industry Institute of Toxicology (CIIT), the U.S. National Institute of Occupational Safety and Health (NIOSH) recommends that formaldehyde be handled in the work place as a potential occupational carcinogen (26). The Consumer Product Safety Commission (CPSC) banned the use of urea-formaldehyde as a home insulating material because it received numerous complaints concerning adverse health effects (27). However, this ban was overruled by a Federal court in July 1983. On the basis of the NIOSH recommendations and the CIIT studies, MSHA has not permitted the use of urea-formaldehyde underground.

Health hazards associated with organic synthetic foam such as Isoschaum could be avoided if foam could be generated from inorganic chemicals that do not give off any harmful vapors. The NCB initiated experimentation in this area. Dilute aqueous solutions of sodium silicate and a hardener composed of magnesium and zinc salts were fed with compressed air into a gun to generate foam. Only 1 pct of an organic surfactant was admixed. The laboratory results were encouraging. However, the expansion factor was only 8:1, much less than 25:1 for the organic synthetic foam; and the cost is much higher because this process is still in the experimental stage. Underground trials have not been reported to date (8).

However, there has been some recent progress. Visitors to the NCB in 1983 reported that the Mining Research and Development Establishment at Bretby has succeeded in developing an aerated cement system called Aqualight, a pumped fill that expands at a rate of 15:1 (28).

MONOLITHIC FILLS

Though roof support by shields offers distinct advantages, cavities may occur both above and in front of the canopies under adverse ground conditions such as--

- Friable immediate roof strata.
- Laminated shale that tends to exfoliate.
- Distinct slip and separation planes.
- Wedges of shale thinning out under massive strong strata.
- Erosional channels, often called rolls.
- Main joints paralleling the longwall face.

Shields can overcome shallow cavities of little expanse by simply advancing until firm ground is reached. Ground stabilization by inserting resin-anchored wood dowels or by polyurethane injection has been effective where roof cavities have steep flanks and are of little extension. However, where the coal face is hading forward, leaving a wide unsupported span between the canopy tips and face, weak roof strata may collapse; and very high and extended cavities may form and become uncontrollable (fig. 20).

To remedy such extremely adverse conditions, West German and British miners have successfully applied strata replacement techniques by applying monolithic packs made of various materials similar in composition to those used for roadside packs (29). The major monolithic approaches to date are (1) the anhydrite system and (2) liquid systems such as "pump packing" or "aqua packing."

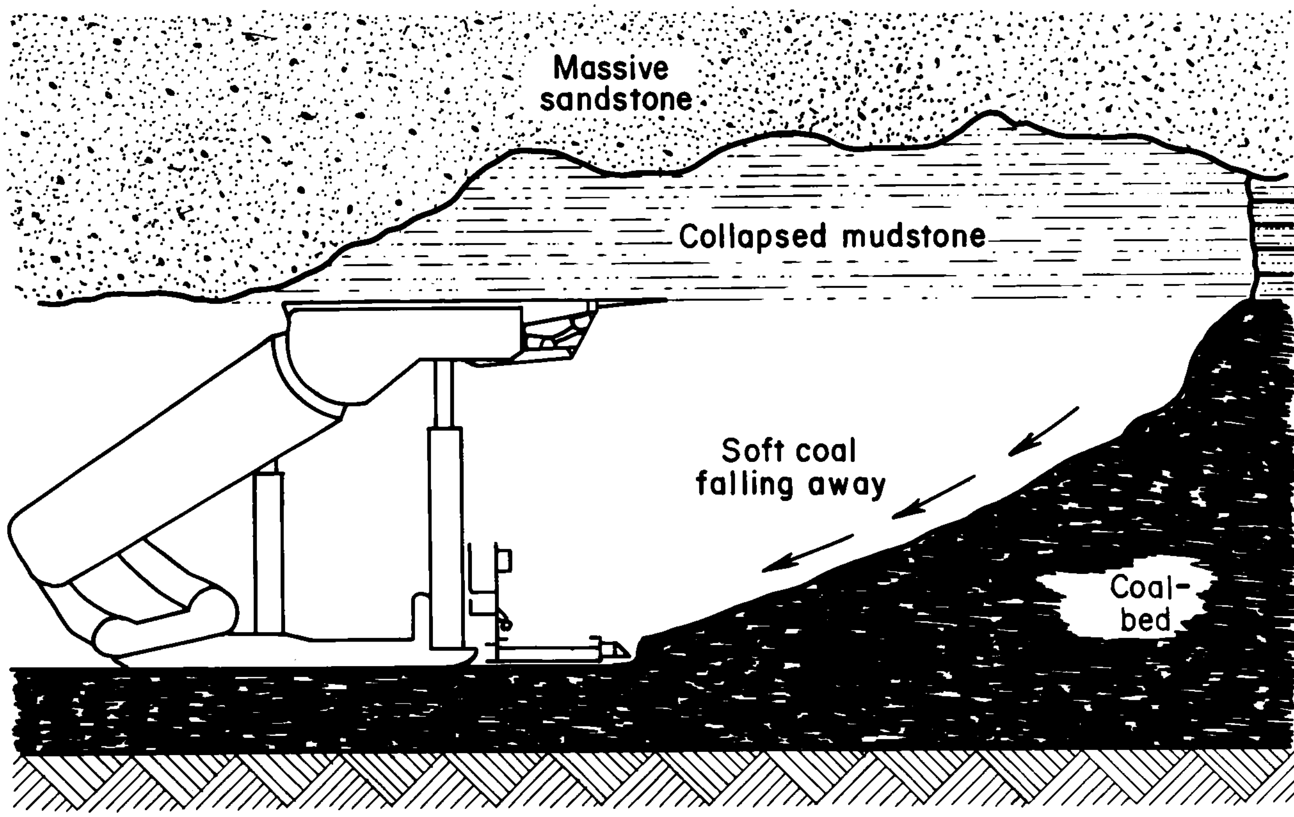

FIGURE 20. - Cavity over coal face hading forward.

Anhydrite-type packs are prepared by blowing a mixture consisting of natural anhydrite, water, and a hardener through 100- or 125-mm airpipe into a frame built with a few timbers and a light netting or brattice cloth stretched in between. The mixture comes through the pipe slightly damp and has an angle of repose 30° off the vertical. It sets up rapidly and may attain a compressive strength of 7,000 kPa after 10 h and 14,000 to 21,000 kPa after a few days.

The dry anhydrite is transported into the mine in bulk, stored underground in a bunker, and fed into a blowing machine capable of sending a continuous stream of anhydrite through the pipeline to the scene of application at a rate of approximately 6 t/h. The hardener is composed of potassium and ferrous sulphates and is apportioned to the anhydrite in a quantity equal to 1 pct of the mixture. The anhydrite-to-water ratio is 10:1.

The pump-packing system, developed in 1973 in the United Kingdom, utilizes a mixture of coal fines, bentonite (flowmat), water, and a type of cement called Packbind to form a monolithic pack. For remedial application only, the Packbind component is used with water. Packbind is a specially formulated cement that sets up rapidly even though substantial amounts of coal may be mixed with it. The mixture is a slurry that is pumped to the scene of application into strongly built shuttering.

Aqua packing (fig. 21) is a variation of pump packing that uses essentially the same pumping equipment but dispenses with the need for coarse aggregates by utilizing increased quantities of water and special cements. The resultant mix crystallizes and sets to produce a pack having properties similar to those of a standard pumped pack. As an alternative to shuttering, the slurry can be

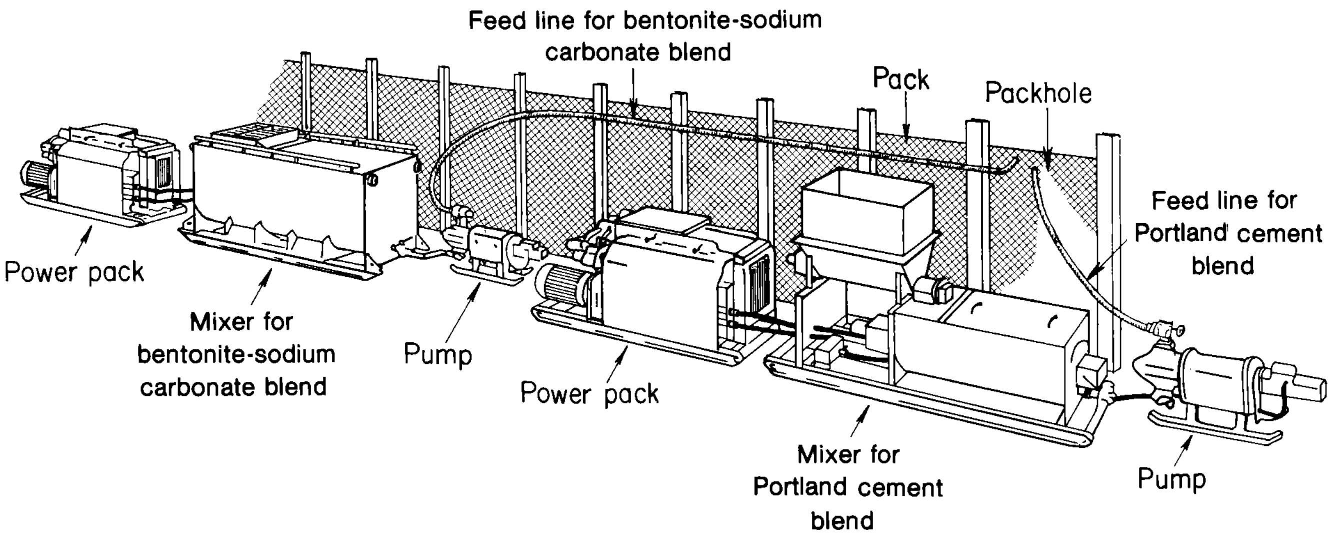

FIGURE 21. - Aqua-packing system.

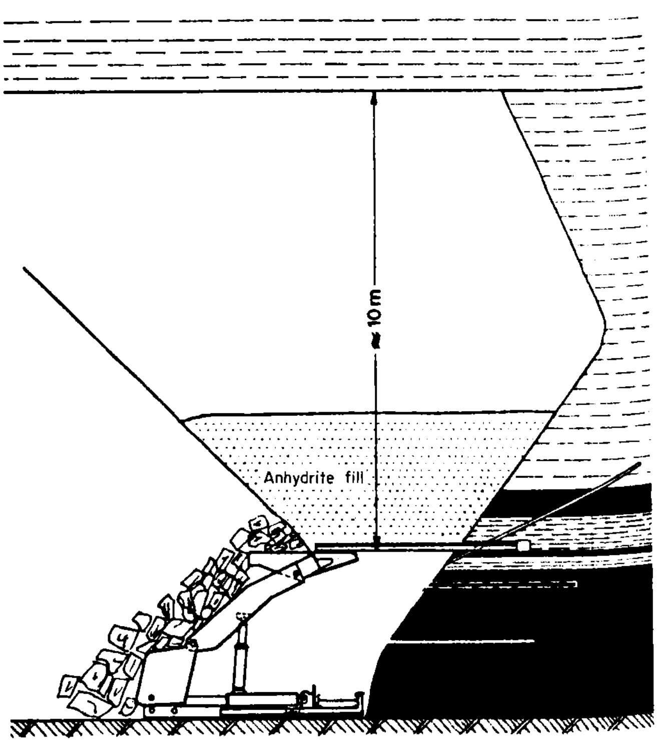

FIGURE 22. - Strata replacement in a high and wide cavity.

pumped into bags tailored to fit the cavity. British mining engineers believe that the maximum compressive strength of pumped pack material, about 8,400 kPa, is adequate.

West German miners successfully applied remedial techniques as described in reports from the early seventies. One of the first longwalls with shields operated in the Dickebank Coalbed, extracting 2.21 m of the seam under a roof of rash with several slip planes (9). Rather than move the bulky face equipment from panel to panel, the entire longwall was swung 180° to form a new face line. During this rotation the weak roof strata dropped out several times.

The following measures were taken to stabilize the ground before resuming mining under a 10-m-high cavity (30) (fig. 22):

- Rails were placed into holes drilled into the face on 1.5-m centers. The rear ends of the rails rested on the shield canopies.

- Bridge boards, laid above the rails, provided a tight shuttering.

- Natural anhydrite was blown into the cavity on top of the bridge boards to form a cover 1.5 to 2.10 m thick.

In addition, the face was secured with resin-anchored wood dowels and polyurethane injections. Without any further delay, the cavity was undermined, and, after 10 shearer passes, normal operating conditions were reestablished.

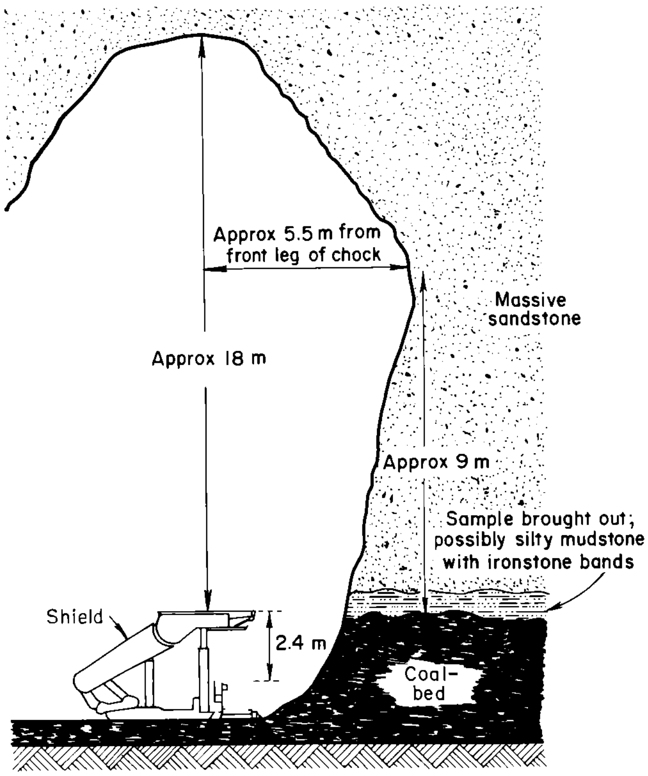

FIGURE 23. - High cavity on shield face.

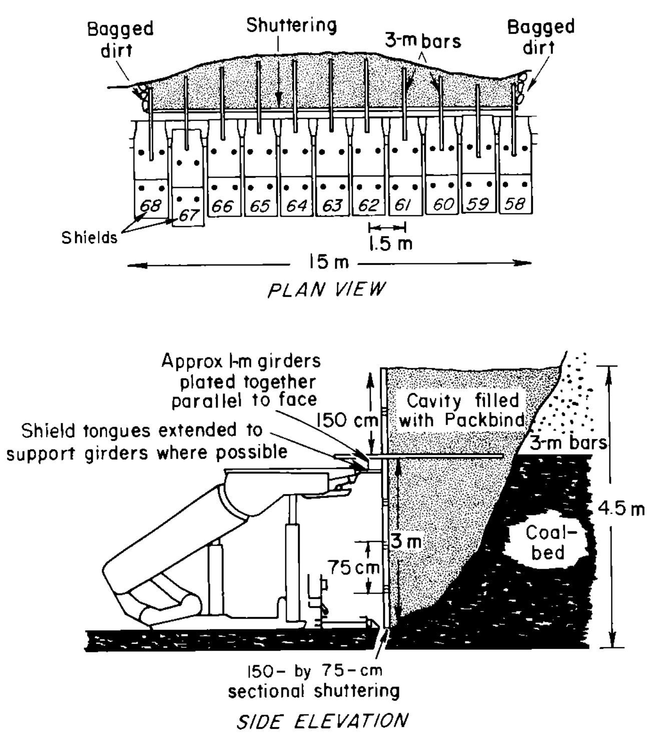

FIGURE 24. - Strata replacement scheme used to fill cavity shown in figure 23.

Figure 23 shows a very high cavity over a heavy-duty shield support on a British face 210 m long and extracting 2.74 m of the Barnsley/Dunsil Coalbed (31). A lens of mudstone was overlain by a strong sandstone. The soft mudstone dropped out from under the massive strata, while the coal face haded forward. The cavity grew progressively worse until it attained a height of 18 m and an extension of 15 m along the face. To stabilize the fall area, management decided to fill the cavity with Packbind, which is quick setting, attains early strength, and can be cut by a shearer.

The stabilization program was conducted in the following phases (fig. 24):

1. Shuttering, consisting of plywood panels, was built in sections of 1.5 by 0.75 m. Ten of these sections formed a base, extending over the 15-m length of the cavity. Four rows of the 0.75-m high sections raised the shuttering to a height of 3 m. The shuttering was braced with proper spragging.

2. Packbind was pumped into the space between the coal face and shuttering. The Packbind was mixed in a grout mixer station located in the return gate (18 m from the face line) and pumped through a 50-mm hose into the shuttered area (110 m down the face). The Packbind-to-water ratio was 1:2 by weight. Packbind was supplied in 25.4-kg bags. The system was cleared with fresh water after every 100 bags pumped. The armored face conveyor was kept running during this procedure to prevent clogging of the bottom race with Packbind, which leaked from the shutter.

3. Rails 3 m long were placed over the Packbind fill, one over each shield canopy.

4. Another row of shuttering, 1.5 m high was erected above the rail level.

5. Packbind was pumped in again, to the top of the shutter, to complete a fill 4.5 m high and 15 m long, which reestablished the face line.

The entire process, from building the shuttering to completing the fill, took 48 h and required 97 t of Packbind.

Mining was resumed cautiously. The shearer cut through the fill while the shields were advanced under brushing contact with the artificial roof. As the coal was reached, normal operation was reestablished.

The major differences between the monolithic systems can be summarized as follows: Anhydrite sets up rapidly, is stronger than pumped-pack and aqua-pack systems, and is installed with relative simplicity, needing only light shuttering. However, pneumatic placement of anhydrite requires a source of compressed air either from a mainline net or from a compressor station underground. Another serious problem that must be addressed is dust propagation at the blowing machine and during pack installation. However, West Germany's Silicosis Research Institute has certified that natural anhydrite from 0 to 6 mm (the type and size used in mines) contains less than 1 pct silica and is not considered to be harmful.

Anhydrite is not widely mined in North America. There is a mining operation in Nova Scotia, and there are large deposits of natural anhydrite in Utah and other places.

Because of differences in the systems and differences in the strata at different mines, determination of the best system for strata replacement will be a site specific choice.

An alternative to monolithic fills is lightweight aggregate concrete blocks such as cinder blocks, stacked up as a dry wall and coated with concrete, that can be cut by a shearer and removed from the face (8).

PLANNED BUREAU RESEARCH

There are several areas in which planned Bureau research may reduce exposure of workers to ground hazards and improve roof control on longwall faces.

The application of inflatables merits attention as to how to fit them into the operational cycle and how to lower material costs. The Bureau plans to investigate cheap versions of inflatables such as dunnage bags and their resistance to rips, tears, and water.

Since the application of Isoschaum foam has been challenged on health grounds, substitution of a material without toxic or carcinogenic side effects will be investigated. Foam generated from anorganic constituents will be a candidate.

Planned studies into the technology of monolithic packs will open a new avenue in the art of strata replacement, which heretofore has been almost unknown in the American mining scene.

The Bureau's research into remedial and strata replacement techniques is directed toward the long-range goals of enhancing safety, productivity, and resource recovery. As mining penetrates to greater depths, growing interaction between overmined and undermined coalbeds is expected to make room-and-pillar mining so vulnerable to ground hazards that extraction by longwall techniques will be the only viable alternative.

SUMMARY AND CONCLUSIONS

Uncontrollable strata problems that lead to stoppages on highly productive longwall faces can result in sizable losses. A longwall face supported by shields and extracting coal with a double-ended ranging-drum shearer has a potential for a daily average output of 3,120 t. Given the current realization of \$36.5/t of utility fuel, the revenue shortfall is \$114,000 for each day of stoppage, and the opportunity loss per day is \$7,000 with \$8 million invested and a 20-pct return on investment, in addition to $6,000 face labor and interest and depreciation related to the equipment.

Techniques designed to maintain the roof intact are wire meshing and chemical rock stabilization. Wire mesh laying has been mechanized in Great Britain to minimize miners' exposure to roof hazards. Mechanical mesh laying could find application in U.S. coal mines during the salvaging phase of roof shield operations that pose an unusual problem because part of the structure is under caved rock. Chemical rock stabilization is indicated when, in disturbed roof conditions and under the effect of mining, coal and adjacent rock tend to break up in blocks along planes of weakness.

Wherever roof deterioration has progressed to the degree that cavities appear above and in front of the canopies, remedial action is indicated. Remote placement of a fill in support of the stratum is an alternative to the usual practice of placing cribbing by hand and thus reduces workers' exposure to hazardous roof. Alternatives developed in Europe that reduce hazardous exposure are SAM's, inflatables, chemical foam, and strata replacement techniques that use monolithic fills. SAM's are placed quickly, but their application requires brief exposure of crews to the cavity roof. The other systems allow remote filling of roof cavities, but need a measure of preparation, such as shuttering, which involves exposure.

Table 3 shows how various remedial materials used in British mines compare with wood cribbing for filling a 10-m^3 void. Ease of transportation for each material is related to its expansion factor. The higher the factor, the easier the material is to transport. SAM's can be folded flat. Inflatables increase enormously in volume. Synthetic foam, through air entrainment during its generation, greatly expands and exerts positive pressure on the surface of the cavity. Monolithic fills of anhydrite or cementitious nature provide little expansion, but air-entrainment techniques or admixtures of a foaming agent may improve their expansion properties.

The relative values in the comparative cost column of table 3 may be valid for the U.S. coal industry. In addition to material costs, the opportunity loss that results from roof deterioration on the face must enter into the evaluation of cost effectiveness in each site-specific case.

The cost of application reflects the relative simplicity of installation. The use of most remedial systems requires little training. European miners are familiar with monolithic fills that are pumped or placed pneumatically. However, in the Uited States, this technology is in its infancy and is in need of development.

The ultimate goal of cavity treatment from a sheltered location is best fulfilled by a combination of remedial systems. SAM's or inflatables can provide rapid "first aid" prior to the application of Isoschaum foam or monolithic fill.

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