An Assessment of Thin Seam Mining System Technology

prepared for

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by

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NOTICE

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16. Abstract (Limit: 200 words)

The objective of this study has been to determine the state of the art in underground, thin-seam-mining systems, to identify new technology and procedures that would reduce the hazards of thin-seam mining and to recommend areas for further research.

The study was carried out in two phases, the first of these being devoted to a survey of authoritative literature. Subsequently, the practices adopted in the various mining countries were analysed and comparisons made to identify machines and methods with the potential for application in the USA.

Published figures on US bituminous coal reserves show the high proportion contained in thin seams and US accident statistics demonstrate that the risk is greater in thin-seam operations. A comparison has been made with the experiences of foreign operations.

The Phase II study proceeded along two separate avenues to examine the distinctive problems associated with seam thicknesses above and below 30 in. Current systems of mining appropriate to seams greater than 30 in were elaborated and compared by means of a simulation exercise in terms of safety, production and cost. For seams below 30 in, less conventional systems, some presently disused, were studied and their potential evaluated; the research and development work necessary to make them viable has been indicated.

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FOREWORD

This report was prepared by British Mining Consultants Limited, UK, under USBM Contract number J0199122. The contract was initiated under the systems engineering element of the Bureau's Mineral Safety Research Sub-program. It was administered under the technical direction of the Twin Cities Research Center with Mr. B. Nelson acting as Technical Project Officer. Mr. D.W. Teets was the contract administrator for the Bureau of Mines. This report is a summary of the work recently completed as a part of this contract during the period September, 1979, to July, 1980. This report was submitted by the authors on 10th July, 1980.

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INTRODUCTION

1. The demand for coal in the USA is likely to increase substantially in the future, owing to the diminishing availability of alternative energy sources. Since a large proportion of the total coal reserves of the country lies in thin seams, it is prudent to appraise the methods and feasibility of mining these reserves.

2. The US Bureau of Mines is concerned about the coal industry's ability to supply the country's demand for coal in a safe and efficient manner and, with other government agencies, it actively sponsors coal mining studies in order to define key problems and to find and evaluate solutions.

3. Underground mining of thin seams has invariably been a more difficult, hazardous and costly process than the working of thicker beds. Historically, thin seams have been worked as far back in time as in the earliest recordings of mining coal. Over the years, efforts to mine thin seams have fluctuated in intensity, according to the political, technical and economic environment prevailing at the time.

4. This report assesses the current state-of-the-art of thin seam mining technology and addresses the technological and economic aspects of candidate mining systems suitable for application in US conditions.

5. The definition of a thin seam varies considerably amongst the coal mining countries. There are, however, fairly well defined minimum thicknesses in which both miners and modern equipment can work. Highly productive mining systems are seldom found in thicknesses of less than 30 in, while the limit for mine workers is probably as low as 14 in to 16 in.

METHOD OF EXECUTION

6. The study was carried out in two phases. The first phase entailed the acquisition of data from sufficiently diverse sources to constitute an extensive and world-wide literature survey. This was achieved through technical libraries and institutes and discussions with various mining groups, including thin seam operators and mining equipment manufacturers in the USA, eminent mining engineers who have visited both Eastern and Western Europe and other mining countries around the world, and British Mining's in-house engineers who have first-hand knowledge of thin seam operations in most continents. The survey addresses the subject of resources, production, accident analysis and the state-of-the-art.

7. The second phase of the study involved an analysis of the data and the identification and evaluation of mining systems for use in US conditions. These are sub-divided into systems suitable for a seam thickness range of 30 in to 36 in and for the thinner range of less than 30 in. Seven candidate systems are described and evaluated for operation in the 30-in to 36-in range (see Chapters XVII to XXII) and five systems in the minus-30-in range (see Chapters XXIII to XXVIII).

8. The systems identified for mining seams above 30 in are longwall, room and pillar and shortwall. These three methods are also used in thicker seams but in this study the equipment described is essentially thin seam versions of the more widely used models. In order to evaluate candidate systems in the 30-in to 36-in range, a block of thin coal was simulated with typical US conditions and the various mining methods described are designed for extracting the imaginary block of coal. This has allowed comparisons to be made between all seven of the conceptual designs and an examination of the effect of variations in some key parameters such as longwall length, depth of mining and output levels.

9. The identification of systems suitable for the minus-30-in range is more difficult. There is no shortage of techniques tried in very thin seams, but few or none have produced results that are considered to be sufficiently successful to warrant direct underground application in the USA. However, four of the five designs described appear to have sufficient potential to make further research and development worthwhile, and their concepts may also be applicable in other seam heights.

Resources and Production

10. The thin seam resources of the USA are appraised and a survey made of world-wide thin seam mining production from underground mines. The enormous resources of thin seam coal in the USA are confirmed: these occur in nearly all states that contain bituminous coal deposits. Attention is focused on the measured and inferred reserves, and several states have been identified as containing the most likely sites for thin seam mining.

11. Only two major areas in the world mine any significant quantity of thin seam coal; namely the USA and Europe. Of these, the USSR is by far the largest producer, accounting for approximately 75% of the total world output from underground thin seams. The European techniques have been developed in much deeper mining conditions and, in certain cases, steeply-inclined seams. In consequence, the longwall system is used almost exclusively. The deposits in the USA are shallower and flatter and this has allowed the industry to continue using room and pillar methods. A unique feature of thin seam mining in the USA is the small average output from thin seam mines. This is in contrast with the deep shaft mines in Europe where thin seams are often worked along with thicker seams in large mines.

Accident Analysis

12. This section of the report has been based on Hudson's paper entitled, "Accident Hazards in Different Seam Thickness in the USA". In order to make a comparative study with European experience, a special exercise was undertaken to examine the accident statistics of five UK mines, all exclusively extracting thin seams. The accident rates of the sample mines were compared with those of all UK mines, but no clear support was found for Hudson's conclusion that the accident hazard is greater in thin seams. To make valid comparisons of accident statistics between countries, it is necessary to have a common base for reporting and

recording. No such common base exists, however, except in the case of fatal accident rates, and these are given for Europe and the USA. In this respect, the recent establishment of a common base for accident statistics among coal mining countries of the EEC has been an advantage. It has not been possible to obtain data for the USSR or any other Eastern European country, in spite of UK membership of coal mining safety committees, whose members include Eastern European countries.

State-of-the-Art

13. The state-of-the-art has been examined under twelve headings. These include a short historical survey of thin seam mining and the reasons for mining thin seams. World-wide deposits have been reviewed in terms of structural geology and other parameters affecting mining conditions, such as methane, spontaneous combustion, cleat, water, etc. In the USA, both the method of access from the surface and mine layout tend to be completely different from European practice. In thin seam mines especially, access is often by a group of roadways in the seam, driven into the hillside, and the subsequent mine layout is predominantly horizontal in nature. Thin seams in Europe are often mined in conjunction with thicker beds in deep shaft mines and, in Eastern Europe, the steep inclination of the seams requires the mine layout to be in the vertical as well as the horizontal plane.

14. The actual cutting mechanisms used to break and load the coal have been described. Many of these have been superseded by sophisticated power loaders that can be operated by remote control. Unmanned systems in seams as thin as 12 in have existed for some 20 years but, even today, they have low output capacities. The latest European technology for the transportation of men, materials and coal, both in and out of the mine, has been examined for possible adaptation to US conditions.

15. A review of productivity and costs in thin seam mines highlights the more difficult financial justification of the deep shaft mines in Europe, compared with the small underground mines in the USA. The literature, although adequate in technical content, provided sparse details on costs.

16. Health and safety have been considered along with environment and it is concluded that, although there is an additional health hazard from working in water in thin seams, a reduction in airborne dust and greater protection from noise have improved mines in thin as well as thicker seams.

17. The state-of-the-art is concluded with descriptions of typical mining systems and coal winning methods. These have been evaluated and compared in tabular form using criteria such as safety, productivity, costs, maximum working depths, minimum working heights, etc.

18. Besides the basic principle of longwall mining, there would appear to be some European equipment and techniques that could be adapted to US conditions.

Design and Layout
(Plus-30-in Seams)

19. Room and pillar mining is the principal method used in underground production in the USA, whilst longwall, in various forms, is the most widely used method in Western Europe and the USSR.

20. In order to evaluate the respective merits of these systems, including shortwall mining, a simulated block of coal, 33 in thick, is considered and seven conceptual designs are described for its extraction - three longwall, three room and pillar and a shortwall. The designs incorporate the best features of existing systems and, in some cases, would require variances from the code of regulations.

21. The designs are considered to be sufficiently practical, however, to give a reliable indication of the best results that can be expected from proven techniques.

22. The simulated block of reserves assumes ideal US thin seam mining conditions in order to simplify comparison of designs.

23. Detailed designs are developed for a typical mining section which would extract a panel of coal within the block. An examination of the controllable variables is made in order to optimise the design parameters and to arrive at the most cost effective arrangement of equipment and panel geometry.

24. Each of the seven conceptual designs includes a mine layout that illustrates how the individual panels are laid out and the sequence of extraction. The layouts represent the overall strategy of the mine plan.

Output and Productivity
(Plus-30-in Seams)

25. Values for output and productivity are generated for each of the seven candidate systems. A target tonnage of 1,500 tons of run-of-mine coal per day has been used so that the systems are compared on similar terms. However, the outputs are not constrained to 1,500 tons per day, but are allowed to achieve a "standard" output that would be expected of the system under the simulated conditions.

26. Labour requirements are evaluated to determine each system's productivity in terms of output per manshift. For the stipulated criteria, the longwall designs generated significantly higher productivities than the other systems.

Costs
(Plus-30-in Seams)

27. Total mining costs for the simulated block of reserves are evolved for each of the designs. These include labour and material charges, and the effect of the capital cost of each system. Use has been made of the concept of "capital recovery" as a cost element in order to compare systems having a high capital cost

with those having a modest capital cost. Evaluations are made at two interest rates - 20% and 10% per annum. The effect of inflation on the financial model is also discussed. Despite the high capital cost of a longwall and an interest rate of 20%, the total mining costs are comparable with room and pillar. When the interest rate is reduced to 10%, the advantage moves further towards longwall.

Comparisons and Sensitivities
(Plus-30-in Seams)

28. Comparisons and sensitivity tests are made for each of the candidate systems to illustrate the effect on productivity and costs by changes in design values.

29. Dependent on the overall objectives of the mining enterprise, such tests can indicate optimisation of resources and further analysis can be carried out to determine the most suitable system compatible with requirements.

Candidate Systems
(Minus-30-in Seams)

30. All the candidate systems described for thinner seam thicknesses have the inherent disadvantage of relatively low bulk output. This may not preclude their economic application in US conditions. However, no attempt has been made to evaluate them quantitatively in this respect. Instead, they are examined against the background of possible technological improvements that might ensure their viability. Two remotely-controlled systems, mole and auger mining, are identified as capable of being developed for higher productivities. These remotely-operated systems are described, their development traced and the results that have been achieved to date are indicated. Specific problem areas have been isolated and reviewed in the light of advanced engineering technology.

31. The scraper box system has been in use for many years in Europe. It is a mining technique that can be operated either remotely or, when used in the 20-in to 30-in thickness range, with miners working on the longwall. The scraper box technique offers considerable advantages, but there are several major problem areas, such as roof control, that have to be solved to realise the potential of the system.

32. Wide-web and full-face mining systems are evaluated and considered in terms of application to US conditions.

FURTHER RESEARCH AND DEVELOPMENT

33. There is considerable scope for research and development work to be done on systems and equipment suitable for the 30-in to 36-in range.

34. A problem that faces operators using present thin seam techniques is the relatively high delays caused by failure of equipment. Improvements are the responsibility of both the mining equipment manufacturers and the operators but these are likely to lead to higher-cost machines.

35. In room and pillar mining, support operations cause considerable dislocation of the production tasks and the use of a continuous miner with an integral bolter may be considered worthy of development, despite the difficulties of roof bolting in thin seams.

36. The use of roof bolts for support in thin seams is difficult and time consuming and, hence, it would be of great value to the industry to have equipment that could drill and install bolts automatically, from the aspects of both productivity and safety.

37. Systems suitable for use in seams less than 30 in thick require considerable research and development to overcome the problems that have been experienced in past trials and operation. These problems have been identified for each system described and a subjective indication of the amount of research and development work required is included in Chapter XXVIII.

CONCLUSIONS AND RECOMMENDATIONS

38. Depending on the actual thickness of a "thin seam", coal can be mined efficiently and safely. The cost of mining thin seams is generally higher than for thicker seams but, if the proceeds per ton are high enough, the operation can be viable.

39. Coal mine operators presently using room and pillar methods should be encouraged to re-examine their high quality thin seam reserves with a view to using the longwall method.

40. With the ultimate depletion of thicker seams and the vast resources of thin seams in the USA, it is prudent to keep thin seam mining techniques under close review.

41. To accomplish this:-

(i) Mining machinery manufacturers should be encouraged to evolve further the most promising prototypes developed in the last 20 years or so for seam thicknesses down to 24 in.

(ii) US Government research should assist in sponsoring this work to the point of conducting underground trials.

P H A S E I

CHAPTER I

RESERVES AND RESOURCES

GENERAL

1. The purpose of this chapter is to make an assessment, on both a national and a regional scale, of the magnitude, quality and distribution of thin seam coal reserves and resources in the USA, and to define those areas that merit further study, with a view to establishing a viable thin seam mining industry at some future date.

2. In view of the increased capital and operating costs that are inevitably associated with thin seam mining, it has been decided to restrict the present study to bituminous coal; it is considered that the calorific value of sub-bituminous coal and lignite would be insufficient to warrant the extra investment required.

3. The available information concerning US coal reserves and resources is so voluminous as to preclude a comprehensive survey in the time allotted for the study. Thus, although maximum use has been made of the data acquired by British Mining Consultants Ltd, the assessment of resources on a local basis, at the level of detail required for a feasibility study, would have to be the subject of a separate report.

4. Although a meaningful definition of the term "thin seam" should be related in some way to currently available mining methods, the upper thickness limit of thin seams is somewhat arbitrary and varies from one country to another. In view of recent US Bureau of Mines usage (138 and 139), and following discussions between the US Bureau of Mines and British Mining, it was decided that the most useful definition of "thin seam" would probably be "less than 36 in". This definition conflicts, however, with the thickness categories used in published US Bureau of Mines and US Geological Survey reserve and resource estimates. In these, "thin seams" are defined as 14 in to 28 in thick, whilst seams of 28 in to 42 in thickness are included within an "intermediate" category. The desired upper thickness limit of 36 in for thin seams falls, therefore, within the intermediate category, making interpretation and evaluation of the published data extremely difficult. For the purpose of this study, it was decided that a thin seam should be defined as being less than 42 in thick, thereby incorporating the published "thin" and "intermediate" categories and facilitating the efficient handling of data. Given access to the computer file of the Office of Coal and Electric Power Statistics, it should be possible to sub-divide the intermediate category into minus 36-in and plus 36-in components. Such an exercise is, however, beyond the scope of this report.

DEFINITIONS

5. The definitions relating to reserves and resources used in this report are those used by the US Geological Survey and US Bureau of Mines. With respect to bituminous coal, they may be summarised as follows:-

Resources

Concentrations of coal in such forms that economic extraction is currently or potentially feasible.

Identified Resources

Beds of bituminous coal 14 in or more thick that occur at depths to 3,000 ft, the existence and quantity of which have been specified within specified degrees of geological assurance as measured, indicated or inferred. Includes also thinner and/or deeper beds that presently are being mined or for which there is evidence that they could be mined commercially.

Reserves

That portion of the identified coal resources that can be economically and legally mined at the time of determination. Also referred to as recoverable reserve.

Reserve Base

Includes beds of bituminous coal 28 in or more thick that occur at depths to 1,000 ft. Also includes thinner and/or deeper beds that presently are being mined or for which there is evidence that they could be mined commercially at this time. Includes only coal from measured and indicated categories of reliability.

6. The following definitions are applicable to both the reserve and identified resource components:-

Measured

Tonnage computed from dimensions revealed in outcrops, trenches, mine workings and drillholes. The points of observation and measurement are closely spaced, and the thickness and extent of coals are well defined such that the tonnage is judged to be accurate within 20% of true tonnage. Although the spacing of the points of observation necessary to demonstrate continuity of the coal differs from region to region according to the character of the coal beds, the points of observation are, in general, no greater than one half mile apart.

Indicated

Tonnage computed partly from specified measurements and partly from projection of visible data for a reasonable distance on the basis of geological evidence. In general, the points of observation are about 1 mile apart, but they may be as much as $1\frac{1}{2}$ miles apart for beds of known continuity.

Inferred

Quantitative estimates based largely on broad knowledge of the geological character of the bed or region. Few measurements of bed thickness are available. The estimates are based primarily on an assumed continuation for which there is geological evidence. In general, inferred coal lies more than $1\frac{1}{2}$ miles from the outcrop or points for which mining or drilling information is available.

NATIONAL APPRAISAL

7. US coal reserve and resource figures are published by the US Bureau of Mines and US Geological Survey. Estimation of the proportion held in thin seams of bituminous coal is, however, complicated by the following factors:-

(i) US Geological Survey figures for total identified resources are classified by thickness and rank separately, but there is no sub-division of thickness categories according to rank.

(ii) US Bureau of Mines figures for the reserve base are quoted in terms of both thickness and rank, but the 14-in to 28-in category is excluded.

8. An estimate of total US thin seam identified bituminous coal resources was given in a recent US Bureau of Mines paper by Nelson and Johnson (138). Recognising the difficulties described in paragraph 7 above, these authors proceeded in the following manner to obtain their estimate for the thin (14-in to 28-in) category:-

(i) The proportion of total identified resources contained in thin (14-in to 28 in) seams was determined, for the measured and indicated categories, from US Geological Survey data. A figure of 7.5% was obtained, equivalent to 121.2 billion tons.

(ii) In order to obtain a break down of this figure according to rank, the percentage contribution of each rank to the US Bureau of Mines reserve base was calculated for the intermediate and thick categories. The resulting percentages for each rank were averaged, and these averages were assumed to be applicable to the thin (14-in to 28-in) category of the US Geological Survey measured and indicated reserves calculated in (i):-

Rank	%	USGS Measured and Indicated Resources (Billion Short Tons)
Lignite	6.4	7.8
Sub-bituminous	38.5	46.6
Bituminous	53.4	64.7
Anthracite	1.7	2.1
Total	100.0	121.2

(iii) Adding 121.2 billion tons to the 436.4 billion tons of measured and indicated thick and intermediate seam resources published by the US Bureau of Mines gave a total of 557.6 billion tons measured and indicated resources for all thickness categories. According to US Geological Survey sources, 91% of total identified resources lie in the 0 to 1,000 ft depth range; 91% of 1,731 billion tons = 1,575.2 billion tons. Subtracting 557.6 billion tons from 1,575.2 billion tons gives a figure of 1,017.6 billion tons which was taken to represent total inferred resources. This process of subtraction was then repeated for each rank:-

Lignite	400.5
Sub-bituminous	228.0
Bituminous	381.3
Anthracite	7.8
Total	1,017.6

(iv) Published US Geological Survey estimates of the distribution of thin, intermediate and thick seams for the inferred category of resources are 58.2%, 20.0% and 21.8% respectively. Applying these percentages to the break down by rank of 1,017.6 billion tons given above allowed the calculation of inferred resources by rank and thickness.

(v) The data for bituminous coal was abstracted and presented as follows:-

Bituminous Coal Resources (Billions of Short Tons)

Thickness Category	Measured and Indicated Resources		Inferred Resources	Total Resources	
	Tonnage	Percentage		Tonnage	Percentage
Thin (14 in to 28 in)	64.7	21.8	221.9	286.6	42.2
Intermediate (28 in to 42 in)	88.5	29.7	76.3	164.8	24.3
Thick (>42 in)	144.5	48.5	83.1	227.6	33.5
Totals	297.7	100.0	381.3	679.0	100.0

9. The assumptions made by Nelson and Johnson in order to arrive at these figures were rather large. Nevertheless, at the present stage of investigation it would be difficult to better their estimate and it is considered that the totals presented above are fairly realistic and perfectly adequate for the purpose of this study.

10. It can be seen from the figures of Nelson and Johnson that some 451.4 billion tons of bituminous coal are contained in thin seams as defined in this report (ie less than 42 in thick), representing 66.5% of total identified bituminous resources lying between 0 and 1,000 ft depth. Of this, approximately 153.2 billion tons lie within the measured and indicated categories. This is a very substantial amount of coal. However, a factor of much greater significance in the economic evaluation of thin seam mining potential is the distribution of these reserves, and the remainder of this chapter is devoted to a preliminary regional analysis of thin seam coal distribution and quality.

REGIONAL APPRAISAL

11. The reserve base in millions of tons (US Bureau of Mines) for seams in the 28-in to 42-in category is as follows:-

Eastern States		Western States	
Alabama	1,052	Arkansas	152
Illinois	6,817	Colorado	1,430
Indiana	1,881	Iowa	1,036
Kentucky	5,298	Missouri	4,612
Maryland	470	Montana	347
Ohio	6,787	New Mexico	486
Pennsylvania	9,296	Oklahoma	454
Tennessee	499	Washington	52
Virginia	1,882	Wyoming	830
West Virginia	11,824		

12. The regional distribution of US thin seam resources in the 14-in to 28-in category was considered by Ketron Inc in a 1978 report prepared for the US Department of Energy (143). Underground mineable resources for those states containing more than 2,000 million tons were listed as follows:-

State	Millions of Short Tons	% of US Identified Thin Seam Resources	% of Total Identified Resources in State
Alabama	5,490.95	6	41
Indiana	2,021.41	2	6
Kentucky (E)	9,275.42	11	41
Missouri	13,516.44	15	67
Ohio	11,976.45	14	35
Pennsylvania	13,030.38	15	21
Virginia	4,132.91	5	52
West Virginia	15,485.20	18	16

13. Together, these eight states contain, in the underground mineable category, 86% of total US thin seam resources. The figures listed include adjustments, wherever possible, for unclassified resources, strippable resources, and out-of-date resource estimates. An indication of coal distribution per county in which thin (14-in to 28-in) seam resources exceed 200 million tons is also given for each state listed. Thus, in Alabama 89% of thin seam resources occurs in five counties, in Kentucky 73% in twelve counties, in Missouri 80% in seventeen counties, in Ohio 79% in sixteen counties, in Pennsylvania 89% in ten counties, in Virginia 92% in four counties and in West Virginia 20% in five counties. A large proportion of these resources is contained in local, thin developments of normally thick seams.

14. Table I contains a listing, by state, of the coal quality and reserve statistics for the better known, exclusively thin (14-in to 42-in) seams. This information was abstracted from the Keystone Coal Manual for 1978. The listing should not be regarded as comprehensive, but it does convey an indication of those states in which a large number of thin seams occur, as well as a regional indication of thin seam quality.

15. It can be seen from the US Bureau of Mines and Ketron data that the bulk of thin (14-in to 42-in) seam resources are concentrated in Alabama, Colorado, Illinois, Indiana, Iowa, Kentucky, Missouri, Ohio, Pennsylvania, Virginia and West Virginia. However, Table I shows that the states of Georgia, Kansas, Oklahoma and Tennessee also contain prominent developments of thin seam bituminous coal, and although these states may not contain very large total resources compared with the major thin seam states, they may, nevertheless, contain local accumulations of sufficient magnitude to support a viable mine, and are therefore worthy of further study.

16. The reserve base (US Bureau of Mines) of low sulphur (<1%) bituminous coal in 28-in to 42-in seams for the USA is given as 10,723.91 million tons (138). In general, the coalfields of the Western Interior Basin (Iowa, Kansas, Missouri, Oklahoma and Texas) contain high sulphur coals, whereas those of the Eastern Interior Basin (Illinois and Indiana) and Appalachian regions contain low to medium sulphur coals. A notable exception to the former is the Croweburg seam of Oklahoma which has an exceptionally low sulphur content (see Table I).

17. The greatest development of high quality, low sulphur, thin seam bituminous coal in the USA appears to occur in the southern counties of West Virginia. Many of these seams also contain high grade coking coal. Other high grade, low sulphur, thin coking coals include the Aetna and Dade seams of Tennessee and Georgia (see Table I).

18. Tables II and III show coal quality for counties in which measured, indicated and inferred resources per seam listed exceed 60 million tons in the 14-in to 28-in and 28-in to 42-in seam thickness categories respectively. These tables constitute a list of potential state/county/seam targets for further investigation with a view to examining thin seam mining potential. The data were abstracted

from the appendices of the Ketron report, and a lower threshold of 60 million tons was chosen in the belief that, in view of current US overall recovery of approximately 30% of in-situ reserves (144), this is the minimum tonnage capable of supporting an economical, efficient mine of 1 million tpa output over 20 years.

CONCLUSIONS

19. Ketron conclude that depletion by mining has not yet created a situation in which thin seam deposits are of considerably better quality than thicker seam deposits (the only exception occurring in the state of Missouri), and that this situation will not change significantly in the foreseeable future. In particular, they stress the fact that sufficient coking coal resources are present in the thicker seams to satisfy demand far beyond the year 2000, and they recommend postponement of the decision to initiate the development of thin seam mining technology until the uncertainties concerning coal's long-term future are resolved. This is a tenable point of view, but it takes no account of social factors such as the possible need for regional development, or of certain economic factors such as the attraction of capital to the coal industry by stimulating thin seam mining technology and thereby encouraging companies and individuals who, with access to thin seam deposits only, would not otherwise be able to develop their resources. Nelson (139) cites two companies, located in Virginia and Arkansas, that have already expressed an interest in thin seam mining.

20. It is recommended that the resource study be extended during Phase II with a view to compiling a short list of areas suitable for the execution of a thin seam mining feasibility study at some future date. Efforts should be concentrated on the states, counties and seams listed in Tables II and III, and a thorough study of thin seam resource potential should be carried out in Georgia, Kansas, Oklahoma and Tennessee. Initial prime targets for examination in the latter states include the Croweburg seam of Oklahoma and the Aetna and Dade seams of Tennessee and Georgia. Availability of existing infrastructure is perhaps a factor to be taken into account in making the final choice. Simultaneously, a nationwide survey could be initiated with the object of determining the degree of interest at company level in the development of thin seam mining technology.

CHAPTER II

PRODUCTION

GENERAL

1. The overall picture of thin seam production throughout the world is a simple one. The two largest producers are the USSR and the USA. Other countries produce much smaller, but still significant, tonnages, namely, Spain, the UK, Czechoslovakia, Poland and Colombia. The most recent estimates of thin seam production are shown below:-

Country	Thin Seam Output in Million Tonnes	Proportion of National Deep-Mined Output	Date and Data Source
		%	
USSR	219.4	47.6	1973 (105)
USA	28.6	10.8	1975 (RFP)
Spain	8.4	70.0	1975 (84)
UK	7.0	7.0	1978 (100)
Czechoslovakia	7 to 9	30.0	1978 (112)
Poland	4.0	2.0	1978 (106)
Colombia	2.0	50.0	1975 (84)
France	4.1	7.8	1962 (26)
Belgium	7.5	38.4	1965 (26)
Germany	1.6	1.1	1965 (26)

The US figure has been adjusted to metric tonnes, and the Czechoslovakian tonnage is given as a range based on the actual national tonnage for 1978 and a forecast of the thin seam proportion (112). The Polish figure is another estimate which is based on the actual national output and information from a recent visit of a senior UK mining engineer.

2. France, Belgium and Germany produced significant tonnages in the early 1960s but have since virtually ceased production from thin seams.

3. A report of the Economic Commission for Europe (ECE) Coal Committee of 1967 (26) contains details of European thin seam production in the early 1960s and indicates that outputs from the Eastern European countries have not declined but have probably increased.

4. Within this broad scenario, the outputs can be discussed in relation to the mining methods and seam thickness.

5. The definition of a thin seam must be considered again, as it was in Chapter I on Resources. This time, the variation takes place between countries, and must be borne in mind when examining statistics.

6. The ECE Coal Committee Report of 1967 included the following thicknesses below which the seam was considered thin in different countries:-

	m	in
Belgium	0.60	24
Germany	0.70	28
UK	0.91	36
France, Poland, Ukraine and Czechoslavakia	1.00	39
USSR	1.20	48
Bulgaria	1.30	51

7. By comparison, the US production from thin seams comprises output from seams up to a maximum of 39 in thick.

8. Over recent years the trend in thin seam production from Western European countries has been one of contraction. In some countries this has accompanied a downward trend in the total output of the country, as in Belgium, Germany and the UK. In the USSR, the thin seam bulk production appears to have increased sufficiently to show a slight increase in its proportion of the total USSR underground tonnage. Another country showing a significant increasing trend is Czechoslovakia, which is planning to increase the proportion of production from thin seams to as much as 37% of the national output (112).

9. The reasons for the downward trend in Western Europe must be linked to economic factors. Advances in technology, which have been applied to the thicker deposits first, have widened the productivity gap between thin and intermediate seams.

10. Each country is discussed in order of its thin seam production, starting with the USSR, the largest producer. Mention is also made of China and Korea, although no quantitative production figures are available.

USSR

11. The total thin seam output from the USSR in 1963 was 166.4 million tonnes or 40.5% of the national deep-mined output and rose in 1973 to 219.4 million tonnes or 47.6% of the national total (26, 105). Thin seam production is concentrated in the Ukraine and, in 1963, the output mined from thin seams in this region was 98.5 million tonnes or 63% of the total underground Ukrainian production. The seam section used to define this output is 1 m (39 in) which differs slightly from the overall USSR definition of 1.2 m (48 in).

12. The Ukraine region largely overlaps the Donetz coalfield and the table below shows the seam thickness, gradient and proportion mined in each category for the mines of the Donetz coalfield in 1963. It indicates a significant output from seams less than 0.7 m (28 in) thick and the proportion from steeper gradients.

Output of Rom Coal from Seams Below 1 m Thick

Angle of Dip	Thousands of Tonnes	Proportion in %
Less than 8^0	25,757	26.2
8^0 to 18^0	34,288	34.8
19^0 to 24^0	8,433	8.6
24^0 to 35^0	4,470	4.5
36^0 to 44^0	6,608	6.7
45^0 to 60^0	9,550	9.7
above 60^0	9,373	9.5
Total	98,479	100.0

Seam Thickness Categories m	Thousands of Tonnes	Proportion in %
Up to 0.5	2,740	2.8
0.51 to 0.7	27,084	27.5
0.71 to 1.00	68,655	69.7
Total	98,479	100.0

13. Although these figures probably do not include all USSR thin seam production, they are, in themselves, formidable tonnages from thin coal and of the 98.5 million tonnes, 38.4 million tonnes came from faces with a gradient of more than 18^0.

14. The system of mining these thin seams is almost exclusively longwall with about equal proportions of advance and retreat. Thin seam mining is likely to remain a strong feature of USSR production in the future, as the thin seams contain high grade coking coals and are the only source of anthracite.

USA

15. The total thin seam tonnage in 1975 was 31.52 million short tons (28.6 million tonnes), which is 10.8% of total underground production. Of this figure 1.61 million short tons were mined from seams less than 30 in thick (138).

16. Since total underground production has reduced slightly from 1972 to 1977, it is likely that a similar drop has taken place in thin seam extraction. This view is supported by the significant reduction in the category "hand cut or shot from solid", because this is the method most widely used in seams less than 30 in thick (82).

17. In contrast, there has been an increase in the quantity and proportion of coal from longwall installations. Despite the application of longwall to very thin seams in Europe, this method of mining has been applied in the USA only to the medium seam range. In 1977 there were 11 installations in the range of 36 in to 42 in or 14% of the total number of longwalls (82).

18. A further attempt to detect a production trend has been made by examination of the outputs from the states that mine most of the thin seams. These include Virginia, Tennessee and Eastern Kentucky, and they showed marginal increases in their annual production from 1973 to 1977. Alabama has remained constant while Pennsylvania and Western Virginia have declined during this period. The overall picture from these six states shows only a marginal decline.

19. The inclinations of thin seams in the USA, which are currently exploited, tend to be flat or slightly dipping.

SPAIN

20. The annual output of deep-mined bituminous coal in Spain in 1978 was 12 million tonnes. The thin coal region is in the north in the Asturian mountains. Here the seams are 3 ft or less in thickness and steeply inclined, except on the top of folds or the bottoms of basins. Output from the Asturia region in 1975 was over 8 million tonnes and this was won principally by hand-got, but also by mechanised longwall (84). The principal form of mechanisation is the USSR TEMP cutter loader and this equipment has mechanised approximately 20% of all faces.

UK

21. Thin seam production, both in terms of total output and proportion, has shown a significant downward trend. The total tonnage from thin seams in 1965 was 33.83 million tonnes, representing some 18.5% of the total annual deep-mined output (26). In 1978 the figure dropped to 7.0 million tonnes or 7% of the total deep-mined output (100).

22. The principal reason for this marked decline has been the unacceptable level of output obtained from seams in the 18-in to 24-in range, which were extensively exploited in the mid-1960s, particularly in the Durham coalfield. The majority of thin seam faces today lie in the 32-in to 36-in range and are equipped with power supports and thin seam versions of shearers and trepanners. In 1978 there were only four installations operating in seams less than 30 in, all equipped with ploughs.

CZECHOSLOVAKIA

23. Coal is mined in two main districts of the country, Karvina, where the seams are thick, and Ostrava, where the seams are thin. The Ostrava coalfield has many mining problems, not least of which is the gassy nature of the seams. They are also tectonically disturbed, steeply inclined and liable to outburst. There is also the problem of water-bearing strata overlying the seams. Despite these difficulties, in 1964 the mining industry produced over 26 million tonnes, of which 24% or 6.3 million tonnes came from thin seams. The indications are that this proportion has probably increased in recent years (112). A likely figure is around 30% which, if applied to the 1978 national output of 28.3 million tonnes, means that thin seam production probably lies within the range 7 million to 9 million tpa.

24. During the period 1964 to 1970 there was a marked increase in the number and production of power loaders. In the more highly inclined seams, ploughs and scraper boxes still produce substantial tonnages (112).

POLAND

25. Thin seams are worked in both the principal coalfields of the country, namely Upper Silesia and Lower Silesia. In the Upper Silesian field the thin seams are found at the top of the coal measures, whilst in the Lower Silesian field they lie near the bottom. In 1964 the gross output from thin seams was 5.4 million tonnes or 4.6% of the country's entire output of coal (26). The trend has been to reduce the output from thin seams, and a recent estimate made by visitors to the mining industry was 4 million tonnes or 2% of the total underground production. This reduced output from thin seams in Poland must be set against a rising total output from the mining industry which increased from 117 million tonnes in 1965 to 192.6 million tonnes in 1978 (106).

26. The thin seam coal is produced mainly from ploughs and a few wide web cutter loaders and shearers.

COLOMBIA

27. Colombia possesses about 80% of the total reserves in South America. In 1975 the national production was around 4 million tpa, approximately half of which was mined from seams around 1 m (39 in) in thickness. The mines are small and operated by antiquated methods. Despite the lack of mechanisation, the output of 2 million tonnes is still a significant tonnage from thin seams.

FRANCE

28. Thin seams are found in all the French coalfields, namely Pas-de-Calais, Lorraine and Province. Economic pressures compelled the French mining industry to abandon thin seam production in the late 1960s. It is of historic interest only to note that production from thin seams in 1962 amounted to more than 4 million tonnes from seams of 1 m (39 in) and less. This represented nearly 8% of the country's total coal production. The total annual output declined from nearly 50 million tonnes in 1962 to nearly 20 million tonnes in 1978. When the thin seams were mined, about half of the thin seam production came from ploughs and scraper boxes, and the other half from blasting and pneumatic picks.

BELGIUM

29. The contraction of thin seam mining in recent years in Belgium mirrors the trend in France. In 1965, thin seams were still mined in two areas, namely Charleloi and Leige. At this time over 7 million tonnes were produced from thin seams and this amounted to nearly 40% of the total output. By 1978, the total production had dropped to less than 7 million tonnes and no significant tonnage was mined from thin seams.

GERMANY

30. Today, thin seam production is concentrated at one mine in the Aachan area, which produces nearly 2 million tpa from seams ranging in thickness from 33 in to 44 in (1). In 1965, the output from some 26 longwall faces in thin seams amounted to 1.6 million tonnes. This was from seams less than 0.7 m (28 in) thick and represented only 1% of the total German output. From 1965 to 1978 the total German output fell from nearly 160 million tonnes to 84 million tonnes (106).

CHINA

31. Information received by visitors to China indicates that little or no thin seam production is currently taking place in China.

32. Historically, many mines were owned and operated by Belgian companies. Today, room and pillar mining is giving way to widespread acceptance of the longwall system and, in an effort to modernise and expand the industry, longwall equipment has been purchased from European countries. It includes powered supports, shearers and ploughs, all of which are for seams in the range of 1.5 m to 2.5 m (5 ft to 8 ft 6 in).

KOREA

33. Production in the whole peninsula now exceeds 60 million tpa, mostly anthracite (109). The seams are steeply inclined, heavily faulted and folded. Mining is difficult and is often carried out using metalliferous sub-level caving methods. Isolated pockets of thin seams are mined but are not mechanised (84).

CONCLUSIONS

34. It is difficult to forecast future trends in thin seam production as so much depends on economic and other factors.

35. New aspects are beginning to make an impact in the USA in the form of environmental controls on the higher sulphur coals burned in utilities. This could encourage the development of thin seam underground mines where the coal has a low sulphur content.

36. In Western Europe the downward trend is likely to continue and, although Eastern Europe has not lost ground in thin seam production, there could be reductions in the future unless some technological breakthrough takes place.

CHAPTER III

ACCIDENT ANALYSIS

GENERAL

1. An accident has been defined as "any unplanned exchange of energy which degrades the system in which it occurs" (30). This is a fundamental definition that makes no reference to injury to personnel. However, it is the effect of the accident on mine personnel that is most noticeable, and the records of such injuries provide the bulk of the statistical information on accidents. In most countries this wider concept of an accident is reflected in mining legislation that demands formal records and reporting of certain dangerous occurrences that may or may not cause personal injury.

2. One of the major factors in determining whether an accident is recorded and reported is the nature of the injury sustained, ie the effect in terms of disability and the time the person is prevented from working by the injury. The severity of the injury may not be directly related to the hazardous circumstance that initially caused the accident. The position of persons at the time of the occurrence is often the critical factor. Transport accidents are examples of this type; a runaway vehicle can either cause serious injury or no injury whatsoever. Secondary factors, such as the prompt and efficient application of first aid and subsequent medical attention, can reduce the chance of more serious medical complications and hence the time an employee is absent from his work.

3. The greater the numbers of people exposed and the longer the period of exposure, the greater the probability is that the number of injury-causing accidents will reflect the hazardous nature of the environment in which the persons are working (98). It is necessary in any analysis of accidents and accident rates that the exposure periods concerned and the numbers of persons exposed be sufficiently large for the results to be statistically significant.

4. In a comparative analysis of accidents in different countries, problems arise with the definitions of accidents. One of the few accident statistics that gives a reliable guide to the safety standards in coal mining countries is the fatality rate. This is usually expressed in terms of 1 million man-hours of exposure or 1 million tons of production. The statistical data for less severe injuries can provide meaningful comparisons between countries of the EEC but wide discrepancies exist between the EEC countries, the USSR and the USA.

US ACCIDENTS

Problems of Comparison in Different Seam Heights

5. Accidents in the USA are recorded and reported in terms of the Code of Federal Regulations 30 CFR 80. The specified reporting procedure does not require, amongst the details to be supplied, a record of the seam height of the

accident. In order to correlate the relative hazards with the seam height, it is necessary to review the available statistics together with additional seam height information.

6. The comparison can be made in a number of ways, one of which is by taking the records of accidents occurring at individual mining operations and comparing them with similar operations working in a different seam height. This can form a guide but could be unreliable owing to the very limited numbers of persons and hours of exposure involved. As some mines work more than one seam and the seams may be of differing heights, the comparisons become less valid.

7. Another approach might be to consider mining a whole area or district where there is a preponderance of workings at a certain height. This would increase the time exposure, and hence make the statistics more significant, but would further introduce the distortion of some mines in the sample operating at seam heights different to the average for the area. A further problem arises in comparing geographical areas as the overall trends may be distorted by differences in geology, mining methods and miners' experience.

Hudson's Study

8. The purpose of Hudson's study was to examine statistically the injury hazards in different coal seam heights in the USA (114). The study collected data from over 1,000 mines and analysed it under three seam heights - low seams less than 36 in, medium seams of between 36 in and 72 in and high seams of over 72 in. The study covered a period from January, 1974 to 30th June, 1976 and the analysis was carried out for the 20 major categories of accidents listed in Codes 01 to 20 of the Federal Regulations. The results of the analysis have been integrated with UK statistics and are shown in Table IV.

9. Owing to the relatively low numbers of incidents involved, no significant variation was found in the frequency of rates for fatalities between the seam heights examined, although the average rate for low seams was higher than for the medium or high seams. The frequency rate of disabling injuries was approximately 100 times higher than the fatality rate and, hence, the statistical data become much more meaningful. In all the categories where sufficient data existed, it was found that the accident rate was significantly higher in the low seams than in the high or medium seams.

10. Possible explanations advanced for an increase in the level of hazard as the seam height decreases were poorer lighting and uncomfortable working conditions.

11. In the case of injuries from falls of roof, it was suggested that it was harder to avoid an imminent fall in the more cramped conditions of a thin seam. Another explanation put forward was the difficulty of providing protective cabs and canopies on low seam face equipment.

12. A further factor discussed in Hudson's study is the possible correlation between the size of the mine, in terms of personnel employed, and the seam height. There is a relatively greater number of small mines in the thin-seam category, while the higher seams tend to be mined at larger mines. The inference is that the larger mines may have better safety programmes and a more effective safety organisation.

13. In contrast to the conclusions for disabling accidents, a reverse trend was apparent for non-disabling accidents. In the majority of accident categories, the frequency rate for the low seam mines was lower than for the higher seam mines. Possible explanations advanced for this trend were:-

(i) "Low" coal accidents are more likely to be serious once they occur, since it is harder to get away from or to correct a potential accident situation, owing to the confined space.

(ii) Errors or omissions in reporting accidents, on the basis that small mines in low seams may not adequately record as many non-disabling accidents as larger mines in the thicker seams.

14. Support for the latter point is drawn from an audit carried out on metal and non-metal underground mines, which showed that the smaller operations were less likely to report accidents than the larger mines.

15. From the analysis of sub-categories of falls of roof, the higher proportions of accidents in low section seams show that more persons were injured whilst setting timber in the thin seams than in the thicker seams. This is explained by the more prevalent use of timber support in low sections. The study refers to the difficulty of using roof bolts as a support medium in thin seams. This is a considerable disadvantage, as such a support provides unobstructed travelling ways, which are necessary for mobile machinery in room and pillar mining.

16. One of the aspects of accident analysis not examined in the Hudson study was the nature of the injury sustained by the miners. This often provides a pointer to effective accident prevention and could be added to the suggested list of further studies found at the end of the Hudson report.

Information from Other US Sources

17. From visits to thin seam mines in the USA and discussions with mine managements, it was apparent that a number of reasons could be found to justify the low accident record at some small mines. Good morale at the mine had the effect of reducing accidents and better geological conditions, in terms of good strong roofs and floors, was also suggested. Finally, the introduction of the scoop car has reduced the amount of physical lifting and handling of materials and supplies.

18. At a large capacity mine, working room and pillar with continuous miners at an extracted height of 48 in, the most typical injury was considered to be a sprained back. This mine also operated a longwall and accidents in this district of the mine were considered to be fewer than in the continuous miner sections, although no data had been extracted from the statistics to substantiate this view.

UK ACCIDENTS

Reporting Procedures

19. The UK Mines and Quarries Act requires notification of accidents that result in fatalities or serious injuries to be given to the mines inspector for the district (31). Serious injuries are defined in terms of fractures and other bodily damage, but there is no reporting requirement for loss of working time. There is no UK equivalent of the US disabling injuries. Additional records are kept by the National Coal Board, who directly mine virtually all the deep-mined coal in the UK. These records contain details of all accidents that result in time being lost from work owing to injury.

20. The overall trend in British coal mines has been a steadily decreasing accident level. In terms of fatalities, the rate has fallen from 4 per 1,000 men employed in the 1850s to 0.25 per 1,000 in the 1970s. The rate is now so low as to make the use of fatal accidents, as an indicator of relative risk in different seam sections, insignificant.

21. No statistics are published relating accident rates to seam thickness. In order to investigate accident trends in thin seams, statistics were obtained from five mines, all of which mine exclusively in seams of 36 in or less. These data were used to construct a special study that attempts to compare UK experience with US experience, as recorded in the Hudson study.

Special UK Study

General

22. A special study was commissioned covering more than $2\frac{1}{2}$ million manshifts over a period of four years. Frequency rates in the study are expressed in terms of accidents per 100,000 shifts. Categories of causes of accidents are similar to the categories used in the USA, except that injuries from falls of roof, machinery and haulage are distinguished in their location between the coal face and elsewhere below ground.

23. The sample mines all operate the longwall system but each uses a different winning and loading technique. This varies from hand loading with props and bars to shearers with powered supports.

24. The results of the sample survey are shown in Table IV, where the accident frequency rate for each category of accident is compared with the British national average for the same period. The table also includes Hudson's analysis.

25. The study does not attempt to establish a strictly statistical comparison between thin seam mines and all others. The figures are simply averages for accidents rates expressed in terms of 100,000 manshifts of exposure and this limitation must be borne in mind in the following discussion.

Falls of Roof at Face

26. In terms of both serious and less serious accidents, the rates for the thin seam mines are higher than for all other mines. This may be attributed to two causes; firstly, the lack of mobility in the lower section to take evasive action. Secondly, thin seam supports tend to be of lighter construction in order to afford the maximum available travelling and working space for the men employed on the face. In consequence, they provide less resistance to the roof, which may be significant in "heavy" strata conditions.

Falls of Roof Elsewhere

27. In terms of serious accidents, the rates are identical between the thin and thicker seams. This is to be expected as, in general, the access roads are driven and constructed at similar heights, irrespective of seam section extracted. Legally, the travelling roads must be at least 5 ft 6 in high. There is, however, a marked reduction in the rate for all accidents in thin seams. One explanation might be that the repairs and enlarging of roadways is less prevalent in a thin seam mine.

Falling Objects

28. For the thin seam mines, the rate for serious accidents from falling objects is approximately half of that for all mines. However, the rates themselves, at 0.04 and 0.09, are low and hence the significance of the difference is suspect. In the case of the less serious accidents from falling objects, the thin seam mines have a lower rate than the other mines.

Haulage and Transport

29. For all mines, the rate for serious accidents in this category is of national concern. The thin seam mines show up more favourably than the national average and the reason may be the more stable roadway conditions normally found in thin seams, consequently causing less disruption to haulage systems. It is noticeable that a relatively small proportion of the accidents in this category take place at the working face itself.

30. The rate in all haulage and transport accidents shows a slightly higher figure for the thin seam mines. On the coal face, in the thin seams, the rates are also higher at 1.5 compared with 0.7. An unfortunate feature of the haulage accidents is that whilst they contribute only 6% of the total, they are responsible for over a third of the serious accidents.

Machinery

31. There is no significant difference in the rates of accidents from the use of machinery in the thin and thicker seam operations.

Use of Tools and Appliances

32. Serious accidents from the use of hand tools in thin seams are rare and in the sample no serious accidents were recorded against this cause. The rate for all mines was very low at 0.04 indicating no significant difference between the relative hazards in diferring seam heights. In the case of all accidents, the rate for the thin seam sample is slightly higher, possibly indicating that the restricted space available is responsible for a higher number of minor injuries owing to awkward positioning.

Handling Supplies

33. The serious accident rates, for both thin seams and all mines, were identical. In the case of all accidents, the rates, whilst accounting for over 20% of the total number of underground accidents, were similar. This would indicate that the extracted seam height does not have a significant effect on the accident potential of handling materials.

Stumbling and Falling

34. Stumbling and falling account for the highest number of total accidents in any single category and this high rate is reflected in the serious accident category. For all accidents, the comparison between thin seams and others shows a slightly higher rate, 32.4, for the thin seams compared with 29.8 for all coal mines. The rate for serious accidents resulting from slips or falls is much higher for thin seams, 0.27 compared with 0.16 for all mines. By definition, these accidents must take place away from the face. No explanation is offered for the higher level in the thin seam mines.

Other Causes

35. In this category no serious accidents occurred in the thin seam mine sample, compared with a national average of 0.06. The latter figure, however, is low and hence no account can be attached to the difference in rate between the seam thicknesses. In the case of all accidents, the rate of 15.3 in thin seams, compared with 12.4 for the other mines, shows a slightly higher risk in thin seams.

Conclusions

36. The results of the special survey indicate that for fatal and serious injuries, the total of all categories for the thin seam mines is slightly lower at 1.04 compared with 1.20 for all mines. A reverse situation is shown in the case of all accidents where the low seam rate of 118.6 is slightly higher than the average for all mines at 113.9. These differences do not appear to be statistically significant.

COMPARISON OF US AND UK STUDIES

37. Although Table IV is of interest for its comparison of thin seam mines and all other mines in the UK, there appears to be little correlation between the US and the UK studies. Bearing in mind the statistical limitations of the UK figures, a significant advantage appears to lie with the thin seam UK mines in terms of haulage. This is at variance with the US experience and probably can be explained by the entirely different roadway heights used in each country.

38. Overall, the UK thin seam mines do not appear to be significantly different, in their propensity for accidents, to the average UK mine.

39. Owing to the marked differences in mining systems prevalent in the UK and USA, there is little value in making direct comparisons, although it may be of value at some stage to follow up the view of some US operators that longwall systems are safer than room and pillar.

SAFETY ASPECTS OF UK REMOTE MINER TRIALS

40. In both the longwall and room and pillar systems, the face operators work under newly-exposed roof. However, in the longwall system, as practised today, the density of support applied to the roof is very high and, in the case of the shield and chock shield supports, men work under virtually a roof of steel. These effective means of roof support probably make longwalls safer than room and pillar. In thin seams, the vertical space for men to work and travel is restricted, and the development of heavy-duty shield supports is not as advanced as in the thicker seam sections. There is thus a need to develop manless systems in thin seams for two reasons:-

(i) to reduce the accident potential to zero, and;

(ii) because in very thin seams it may be impossible for men to operate.

41. A number of experiments and trials have been conducted in the UK with the object of removing men from the coal face. These include trials with scraper boxes in Durham in the 1950s, remotely-operated longwall faces (ROLF), and the "Collins" miner trials of the 1960s (3, 2).

42. From a strictly manless standpoint, the "Collins" miner is of greatest interest. This is because it was a later development than the manless scraper boxes, produced higher outputs and was also better documented. The ROLF faces had, in fact, men on the face line at various stages of the trials and rarely operated manless.

43. The "Collins" miner trials at Rothwell Colliery in 1965-66 (2) demonstrated the reduced accident potential of such a system. The trials were carried out in a 36-in seam that was being worked concurrently with a longwall shearer face in another part of the mine.

44. The work carried out at Rothwell established the feasibility of the remote miner concept. Although only 34,288 saleable long tons were produced during the trial, there were no serious accidents and only three minor accidents resulted in lost time. The statistical details of this good accident record are not clear but it does appear that the Collins miner district was considered to be much safer than other parts of the same mine.

45. During the period of the trial there were two major influences that may have affected the accident rate. The project was highly supervised and this probably improved the safety of the district. By contrast, however, the workmen and supervisors were working with a new technique and new equipment that was unfamiliar to them. In practice, it is possible that these two influences cancelled out each other.

46. Another safety-related aspect of the trial was the very low rate of only 4.4% absenteeism during the period. This compares very favourably with an average rate of 17% for the mine at the same period. The better attendance can be partially explained by the much improved physical working environment, where the men worked in a spacious, well-lit area and were some distance from the noise and dust of the cutting process. Lower absenteeism is often associated with high morale and results in more stable working teams which lead to improved co-ordination and safety.

EUROPEAN ECONOMIC COMMUNITY ACCIDENTS

47. Owing to the general decline of coal mining activities within the countries of the European Economic Community (EEC), only three members have an active mining industry, excluding the UK. These are the Federal Republic of Germany, France and Belgium.

48. In the early 1960s Belgium had a substantially higher proportion of production from thin seams than the other countries, with over 25% of its output coming from seams of 0.6 m (24 in) or less (101) compared with 1.1% for Germany and 7.8% for France.

49. Today, there are only small production outputs from thin seams within the EEC countries, other than the UK. Accident statistics are presented in a common form but because of the limited thin seam production, no useful comparisons are available for use in this study. The table below is of general interest only. It shows the high fatality rate in Germany and the high accident rates in both France and Belgium. The latter countries have a comparatively low proportion of power-supported longwalls and this could have a significant affect on their accident rates.

EEC Statistics for 1977

	Production (million tonnes)	Accident Rates Per 1 Million Man-Hours of Exposure		Mechanised Mining	Powered Supports
		Fatal	Total Accidents	%	%
Germany	91.2	0.36	127.8	98.8	29.6
France	21.3	0.19	228.6	87.0	42.0
Belgium	7.1	0.10	328.7	100.0	61.1
UK	120.8	0.11	138.6	93.8	95.2
Total EEC	240.4	0.2	152.4	95.3	87.4

USSR ACCIDENTS

50. No direct figures on the prevailing accident rates have been obtained for the USSR coal mining industry. However, it can be deduced from the published literature that the Soviet Union's mining industry has its safety problems. The fact that thin, intrinsically uneconomic seams are mined in order to destress ground and alleviate outburst conditions in adjacent seams, shows that considerable hazards exist from outbursts.

51. One reference (87) described remote augering of thin inclined seams and laid strong emphasis on the fact that during the trials, when some 2,000 holes were bored, there were no recorded accidents. It is reasonable to assume that had this coal been mined in any other fashion, accidents would have occurred.

52. This experience with augering is similar to the UK experience with the Collins miner, in that the remote operation was much safer than other mining techniques.

SOUTH AFRICAN ACCIDENTS

53. Accidents that result in fractures, amputations or absence from work for a period of more than 14 days are legally reportable under the South African Mines and Works Act.

54. Coal mining in South Africa is predominantly in two regions, the Eastern Transvaal and Orange Free State, and the Province of Natal. In the Transvaal, the seams worked are thick and in Natal the thinner coking seams are worked. The accident rate in the thicker mines is lower than in the thinner mines, except where the thinner No 5 seam is worked for blend coking coal in the Transvaal.

55. The predominant method of working in South Africa is bord and pillar. The thick seam mines have largely adopted conventional mechanised working, whilst hand loading is still common in the thin seam (42 in) operations.

56. Accidents from roof falls are more common in the thin seam operations, mainly due to weaker roofs. Haulage and transport accident frequencies are also high due to the use of track equipment and tubs in low seams.

57. The table below shows the combined fatal and serious accident rates for the Transvaal and Orange Free State, and for Natal, arising from falls of roof and face, haulage and material transport.

Casualty Rate Per Thousand Men Employed Per Annum - 1977	
Transvaal and Orange Free State (thicker seams)	10.159
Natal (thinner seams)	17.185

SOUTH AMERICA ACCIDENTS

58. Approximately half the coal production of South America is produced in Colombia. The southern coalfields of Colombia are predominantly thin seam operations with a high proportion of steep workings.

59. The collection of accident statistics in Colombia is not reliable as there is no legal obligation for the mine owners or operators to report and record accidents. Information gathered by British Mining Consultants on a recent project carried out in that country indicates a very high rate for all accidents of some 380 accidents per 100,000 shifts worked.

60. The high accident rate cannot be attributed completely to the antiquated methods used in thin seams but rather to the lack of a tight control on standards of work.

COAL INDUSTRY COMPARISONS

61. It is not the purpose of this study to examine the overall safety record of coal mining industries but a comparison on a limited basis is of interest. The only comparable statistics are those derived from fatalities and these are tabulated below. They have been compiled from the data sources, reference numbers (27) and (61). The variations in both accident classification and accident reporting between the USA and Western Europe make further comparisons invalid.

Underground Fatality Rates Per Million Man-Hours - 1977

USA	0.44
Germany, France and Belgium	0.29
UK	0.11

Underground Fatality Rate Per Million Tons Output

USA all underground coal mines	0.39
Germany, France and Belgium	0.69
UK	0.27

CONCLUSIONS

62. US experience indicates that the accident frequency rate per million man-hours of exposure in thin seams is higher than in medium or thick seam mines.

63. If the accident frequency rate is calculated on the basis of accidents per million tons, then, due to the lower productivity in thin seams compared with medium or thick seams, the rate will be substantially higher.

64. Longwall experience in the UK shows that there does not appear to be an increase in the hazard potential in thin seams compared with thicker seams.

65. The higher accident hazard in the USA in thin seams is partly due to the difficulty of working by means of the room and pillar system which involves frequent moving of large items of machinery in confined spaces. Inherent difficulties in supporting the roof is another contributory factor.

66. UK and USSR trials with remote mining systems have indicated that, where men can be removed from the face operations, then the expected improvement in safety will be realised.

67. In order to make substantial safety improvements in thin seams in the USA, it would appear that the development of remote control equipment is necessary, preferably using systems where the operators and other personnel can carry out their functions from a well-supported and safe position that offers a comfortable working height.

CHAPTER IV

STATE-OF-THE-ART - HISTORY

PRE-MECHANISATION

1. Over the last 50 years, high bulk production of coal from thin seams has been concentrated in the USA and certain European countries, principally the USSR, the UK, Germany, France, Belgium, Czechoslovakia, Poland and Spain. Traditionally, thin seams have been mined in other countries but on a relatively small scale and using, even today, antiquated methods. Colombia is in this category and a small anthracite mine in Ireland currently mines a seam 12-in to 24 in thick, using the hand-filling method.

2. In the UK, thin seams were mined along with the thicker beds during the industrial revolution. This started with Newcomen's steam engine in the 1720s and reach its zenith in the coal mining industry around 1900. In this period, thin seams, which are found as relatively flat deposits, were won from hand-got places or sections. These places widened out to resemble shortwalls. They were undercut by hand pick, blasted down and hand-loaded into tubs which were then hand-trammed along a roadway to a central collecting point and hauled out of the mine by ponies.

3. Later, the places or stalls were developed side by side and linked to one another at the face, either in line or in echelon. This was the forerunner of the longwall system in the UK.

EARLY MECHANISATION

4. The most difficult operation was holing-out the coal before blasting, and this was the first operation to be mechanised, by the introduction of coal cutters. These were introduced around 1850 and one machine undercut the longwall, which, even then, had reached lengths of up to 450 ft.

5. In continental Europe the coal is more disturbed tectonically and softer than in the UK. This led to the development of the pneumatic pick in contrast to the coal cutter, a method of winning softer coal that is still widely used throughout the world.

6. The steeper pitching seams in Western Europe led to mining layouts that are still in use. These incorporated much shorter longwalls, with lengths of between 120 ft and 150 ft.

7. Mining methods that used either coal cutters or pneumatic picks still required hand-loading, and in thin seams such an operation was difficult. This promoted the development into conveying that took place towards the end of the last century. The first belt conveyor was introduced underground in the UK in 1906, and small chain conveyors followed almost immediately. In Germany, prior to 1914, a shaker conveyor was developed to convey coal down gentle gradients.

8. By 1939, the longwall system was the predominant system in use throughout Europe, albeit with shorter walls in the steeper seams and longer ones in the flatter deposits. In the UK, coal was undercut, blasted down by explosives and hand-filled on to a conveyor. Three main types were used: belt, chain and shaker. In the rest of Europe, coal was won by pneumatic pick and hand-loaded on to shaker or chain conveyors. Few belts were in use at that time.

9. Coal mining in the USA began in the same way as the rest of the world, by digging seams from the outcrop. Coal pillars were used as the main form of support for the workings, and the shallower depths of the deposits, combined with the abundance of reserves, has allowed this method of mining to continue into the mechanised era of today.

10. Although there are fundamental differences between room and pillar and longwall, the principles applied in mechanising face operations were similar. Each manual task was replaced by a machine in order of priority of the greatest need.

11. Coal cutting was introduced in the last quarter of the 19th century, together with coal punching machines. Mobility was essential and an early development was a rail track-mounted arc-wall coal cutter. This was followed, in the early 1920s, by a Joy loading machine, also mounted on rails.

12. For thin seams, low height coal cutters were available throughout this period of mining mechanisation, but it was not until the early 1930s that the first low seam loader was produced, followed by a much improved version in 1939.

MECHANISATION - 1945 TO 1960

13. The second world war (1939 to 1945) provided a stimulus to increase bulk production, and experiments in thin seam mechanisation were widespread in Europe. The objectives were both to cut the coal and load it on to the face conveyor. This phase of development produced the Meco Moore cutter loader for use in thicker seams of 4 ft 6 in and upwards. In the thinner seams it was more difficult to combine the two operations and, here, the effort was centred around adapting the longwall cutter to load the blasted undercut coals by fitting flights or loading paddles to its cutting chain. Another purpose-designed machine used for this operation was the Huwood loader.

14. An important development, soon after the war, was the Gloster Getter. This was a multi-jib machine that cut out the complete seams, usually about 36 in, and attempted to load the cut coal on to a belt conveyor by means of a curved chute. Although it had limited success, it proved to be the forerunner of continuous mechanised longwall mining. Previously, longwalling was a cyclic operation with the three main operations being carried out on different working shifts. The wall was undercut on the nightshift and blasted and hand-filled on the dayshift. The afternoon shift moved the face conveyor in to the new track and constructed goaf pack walls from material that fell from the goaf. Advancement of the roadways, at each end of the face, also took place on this shift.

15. The Gloster Getter was the first machine to be used with a system that did not perform cyclical operations on specified shifts. The walls were shorter than normal, about 300 ft, and worked on a continuous system whereby the teams of men performed all operations on their shift. When the Getter had cut and loaded a 2-ft slice of the face, the back-up cycles of conveyor moving, packing, and roadway formation were completed. The system was given the name "composite mining" and resulted in doubling the face output per manshift. Later, lightweight flexible scraper conveyors were used on the face, so setting the scene for the development in the UK of the modern mechanised longwall with its ability for virtually continuous coal production.

16. Immediately after the second world war, cross fertilisation of mining methods took place between Germany, France, Belgium and the UK. Germany was first to use an armoured flexible conveyor and, because its coal was softer than the UK, it developed its thin seam techniques around the plough.

17. For thin seams, the Haarman scraper box equipment was developed in Germany and, subsequently, used extensively in the UK. It is still in use in the USSR and other Eastern European countries.

18. The impetus of the war also applied to Germany, and pre-war experiments with the plough were linked with the development of a stronger face chain conveyor. By the end of the war there were some three different types of ploughs in use on a total of 15 longwalls.

19. In more steeply inclined seams, the plough principle appeared in many modified forms, such as the chain saw and the hook plough.

20. Success with ploughs in the UK was tempered by the harder coals, and this led to the development of activated ploughs. These incorporated some form of blade movement within the plough unit itself. Examples of these were the Huwood Slicer, Samson Stripper and the Lothians plough, all designed for thicker seams. In thinner seams, the Potts hydraulic plough and the auto-percussive plough were tried, but with limited success.

21. In the USA, immediately after the second world war, coal cutting was greatly improved by the introduction of a high-speed rubber-tyred machine; this improved mobility still further, which is so essential in room and pillar mining. At this time, much development took place in augering in the highwall of an openpit. Single holes reached lengths of 200 ft and the equipment proved the principle that an auger could both cut and load coal. More sophisticated machines incorporated two or more cutting heads and one model was capable of cutting drivages up to 900-ft long (18).

22. In the 1950s, a series of machines was introduced to mechanise the combined functions of cutting, drilling, blasting and loading. Thin seam versions appeared in the form of the COLMOL, the Wilcox miner and the Jeffrey 100L auger miner. The latter two machines provided the bulk of thin seam coal production until the 1970s.

MECHANISATION - 1960 TO PRESENT

23. Although many adaptations of thin seam ploughs produced coal in the UK after 1960, it was in the 1960s that real progress was made with machines that incorporated a positive cutting action. These were developed from the Anderson shearer. At the start of this period there were some further trials of multi-jib coal cutters that cut out the complete thin seam and were now mounted on a reliable armoured face conveyor. There were few successful applications, however, due principally to the lack of horsepower for high cutting speeds and the inability to load the coal on to the conveyor.

24. Soon after the introduction of the Anderson shearer there was the development of the Anderson-Boyes floor-mounted trepanner, which was designed specifically for thinner seams of between 3 ft and 3 ft 6 in.

25. Variations of the cutting principles adapted, in both the shearer and trepanner, were incorporated in equipment for even thinner seams. Examples were the Dranyam vertical drum shearer, and the trepan shearer, which had a vertical drum at one end and a trepanner or auger head at the other.

26. Alongside these power loader developments, significant improvements were taking place in self-advancing powered supports and in armoured face conveyors, and these laid the foundations for today's face performances of between 1,500 tonnes and 2,000 tonnes per day from a 34-in seam.

27. One major experiment was undertaken in the 1960s in thin seam mining that did not employ the longwall principle. This was the Collins miner which utilised the auger or mole principle. It was an ambitious project from which many lessons were learnt, and which still may point the way for future developments in thin seams of around 24 in.

28. Many of the developments that took place between 1945 and 1960 in Western Europe spread into Eastern Europe. From a study of the literature, the gestation period for Western European thin seam techniques to be adopted and recorded in Eastern European publications is about 12 years. The time lag between West and East is further borne out by the fact that the Meco-Moore, which ceased production in the UK soon after 1960, is still in use in Eastern Europe as the Kombine cutter loader.

29. In level thin seams in the USSR, current production comes from ploughs, scraper boxes and vertical drum shearers. In the highly-inclined thin seams, the USSR has devised some unique equipment; hydraulic mining, augering and chain saws are examples. Their latest development appears to bear strong similarities to the Collins miner. However, recent visitors to the USSR have reported that there is no new thin seam equipment in widespread use and, moreover, there is a slowing down in the R & D programme for thin seam equipment.

30. In the USA, the Wilcox miner, which was introduced in the 1950s, is still one of the most versatile machines for seams down to 28 in, and outputs of 300 tons per machine shift are still possible.

31. The ubiquitous continuous miner has now been miniaturised to operate in 28-in seams, provided the roof and floor do not converge. These machines have chassis heights of 24 in which are similar to low seam scoop cars and other thin seam equipment, such as personnel carriers.

32. One of the most recent developments in thin seam room and pillar mining is continuous haulage. This system of relatively short chain conveyors provides articulation at a number of points to facilitate room formation and pillar extraction.

33. Another significant technological advance is the remote control of a continuous miner. This can be accomplished by either radio or direct wire and enables the operators to remain in supported and dust-free positions.

34. The thin seam techniques in use today in both the USA and Europe are essentially scaled-down versions of thicker seam equipment. It would appear that the current limit for viable bulk tonnages using this equipment is 28 in in the USA and 32 in in the UK. Higher production and productivity from these seam thicknesses, together with the exploitation of even thinner seams, would appear, at this stage, to require the development of techniques that would represent a new breakthrough in technology.

CHAPTER V

STATE-OF-THE-ART - REASONS FOR MINING

GENERAL

1. Thick or medium seams are frequently more attractive economically to mine than thin seams. However, there are a number of reasons for mining thin seams:-

(i) Political reasons can apply in a country with thin seams of premium coal. If these seams have been mined traditionally and a mining industry exists, it may be politically expedient to continue production despite the availability of cheaper foreign coal of even better quality.

(ii) Where a country has no foreign currency available to import coal, economic reasons can force it to mine indigenous reserves in thin seams, especially if these reserves form a major proportion of the country's total reserves.

(iii) An industrial circumstance may exist in a country or region where there are no alternative sources of coal other than local thin seams. This is most likely in the case of premium fuels for use as metallurgical coals in steel making or in specialist petro-chemicals.

(iv) A fourth category could be termed technical and apply to the situation where the natural mining conditions in the thinner seams are good and where, with suitable technology, they can be mined to give high productivity. A unique example in this category is the mining of very thin seams (less than 18 in) in the USSR in order to prevent outbursts in adjacent thicker seams.

2. However, rarely are thin seams mined for any one reason alone, usually the situation is more complex.

3. The largest tonnages from thin seams are produced in Eastern Europe and here the production trend is to maintain or even increase these levels. In Western Europe there has been such a decline in thin seam activity that some countries that produced significant tonnages in the mid 1960s, now produce virtually none.

EUROPE TO 1914

4. To understand fully the reasons for mining thin seams, it is necessary to examine first the situation in Europe at the time when thin seam activity was at its peak.

5. Immediately prior to the first world war (1914 to 1918), the total UK production reached a peak of 281 million tpa, a substantial quantity being won from thin seams, especially in Durham. At this time, new mines were being sunk and existing mines were being developed deeper to satisfy the increasing demands for coking coals, most of which lay in thin seams. Coal-cutting machines, pneumatic picks and conveyors began to ease the problems of mining these seams. Other Western European countries had similarly reached peak output levels at this time.

6. Where multi-seam working was employed, the trend was towards the longwall system and this subsequently became the standard system of working throughout Europe.

7. Eastern European countries lagged behind in this surge of coal mining activity. The USSR attempted to industrialise by importing capital and technical aid to stimulate the Ukraine coal and steel industry. They turned to Belgium for assistance and by 1914 considerable progress had been made and large Belgian interests were invested in the Ukraine. Belgium was particularly qualified as a country as they were mining thin seams effectively at this time.

EUROPE 1914 TO 1939

8. The first world war completely changed the European situation. In Western Europe there was over production of coal and steel, and reparations in the form of coal and steel from Germany made the situation worse. As a result, the UK output fell to nearly 50% of its pre-1914 level. German industry, without its traditional empire markets, suffered severe cutbacks, whilst France and Belgium were trying to rebuild war-damaged mines and equipment.

9. No new capital was available and every mine was constrained to mine only the coal seams already developed, which included thin seams. Industrial recovery was very slow in all countries in Europe until 1936 when preparation for war stimulated heavy industry and the coal mines were operated to full capacity again.

10. In Eastern Europe, industrial development between the wars was handicapped by revolution and political change in the USSR, and the break-up of the Austro-Hungarian empire. This created new states and many political and economic problems, but the area that did make industrial progress during this period was the province of Silesia in Poland. Both the USSR and Germany had developed this region in the recent past and it was well equipped technically to develop into a strong industrial region.

EUROPE - AFTER 1940

11. The second world war again created a tremendous stimulus to technical development; manpower was scarce and coal was urgently required. Germany overran Silesia, Ukraine, Belgium and France, controlling their large mining industries. At the end of hostilities, mines suffered from lack of development, safety standards had become lax and bad practices had been allowed to increase. Once again, reparations in coal were demanded and collected by Eastern European states.

12. Political changes in Eastern Europe after the war isolated them from technical developments in the West. In Western Europe all the mining industries, including the UK, recovered relatively quickly, and by 1965 had started to over-produce. At this time a greater percentage of thin seams was being mined than at any other period. From 1955 to 1965, the greatest technological developments in the mechanisation and automation of thin seams took place. Cheap oil and natural gas had a disastrous effect on the coal industries of Western Europe and, in spite of two oil shortage crises since, the industry has continued to decline. National subsidies, in various forms, have been necessary to maintain or control the run down of the coal industries in Germany, France, Holland, Belgium and, to a lesser degree, in the UK. As a result of this economic pressure, thin seam production was greatly curtailed, and today relatively small quantities of coal are mined from thin seams.

13. Within the last five years, determined efforts have been made by the UK and Germany to arrest the contraction of their coal industries. The UK plans to increase capacity but this will not take place in thin seams but in thicker deposits suitable for power generation.

14. Eastern Europe, in the meantime, had not reached their targets, nor were they affected by cheap oil and gas, and they have continued to exploit thin seams. This is currently being done in the flatter seams with equipment designed and proved in Western Europe. In steep and very thin seams, the USSR is currently attempting to develop suitable mechanised equipment without any real overall success to date.

15. The effect of recent discoveries of vast deposits of coal in Eastern USSR, which can be surface-mined, now poses a serious problem for the thin seam mining areas. Thin seams of coking coal and anthracite will continue to be mined from the Ukraine, not only for industrial reasons but for social and political ones. It is possible, however, that there could be a gradual decline in thin seam output.

16. The coal industry of Western Europe is now under pressure, not only from oil, gas and nuclear power, but from cheap imported coal from Australia, Poland, South Africa and the USA. It will need very considerable political and financial effort to bring about an up-turn in the production from the industry. In this situation, thin seam mining will still tend to contract in Western Europe in the foreseeable future.

USA - 1900 TO DATE

17. The large, good quality, shallow deposits in the Eastern States permitted the room and pillar system of mining to continue as the predominant system.

18. The parallel situations for Europe have not applied to the USA, though, since the turn of the century, two world wars have first stimulated the mining industry, then caused subsequent contractions. Indigenous and imported oil and gas supplies have smoothed out the peaks and troughs in coal production.

19. The US coal industry is more flexible than that of Western Europe. Its mines can be opened and closed relatively quickly and new mines can be brought into production in three to five years compared with seven to ten years in the UK.

20. The discovery and exploitation of the vast coalfields of the Western USA poses problems for the deep mines of the Eastern States which operate the poorer quality, thin seams. However, new mining regulations and environmental controls may accelerate the pace of technical development and innovation in the good quality, thin seam, mining areas of the USA. Experience from Europe would indicate that a lead time of from five to ten years is needed to develop new equipment that is tested, reliable and acceptable. Even with a flexible industry, the introduction of new mining systems and new machinery will take a few years to become established. It is essential to anticipate, well in advance, the need to mine appreciable quantities of coal from thin seams, in order to have the time to develop suitable systems and machines.

CHAPTER VI

STATE-OF-THE-ART - DEPOSITS

GENERAL

1. In most of the major coal-producing countries of the world there is a wide range of coal seam thicknesses which may vary from a few inches to, in some cases, several hundred feet.

2. The mining conditions in any particular locality are influenced by a combination of regional and local geological factors which, at all depths, are quite independent of seam thickness (26).

3. Table V summarises significant geological parameters of thin seam coal deposits on a world scale. The data presented are intended for comparison on an international basis and take no account of local variations in geology.

SEAM THICKNESS

4. A seam that lies within the category of "thin" can often vary in thickness across the area of working of a mine or a region of the country. The most satisfactory situation for the mining engineer is where the thickness is consistent, as this simplifies the design, selection and operation of the mining equipment.

5. The variation in the definition of a thin seam, from country to country, is quite marked. Belgium uses the very low thickness of 0.6 m (24 in), whilst in Bulgaria the figure is more than double at 1.3 m (51 in).

6. In Bulgaria, the variation in thickness in a seam can take place within very short distances. Probably the largest active thin seam coalfield, the Donetz coalfield in the Ukraine, also suffers abrupt variations in thickness due to washouts.

SEAM DIP

7. The world's deposits of thin coal have dips varying from flat through to vertical. In the USA, the flatter gradients predominate and this has allowed mine development to take place within the seam.

8. In the UK, the average dip is around 6^0, although in localised areas, dips of up to 45^0 are found.

9. In Germany, dips vary; 63% of the reserves have a dip between 0^0 to 10^0 and 27.5% of the reserves dip at greater than 20^0. The steeper gradients in the Rhur led to horizon mining layouts, which spread to the UK in the early 1950s.

10. The dip of France's reserves are even steeper, with nearly 50% in the 0^0 to 20^0 range and the other 50% between 20^0 and 45^0.

11. In the USSR, a wide range of inclinations is found from 0^0 to 70^0. In the steep inclinations, a higher proportion (63.3%) of the total number of operations are in the range up to 0.7 m (28 in) than in the gently sloping or moderately inclined seams. In terms of volume production, 26% of the Donetz output comes from seams having an inclination of 0^0 to 8^0, 34.8% from 8^0 to 18^0 and 19.2% greater than 45^0.

12. Other steep thin seams are found in Spain and Korea.

SEAM DEPTHS

13. Thin seams are mined at all depths up to 1,200 m (3,937 ft), which is not an uncommon depth in the USSR, Belgium and the UK.

14. In the USSR, the calculation of resources includes coal at depths down to 1,800 m (5,906 ft) (23).

15. Underground mining in the USA is limited in depth owing to the relatively abundant reserves of coal that are available at shallow depths. The normal limit for coal resource calculations in the USA is 1,000 ft, though there are probably substantial resources below this level. Occasionally, the depth of cover can increase abruptly if the mine workings enter the side of a mountain and reach a position immediately below high topographical terrain.

16. Owing to the exhaustion of reserves in the USSR, the depth of working is reported to be increasing at the rate of 12 m (39 ft) per year.

17. The average working depths of coal measures in Europe and the USA are as follows:-

	m	ft
UK	340	1,115
Germany	730	2,395
Poland	320	1,050
Donetz field of the USSR	530	1,739 (23, 26)

COAL STRENGTH

18. The strength of coal seams is of considerable importance as it determines the techniques and equipment that can be used in the mine. The strength or resistance to mechanical breakage can be increased or decreased by the effect of pressure from the overlying strata. This is evident when a "new" longwall starts off and the coal is hard and difficult to shear. Later, as abutment pressures develop ahead of the face, it becomes much softer.

19. In Eastern Europe, the breaking strength of the coal and rock is measured on the "Protodyakonov's" scale, which is a combination of the uniaxial compressive strength and the score hardness.

20. The coals in Western Europe are generally soft and suitable for ploughing, except in the Saar field of Germany. In the UK, the coal is considerably stronger and there has been only limited success with coal ploughs. The average compressive strength of UK coals is about 3,500 lb/in^2 but some are as hard as 5,500 lb/in^2.

21. In the USA, the coal strength is, on average, harder than that of continental Europe but possibly not as hard as the UK coals. Some sources consider that about one third of all US coals might be suitable for ploughing.

ROOF CONDITIONS

22. Roof conditions are variously referred to as "soft" or "hard", "strong" or "weak" and "friable", but these are relative terms and subjective to the observer's experience. A more accurate description is the use of mining geological terms, such as sandstone, sandy shale, mudstone, etc. Table V was compiled from the ECE report of 1967 (26) and the references to the roof and floor are usually a combination of subjective and objective observation, such as hard sandstone, etc.

23. A more quantitative description of roof conditions can be obtained from the percentages of sandstone and limestone in the immediate roof beds, the thickness or frequency of laminations in the case of shale tops and the compressive strengths as measured from laboratory samples.

24. In general, the softer shales form a weak roof and the sandstones and limestones a more secure roof. In weak conditions, a common practice, on longwalls, is to leave a thin layer, of from 2 in to 6 in, of coal as a protection.

25. Another means of comparison between the US and other mining conditions is the type of support necessary for the roof of a roadway. In the UK, Western Europe and the USSR steel arches are the most common means of support, whilst in the USA the less positive system of roof bolting gives adequate support. This comparison is not truly representative as room and pillar mining in the USA does not, by design, induce heavy strata pressures on the roadway, as is the case with longwall. This is borne out by the frequent need for extra support in the longwall roadways in the USA.

FLOOR

26. The strength of the floor of a seam is an important consideration in choosing the type of mining system and the equipment to be used. Strong hard floors tend to keep the power loader or coal cutter in the seam. Hard floors make excellent roadways for rubber-tyred equipment in room and pillar working, whereas soft floors, especially in wet conditions, break up, making travelling and tramming

extremely difficult. Soft floors in thin coal have the advantage that they can be cut out either with the seam or in a separate operation to gain more space for men and equipment (11).

27. The strength of the floors is largely dependent on the petrographic composition of the floor material. The sandstones and sandy shales make strong floors, whilst clay floors tend to be weak. Where mining is being carried out at depth, the strata pressure can induce flow in a clay floor which then further weakens causing it to swell and produce floor lift.

28. In the USSR, floors are described as being mostly clay and shales, whilst in Poland, the floors are conglomerates, sandstones and siltstones. In Germany, the floors are shales and sandy shales and in France, where thin seams have been mined, large amounts of floor were cut out to provide vertical clearance, indicating that the floors were relatively soft.

29. In the UK, soft floors predominate and, at depth, cause considerable problems due to floor lift in the roadways. They are composed mainly of clays which break up when wet. A combination of soft floors and strong coal in the UK has led to considerable difficulties with ploughs, making horizon control difficult.

30. In the USA, the floors are of medium hardness which is evidenced by the general success of mobile machinery in room and pillar mining.

OVERBURDEN

31. The strata between the coal seams and the surface has little effect on the actual mining conditions as these are governed more by the immediate roof and the beds which extend about 20 ft to 30 ft above the seam.

32. However, the overburden strata does have a considerable influence on the capital cost of gaining access to the coal seams, especially when it contains water-bearing unconsolidated beds. Water problems in the UK and Germany are often associated with water-bearing sandstones or gravels which make shaft sinking expensive. Cement injection and freezing are widely used in such conditions. In order to justify the heavy cost of vertical shafts, and cross measure drifts, the mines in Western Europe have tended to concentrate output in relatively few high-volume producers. This contrasts with the USA where there are many small mines.

EXTENT OF DEPOSITS

33. Thin seams cover extensive areas in Europe and the UK. A feature of European seams is the general uniformity of thickness of seams throughout a coalfield. This contrasts with the USA where the same seams show regional variations in thickness (Keystone Reports).

34. The Western European coalfields are small compared with the giant USSR coal basins of Donetz and Lvov-Volynsky.

WATER

35. Water occurs in underground workings in varying quantities in Eastern and Western Europe. As the workings become deeper there is a tendency for less water.

36. In the UK, although most mines are relatively dry with only nuisance water at the working faces, some mines are very wet and pump many times the tonnage of coal produced. This also applies to the German mines, and in the USSR some mines are reported as being dry whilst others report large volumes of water being pumped out of the workings (26).

37. In the USA, many of the thin seam mines are above the level of the drainage table and are hence relatively free of water. Water, other than nuisance quantities, does not generally present problems in the USA, though in some mines water enters the workings via fault planes and gives rise to extensive pumping (60).

FAULTS

38. There is no universal measure of geological fault intensity with regard to mining operations, and hence the terms used in Table V are relative or subjective.

39. Faults can vary in displacement from fractions of an inch to hundreds of feet. Mining areas are often divided into blocks, which must be mined virtually independently of each other. Smaller faults can cause severe mining problems, as in the case of a fault which has a displacement greater than the seam thickness.

40. The disruptive effect of a fault varies with the system of mining. Room and pillar is regarded as a more flexible mining method than longwall as areas of weak strata in the vicinity of a fault can be disregarded without undue dislocation to production or to the plan of the mine. In addition, irregular areas of coal cut off by faulting can be extracted easily. With the longwall system, faulting can cause serious problems on the face, which must maintain its straight line independently of geological disturbances. In some UK coalfields, faults are "carried" on longwalls as a matter of course and, in such cases, the faults usually run at right angles to the line of face. If a fault is encountered lying parallel to the wall it may stop the advance of the wall and cause it to be redeveloped.

41. Since the longwall system of mining is extensively practised in Western Europe, the UK and the USSR, it can be deduced that reasonably-sized areas of coal are available which allow walls of up to 900 ft in length to be advanced or retreated distances of more than 3,000 ft. The 1967 ECE report (26) records that only Czechoslovakia, Bulgaria and Romania have highly-faulted coalfields; however, the UK coalfields of Scotland and South Wales are also severely affected.

42. In Spain, the thin seams occur in the mountainous regions in the north where the seams are steep and also badly faulted.

43. In South America, where the coalfields are small, working tends to be difficult due to tectonic disturbances.

44. From work carried out in China by British Mining there would appear to be few faults.

45. In the USA, in general, the frequency of faulting is low and few of the mines have problems associated with this form of geological discontinuity.

CLEAT

46. Regular vertical laminations in the coal seam itself are referred to as the cleat of the coal. Normally the cleat is more heavily pronounced in one direction, with a secondary series of vertical planes at right angles to the major cleat line.

47. The presence of a well-defined cleat is sometimes an advantage as it permits easy cutting or winning of the coal, due to the cleat lines forming planes of weakness within the seam. The presence of a pronounced cleat is particularly important with plough-type equipment, as the wedge action of the plough can take advantage of the planes of weakness.

48. In longwall mining, a cleat line parallel to the face, known as a "bord" face, can present problems of slabbing in medium to thick seams. Even in thin seams, a "bord" face with an inherently weak roof is more difficult to support. For this reason, many seams in the UK are mined on end only; that is when the cleat line is at right angles to the wall.

49. In most European countries the coal seams exhibit a cleat line; this tends to be less well defined in the USA.

SPONTANEOUS COMBUSTION

50. Spontaneous combustion is caused by heat generated from the oxidation of coal. Although much research work has been carried out on spontaneous combustion, the full mechanism of the process is still not fully understood (60). It has been established, however, that spontaneous combustion is related to the chemical composition of the coal and particularly its oxygen content.

51. Spontaneous combustion occurs when sufficient coal material is available and the air supply is such that it provides sufficient oxygen for the combustion reaction but is insufficient to carry away the heat produced.

52. In the UK it is more prevalent in thick seams; the thick coal beds of Warwickshire are notorious for spontaneous combustion problems.

53. In thin seams, the lower incidence of spontaneous combustion is due to less coal remaining in the goaf and better consolidation of pack walls alongside the roadways. In addition, whilst the bulk output of thin coal operations tends to be

lower than those of thicker seams, the rate of travel of the walls is usually greater; this provides better control of the incubation period and, consequently, reduces the chance of spontaneous combustion.

54. The German coalfields have had similar problems with spontaneous combustion and case histories recently studied indicate a prevalence towards thicker seams (60).

55. Spontaneous combustion is known to be a problem in the USSR mines in some areas, but details of occurrences, in relation to seam thickness, have not been found in the USSR literature.

56. Spontaneous combustion occurs in the USA and records show that many of the thick seams in the mid-western states of Colorado, Wyoming, Utah and New Mexico are susceptible.

57. The room and pillar system of mining, with back areas well ventilated, tends to have a low risk of spontaneous combustion. This risk is increased where depillaring takes place and back ventilation is less controlled. In the USA, an additional risk exists where the longwall system is practised in conjunction with bleeder roads. This allows air to migrate through the goaf, an area of the mine that should be effectively sealed off in order to minimise risk.

METHANE

58. In the world-wide survey it was found that most coalfields exhibit methane problems to some degree. Emission rates reach more than 1,000 ft^3/ton in some instances but a more normal rate is 500 ft^3/ton. Unlike the UK, some of the mines in Eastern Europe are classified as gas free. In the Donetz coalfield in the USSR, 28.6% of all mines are in this category. One third of the Donetz mines have gas makes of more than 530 ft^3/ton.

59. The structural geology of an area has a major bearing on the methane emission as impermeable strata layers above the seam can act as seals to keep the methane in the surrounding strata.

60. Most of the European thin coals are mined at depths of between 300 m and 800 m (984 ft and 2,625 ft), and substantial methane emissions are common. However, Germany may be at variance in this respect as the ECE report states that thin seams tend to be less gassy, probably due to their higher rank than thicker seams.

61. In the USA, with many shallow mines, the incidence of methane is relatively rare. In room and pillar mining, the main source of methane gas is the coal itself as little gas comes from the surrounding, undisturbed strata (141). However, when pillar extraction or longwall mining is practised, this induces strata movements which open up passages for methane to enter the mine atmosphere.

62. The methane content of some of the US shallow coals has been evaluated (142) giving, for the Pittsburgh seam, a gas content from 64 ft^3/ton to 224 ft^3/ton at depths from 312 ft to 850 ft, the methane content increasing with depth.

63. In recent discussions with the Methane Control Section of the US Bureau of Mines, Mr. M. Deul intimated a considerable amount of new research work had been carried out on the subject of methane in US coal mines and that this would be published in the near future.

CONCLUSIONS

64. The occurrence of thin coal is widely distributed, but in countries with abundant reserves of thick coal, the thin seams are less well documented.

65. There is a great variety in the characteristics of thin seams and in their structural geology, but frequently these parameters are similar to thicker coals in the same coalfield.

66. Where the natural mining conditions of a coalfield are difficult, these difficulties can be accentuated in thin seams as in the case of geological faults, floor lift, etc.

CHAPTER VII

STATE-OF-THE-ART - SEQUENCE OF MINING OPERATIONS

GENERAL

1. The sequence of mining operations begins with the acquisition of coal leases and the legal rights to mine. Although, in practice, the sequence does not rigidly follow a fixed pattern, the next stage is normally the exploration of the lease area, in order to build up a picture of the structural geology and disposition of the coal beds in the area.

2. After all the relevant geological information has been obtained and charted, it is necessary to determine the position and type of access to the target seam or seams. The access, either drift, adit or shaft, is decided in the light of the mining methods planned and the surface infrastructure, such as roads, rivers, railroads, townships and the general topography of the area.

3. The complete mine design, both surface and underground, is an inter-related process in which each facet of mining is considered. Some of the design parameters are said to be "fixed" and these include details of the available coal seams and their geological structure, together with the surface features. The term "variable" parameter is used to describe those items where a degree of freedom exists in the planning, such as the choice of room and pillar, longwall or another system, also the widths of working panels, designed thickness of extraction, method and direction of working, size of pillars and the general layout of the mine (118).

STRUCTURAL GEOLOGY

4. The structural geology has possibly the greatest impact on the overall mine design. The gradient and depth of the coal measures determine the length of the access slopes. These can be driven either as adits in thin seams or as cross measure drifts. In the case of deeper deposits, vertical shafts may be required. Hardness and permeability of the overlying strata, and the distance that has to be driven, largely determine the capital cost of developing a mine.

5. In order to achieve an acceptable capital charge on each ton of production, the deeper mines must, of necessity, be high bulk producers. This means that the mining of the deeper seams is often only possible by large companies with adequate financial resources.

6. Another factor, which affects the size of a mining company required to develop a mine, is seam thickness. In the USA, there is a correlation between the thickness of seam worked and the size of operating company; small companies tend to mine shallow, thin seams (143, 145). It is unlikely that thin coal at greater depths will be mined without the concurrent extraction of adjacent thicker seams.

Inclination of Coal Seams

7. The inclination or dip of the coal seams has a major influence on the mining method and the mine layout. In room and pillar mining, where the seams are flat or moderately inclined at about 6^0 or less, cat-mounted or rubber-tyred mobile equipment can be used in the working panels. Where the inclination is steeper, mobile equipment can still be used, but there is a considerable fall off in efficiency due to loss of traction. This is exemplified in the case of continuous miners when advancing upgrade. Steep inclinations adversely affect the tramming operation but some four-wheel-drive equipment is claimed to operate in inclinations up to 24^0 (97).

8. In longwall mining, the effect of gradient is less marked. Modern mechanised equipment can operate in inclinations of over 45^0.

9. In moderately-steep and steep seams, the role of the armoured face conveyor changes from that of conveying the coal produced, to that of restraining the gravity-induced flow of the coal. In steep conditions, the mechanical means of face conveying can be removed and replaced by free-falling gravity, usually in a system that can be manlessly operated. These conditions permit relatively simple equipment to be used.

Faulting

10. Geological faults can have a disruptive effect on all mining methods. Where faults of large throws cross a mining area or lease, they divide the area into natural mining blocks, the shape and size of which control the distance that panels can advance. The effect of this can be significant since a mine layout is normally arranged to have as few panels as possible, in order to reduce down-time when moving from panel to panel. If the blocks are irregular, some of the panels will have off-standard widths. Considerable advantages lie with the room and pillar system in that additional roads can be driven or discontinued, so as to mine out irregularly-shaped areas. This is facilitated by the inherent mobility of the mining equipment, and the normal arrangement of the conveyor road in the centre of the panel.

11. Provided major fault planes are accurately marked at the mine planning stage, then longwalls can usually be laid out satisfactorily. Uncharted faults can require changes in the face length and this becomes a major operation, as additional equipment, in the form of conveyor sections and supports, has to be added or removed. The main gate or conveyor road is normally at the lower end of the longwall face and, if the unexpected fault requires that this road be changed in direction, then additional conveyors may be necessary, making the alterations to the whole panel very expensive.

MINE LAYOUT

General

12. There are two distinctive types of mine layout - horizon, where the main roads are driven level, and not necessarily in the seam, and in-seam, where the main service roads are driven in the coal seam.

Horizon Mining

13. Horizon mining is used in a variety of forms where the ore body or coal seams are steeply inclined. It was developed to its present form in Germany. The equipment used for the haulage of coal and transport of men and materials can be chosen to operate on flat gradients independently of the inclination of the seam. However, the driving of these roads in stone is expensive and there is no contribution, in terms of coal produced, to help offset the cost of drivage.

14. Horizon mining is particularly effective where a number of inclined seams lie close together. The drivage of horizontal roads will intersect them and provide a common service to each seam.

15. One of the original advantages of horizon mining was the ability to use locomotive haulage, but this form of transport, for coal clearance, has largely been superseded by high capacity belt conveyors, which now operate at gradients up to 15^0.

In-Seam Mining

16. This system is commonly applied in the USA and in the UK. The main transport and access roads are driven in coal or, where this does not give sufficient height, in coal and stone. Seam gradients must be sufficiently flat to enable transport vehicles and belt conveyors to operate safely and efficiently. The major advantages of in-seam mining are the production of coal during development operations, and direct underground exploration during the development stage.

17. Sometimes, when there is a definite dip in the coal measures, in-seam roadways are still driven, but on a cross gradient or apparent dip. The direction of such roads can often determine the overall mine layout. Alternatively, for reasons of drainage and haulage, main development roadways can be driven on the true dip and longwalls laid off to either side.

18. To minimise the cost of in-seam development in longwall mining, the roadways of a coal-producing panel can be utilised for laying out walls on the flanks. If the mine is wet, it is more usual to develop to the dip with roadways. This presupposes that the reserves lie to the dip. Ideally, in wet mines, development should take place to the rise.

19. The concept of in-seam development in thin coal led to the idea of the dirt-absorbing heading (15). This essentially consists of a short advancing longwall face, which is sufficiently long to enable one roadway to be enlarged by

mechanically stowing the debris behind the coal face. It has the advantage of producing coal to offset the cost of the roadway drivage but, to date, has not proved capable of fast enough travel for effective mine development.

Financial Considerations

20. A mine is the site of a business operation whose object is to provide a return on invested money. This holds true even in the USSR, although the objectives there are to provide the national economy with coal at the lowest overall cost, commensurate with the allocation of financial resources.

21. In order to obtain the optimum financial return on investment, it is necessary that the development costs of a mine and the working costs of production, when discounted financially, should be minimal. The time taken to build up to full production, and hence the generation of revenues from the sale of coal, should also be minimal. Initial expenditure on plant, access slopes, shafts and the like, can be reduced by specifying equipment of a lower standard and adopting small access roads. The saving in cost on these items must be balanced against the higher working costs caused by equipment replacements and extra future entries that may be required.

22. In thin seams, the height of the main access roadways is an example. If the roadways are driven at seam height from the outcrop, then initial expenditure will be minimal. If, however, the entries have to serve for a long period of time and carry high volumes of air and traffic, then a good financial case may be made for providing one or more higher entries. If the mine has limited reserves and therefore a short planned life, the extra expenditure is unlikely to be cost effective.

23. Where technical constraints do not rule out any particular mining method, the choice is a business decision. Parameters that must be considered are: cost of money, cost of equipment, bulk output, continuity of production, manpower, productivity, maintenance cost, utilisation of reserves and useful life of the mine (72).

ROOM AND PILLAR

General

24. The choice of the mining method, on technical grounds, is mainly based on the structural geology of the coal reserves. The size of pillars required for room and pillar mining increases with depth. Considerable research work has been done on pillar design, particularly in South Africa, following a major disaster involving the collapse of a large pillared area. The increases required in pillar size for greater depths, reduce the percentage extraction or utilisation of reserves (145). Although this is not a critical problem in most US mines, it is being solved, in a few, by the adoption of longwall techniques.

25. In all US room and pillar mines, the layout and method of operation must conform to the mining regulations. These demand minimum ventilation quantities and specify separate roads for intake, return, belt conveyors and trolley-wire locomotives. This means, in practice, that a working panel must contain at least three roads. The regulations do not specify a minimum height for roads but allow roads to be taken at the seam thickness.

Thin Seam Room and Pillar

26. A considerable amount of labour and cost is expended on the construction of stoppings and the provision of services, which must be extended as the workings advance. If this cost is evaluated as a unit cost per ton of production, then it will be reduced by increasing the number of roadways in the section.

27. However, an increase in the number of roadways increases the average tramming distance from the face to the ratio feeder, and hence slightly lowers the effective carrying capacity of the face haulage equipment. In conventional cut, blast and load operations, advantages are obtained from extra working faces, as these allow a buffer of prepared coal to be available in the section. This forms an insurance against coal cutter breakdowns and physically separates blasting operations from other operations in the area.

28. A limitation on the number of roads is provided by the maximum length of shuttle car cable or, in the case of continuous haulage section, the range provided by the haulage system.

Extraction and Stability

29. The stability of room and pillar workings is a function of the immediate roof but, more importantly, of the size and strength of the coal pillars. Wider roads, for a given percentage extraction, mean that the size of the pillars is increased. Hence, it is in the interest of both productivity and the utilisation of reserves to have the widest possible rooms consistent with good control of the immediate roof.

30. The strength of coal pillars is also a function of the pillar height (145). Shorter pillars are stronger for the same plan area. This means that in thin seams the pillars can be made smaller for a given safe width of room. This increases the percentage of extraction and hence the bulk tonnage of coal that can be extracted per foot advance across the width of the panel.

Shape of Pillars

31. Where shuttle cars or scoops are used, it is normal to lay out the pillars on a rectangular plan and to make allowances for the pillar corners to be trimmed, in order to facilitate the turning of vehicles. Where continuous haulages are employed, with narrow entries, it is sometimes necessary to modify the plan and carry the crosscuts on a 60^0 line in order that the conveyor section can be driven

into the entries. This change in pillar shape slightly reduces their strength and hence the percentage extraction, based on the parameters of depth, seam height and room and pillar dimensions.

Example of Thin Seam Room and Pillar

32. A typical example of US operations in a thin seam is contained in the article "How a Small Operator Gets Big Productivity", Coal Mining and Processing, July 1978 (147).

33. The system of mining is room and pillar using a thin seam continuous miner and continuous haulage in a seam less than 40 in thick. The mining plan involves driving entries on 60-ft centres with crosscuts on 60-ft centres, turned off at 90^0. Owing to the use of continuous haulage equipment, only three entries are driven, but extraction is increased by driving rooms into the panel barriers.

34. Variable roof conditions can reduce the width of the rooms from the standard of 30 ft down to 20 ft. Originally, a Wilcox auger miner was used but disadvantages arose from the need for wide rooms for high efficiency. In order to maintain productivity, a Jeffrey 101, which has a 10-ft-wide head, is now used for main production, but the Wilcox is retained for coal less than 30 in thick.

35. Support is by a combination of roof bolts and timber. Long cutting runs of up to 28 ft are possible, due to the use of remote control, which allows the operator to be positioned under supported roof.

36. Haulage at the face is by "continuous" chain which is operated by one man. Outbye haulage to the surface is by two belt conveyors. Frequent belt move-ups are reduced by using an overlapping 300-ft chain conveyor at the end of the continuous haulage and this enables belt extensions to be limited to one per month.

37. The system is reported to have a productivity of 60 tons/manshift and, on occasions, has reached 100 tons/manshift.

LONGWALL

General

38. Longwall extraction is achieved by advancing or retreating a single face varying in length from 300 ft to 900 ft. In mechanised longwall, coal is hauled from the point on the face where mining is taking place, along the length of the face to one end. Here it is transferred to another chain conveyor, called the stage loader, which then loads the coal on to a conveyor belt. There are two main types of layout, advancing and retreating.

Advancing Longwall

39. In the advancing system the access roadways, termed the gates, are formed at each end of the mined-out area. There are normally only two entries per face, one at each end, and the area between the roads is usually allowed to cave behind the longwall face.

40. The roof immediately adjacent to the face is supported by powered supports, commonly of the chock or shield type. These are lowered off, advanced and reset in order to follow the advance of the face. After the supports have been advanced, the roof is then allowed to cave.

41. The gates are formed by either cutting or blasting down roof, termed ripping, or taking up floor, termed dinting. The rock debris is then packed in the mined-out area alongside the roadways. This stabilises the roads and forms a ventilation seal across the face to prevent air short circuiting through the waste.

42. A major advantage of advancing longwall is that as soon as a face line has been won out and the equipment, in the form of conveyors, supports, cutting machines and ancillaries, has been installed, production can commence. A major disadvantage of the system is the interaction of activities at the face ends between face operations and the formation of the gates. The rate of roadway formation should be equal to or greater than the rate of advance of the face. A modification to the system provides for the gates to be pre-driven a short distance in front of the face, normally about 60 ft, and this helps to minimise face end interaction problems.

43. In recent years, face activities have been simplified by the development of cutting machines that can overcut the armoured face conveyor drives at each end of the face and this has eliminated the need to mine special stable holes at each end of the face. Where ranging arm shearers are employed, the shearer can be used to profile out the roof of the access road, which eliminates much of the gate formation activities, but has the disadvantage that rock debris is conveyed out of the mine with the coal.

Retreat Faces

44. Retreat faces are operated in a similar manner to advancing faces but differ in that the access roadways are pre-formed. The access roads are driven to a predetermined boundary and a face line established between the two roads. Face equipment is then installed and the wall retreated between the pre-developed roadways, so extracting the pillar of coal.

45. Retreat mining has the principal advantage that there is no interaction between roadway formation and coal-producing activities. A further advantage, in European practice, is that the waste behind the face is not ventilated and this cuts off any oxygen supply to coal left in the waste, so reducing the risk of spontaneous combustion. This practice is not allowed in the USA at present, due to mining regulations which demand that the waste area be ventilated by bleeders, but dispensation has been granted in some instances.

46. A further advantage of retreat mining is that the ground to be mined is fully explored in the pre-development stage; this minimises disruption of production, due to undetected geological features, such as faults and washouts.

47. The major disadvantage of retreat mining is the greater development time and initial costs required before the start of production.

Thin Seam Longwall

48. In thin seams, the choice between advancing or retreating longwall is even more significant. This is due to the higher cost of the development roadways in a retreat layout caused by mining more rock above the thin seam.

49. In order to reduce the high cost of roadway formation, many layouts have been evolved to enable the gate roads to be used twice. Basically, they consist of a combination of advance and retreat walls laid out adjacent to one another. This virtually halves the roadway formation cost but increases the amount of maintenance required to keep the gates open. The possibility of re-using roadways is greater in thin coal because of less total strata movement. Roadway convergence is less, after the first panel has been mined out, and this can provide a suitable roadway to be re-used by an adjacent retreat face.

50. The adoption of layouts that incorporate the re-use of roads is limited in the USA owing to mining regulations that were formulated principally for room and pillar workings.

51. In the UK, one of the simplest layouts for the re-use of roadways is mining out two advancing walls leaving a wide pillar between for subsequent retreat. An older longwall system is the double unit face. In this configuration two faces work adjacent panels and simultaneously use the centre road for the belt conveyor. This results in two faces requiring only three roads. This arrangement was used frequently in the UK prior to the advent of full mechanisation, when the rate of face advance was lower. The technique is not in present use, because it requires a high degree of organisation to co-ordinate face activities so that both faces travel at the same rate and are kept in a straight line.

52. Another method is the semi-advance/semi-retreat combination, where one road of a longwall is pre-formed by an earlier panel and the other is formed with the current mining operation.

53. There are many possible combinations of layout available to achieve the re-use of roads. The choice of method is dependent on local circumstances and an ideal method cannot be advocated to suit all combinations of factors (118).

Face Length

54. The length of a longwall face affects the cost structure of the district as, the longer the face, the greater is the cost of the face equipment. However, the length of the face line also affects the cutting machine utilisation time. With a

longer face, more time is spent cutting compared with the time the machine is idle during turn-around operations at the face ends.

55. To minimise the unit cost of production for roadway formation, the length of the face should be as long as possible. The determination of the optimum face length is thus a balance between the extra cost of equipment, the reduced roadway costs and increased productivity.

56. The length of the wall affects the reliability of the system as the longer the face, the greater the load that is placed on the face conveyor. Increased face length, whilst reducing the need for roadway construction, is not always conducive to good mining conditions at the face. Normally the mining conditions are best when the highest rate of advance is achieved and there is thus the minimum of time for the immediate strata to deteriorate.

Example of Thin Seam Advancing Longwall

57. The article "Thin Seam Mining and Dust Suppression, Langwith Colliery", Colliery Guardian, December 1975 (11), describes an advancing longwall, 915 ft long, working a 31-in-thick seam with a 36-in power loader at a depth of 700 ft.

58. The cutting machine used was the AB double-ended conveyor-mounted trepanner (DECMT), fitted with a water-cooled motor. The machine was equipped with 30-in-diameter augers with additional clearance being gained by top and bottom turret discs.

59. The face powered supports consisted of five-leg, Dowty chocks, designed to operate in thin seams. The chocks were double telescopic and rated at 150 tons on the first extension. The supports were set at 3-ft 6-in centres and attached to the armoured face conveyor by relay bars to maximise the available travelling passage.

60. The conveyor was a $7\frac{1}{2}$-in high, $24\frac{7}{8}$-in wide, open-bottom type, using two outboard 18-mm chains. At a chain speed of 169 ft/min, the conveyor was rated at 300 tph. The drive for the conveyor was by two 120-hp-drive units, one at each end.

61. The roadways were formed by the advance heading techniques. The main gate or loader gate was blasted and loaded mechanically and the tail gate cut by a Dosco road header. An advantage of the Dosco is that the machine-cut rock is small in size and can pass under the trepanner, which is mounted on the armoured face conveyor.

Results

Face manpower including headings	21
Face available time (minutes)	369
Seam height (inches)	31
Extracted height (inches)	36
Face length (feet)	915
Tonnes per shift (clean coal)	293
OMS (tonnes)	11.07

62. Owing to the fact that an average of 5 in of rock was being mined with the coal, severe dust problems were initially experienced with this installation. Extensive modifications were carried out on the trepanner cutter to introduce pick face flushing (PFF) and this resulted in substantial improvements to the dust counts on the face.

Example of Thin Seam Retreat Longwall

63. Although this example is in slightly thicker coal of 1.2 m (47 in), it has been chosen to illustrate the general principles and high performance that can be obtained from retreat faces (148). The longwall face is in Royston mine in Yorkshire where the face length is limited to approximately 130 ft for reasons of surface subsidence.

64. The equipment used in this operation is as follows:-

Cutting machine	- Eickhoff 172-kW (230-hp) single-ended ranging-drum shear
Armoured face conveyor	- 620-mm ($24\frac{7}{8}$-in) pans, 18-mm twin outboard chain
Supports	- Dowty 6 x 180-ton immediate forward support (IFS)

65. All the faces at Royston are laid out on a standard length and outputs have been consistently high. Over 1,300 tonnes per day on single-shift working has been achieved over sustained periods. The mining conditions are good and suitable for retreat as both the floor and the roof are strong and competent.

OTHER LAYOUTS

Remote Partial Extraction

66. Partial extraction, including room and pillar, has been carried out in the UK, mainly to limit subsidence of surface objects, as damage can result in expensive claims. Where mining is carried out under densely-populated areas or surface features such as high-speed railroads or rivers, the mine design is adjusted to give the minimum surface disturbance. This can be done in several ways.

67. In the case of longwall operations at depth, the ribs between longwall panels and the length of the longwalls can be adjusted (118).

68. Another approach is the partial extraction method which was employed by the Collins miner in a 30-in seam. In this case, the auger headings were cut from a single pre-developed roadway (2). This allowed the extraction of headings on both sides of the entry road, so maximising the extraction per unit length of development. Strata stability was planned into the design by leaving ribs of coal between the headings of approximately equal widths to the headings.

Multi-seam Extraction

69. Many coal seams exist as level beds in a vertical sequence of coal measure strata. The working of one seam normally has an effect on the others due to the disturbance of the stress distribution patterns. The effect of mining is to reduce the vertical stresses over and under the areas that have been mined out. Subsequent redistribution causes high stress abutment pressures on the ribs of the surrounding unworked coal or pillars that have been left (118). As the first seams to be worked are normally thick, for economic reasons, the subsequent interaction problems are normally found in the thin seams.

Multi Seam - Room and Pillar

70. In the USA it has been estimated that large volumes of coal have been sterilised due to the working of one seam in multi-seam areas (67). The situation has been compounded where the initial extraction was of an irregular pattern, producing uneven stress distributions.

71. The problems of uneven stress redistributions are more pronounced where pillar extraction has taken place. During pillar extraction it is normal to leave thin ribs of coal to assist in roof control during the mining operation. If conditions deteriorate, the widths of these ribs are increased and sometimes whole pillars or groups of pillars are left. This results in very high stresses being developed within the remnant pillars, which can cause serious strata control problems in the later mining of adjacent seams.

72. Research carried out in South Africa (149) in multi-seam workings indicates that layouts in the different seams should follow the general recommendations outlined below:-

(i) Where workings are being carried out in superimposed seams, the safety factor for the pillars, that is the strength of the pillar divided by the anticipated load, should be 1.7 or greater.

(ii) Where the thickness of the strata between the seams is less than the pillar centre distance, the pillars in the adjacent seams should be superimposed.

(iii) Where the strata between the seams is less than 1.5 to 2.0 times the width of the rooms, a danger exists of failure of the intermediate strata. This danger substantially increases the effective height of the pillars and hence the strength has to be based on the total pillar height, which involves new considerations and calculations. In the case of thin stratum between seams, the ability to mine the second seam is dependent on the strength of the intermediate strata.

73. Where superimposed pillars are planned, great care and control of mining has to be exercised to ensure the workings are, in fact, in a vertical line.

Multi Seam - Longwall

74. The working of multi-seam coalfields is widely practised in Europe and the USSR where the longwall method predominates. The major advantage of the longwall method is that, within the panel, complete extraction is obtained and no pillars are left to create local areas of high stress.

75. Adjacent seams have been successfully worked where the strata between them is less than 6 ft. In this situation, the normal sequence of operations is to mine out the top seam and then retreat a panel in the lower seam within the "shadow" of the destressed zone.

76. A frequent problem in older coalfields, where a number of seams have been mined intensively over a period of years, is the adverse effects of old pillar edges (112).

77. The normal procedure to avoid this problem is to design the initial workings so that the minimum of pillars is left and then site roadways in subsequent seams so they are situated in destressed areas.

CONCLUSIONS

78. The choice of a thin seam mining system for use in either a new mine or the extension of an existing mine depends on a number of factors. In conditions where the structural geology permits either longwall or room and pillar to be used, the choice could depend on financial evaluation. Basically, this would be a comparison of the higher capital costs of longwall, counterbalanced by the potential of larger bulk outputs.

79. A potential thin seam mining layout should include the evaluation of an enlarged man and materials transport roadway, sufficiently high to allow the use of a high-speed rail system.

80. Where reserves would be conserved by multi-seam mining, the longwall system is particularly suitable.

CHAPTER VIII

STATE-OF-THE-ART - CUTTING MECHANISMS

GENERAL

1. Numerous cutting mechanisms have been developed and applied to thin seams. In many respects the design criteria are common to both thin-seam and thick-seam cutting machines. Six broad classifications are identified and discussed in turn: chain jibs and their adaptations, rotating drums, augers or trepanners, ploughs and their adaptations such as scraper boxes and chain saws, other mechanical devices and, finally, hydraulic mining equipment.

2. In each of the above mechanisms, two cutting actions normally occur which cause mechanical failure of the coal. These are cleaving or tensional failure and compressive failure. At the bit, two component forces are applied: one, maintaining the depth of cut, and regarded as the normal force, acting at right angles, and a second cutting force, propelling the tool in the line of cutting. The relationship between the forces is dependent on the geometry of the tool, the depth of cut and the strength and directional properties of the coal being cut.

CUTTING PARAMETERS

3. In a series of experiments conducted by the NCB's Mining Research and Development Establishment (135), the relationship between various parameters of cutting was examined. The following broad conclusions were reached:-

Depth of Cut

(i) Shallow cuts, represented by low pick penetrations, are highly inefficient in terms of power expended per unit of coal broken and produce a high proportion of small-sized coal and large quantities of dust.

Cleat

(ii) The most efficient orientation of the cutting tool to the cleat is when the line of attack is at 45^0 to the cleat. Cleat orientation has little bearing on trepanners or shearers where much of the cutting is done perpendicular to the bedding planes but it is highly significant with ploughs and similar equipment.

Pick Speed

(iii) Pick speed does not affect cutting forces or coal yield significantly, although high pick speed can "throw" additional dust into the air stream.

Strata Pressure

(iv) The strata pressure exerted by roof and floor beds can either weaken or strengthen the coal and affect its cuttability.

Spacing

(v) Picks cutting in an array help one another, provided the penetration by individual picks is one-third to one-half of the line spacing; hence, deep cuts with wide line spacing are more efficient than shallow cuts at close spacing.

Machine Design

(vi) Coal cutting machines should be designed to take as deep a cut with each pick as is compatible with the strength of the picks and the strength and stability of the machine.

CHAIN JIBS

Basic Design

4. Chain jib cutting mechanisms are based on conventional cutter jibs, which were used for undercutting longwalls prior to blasting.

5. A number of machines have been produced using chain jibs either to trim, selectively cut, or back shear the web of coal being removed from the face. In the 1950s, the Anderson-Boyes Meco-Moore was a popular multi-jib cutter loader, which used a series of chain jibs to cut out the complete seam.

6. The chain provides a carrying medium for a series of picks or cutting tools mounted in pick boxes. The frame of the jib provides the normal force which holds the bit into the coal whilst the chain's motion provides the cutting force. The action, apart from the area around the nose of the jib, is one of fixed-depth linear cutting. The arrangement or lacing of the chain determines the breakout geometry of each line of cut.

7. The chain also acts as a conveyor; the pieces of coal broken out of the face are entrained in the chain and discharged as the chain comes out of the cutting area.

8. The limited space available for conveying restricts the length of jib that can be self cleaned in this manner. If the depth of cut is too great, the broken coal will completely fill the vacant space between the solid coal and the chain, and the mechanism will jam.

9. These features of the chain jib machine limit the depth of penetration of the picks and tend to produce many fines and large amounts of dust. The amount of dust that becomes airborne can be reduced by shielding and the use of water sprays.

Application to Multi-jibs

10. The machines used to power the multi-jib cutters were converted longwall undercutters but undue strain was placed on them which resulted in high maintenance costs and excessive down time (2).

11. Multi-jib cutter loaders have been used extensively in the UK mining industry. They were a convenient stage on the way to full mechanisation as they could be directly used on what was formerly a hand-loaded longwall face. The machine was compatible with bottom belt face conveyors and the timber support system. The technique has been largely superseded by purpose-designed thin seam equipment, but a few isolated machines are still in use in the UK, with armoured face conveyors.

Application to Continuous Miners

12. Originally, cutting chains, forming a chain mat, were widely used on continuous miners. They have, to a large extent, been superseded by machines using rotating drums.

13. The use of chains on continuous miner cutter heads led to excessive wear, high replacement costs and the production of more fines and airborne dust. The major reason for the adoption of drums, however, was their inherently stronger construction.

In-seam Miner

14. The in-seam miner employs a single heavy-duty chain which carries the pick boxes and the conveying scoops. The chain traverses the whole width of the heading and is driven by sprockets mounted at each end of the unit. The chain sweeps the lower section of the face of the heading in one direction and the top half of the face in the opposite direction (13).

15. Coal cut from the picks is carried along by the scoops of the bottom strand of the chain, and discharged into a small scraper chain conveyor. Steering the miner is achieved by tilting the main frame hydraulically in both the vertical and horizontal planes and advancement is by hydraulic thrust rams.

16. The machine is used to win out longwalls prior to installing the powered supports or, in some instances, to mine stableholes at either end of advancing longwalls. It has also been used in the drivage of a dirt-absorbing heading (15).

17. The machine length can be adjusted to drive headings from 14 ft to 30 ft in width at a rate of advance of 2 in/min to 3 in/min.

18. Some advantages of the machine are the low pick speed (220 ft/min), which minimises the dust make, and the narrow width (3 ft 6 in), which enables early support of the newly-exposed roof.

The Yarmak Miner

19. This machine was invented by a Mr. Yarmak of the NCB. Underground trials were carried out, but these were discontinued principally due to horizon steering problems (19).

20. The integrated mining system consisted of powered supports to which a single-strand flight conveyor was attached. The cutting tools were attached to the flight bars, which cut the face on both the top and bottom conveyor strand.

21. The machine, which was designed to extract a height of 30 in, was controlled by a man on the face who was positioned under the rear canopies of the support system. The trials proved that the cutting and loading principle was effective and that advance rates of 2 in/min could be obtained.

The Gloster-Getter and Meco-Moore Machines

22. These machines were used in production quantities in the UK prior to the development of the shearer. They both used chain-cutting jibs to profile out a web of coal by under cutting, back shearing and, in the case of the Gloster machine, top cutting. The cut-out coal was then transferred by means of a short conveyor or chute on to the belt face conveyor.

23. The profusion of cutter chains made them complex machines and whilst they were compatible with timber support systems, they were not suitable for use with armoured face conveyors and powered supports.

ROTATING DRUMS

Horizontal Axis

24. The rotating drum mechanism is the most common cutting device used in underground mechanised coal mining. The usual arrangement is to have the axis of the drum horizontal so that the picks or bits rotate to describe vertical circles. The drums normally incorporate a scroll, which acts as a screw conveyor, removing the broken coal from the cutting area. The use of a horizontal-axis rotating drum provides the basis for a very simple and mechanically robust machine.

25. As the drum rotates it is advanced into the coal face. The increment of advance per rotation of the drum, together with the arrangement of cutting picks, determines the maximum pick penetration. It has been shown in numerous experiments and tests that the production of dust, and the efficiency of the cutting process, are affected by the depth of tool penetration. The greater the penetration, the more efficient is the cutting process, in terms of specific energy expended per unit mass of coal removed from the face.

26. The arrangement has several disadvantages which limit its performance, both from a theoretical and practical view point:-

(i) The penetration of the cutting tools is only at the maximum in the middle of the face, and is zero at the start and end of each passage of the cutting tool.

(ii) The whole of the vertical seam section is traversed by the cutting tools. If there are any particularly hard or abrasive bands of stone in the coal, these will be cut with the coal, causing extra dust and heavy pick wear, and producing possible gas ignition hazards.

(iii) In thin seams there is difficulty in constructing a longwall shearer drum that will give adequate pick penetration, have sufficient depth of vane to act as an efficient conveyor and yet leave enough space for a strong centre shaft that will transmit the power and carry the reactive forces. These considerations would appear to limit the diameter across the pick points to between 26 in and 30 in.

(iv) The horizontal axis drum does not cut towards a free face, like a plough blade, and requires more cutting energy. High specific energy cutting is normally a high dust producer, owing to additional energy being used in breaking the coal into small pieces.

Horizontal Axis Drum Applications

Continuous Miners

27. Horizontal axis drums are used extensively on coal mining machinery. The principal use is in the cutting heads of continuous miners and the cutting elements of longwall shearers.

28. The mode of operation of the continuous miners is to sump the drum into the face at the top of the seam. The normal force for cutting is provided by the tractive effort of the machine via the tram drive and cat tracks. In one model, the head is sumped into the coal hydraulically with the cat tracks locked. When the drum is sumped to the required depth, which is approximately two-thirds the drum's diameter, the head is forced down through the coal to the bottom of the face. The head is then retracked to trim the floor to the required profile.

29. The cutting forces available are dependent on the weight and the power of the machine. During sumping, the forces available tend to limit the pick penetration and this produces more dust. As the dust is released at the top of the seam it is more difficult to suppress.

30. During the downward shearing portion of the cutting cycle, the weight of cutting force is provided by the weight of the head and the chain chassis weight. This results in a greater pick penetration, larger coal and less dust.

31. The design of the continuous miner's cutting head makes the provision of internal water passages for pick face flushing difficult. Dust suppression measures rely on external sprays, scrubbers and auxiliary ventilation.

Continuous Miners in Thin Seams

32. In thin seams the proportion of coal cut during sumping increases, and, as the seam height approaches the diameter of the drum, the downward shearing action disappears completely. This tends to reduce the effectiveness of the cutting action in terms of energy used, coal size produced and airborne dust.

33. The drum sizes are reduced as much as possible; the smallest commercially available size is 28 in. However, this leads to smaller drive components, less power and lower capacity machines.

Longwall Shearers

34. A longwall shearer drum performs the combined functions of cutting the coal and axially conveying it from the face to the armoured face conveyor by means of scroll vanes incorporated in the drum.

35. The disposition of the cutting picks across the face of the drum is not uniform and the normal configuration is a ring of closely-spaced picks at the face side of the drum to ensure clearance. Across the remaining face of the drum the pick spacing is less concentrated and this promotes breakout between the pick lines.

36. The design of shearer drums is a specialised art and much research has been carried out to determine the most efficient vane configuration and pick arrangement. The most recent work has been carried out on wider drums for use in wide-web shearing. These have now been tried and proved up to widths of 40 in.

37. The screw action of the drum is assisted by cowls and dozer doors. The cowl is contoured to fit at the back of the drum and close enough to prevent coal being left in the track behind the machine as it traverses the face. It is required for the bi-directional cutting method. Uni-directional machines usually use dozer doors.

38. Modern drum design incorporates an internal water feed to the pick point. Where incendive sparking may be a hazard, hollow shaft drums are available, which use the venturi principle to entrain an air and water mixture to the face side of the drum.

Thin Seam Shearers

39. There are two principal limitations to the height in which a shearer can operate: one is the height of the machine itself and the other is the ability of the drum to cut and load the coal.

40. Conventional conveyor-mounted machines must have sufficient clearance from both the beams of the roof supports above the machine and the armoured face conveyor under the machine. If allowance is made for a small amount of floor lift and some spillage under the conveyor, the practical limit for efficient operations is 36 in.

41. To improve the flexibility of the shearer, floor-mounted machines have been developed. They are sometimes known as in-web or buttock miners and have higher horsepowers (5). The machines are bi-directional with drums at each end mounted on ranging arms. This design allows more clearance under the support beams and between the body of the machine and the armoured face conveyor. To date, the minimum practical working height is 34 in.

The Montgomery ("Mini Miner") System

42. Some novel features are incorporated in the Montgomery miner, which has been developed for thin seam room and pillar mining.

43. The machine consists of a horizontal axis drum which is traversed across the face of a heading. The drum drive mechanism is supported on a traverse bar and the cut coal is then gathered by scroll conveyors which extend on each side of the machine. Two conveyor scrolls and a cutting traverse bar are carried on a frame that enables the machine to be moved from face to face. The system is designed to operate with a continuous chain haulage.

44. Initial experience has been in a 30-in seam which the machine mined without removing any of the roof or floor, but to date no production results have been published. The manufacturers claim that the working height may be reduced to 26 in to 28 in with reasonable mining conditions (134).

The Dawson Miller and Gullick Miner Systems

45. Both these systems were designed around the concept of a very narrow web machine drawn across a face at high haulage speeds.

46. In the Gullick design (111), the drum drive mechanism was mounted on a series of cantilevers attached to the powered supports. Horizon steering was achieved by raising and lowering the cantilevers.

47. The machines were designed for short faces in the region of 15 ft to 60 ft long, to form "stables" at the ends of longwall faces.

48. The problems of integrating main face operations, stable machine operations and ripping, led to the development of stableless faces and, consequently, the machines became redundant.

Vertical Axis Drum Applications

Advantages

49. In order to retain the basic robustness and simplicity of the rotating drum concept, various efforts have gone into the design and testing of vertical axis drums. The "Dranyam" was developed in the UK (102) and a similar machine was developed in the USSR (126). The picks of the vertical axis machine rotate in the horizontal plane. This allows pick lacing to be tailored to suit hard and soft bands in the seam.

50. The vertical drum machine cuts to a free face and would be expected to produce a larger product. This was borne out by tests (102) which showed an improvement over a standard horizontal shaft shearer.

51. Owing to the horizontal cutting action, it is not necessary to convey the product axially along the drum, and hence the extraction height limitations of the standard drum shearer do not apply.

Dranyam

52. This machine consisted of a conventional conveyor-mounted coal cutter with the vertical axis drum attached to a short horizontal jib. The jib was in the place of the normal undercutting jib and served as a support bracket for the vertical drum. The drum was driven by a chain from the normal cutting chain drive sprocket and, in addition to driving the drum, the chain cut the necessary clearance for the support bracket.

53. Trials of the machine were carried out in 1960. Encouraging results were obtained in terms of coal sizing and output, which was in the region of 8 to 9 tonnes per manshift, but mechanical difficulties were experienced with drum bearings, and the centre shaft had to be redesigned (103).

54. The original machine was designed for an extracted height of 48 in to 54 in but this could be substantially reduced by reducing the axial height of the drum. Before further developments could be made, the Dranyam was overtaken by improvements to the shearer and the longwall trepanner.

Other Drum Configurations

55. A prototype double conical drum was suggested in the USSR to attempt to obtain the advantages of the vertical and horizontal drum arrangements. The geometry of the drum was arranged to cut out a square profile. The problem with the concept was the variation of pick speeds across the conical faces and the mechanical difficulty of introducing the driving power at an angle at the bottom of the face. There is no reported information as to the performance of the machine or if, in fact, it was ever constructed.

56. Many other machines were designed and assembled for the US market in the 1950s and 1960s. Most of them were weird in appearance and quickly superseded by the continuous miner.

AUGER MACHINES

57. An auger cuts and loads the coal like a large, slowly-rotating drill. The paths of the cutting tools are circular and, with a constant rate of both axial advance and rotational speed, the penetration of the picks is constant. This is in contrast to the variable depth of penetration that occurs with the horizontal or vertical drums.

Simple Augers

58. These are used extensively in the USA, on the surface, for extending mining operations under the high wall of openpit operations and where access to the coal is obtained from shallow cuts. The auger is driven from the outer end of the hole, and additional lengths of drill section are installed behind the cutting section as the auger advances into the hole. The cutting tools on the head of the auger break the coal, and the scroll sections behind the head convey the broken coal out of the hole. The coal is then elevated using a short conveyor and removed from the site.

59. The operation of surface augers is highly productive in terms of output per man and the machines have a low capital cost.

Underground Augers

60. Development work has been carried out in the USA to extend the simple auger method underground. The work involved modifying the equipment to make it fit into the restricted space available underground and also to make it comply with underground safety standards.

61. Trials were carried out in Ireland Mine, West Virginia, on an auger miner to increase pillar extraction (42). Despite problems in directional control caused by the varying strength properties of the coal, the machine was able to mine 42-in diameter holes up to a depth of 100 ft from the rib side.

Trepanners

62. The trepanner, which is similar in principle to the auger, operates by cutting an annulus of large diameter. A core is formed in the centre and this is then cracked by heavy duty breaker picks in the trepan head. The resultant product contains a higher proportion of larger sizes and this attracted a financial premium in the UK in the mid-1960s.

63. The circular profile cut by the trepanner head was originally trimmed using small cutter chains, but in the later models rotating discs are used. The trepanner has been, and still is, a consistently high producer of coal in the thinner seams and is used in seams down to 34 in.

Auger-Type Continuous Miners

64. The Wilcox miner is a successful thin seam continuous miner that uses the auger principle. The cutting mechanism consists of two augers which are sumped axially into the face. The machine is then traversed across the face of the heading, using ropes, in a similar manner to a shortwall coal cutter. The augers have picks or cutting tools mounted on the periphery of the auger scrolls and these cut the coal during the traversing operation. Production rates of 300 tonnes per machine shift can be achieved by this machine in seams as thin as 30 in.

65. The auger principle was also used in the "borer"-type heavy continuous miners. Examples of these were the Goodman and Marietta borers, which were equipped with twin trepanning heads and peripheral chain cutters. The borer-type continuous miner has now been superseded by the drum type because of its simple, more robust construction and its ability to cut hard coal.

The Collins Miner

66. In the UK, the Collins miner was developed as a multi-auger boring machine. The cutting mechanism consisted of three augers, mounted side by side, at the front of the unit, the centre auger being set slightly ahead of the side augers. The cusps in the roof and on the floor, which were formed between the centre and side augers, were removed by static blades attached to the main body of the machine (2).

67. The design from the outset was for a remotely-operated unit, and sensing and directional control devices were built into the machine. Vertical sensing was by nucleonic gamma-rays. This technique, which measures the thickness of roof coal, has been further refined and is now used for automatic steering on longwall shearers. Horizontal steering was achieved optically by observing a target on the back of the machine and then making adjustments by means of hydraulic jacks in the side of the machine.

Russian Augering Techniques

68. In the USSR considerable research has been carried out and several auger-type machines have been produced. The BUG is a production machine which is essentially a simple auger. The unit is designed to drill holes up to 165 ft long but the average length achieved is only 120 ft. The principal reason for the shortfall is the loss in power owing to excessive friction against the sides of the boreholes (87).

69. The "Tentek" machine was designed to increase output and to overcome frictional losses by placing the scroll conveyor inside a metal tube. The unit was fitted with auger cutters which were rotated through 90^0 at the inner end of the

hole. This allowed the auger head to cut, as in shearer drums, on retraction of the unit from the hole. "Unsatisfactory" results were obtained owing to steering difficulties (88).

70. After a visit to the USSR in the early 1970s, Collins, the designer of the Collins miner, disclosed that no further developments were being made in the USSR in augering and remote extraction.

COAL PLOUGHS

71. Static coal ploughs, which were initially developed in Germany, have no on-board power source. The plough consists of a number of cutter blades or tools, mounted on a guided tool holder or plough body, which is hauled across the coal face by means of a heavy-duty chain, driven by motorised sprockets at each end of the face. The plough body is guided in a variety of ways, depending on the model, and there are also variations in the position of the haulage chain. The incremental cut per traverse of the plough can be adjusted to suit the hardness of the coal.

72. One major difficulty inherent with ploughs is horizon control. If the floor of the seam is similar to or weaker than the coal, the plough will tend to cut into the floor. This can be corrected by adjustment to the cutter blades; further steering is available from goaf side jacks on the armoured face conveyor.

73. The plough is very dependent on the directional properties or cleat of the coal. Where the cleat of the coal is well developed and the longwall is parallel to the major axis of the cleat, the face is "on-bord" and the plough is able to exploit the weaknesses of the coal and literally "peel" it off in slices.

74. Because the plough breaks the coal to a free face and uses to advantage any weaknesses in the coal, it produces a much larger sized product. The reduction of fines is accompanied by a reduced make of dust and a product that is more easily cleaned in a coal preparation plant. Ploughs also require a coal that parts easily from the roof of the seam.

75. The characteristics of the coal thus play a much more significant role with ploughs than with other cutting mechanisms but, in suitable conditions, much less energy is required. However, power requirements can rise substantially owing to the friction of the haulage chain and the plough body.

76. Ploughs have been, and still are, operated in some of the thinnest seams. In 1965, a 24-in to 28-in thick seam was worked at Lofthouse Colliery, Yorkshire, (28). The USSR have worked ploughs in sections down to 20 in. The lower limit of height worked is set, not by the plough mechanism, but by the powered supports and the requirement of a suitable space for men to travel on the face. Although it is possible to operate the powered supports by remote control, men must travel along the wall to operate the horizon control jacks mounted on the conveyor.

77. Dust suppression on plough faces can be difficult. Although a plough produces, in general, less dust per ton of output than drum machines, the rapid travel of the plough through the face can provoke an airborne pick up. In such circumstances it may be difficult to apply effective water jets to the plough blade.

78. Dust suppression using stationary sprays along the face line has been used. This technique requires sequential cut-off valves to prevent water being needlessly sprayed on to the face when the plough is in another section of the face.

79. The speeds of the plough and the armoured face conveyor must be properly integrated to ensure effective coal clearance. High-speed ploughs have been introduced in order to provide a sufficient relative speed, when both plough and conveyor are travelling in the same direction. When these two moving components are travelling in opposite directions, the limiting factor becomes the carrying capacity of the conveyor.

Scraper Boxes

80. The scraper box performs two functions: it breaks the coal off the face in a manner similar to a plough and the boxes scrape the coal to the end of the face without the use of a separate face conveyor.

81. The boxes consist of open-ended frames that run along the floor of the seam. Flaps are incorporated in the rear of the boxes, which close when the box is pulled towards the loading end of the face. Plough-type blades are attached to the face side of the boxes and actually cut the coal. A number of similar box units are spaced out along the face. As the boxes are pulled back and forward through the face, by means of an endless steel haulage rope, coal is transferred from one box to the next and subsequently to the end of the face.

82. The major variation in scraper box design occurs in the manner in which the boxes are trapped against the face. In the earlier Haarman design, the boxes were held against the face by a skid board, which unfortunately occupied a considerable proportion of the face height. This presented a major obstruction to men wishing to make cutter blade adjustments for horizon control (3).

Tension Chain

83. The tension chain technique replaces the skid board by a tensioned chain. The ends of the chain are kept in front of the face, causing the face to assume a slight bow, and the force pushing the boxes into the face can be regulated by increasing or decreasing the tensions in the chain.

84. The mechanism can work in any seam thickness but is dependent, to an even greater extent than the plough, on the directional and strength properties of the coal seam and the floor. The absolute capacity in tons per hour of the scraper box system is relatively low due to the crude conveying system (3). Substantial increases in capacity result from its application in steeper gradients, where gravity assists conveying. This feature led to the development of the cable chain saw.

Cable Chain Saw

85. The use of the cable chain saw cutting mechanism has been limited to highly inclined seams. The cable chain employs a series of cutter bits attached to a cable or chain, which is drawn across the face by either a winch or a motorised chain sprocket.

86. By maintaining the ends of the chain in front of the face, the face assumes a curved shape, such that the tension in the chain or cable provides the necessary components to force the cutting bits into the face and propel the bits along the face.

87. As the cutting mechanism has no part in conveying the coal off the face, the system has only been applied in steeply inclined seams where the coal can naturally gravitate to the end of the face.

88. The cutting mechanism has, in common with the other static blade systems, an application where the relative strengths of the floor and coal keep the cutter bits in the coal.

HYDRAULIC MINING

89. The use of jets of water as a cutting mechanism for coal has interested mining engineers in many countries. Hydraulic mining holds the prospect of very low dust production and the elimination of the danger of a spark from a cutting tool igniting any gas that may be present. The concept also offers a means of cheap haulage for the coal, from the face and along district and main roads.

90. Hydraulic mining is known to be practised in Canada, the USSR, China, Japan and Germany. In all cases it is used in situations where the seam or seams are sufficiently inclined to enable the water to carry the coal along either a pipe column or an open flume.

91. At the Hansa hydro mine in Germany, seams of 48 in in thickness have been mined and, although this cannot be classified as thin, it is sufficiently low to have provided useful experience.

92. Hydraulic mining requires a fairly soft coal to enable the water jets to remove substantial tonnages, but experiments have indicated that increased pressures would enable harder coals to be won by this method.

93. In the Hansa operation the water jet or monitor is located in a roadway specially driven for monitoring operations. The monitor operator is positioned in sight of the monitor but operates it by remote control. This offers protection from the danger of injury from uncontrolled movements of the monitor. Further safety features are the low level of airborne dust and the reduced risk of ignition by incendive sparking.

94. The distance at which the monitor can effectively operate and break out coal is highly significant to the economics of the operation as it determines the layout of the monitor drivages and, consequently, the amount of development work necessary.

95. To date, hydraulic mining has had limited application throughout the world owing to its dependence on very specific natural conditions, such as the softness of the coal, the hardness of the roof and floor and the inclination of the deposit.

CONCLUSIONS

96. It is difficult to identify any new form of breaking out coal that might utilise a basic cutting principle not included in this survey.

97. However, this will not preclude the future development of improved equipment, based on existing techniques.

CHAPTER IX

STATE-OF-THE-ART - TRANSPORTATION

GENERAL

1. Underground transportation in a thin seam US mine differs in many respects from its counterparts in other countries. In the USA, the main transport system is normally within the confines of the coal seam, with only a limited amount of extra height being made available to allow the passage of machinery or vehicles. The various elements of the mine transport system tend to be housed in separate roadways. This arrangement is facilitated by the multiple-entry system which has other advantages, not least that of ventilation. A common configuration is five entries. The coal is transported on a belt conveyor in the middle entry while men and materials travel in either one or two of the other roads. The two inner roads are usually used for intake air and the flank roads for return air. If the mine has cold winter conditions, then the men and materials transport systems may be in the returns. If methane is a problem, then the gas-free intakes would be used.

2. In European practice, the main access roads must, by law, be driven to give a walking height, and, in the UK, this is set at a minimum of 5 ft 6 in (31). The longwall method requires only one roadway at each end of the wall but main or trunk roadways are often duplicated, principally for ventilation.

3. Underground transportation is required in a coal mine for a variety of duties. The most common are to convey coal and debris out, materials in, and men both in and out. Other more specialised duties include the transportation of prepared stowing debris in to the mine from the surface, piped water transport systems for drainage out of the mine and firefighting, and dust suppression water in to the mine. Methane is also piped out and, in some mines, compressed air is piped in.

TRANSPORT OF COAL

US Practice

4. Coal transport in main roads in the USA is usually by conveyor belts, by rail-mounted rolling stock or a combination of both. Table VI shows the distribution of the various systems in use in underground US mines. It can be seen that there is an increase in the number of conveyors and a reduction in the number of mine cars.

5. For thin seams, belt conveyors are virtually essential, as they can operate in relatively low roadways, down to 24 in high. They can provide high carrying capacities and are compatible with the production rates of continuous miners. With the introduction of continuous haulage behind the mining machine, outbye belt conveyors become an extension of the face haulage system.

6. Belt conveyors in US mines are frequently rope-mounted and often supported from roof bolts. These features reduce the amount of maintenance required, although not eliminating it completely.

7. Where the conveyor belt is of approved fire-resistant material, equipped with adequate signalling and protective devices, and is installed and maintained to good standards, the use of conveyors provides a safe and efficient method of mineral transport. One of the most significant recent developments in coal transport in the USA has been the introduction of continuous haulage behind the continuous miner. This equipment is of special significance in thin seams as it obviates the need for shuttle cars. The system consists of a series of scraper chains or belt conveyors, normally three conveyors, one directly behind the continuous miner, one which transfers the coal to the section belt conveyor and an intermediate conveyor linking the two (108) (122). The articulations in the systems provide the flexibility to mine rooms and also depillar if required.

European Practice

8. In Europe there is little or no difference in the coal transport system between mines extracting thin and thicker seams. This is due to the prevalence of longwall mining. Systems do vary, but for reasons other than seam thickness. Mineral transport from the working face is usually by belt conveyors. The conveyors lead either direct to the coal winding shaft or to underground loading points, where the coal is then transported by rail to the shaft. If the mine has a slope entry, then the main belt conveying system usually delivers the coal direct to the mine surface.

9. Recent developments in high-capacity trunk belt conveyors have enabled UK coal transport systems to utilise surface slopes or drifts to haul large tonnages from greater depths, often from a combination of existing mines. An example of this is the new mine under construction at Selby in the UK, where two 14-km (9-mile), 2,000-tph trunk belt conveyors are being installed, one in each of two adjacent surface drifts. The vertical lift, when both conveyors have reached their final length, will be approximately 3,000 ft (123).

10. The use of bunkers is becoming increasingly important with the growing number of high-capacity faces. Where several longwall faces feed on to one main belt line, the combined peak feed rate is normally greater than that of the main trunk belt. Bunkers are used to reduce these peak loads on the main conveyors and to provide a buffer to the coal faces, so reducing the effect of delays on the main conveying system.

11. The most recent development in the UK has been the use of computers to control both conveyors and bunkers. By adjusting the discharge and feed rates to bunkers, overloading of conveyors and consequent downtime and spillage are greatly reduced. The use of the control and monitoring techniques also reduces the number of operators required as both the conveyors and the bunkers are unmanned.

12. Elsewhere in Europe much greater use is still made of mine cars. These represented a major improvement in coal transport prior to 1950. At that time horizon mining methods were under development for the steeply-pitching seams of the Rhur coalfield. These incorporated level trunk roadways, suitable for locomotive haulage. However, the increased popularity of trunk conveyors is also in evidence on the continent of Europe and many coal winding shafts have been converted from mine cars in cages to coal skips. The latter facilitates the extension of the belt conveying system to the shaft bottom, and eliminates the less efficient locomotive haulage.

TRANSPORT OF MATERIALS

US Practice

13. The supply of materials and machinery is necessary for the smooth operation of mining activities at the coal face. Handling of supplies can be responsible for a considerable number of accidents when the work is left to new and inexperienced personnel, or the handling systems are poor.

14. Supplies are normally transported by a combination of rail and rubber-tyred vehicles (85). Where a rail system is used, it is necessary, in many cases, to transfer the material from the rail car to a scoop or utility vehicle to take the material to its point of use. This double handling of material is expensive in labour, and when heavy items have to be handled, such as parts of a shuttle car, wheel units or other machinery, improvised means are frequently employed to lift or drag the parts from one vehicle to another. This operation can be a source of serious accidents.

15. In thin seams, where the supply roads are at the seam height, the handling of materials becomes more difficult, more labour intensive and more hazardous. Increasing use is being made of rubber-tyred vehicles to move supplies direct from the mine entry to reduce double-handling. This facility is available to operators where the distance from the entrance to the working sections is suitable for battery-powered vehicles and rail transport has not been provided for any other purpose.

European Practice

16. The normal UK practice is to transport supplies by rail using locomotive and/or endless rope. The main roadways of the mine are often suitable for locomotive transport, but the supply roads to the individual longwalls are rarely adequate, owing to severe gradients, poor track or insufficient clearances due to ground movement from mining operations.

17. Overhead monorails and floor-mounted captive-rail systems have been used for a number of years in a few mines and now promise to spread to others. This follows a period of relative inactivity in the development of new and better supply systems. Overhead systems are particularly useful in conditions where floor

heave is prevalent. Conversely, floor-mounted equipment is used where roadway distortion takes place but the floor is relatively stable. Germany and France have developed these systems and, consequently, they are in widespread use in Western Europe.

18. The transport systems in thin seams do not differ from those in thick or medium seams as the roadways are made roughly the same height in each case.

19. In any one mine, a combination of transport modes, such as locomotive and endless rope, means that materials are frequently transferred from vehicle to vehicle. This is labour intensive, but, because of the availability of height and the relative permanence of transfer points, it is often possible to install mechanical lifting equipment at transfer stations.

20. Increasing attention is being given to the use of rubber-tyred vehicles and new operations are making greater use of this type of transport. Although the road widths and floor conditions in the UK do not readily adapt themselves to the general use of rubber-tyred vehicles, trials are underway and experience is being gathered from mines in France that have developed roadway consolidation techniques for use with free-steered vehicles.

TRANSPORT OF MEN

US Practice

21. The efficient transport of men is necessary in order that the maximum available time out of a shift is spent at the working face. The transport of men is often carried out using the same methods and equipment that are provided for the transport of materials. This is normally a rail system incorporating battery- or trolley-wire powered vehicles, which starts at either the surface of the adit or the shaft bottom and extends inbye as far as possible. Rail transport offers fast and efficient means of entry and exit from the mine. In thin seams it is necessary for men to lie flat owing to lack of headroom. Specially-constructed man carriages are in use, which are more comfortable and offer protection from the top and sides of the roadway. Beyond the rail system, rubber-tyred battery man carriers are used to transport the miners to within a few yards of their work place. Alternatively, utility scoops are used where men ride in the scoop or are drawn behind in a trailer or a sledge made from discarded conveyor belting (85). A typical travelling time for a two-mile journey in a thin seam mine equipped with efficient manriding facilities is 30 minutes.

22. One danger or hazard in using utility or production equipment to transport men, is the possibility of the unit's discharge mechanism being activated whilst men are riding in the vehicle and, for this reason, purpose-designed vehicles are to be preferred. An alternative method of outbye transport is by means of the coal transport belt conveyor system. This can be made safe by the provision of platforms for boarding and alighting and is a cheap and flexible way of transporting men either out of a section or out of the whole mine.

28. If the operations were extended to three shifts per day over a seven-day week, the operating time would approximately double, though an allowance would have to be made for the essential maintenance that is normally done on the non-producing shifts.

29. As a result of working 21 shifts per week, as opposed to 10, the output would approximately double. This would result in a reduction in the average working life of the equipment, probably from eight years to five years. (The working life of the equipment is a combination of service and time factor.)

30. The reduction in equipment life from eight to five years would increase the capital recovery factor by approximately 28.3%. This increase would be more than offset by the increase in output which would result in the unit cost of the capital recovery charge reducing to 64.15% of its previous value.

31. On the assumption that the material cost per ton remains static and labour charges are increased by 14% to compensate for working on a seven-day-per-week basis, the unit costs would be as shown in Table XVI. It can been seen that such a working arrangement reduces the total operating costs for all the systems, but has the greatest effect on those using equipment with the highest capital cost.

Effect of Change in Interest Rate

32. The capital recovery factor used in the calculations was based on an interest rate of 20%, which approximates to the prime borrowing rate at the time of the preparation of this report. Many previous exercises have been based on a rate of 10%.

33. If a 10% rate were used on an eight-year repayment, the depreciation/capital repayment charge would fall from approximately 26% of the capital base to about 19%. The cost structure of the system based on an interest rate of 10% is shown in Table XVII.

Effect of Inflation

34. Inflation is constantly increasing costs, though not all costs are affected in the same manner. The effect of inflation on material and labour costs tends to increase these year by year. In the case of equipment purchased on a fixed-term loan, the repayments would be fixed and therefore proofed against inflation. This effectively reduces the rate of interest in present-day monetary terms.

35. The purchase of new equipment would cost more owing to inflation. However, this would be purchased on new loan terms which, again, would show a progressively reducing repayment cost in constant-value terms.

CHAPTER XXI

COMPARISON OF CONCEPTUAL LAYOUTS AND DESIGN

INTRODUCTION

1. The purpose of this chapter is to compare major features of the conceptual layouts and designs and to highlight the differences and advantages each offers. It should be noted that the comparisons, particularly those of a quantitative nature, are those assumed to occur under the "average" conditions simulated in the block of coal. The effect of changing the physical and other parameters is specifically addressed in Chapter XXII.

2. Each system is examined in the following table:-

Criteria	Longwall			Room and Pillar			Shortwall
	Z-System	Single-Entry	Multiple-Entry	9-Entry	5-Entry	9-Entry Retreat	
Health and safety	A	A	B	C	C	D	B
Operating cost, \$/ton	15.90	16.80	13.46	16.99	14.33	18.46	15.35
Capital cost, 10^6\$	10.68	11.18	9.12	6.47	5.03	6.47	9.93
Annual output, 10^3 tons	361	348	358	348	351	308	441
Productivity, tons/manshift	28.00	26.50	28.80	16.80	20.60	15.00	21.30
Time needed to reach full production (1)	4	18	27	5	3	7	3
Continuity of production (2)	94.00	88	93	100	100	100	100
Utilisation of reserves (3)	92.80	87.80	82.00	64.30	59.40	74.40	81.90
Required legal variances	Moderate	Major	-	-	-	-	-
Environmental impact	B	B	C	C	C	D	C

Legend: A, B, C and D represent rank order of desirability where A is the best or most desirable and D the least.

1 = The time in months to reach full output.

2 = Percentage of time the section is available for production as opposed to being moved.

3 = Percentage of coal removed from the in-situ reserve.

8. The use of remote control does not solve all the problems of restricted height as it is necessary to repair the equipment on site, should a malfunction occur. Some of the equipment may be permanently positioned on the face, such as longwall powered supports, in which case routine maintenance must also be carried out in cramped conditions. Local enlarging of the space around thin seam equipment can be accomplished by blasting but this is time-consuming and liable to damage the machinery.

9. Where the available space totally precludes access even for repair and maintenance, then a remote extraction technique is essential and the design of the system must be such as to enable the equipment to be withdrawn from the low areas.

10. Equipment suitable for total remote extraction includes conventional augers, Collins-type augers, chain ploughs, scraper boxes and specially-adapted continuous miners. It is also possible to adapt hydraulic mining to thin seams, although the thinnest example of this method is a currently-worked seam approximately 52 in thick.

REMOTE EXTRACTION

Design Parameters

11. An essential feature of remote extraction is that men do not enter the unsupported working area. This means that the excavations in the seam must be self-supporting. Another requirement is that some alternative to the conventional scraper chain conveyor is necessary to convey the coal off the working face. In simple augering, the scroll satisfies this requirement, while, in some of the other systems that operate on steep gradients, gravity is utilised.

Augers

12. The auger is most suitable for remote extraction. The circular shape of the excavation is ideal for self-support. There is no practical limit on the minimum hole diameter that may be bored; the limitation is an economic one. Thus, a 20-in diameter hole would require to be four times as long as a 42-in diameter hole. Despite the expectation of faster drilling speeds in smaller holes, the relationship between hole diameter and output is that the latter varies as the square of the diameter. This limitation of a single auger hole led to the development of twin- and triple-headed augers for thin seams.

13. Augering experience was initially gained on openpit highwalls, where diesel-powered units achieved drivage in excess of 200 ft. One of the limitations is the problem of deviation of the line cut by the auger in both the horizontal and vertical planes. Deviation is dependent on the cutting characteristics of both the seam and the host rock and on the rigidity of the auger sections. In boring small diameter holes, the auger scroll sections have a lower degree of rigidity than in larger holes. Thus deviation problems are likely to be greater in thin seams. Deviation would tend to limit the length of hole and aggravate the time spent on repositioning the auger for the next hole.

14. The limitation in hole length would increase the amount of roadway development necessary for the exploitation of a block of coal.

15. These factors, together with the inherent lack of production capacity, preclude the viable use of a single-hole auger in a thin seam underground. Despite the constrictions to production of single augers, it is nevertheless important to develop the capability to steer a single borehole in a thin seam as such a hole forms the start line of other remote extraction systems.

Collins-Miner-Type Machines

16. One of the forerunners of the Collins-miner-type machine was a machine developed in the late 1950s by Union Carbide Co of the USA. It was track-mounted, incorporated four overlapping cutting heads and was controlled remotely from the mouth of the drivage. The auger concept, to bore long unsupported excavations, is extended with the Collins-miner-type machines. By using a series of circular cutting elements mounted side by side in the head of the machine, the problem of greatly reduced tonnage with diminishing height is minimised. Although the Collins miner as such is no longer in use in the UK, it did produce significant tonnages from a 30-in extraction in Yorkshire in the mid 1960s. It would appear that similar machines are probably still in current use in the USSR (87)(88).

17. All three of the Collins miner installations have been adequately described (2). The basic design of the equipment consisted of a 120-hp cutting head, capable of driving an excavation 6 ft 3 in wide by 30 in high. A small internal conveyor transferred the newly-cut coal to the extensible belt conveyor which was pulled up each hole behind the cutting head.

18. A system of push rods at the mouth of the hole transmitted the thrust for the cutting head. The planned length of each hole was 300 ft. All operations were remotely controlled from the mouth of the hole, including steering and ventilation.

19. Because of the shallow depth of working (495 ft) and the substantial coal pillar of 7 ft 6 in between holes, there were few problems associated with the collapse of the unsupported holes.

20. Although operating difficulties were experienced with steering the cutting head and with the belt conveyor, the machine amply demonstrated that remote extraction could be carried out, and at a rate of productivity better than a conventional longwall face working under the same conditions (2).

Scraper Boxes

21. Scraper boxes, known as the Haarman scraper, have been used in the UK, Belgium and France and are still in use in the USSR. The system is simple and consists of a series of scraper buckets that are pulled backwards and forwards across the longwall face. Coal is won from the face by means of plough blades attached to the face side of the buckets. This falls into the track of the boxes and is collected and transferred from box to box and finally delivered to the end of the face.

22. Where the boxes have been used in seams higher than 16 in, men have been employed on the walls to set supports. For seams less than 16 in, it is impossible for men to work effectively. In these heights the system becomes remote extraction and operates without roof supports.

23. In the early 1960s, several mines in the Durham coalfield of the UK operated unmanned longwalls, using modified versions of the Haarman scraper, called the tension chain scraper (3). The walls were won out by drilling holes between two entries 120 ft apart. Reamer tools were then used to make sufficient room in the 14-in seams to install the boxes. Advances of 5 ft to 6 ft per shift were achieved, but, owing to the friable nature of the roof of the seams, the faces were only able to proceed a distance of between 27 ft and 45 ft before the roof collapsed and closed the wall. When this took place a new hole was bored and reamed and the equipment re-installed. The time and cost of re-opening faces adversely affected the economics of the system but the experience gained proved that scraper boxes could cut and load effectively in very thin seams.

24. For the scraper box technique to operate successfully without men being present on the wall, it is necessary to have a roof that will converge without caving. An alternative to a supportless face is the provision of a measure of positive roof control by either props or wedges remotely placed between the roof and floor and abandoned, or a simplified powered support that is maintenance free and capable of remote operation. Another development could be to use an hydraulic or pneumatically-inflated tube along the face.

Chain Saws and Ram Ploughs

25. Both chain saws and ram ploughs have been used in the USSR for mining thin seams. The applications to date have all been in steeply inclined deposits where the coal, once broken from the face, slides to the end of the wall. In the USSR, ram ploughs remotely extract seams as thin as 12 in. At this thickness, the lower half of the wall is curved forward to allow the caving roof to fall away from the face of the wall, thus keeping a clear track for the saw or plough (86).

26. The plough or saw unit is so designed that it can accommodate variations in seam thickness. Although the rates of face advance are low, this is not the sole criterion in evaluating the technique. Another, and probably more important, factor is the need to extract thin seams as a precautionary measure against rock bursts in adjacent thicker seams.

27. Only limited application of chain saws and ram ploughs is seen in the USA owing to the bulk of the reserves being flat or moderately dipping seams.

REMOTE CONTROL

Room and Pillar

28. Most of the commercially-available continuous miners produced in the USA are now available with remote control. The normal types are electric or hydraulic control cable and radio control.

29. The principal advantage of remote control is that the operator no longer has to be stationed on the machine. This allows longer passes and obviates operator exposure under unsupported roof.

30. In conjunction with remote control, associated equipment has been developed to monitor methane on the continuous miner. A sensing device is also available to assist the operator in positioning the machine correctly within the seam. There is, however, still the requirement that the operator be able to see the machine in order to steer it.

31. This remote control and monitoring equipment, although available for all ranges of continuous miners, is especially valuable for thin seam machines. It allows the operator to restrict his physical movement in cramped conditions. The limit of workability, however, is still that of miniaturised equipment, which is at present about 28 in. For complete remote operation of room and pillar mining, the latest techniques of remote control and guidance sensing must be incorporated.

Longwall

32. The NCB has carried out a considerable amount of development work in the UK to produce an unmanned longwall face. Four longwalls were equipped and operated in the mid 1960s. The initial remotely-operated longwall faces (ROLF) were not designed specifically to mine thin seams, but the fourth installation, ROLF 4, was put to work and successfully mined a 30-in seam in the Yorkshire coalfield. This installation was an ideal example of what could be achieved when sophisticated modern mining technology was applied to an unmanned longwall. The manless aspect of the project only referred to the operations of the shearer, the armoured face conveyor and the powered supports. Men were still required to work in stable holes at either end of the longwall and for the maintenance of the powered supports.

33. The experience gained from ROLF indicated that electro-hydraulic control of powered supports could be made reliable but that further developments were required for complete automatic operation of the power loading machine.

34. Successful remote control of a power loader, either shearer or trepanner, requires a knowledge of the status of the machine, ie its orientation and position in the seam. In manual operations, the operator obtains this knowledge visually, though often aided by the sound of the machine as it cuts. A good operator interprets this and makes adjustments to the controls timeously and it is this element of anticipation that is difficult to build in to remote control systems. More recently developed remote control equipment for shearers and powered supports is proving reliable, but so far it has been difficult to tailor such techniques to a plough face, owing to the need for manual adjustment along the whole length of the armoured face conveyor.

MANUAL CONTROL

35. In slightly thicker seams in the range of 34 in to 42 in, it is possible to produce viable outputs from manually-operated room and pillar and longwall systems.

36. The longwall conveyor-mounted trepanner (120 hp) has been particularly successful in these heights and a new design of longwall shearer has been specially developed for these conditions. This is the floor-mounted buttock or in-web shearer, which has more power (270 hp) and is expected to perform well in these seam thicknesses.

37. Another variation of manned longwall operation in thin seams uses the Russian TEMP coal-cutting machine. A number of limitations apply to this equipment that tend to restrict the production of high tonnages. The seam must be steeply inclined, as face conveyors are not used, and the cut coal must gravitate to the bottom of the wall. The cutting element consists of two rotating drums, one driving the other through a cutting chain, and the arrangement is susceptible to breakdown in harder coals. The low power (50 kW) of the unit restricts the cutting speed to roughly half that of the conventional shearer.

38. Despite these limitations, the latest machine can operate in seams down to 18 in. At this thickness the manual operation of the support system tends to become the limiting factor.

CONCLUSIONS

39. Equipment is available to extract coal in seams as thin as 12 in.

40. Many of the remote extraction systems designed for seams of 12 in to 18 in in thickness require steeply-pitching beds in order that gravity can haul the coal off the wall; this condition tends to restrict the application of such systems in the USA. However, remote augering has been used extensively in the USA and could be a technique worth further development in underground mines.

41. Remote control methods are still not fully developed, but they are likely to play an important role in the future in the seam range 20 in to 30 in.

CHAPTER XI

STATE-OF-THE-ART - AVAILABILITY AND RELIABILITY

AVAILABILITY

General

1. This section examines the availability of thin seam equipment that is either proven or still on trial. To be classed as proven the machinery must be reliable and capable of producing satisfactory results from the mining conditions in which it operates.

Room and Pillar Equipment

2. Continuous miners are available from three manufacturers, in height ranges down to 28 in. Two are capable of being operated both manually and remotely; the third machine is equipped for remote operation only. Remote operation is by direct single wires, a multiplex system or radio.

3. New equipment under development by one manufacturer is a low seam auger miner, incorporating hydraulic controls. It is designed to improve mobility and provide remote control. Trials have been held but it is not yet sufficiently reliable for common application (108).

4. Another new development is a low-vein, circular, kerf-cutting machine suitable for seams 18-in to 20-in thick. The first underground trial was encouraging and this has led to a second. The work is being carried out under US Bureau of Mines' contract No H0 252 044.

5. A new range of equipment has been developed to convey the coal from a continuous miner in a height of 20 in. This includes a shuttle car, a ram scoop car and a continuous haulage system. These are now available as proven items, although the continuous haulage system has been slow to gain acceptance, despite its initial introduction in 1970 (39).

6. Roof-bolting equipment is currently available to operate in thin seams but is not yet widely applied. Wooden supports are still in widespread use (51).

7. Manriding and supplies vehicles of the rubber-tyred, battery-operated type are now proven and available. Various designs for thin seam conditions can operate down to 28 in.

8. Rope-mounted belt conveyors are widely used as the main road coal haulage system. Where the entries have been driven wide enough, and there is sufficient clearance to comply with the regulations, rail or rubber-tyred manriding can take place alongside the belt conveyor.

9. Although much of this equipment is available for operation down to 28 in, the underground conditions have to be nearly perfect to obtain maximum efficiency. A more practical minimum seam thickness is 32 in. Results tend to fall off rapidly below 40 in, at which height difficulty is experienced with mobility, lifting and repairing equipment. As a consequence, there is a tendency to cut into the roof and/or the floor when conditions permit.

Longwall Equipment

10. The two major longwall techniques are shearing and ploughing. Ploughs can operate in any seam section from which coal can be satisfactorily conveyed. On faces with armoured face conveyors and powered supports, 32 in is the minimum seam section recommended. Equipment for these conditions is readily available and proven.

11. Scraper boxes on a longwall face, in conjunction with some form of roof support, either props or chocks, can operate as low as the minimum support design height and at the limit of manned operation, which is approximately 16 in. Without supports, scraper boxes can operate in a 12-in seam.

12. The limitations of the plough system are its capability to win coal off the face at a sufficient rate, and the ability to control its cutting horizon. If these two criteria can be met, the plough is a very versatile coal-getting machine and can mine coal down to between 24 in and 28 in.

13. Where the coal is too hard for the plough, the shearer is the most popular machine. Acceptable results have been obtained from seams of 30 in, using a single-drum machine, cutting bi-directionally, with remote haulage (151). The decline in thin seam mining has virtually halted further developments of this machine, but UK manufacturers could produce them if required.

14. Improvements to the shearer loader have concentrated on the thicker seam models, resulting in larger-horsepower, double-ended, ranging-drum machines. These can be supplied for seams in the range 42 in to 44 in. About seven years ago, a new UK development modified the geometry of the shearer to allow it to be floor-mounted instead of conveyor-mounted. This in-web or buttock shearer can, at present, operate in seams down to 34 in. Three manufacturers, two UK and one German, make the design, and upwards of fifty machines now operate in the UK. Experience to date in the USA has not shown satisfactory results as exampled in the US Bureau of Mines-sponsored trials in Kentucky.

15. An early variation of the shearer for thin seams was a vertical drum machine, known as the Dranyam. The drum was attached to a standard longwall coal cutter, which was mounted on the armoured face conveyor. This machine is no longer in use or readily available from the UK manufacturer, but the design details are still available for a rebuild should this be requested.

16. In the early 1960s, many adaptations were made to the standard longwall coal cutter. The multi-jib version is still in use in Europe and available from manufacturers.

17. Scraper boxes can be used as a conveyor only or combined plough and conveyor. They are simple to manufacture locally and operate with readily available rope winches, making them a simple, cheap system for thin seams.

Other Systems

Collins Miner

18. This machine is no longer in use in the UK but similar equipment is in use in the USSR. Modifications to its nucleonic steering mechanism and to certain of its other weaknesses, could improve it sufficiently for viable operation in thin seams. A recent re-design of this equipment was proposed for thicker seams, but to date it has not been developed further.

Yarmac Miner

19. Only one machine was built and tried for a short longwall system, but problems of steering were encountered. This machine was scrapped some years ago, but sufficient detailed information remains to rebuild this equipment, if required (109).

Heading Machines

20. The Dawson Miller and Gullick machines were integrated heading machines to cut headings from 15 ft to 60 ft wide using a narrow web shearer system. The machines were scrapped but could be rebuilt if required (110 and 111).

Augers

21. Expertise in augering is to be found in the USA but development of new techniques using this principle would require considerable time and expenditure. Nothing is currently available that would be suitable for viable thin seam extraction underground.

Hydraulic Mining

22. The principles of hydraulic mining are now well known and the equipment is available. Its adaptation to thin seams has not proved eminently successful.

RELIABILITY

General

23. Some criteria of equipment used in thin seam systems and techniques can be measured quantitatively. Obvious examples of such parameters are bulk output and productivity, expressed in output per manshift. However, the reliability of

equipment is more difficult to measure accurately, though it is a most important aspect of the system, more so in thin seams, where the breakdown of equipment and its subsequent repair or replacement is often more inconvenient and costly than in a thicker seam. It is not uncommon in thin seams to have to resort to blasting and filling out in order to make sufficient space round a piece of equipment in preparation for its repair, either mechanically or electrically.

24. Three critical aspects of equipment reliability in thin seams are design, testing and maintenance.

Design

25. The basic design of thin seam equipment requires special consideration. The ideal design enables the coal cutting mechanism to be withdrawn into a more spacious area which will allow easier repair or maintenance.

26. The Collins miner satisfied this design requirement, as it could be withdrawn into the main development entry. A remotely-hauled shearer can often still limp to one or either end of the face by means of its external haulage housed in one of the gate roads.

27. Within the coal-getting machine itself, due attention must be paid to good design. Often, thin seams require the "miniaturisation" of existing equipment and this can mean slimmer shafts, gears and bearings. The remotely-hauled longwall shearer is a good example of how a basic function of the machine has been transferred from the cramped dimensions of the machine shell to the more spacious roadway, in this case by means of haulage. Again, the Collins miner featured this design logic by ensuring that as many functions as possible were carried out from the development roadway, such as launching the cutting head, etc.

28. Another important aspect of design is to know in the first instance what is worth designing. In the past there was a tendency for each small manufacturing company to design and develop its own brand of machine. The amalgamation and rationalisation of manufacturing companies reduced the variety of new designs, but made the ones that were developed more reliable. The NCB has a commendable system for new designs, whereby a series of committees, composed of senior operators as well as designers, decide the policy for new designs in the various fields of machinery development, such as face equipment, mine transportation, roadway drivage, etc.

29. There is a tendency in coal mining to design equipment very much stronger than seems necessary at first sight. This policy of over-designing, as it is sometimes termed, has been prevalent in the USA, especially on longwalls. The over-designing takes care of the excessive wear and tear caused by the underground usage of the equipment. The problem in thin seams is that the equipment is manufactured to a lighter and not a heavier design, in order to squeeze into the cramped conditions. This means that stronger materials must be used and better mechanical and electrical engineering design features developed.

Testing

30. Although much machinery is tested on a laboratory test bed before it is put to use in the mine, this is often no substitute for actual underground operation. Testing does, however, filter out some design weaknesses. Independent component testing has been followed by machine testing, and now systems testing can be carried out, to ensure compatibility between the various elements of the system. Examples are testing for correct fixing of the powered supports to the armoured face conveyor and of the power loader to the armoured face conveyor.

31. In the case of continuous haulage behind a continuous miner, there must be compatibility between the production capacity of the miner and the carrying capacity of the chain conveyor. This type of compatibility testing can be checked on the mine surface.

32. Another aspect of testing in thin seam mining is post-failure testing of the remaining parts of the failed element. A good example is the metallurgical testing of the links of conveyors that have persistently broken. Sample testing throughout the working life of a component can often predict the end of its useful life; non-destructive testing is normally applied to very large components.

Maintenance

33. Maintenance can be considered under two basic forms, preventive or breakdown. Preventive maintenance aims to avoid unplanned stoppages to production caused by the failure of mechanical or electrical components. Most mines carry out some preventive maintenance, even if it is only the simple, but effective, precaution of using a grease gun.

34. A detailed planned preventive maintenance system is now mandatory in UK mines. It consists of an individual check list for each item of mechanical and electrical equipment at the mine, and an organisation that ensures that the inspections are carried out and recorded in accordance with the check list.

35. The check lists detail the type and frequency of inspections. This can vary from a cursory, daily, outward inspection, to a major annual internal inspection.

36. The organisation consists of mechanics and electricians to carry out the inspections, office staff to keep the records and a senior engineer responsible for implementation.

37. Although this scheme of planned maintenance originated from the need to improve the safety of mine equipment, it now has beneficial effects on production performance.

38. A special investigation into downtime on face equipment was conducted by British Mining. The two areas of the NCB that mine most of the thin seams in the UK were examined. The table below shows average minutes lost per machine shift by class of delay for the 13 weeks ended 30th June, 1979, based on delays of 20 min or more:-

NCB Area	Power Loader	Face Conveyor	Geological (Weak Top)	Powered Supports	Coal Clearance	Total
North East	20.6	12.5	22.2	3.1	14.0	77.3
Barnsley	15.4	10.3	39.1	2.7	15.9	98.1
UK total	17.3	13.3	27.5	3.1	15.0	90.1

39. The data are inconclusive. On the coal face most equipment downtime is either on the power loader or on the armoured face conveyor; the table shows one area is above the UK average and the other below.

40. Any planned preventive maintenance system is only as effective as the engineers who carry out the inspections. Sometimes, owing to insufficient training or other reasons, their weekend inspections, especially internally on the power loader, appear to induce breakdown on a Monday morning. This has led to the view that it is best to leave well alone and to rely on the complete cyclic replacement of equipment, such as power loaders.

41. For thin seam equipment this viewpoint could be especially applicable where it is difficult to carry out detailed inspections in the cramped conditions.

CONCLUSIONS

42. The reliability of modern coal face equipment is much improved. Many of the purpose-designed thin seam techniques developed in the 1950s and 1960s used equipment that was not reliable. One reason was the wide variety of machine designs. The subsequent standardisation on a basic design of the continuous miner, the shearer and the plough has been a major factor in the improvement of the reliability of these machines.

43. However, new basic engineering developments in the fields of hydraulics, electro-hydraulics, electronics, etc, make it possible for the designer to build reliability into new equipment.

44. Reliability must be a major design feature of any new thin seam equipment.

CHAPTER XII

STATE-OF-THE-ART - PRODUCTIVITY

GENERAL

1. Coal mining productivity is normally measured in terms of output per manshift (OMS) and, less frequently, in output per manhour; the measure of output per machine shift, though not strictly one of productivity, is also used. The output can be expressed either in run-of-mine or clean tons. The difference can be significant, as, with a metallurgical coal, 50% of the run-of-mine coal can be discarded during coal benefaction. The number of manshifts is the divisor in the OMS ratio and these can be obtained from a variety of manpower categories. The most common are manshifts worked at the coal face, section manshifts, the underground total, and the mine total, which includes surface activities. In all cases, overtime shifts are included in the divisor.

2. It is apparent that meaningful comparisons between mines, companies and especially countries, can be difficult when there are wide variations in the run-of-mine coal discard and in the definitions of the manpower categories.

3. A further misrepresentation can take place when different systems are compared, such as advancing and retreating longwall. Frequently the two are compared without making due allowance or adjustments for the labour expended in the pre-development of the access roads in the case of retreat.

4. Finally, the productivity of a section or a complete mine can be completely at variance with its counterpart nearby. Reasons for this could include variances in geological conditions and widely differing mine management abilities.

5. All these factors can affect productivity and this means that even published data of outputs per manshift can often be misleading.

6. The level of productivity in underground operations in the USA has changed considerably over the years. Increases in the 1960s corresponded with the introduction of new mining machinery and techniques, but in more recent years the decline has coincided with the Federal Coal Mine Health and Safety Act 1969 (145). The productivity of the average US underground mine has reduced from a peak in 1969 of 15.6 tons per man-day to 9.10 tons per man-day in 1976 (66).

US THIN SEAM PRODUCTIVITY

Factors Affecting Productivity

7. In thin seams a greater area of ground has to be mined in order to extract an equivalent tonnage to that from thicker seams. Many of the tasks that have to be performed in an underground mine are related to linear advance, and so for a given output they must be carried out at more frequent intervals in a thin seam.

Examples of this type of work are the extensions of rail track, conveyor lines, water and power lines. Other tasks, such as cleaning spillage and rock dusting, are directly related to area extracted and not tonnage mined. These aspects militate against high productivities in thin seam room and pillar mining. They are further aggravated by the reduced capacities of low-profile continuous miners.

8. The effect of seam thickness on productivity is demonstrated by the table below (145):-

Seam Thickness	Productivity of Underground Bituminous Mines (Tons/Man-Hour)		
in	1948/52	1953/65	1966/69
less than 25	0.49	1.04	0.77
25 to 36	0.55	0.93	1.53
37 to 48	0.64	1.07	1.77
49 to 60	0.74	1.35	2.20
61 to 72	0.79	1.47	2.22
73 to 84	0.85	1.52	2.35
85 to 96	0.93	1.46	2.33
97 to 108	0.96	1.88	3.03
109 or more	0.88	1.42	2.39
All mines	0.73	1.26	2.06

9. For the 1966/69 period, the productivity of 1.53 tons per manhour for mines working thin seams of 25 in to 36 in probably represents an OMS of about 10 tons; this assumes a time available at the face of 6.5 hours. Although there has been a substantial decline in the productivity of all mines since 1966/69, it is interesting to record that many thin seam mines still operate at 10 tons per manshift. This has been verified from recent mine visits.

10. It is probable that, despite mining thin coal, small adit mines developed from the hillside have minimised the decline in productivity that has overtaken the industry in general. The reasons for this are likely to be better mining conditions, tighter supervision and higher morale.

Thin Seam Examples

11. The trend in small, thin seam mines is illustrated by several articles on individual mining operations, which record high levels of productivity. At the No 11 mine of Pyro Coal Company, West Kentucky, 500 tons per section shift are quoted from a 42-in seam using the conventional room and pillar methods (42 tons per manshift). In the same mine section, when the seam was 10 in higher at 52 in, 700 tons were achieved with the same 12-man crew (33).

12. Solar Fuel Company, Somerset, Pennsylvania, is recorded as producing up to 800 tons per section shift using a remotely-controlled continuous miner in the 42-in Upper Freeport seam. Continuous chain haulage was used and the average productivity is quoted as 25 tons per manshift, although it is not clear if this figure refers to the section or includes outbye labour (34).

13. At Kessler Coals Inc, Whitesville, West Virginia, production from the 36-in thick Peerless seam varies from 4,500 tons to 5,500 tons per section, for a 22-working-day month. Assuming a crew of nine men, which is a normal figure for continuous miners using continuous haulage, this equates to a section OMS of between 23 tons and 28 tons.

14. A recently-developed mine in West Virginia, owned by Marson Coal Company, is extracting the 36-in to 40-in thick Sewel seam and has recorded outputs per section shift of 425 tons to 450 tons. These high tonnages have been produced by a section crew of only seven men, which is equivalent to 60 tons per manshift.

15. Further examples of high productivities are available, but often these are exceptions to the average performance in thin seams, which tends to level off at between 120 and 150 tons per section shift from 30 in to 36 in of coal.

Productivity By Machines

16. The general decline in productivity applies not only to productivity expressed in terms of tons per manshift, but also in tons per section shift. Two interesting aspects of machine productivity in the USA are, firstly, the performance trend of each machine or method over recent years and, secondly, the comparison between machine types. The table below illustrates this (82).

Average Output/Section/Year
(1,000 Net Tons)

Year	Continuous Miners	Longwalls	Conventional
1972	96	194	60
1973	96	189	70
1974	87	181	60
1975	84	182	59
1976	79	197	52

17. The productivity of the continuous miner has dropped by nearly 20% over this period and a reduction of 13% has taken place in conventional sections. Production levels from longwalls have remained fairly static; this may indicate that longwalls are more able to work efficiently within the ever-tightening framework of federal mining legislation.

FOREIGN PRODUCTIVITY

UK Productivity

18. The bulk of the UK's coal production is obtained by longwall mining and productivity is expressed in terms of face OMS, which includes supervision and maintenance shifts. Section productivity is rarely used, but the other commonly-quoted figure is the overall productivity for the mine, which includes the surface activities. Within each mine, the efficiency of different operations is controlled by a parameter called manshift per 1,000 tonnes.

19. In 1978 the average clean output per annum from each longwall operating in heights of less than 36 in was approximately 100,000 tonnes and the number of mechanised longwalls in this seam thickness was 42 (15 shearers, 11 trepanners and 16 ploughs). This annual output per thin seam face can be compared favourably with the UK average for all faces, which is approximately 150,000 tpa.

20. It can be concluded that a well-designed and well-managed wall in a seam of around 34 in to 36 in can produce comparable productivities to the average UK face.

Individual UK Examples

21. Thin seam extraction tends to be concentrated in certain areas of the UK. The table below shows OMS figures for the best thin seam faces in the Barnsley Area over a three-month period in 1975/76:-

Mine	Seam Thickness	Working Thickness	OMS	Machine
	in	in	tonnes	
Woolley	31	35	10.2	remote hauled shearer
Wentworth	25	30	10.6	remote hauled shearer
North Cawber	27	34	15.2	trepanner
North Cawber	30	35	22.5	in-web shearer
Ferry Moor	39	44	33.8	shearer

22. Other longwalls in the UK have produced comparable figures. More recently a trepanner has consistently produced 1,500 tonnes of clean coal per day from a 34-in seam.

Productivity in the Western European Countries

23. The overall underground productivity, measured in tonnes per manshift, in Western Europe has increased from 1.63 tonnes in 1958 to 3.82 tonnes in 1977. This has coincided with a decline in total output from 252 million tonnes to 120 million tonnes, caused by the wholesale closure of uneconomic mines, many of which were mining the thinner sections of coal. The closures have resulted in the total suspension of coal mining operations in the Netherlands and Italy (27).

24. So little thin seam mining is currently carried out in Western Europe that examples are rare. The best documented operation is the Dutch-owned Sophia Jacoba mine in the German district of Aachen. Here a total of 1.9 million tpa of clean coal is produced from six longwalls which average about 1,350 tpd from seams of between 33 in and 44 in thick (1). The faces are equipped with ploughs that can operate in a minimum height of 27 in. The gross tonnage is virtually halved to produce the clean output, yet this still allows an overall OMS of over 3 tonnes to be achieved.

USSR Productivity

25. The productivity of the USSR coal mining industry is low and the figures that are available must be interpreted with caution (105). The figures normally relate to run-of-mine coal and the labour base merely reflects the actual "production workers".

26. The overall productivity from "production workers" is given as 52.6 tonnes per man-month for underground operations, with a working year consisting of 330 shifts. This is equivalent to an OMS of about 2 tonnes. However, it is estimated that the manshifts and tonnages used in the calculation inflate the published figures by a factor of three. For comparative purposes, this reduces the overall OMS to between 0.7 and 0.8 tonnes.

27. In the thin seams productivity is lower still. Professor Astakhov, in the geological appraisal of the major coal basins in the USSR, states that, whilst the Donbass, the major centre for thinner seam working, produces 32% of the nation's output (40% hard coal equivalent) the same region consumed 47% of the basic funds of the coal-producing industry (23).

28. Professor A.V.D. Spochiniskii, Mining Institute (92), presents graphs of productivity against seam height. These show that for shearers the productivity increases from 9 tonnes per manshift in a 39-in seam to 14.5 tonnes per manshift in a 117-in seam, where powered supports are used. Where the equipment is used with hand-set supports, productivity increases from 5 tonnes per manshift at 31 in to 10.6 tonnes per manshift at 117 in, although the increase is not linear with seam height. He also claims that if the working of thin seams were terminated, then the productivity of the industry would increase substantially. Thin seams could be then worked in the future, when more suitable technology was available.

29. An example of a highly-productive thin seam longwall is described by V.M. Ivashen (90). The average productivity for March to December, 1976, is quoted as 314 tonnes per man-month in a working height of 25 in to 33 in. The 623 ft long face, in level conditions, had a strong roof and floor. The equipment on the face consisted of a shearer, an armoured face conveyor and powered supports and could be regarded as a normal mechanised face for Western Europe. The average daily tonnage was over 900 tonnes with peaks above this, and the productivity, in terms of OMS, equated to over 12 tonnes for a 25.5-day month.

30. It would appear that the average USSR productivity is much lower than that of Western Europe and that high bulk outputs are achieved by employing a large amount of labour. No doubt other examples exist of more productive operations.

CONCLUSIONS

31. In all countries mining thin seam coal, there would appear to be a correlation between seam thickness and labour productivity. There are, however, examples of thin seam faces operating at high productivities. In these cases the mining conditions are usually more favourable.

32. The good results that can be obtained reflect the production potential of mining equipment, something that is not normally reached on the average installation, due to factors such as difficult geological conditions, poor equipment maintenance and downtime on the transport systems.

CHAPTER XIII

STATE-OF-THE-ART - COSTS

GENERAL

1. The subject of mining costs is a complex one, as mining companies adopt their own systems of costing and assign different items of expenditure to different categories. This makes the comparison of costs between companies difficult. In a country where the coal industry is nationalised, the costing system is standardised but comparisons between countries can still be difficult. A basic problem is the assessment of the real values of cost data. For example, in 1973 the "official" exchange rate of the USSR rouble was 1 rouble = US$1.34, yet the estimated buying power of 1 rouble, relative to prices in the USA for hard goods and food, was between US¢20 and US¢50 (105).

2. A common breakdown of operating costs is direct labour, materials and administration. In 1979, in the NCB mines, the proportions in each category were: direct labour 46%, materials (including power, heat and light) 32% and administration 18%. The remaining 4% was depreciation.

3. The operating costs may not, however, include the cost of capital moneys such as interest on loans, etc. This can be a substantial cost item when major new mine development takes place or expensive heavy machinery is bought.

4. In common with other heavy engineering plants, a coal mine has large fixed costs and a relatively small proportion of variable costs. This feature of a mine makes it imperative that output targets are achieved, because nearly all the profits come from marginal tonnage, ie tonnage mined over and above the base tonnage.

5. In the State-controlled economies of some Eastern European countries, costs are unreliable, as many of the charges are not the true cost of an operation, but are manipulated for other reasons.

US COSTS

6. Costs in the USA are the responsibility of the individual mining companies and, for the most part, are confidential company data. Several exercises have been carried out, under the auspices of US federal agencies, to estimate the costs of different mining systems, in order to determine the most economic.

7. The 1977 and 1978 increases in the price of coal coincided with an upsurge in the number of small operating mines, which tend to work the thinner seams (82). Even more recently, however, the market price of coal has dropped dramatically and, as a result, the total small mine production has been reduced. The close correlation between mining costs at underground mines and market price is due, in general, to the smaller profit margins at underground mines compared with surface

mines. This is borne out by an examination of the average value of coal per ton fob mine for underground ($26.28 in 1975) and surface mines ($13.43 in 1975), and the average value as sold on the open market ($18.02 in 1975) and in term contracts ($26.99 in 1975).

8. In the US Bureau of Mines' study, "Comparative Shortwall and Room and Pillar Mining Cost", costs are evaluated for working a 42-in to 54-in thick seam (74). After a comprehensive and detailed examination of costs, the report concludes that it is reasonable to assume that the extra marginal tonnage from the shortwall more than offsets the larger initial capital cost of equipment.

9. Although considerations other than cost can lead, in certain instances, to the preference for longwall, the basic financial requirement is to recoup the higher initial capital costs by increases in marginal tonnage.

10. Discounted cash flow analysis is commonly used in the evaluation of new mine projects and an example of this approach is detailed in the US Bureau of Mines' information circular 8689 (72).

Application to Thin Seams

11. The typical thin seam underground mine is small; in the period 1966/69 it produced only 22,000 tpa with 10 men (145). The funding of a mine of this size is much less onerous than for a large mine, and this tends to encourage the formation of small mining companies. Information obtained verbally from a recent visit (1979) to a mine, which was extracting a 30-in seam with only 35 men, indicates that a large proportion of the mine's operating cost of $37/ton consisted of overhead administrative charges. These were allocated by the company, which, in this instance, was large and operated several other mines.

12. In general, the problem of funding mine development is an acute one, and this will tend to perpetuate the existing trend for small mines in thin seams, whether they be owned by small or large mining companies.

UK COSTS

13. The nationalised UK coal industry is managed by the NCB on behalf of the State. It is divided into "areas" which are responsible for between 15 and 20 mines in each coalfield.

14. A standardised system of costing is used throughout the industry and this provides for common services and administration charges to be allocated on a proportional tonnage basis.

15. When the industry was nationalised in 1947, the legislation reflected a policy of supplying the nation's requirements for coal without making either a profit or a loss. In practice, this policy allows some mines to run at a loss provided others make a profit, and it has assisted in maintaining even the limited amount of thin seam extraction currently undertaken.

16. The following table shows the overall breakdown of operating costs in the UK for the financial year 1978/79:-

	Total Cost		Cost Per Clean Tonne		% of Total Cost
	£10^6	$$10^6$	£	$	
Wages, including allowances in kind	957.4	1,914.8	7.88	15.76	32
Wages, charges	401.4	802.8	3.30	6.60	14
Materials and repairs	589.4	1,178.8	4.85	9.70	20
Mining and civil engineering contract work	229.2	458.4	1.89	3.78	8
Power, heat and light	117.2	234.4	0.96	1.92	4
Salaries and related expenses	81.1	162.2	0.67	1.34	3
Other operating costs	233.7	467.4	1.92	3.84	8
Overheads and services	214.4	428.8	1.77	3.54	7
Depreciation	105.0	210.0	0.86	1.72	4
Total costs	2,928.8	5,857.6	24.10	48.20	100
Operating profit	55.6		0.46		

Exchange rate: £1 = $2

17. No detailed costs are currently published for thin seam mines in the UK, but a representative figure for an efficient mine extracting a 36-in seam in 1979 would be between $30 and $40 per tonne. It would produce 500,000 tpa with 500 men at an overall OMS of about 5 tonnes.

18. The actual selling price of coal in the UK and in other Western European countries is usually less than the cost of production. Government subsidies reduce the cost of coal to the consumer; these can be either in the form of regional assistance, to provide employment, or aimed at specific markets, to encourage the use of coal in preference to other forms of energy. The following table gives the subsidies per tonne of European coal production in 1978 (150):-

	£	$
UK	1	2
Germany	11.9	23.8
France	14.7	29.4
Belgium	24.1	48.2

Exchange rate: £1 = $2

Regional Costs

19. Because of its relevance to thin seam mining, the technical aspects of the Collins miner are discussed in detail elsewhere in the report. However, the cost data, published in the mid-1960s, are of little value, except, possibly, the comparison with a contemporary shearer face in the same mine.

	Collins Miner	Longwall Shearer
	£/tonne	£/tonne
Proceeds	4.30	4.23
Development	0.69	0.03
Installation	0.05	0.02
Wages	0.60	0.58
Materials	0.20	0.20
Depreciation	0.11	0.09
Total cost	1.65	0.92

For the Collins miner, the table indicates the high depreciation cost arising from the cost of the miner and an apparently excessive development cost for roadway drivages. From practical experience of thin seam mining, it is difficult to understand the very low figures for the longwall shearer in both these categories. It is suggested that updated figures for both systems could be comparable.

20. In thin seams, the effect of mining either roof or floor, in order to achieve a minimum working height for access of men and machinery, results in a dilution of the coal product. Where the product is sold untreated directly to a power station utility, the price received is considerably reduced by the increase in ash content. The following table shows the effect of mining from 0 in to 6 in of dirt within an extracted height of 36 in (5):-

Thickness of Coal Extracted	Thickness of Waste Extracted	By Weight Ash	Market Price/Tonne October 1976	
in	in	%	£	$
36	0	19.0	14.30	28.6
35	1	21.4	13.70	27.4
34	2	23.8	13.40	26.8
33	3	26.0	13.00	26.0
32	4	28.3	12.50	25.0
31	5	30.1	12.10	24.2
30	6	32.6	11.60	23.2

Exchange rate: £1 = $2

21. The loss in proceeds from £14.30 to £11.60 ($28.6 to $23.2) has a marked effect on the economics of thin seam mining. Even if a preparation plant were installed to upgrade the coal, the extra costs of washing and the loss in bulk tonnage, would still adversely affect the economics of mining. Thus, in a thin seam the choice of working height is determined by a number of factors, not least of which is an economic evaluation of the coal produced.

USSR COSTS

22. The total cost of coal production in the USSR is not available, owing to the lack of published statistics and the method of presentation of published information (105). There are, however, several indicators that show that USSR coal is expensive in comparison with other world producers.

23. The low overall labour productivity is a major indicator of high mining costs. In terms of manpower, 1.2 million persons are involved in the production of less than 400 million tpa of deep-mined output.

24. Soviet literature frequently quotes information in comparative terms. For example, the cost of mining in the Donbass region, the principal thin seam area, is higher than the USSR national average. This coalfield produced, in 1975, 32% of the country's coal at an average height of 43 in, but used 47% of the country's basic funds allocated to coal mining (23). This is attributed directly to thin seams and deeper workings. The higher costs are offset to some extent by the better quality of the Donbass coal; the bulk tonnage amounted to 32% of the total national output, yet represented 40% of the country's coal energy production.

25. The actual face extraction cost is termed the "prime" cost of production and the paper "The Working and Total Mechanisation of Thin Coal Seams" illustrates graphs of the prime cost in varying seam thicknesses (92). At a working height of approximately 31 in this varies from 3 roubles/tonne to 3.6 roubles/tonne, and decreases as working height is increased. The prime cost, however, does not include the development cost for the working faces, as it is stated that, for thin seams, the development cost per tonne approaches the prime cost. To the prime cost and the development cost must be added all other mine costs, including surface costs and overheads, in order to get the true cost of USSR coal.

26. The tremendous expansion of USSR production in the remote eastern regions of the country is an indication of the high cost and difficulties that are features of thin seam mining in such coalfields as the Donbass.

CONCLUSIONS

27. In countries where mechanised mining is practised, the production costs in thin seams are higher than in thicker seams. Within the thin seam category, large cost variations occur depending on the mining conditions and the face equipment available.

28. In the main, the high costs of thin seam mining are offset by the higher selling price of the coal, but also regional employment problems can often maintain uneconomic thin seam production, and, in some cases, the proximity of thin seam mines to the centres of energy consumption provides sufficient savings in transport costs to keep the mines competitive.

CHAPTER XIV

STATE-OF-THE-ART - HEALTH, SAFETY AND ENVIRONMENT

GENERAL

1. The term safety is used to denote an absence of risk or hazard. Hazards that result in physical injuries are easier to identify than those that affect the health of workers. The reason for this is that an injury normally occurs as a result of some violent event, and the object that caused the accident can be directly identified. The injury risk of working in a particular industry, or occupation within an industry, can be assessed statistically in terms of accidents per recorded hours of exposure. The relationship between accidents and seam height is discussed in Chapter III of this report.

2. Health hazards, however, are not always as apparent as physical danger, such as might arise from a weak roof at a coal face. In many cases the detrimental effect on health takes place over a period of years, and, until some loss or impairment of body function has occurred, the employee may not be aware that the process is taking place.

3. The more obvious hazard to health is that affecting the respiratory system, ie "black lung" or pneumoconiosis. In thin seams another health problem is beat diseases, which are caused by working and travelling in unnatural positions. Other environmentally-related health problems are those associated with working in close contact with water or oil, the danger to the eyes from particles picked up by high air velocities, noise and poor illumination.

RESPIRATORY HEALTH

4. Hazards to respiratory health in coal mining come mainly from the inhalation of respirable dust particles. These are considered to be 5 microns in diameter or less, as it has been found that larger particles will not reach the transfer surfaces of the lungs (136). Medical research has shown a very strong correlation between the respirable dust concentration in grams per cubic metre and the incidence of pneumoconiosis. Several aspects are still being researched, including the combined effect of dust and noxious gases and the chemical composition of the dust (137).

5. US legislation lays down a maximum dust exposure of 2 mg/m^3 of respirable dust throughout the working shift. The basic standard in UK mines is a maximum of 7 mg/m^3 measured at the coal face; no measurements are made of individual exposure over the whole working shift.

Application to Thin Seams

6. The general remarks on the relationship between health and dust apply to all seam sections. However, the problem can be more acute in thin seams owing to higher velocities. In the USA, a minimum of 9,000 ft^3/min of air is required at the last through crosscut, but this requirement is not related to seam height. A consequence of this is that in thinner seams, increases in velocity are needed to supply the required quantity of air. In some thin seam mines the dilution of methane to the statutory level of 1% can also contribute to high ventilation velocities.

7. Normally, an increase in the air quantity dilutes the concentration of respirable dust. However, higher velocities can produce a counter effect by causing dust pick-up (136). As the velocity is increased, a cross-over point is reached, where the increase in quantity actually increases the dust count. The actual velocity at which the cross-over occurs varies with the source and the type of dust. When the dust is dry, velocities above 2 m/s cause appreciable pick-up but, when the dust is wet, higher velocities above 4 m/s can be tolerated. The size of the particle also affects the pick-up of dust; within certain limits, large-sized particles are more easily swept into the air than the smaller ones, which adhere to surfaces.

8. A further feature of dust pick-up in thin seams is the severe restrictions in cross-sectional area that can be caused by items of equipment in the roadways. An example is a longwall face where the cross-section is reduced owing to the seam height. In the vicinity of the cutting machine, the area is further reduced, causing funnelling of the air and an increase in velocity at the point where freshly-cut coal is first exposed to the ventilation. It is, thus, particularly important, in a thin seam, to fit and operate adequate dust suppression equipment on the longwall power loader. This consists of pick face flushing and venturi-type sprays for shearers and trepanners. A novel development is a miner's helmet incorporating a curtain of clean air. However, the larger dimensions of this helmet would make it more difficult to wear in thin seams.

BEAT DISEASES

9. The beat diseases are more common in thin seams because of miners working on their knees and elbows. The use of correctly fitting, comfortable knee pads has reduced the incidence considerably. The most significant improvement in the future is likely to come from the widespread adoption of remote control equipment.

WATER

10. In thick and medium seams, water on the floor is merely a nuisance. In thin seams, however, the problem is more severe when miners become sodden from crawling on a wet floor. The widespread use of hydraulics in face machinery

introduces the risk of skin diseases such as dermatitis, and spillage must be kept to the minimum. Most hydraulic fluids are considered safe from a medical standpoint but protective gloves are still recommended.

11. It is difficult to evaluate the effect of working in water on the general health of the miners. Such complaints as colds, influenza and rheumatism can develop. Where the ventilating air current is cold, this can accentuate the problem as the miners travel out of the mine at the end of their shift.

12. The health hazards caused by water in the mine make it imperative that every precaution be taken in order to confine the strata water to ditches and pipes. In the case of dust suppression water, the equipment must be maintained to avoid spillage and leaks.

13. Besides effective mine drainage, using pumps, the layout of the mine can encourage water to drain naturally away from the face. This may not always be practicable owing to constraints such as the position of the surface access and the cleat of the coal. The latter consideration may require that the faces travel to the dip, which is not usually conducive to good mine drainage.

NOISE

14. In the 1969 Federal Coal Mine Health and Safety Act, limitations were set for the permissible noise levels in the working environment. The Act lays down that, where the noise level at the operator's position exceeds 90 dba, the period of exposure must be limited, and, where an operator is subjected to differing sound levels during his shift, the cumulative effect over an eight-hour period must not average more than 90 dba.

15. The purpose of the legislation is to prevent ear damage and is based on medical research. Experiments have been conducted to reduce the noise level produced by thin seam mining equipment in the USA (47). These demonstrated that an auger-type continuous miner, which, prior to modification, was not permissible for a full shift's operation, could, by the addition of sound-absorbing materials, be made permissible.

Protective Devices

16. It is possible for operators in most industries to be protected from excessive noise by wearing ear muffs. This is not so practical in coal mines, particularly in thin seams where the wearing of additional items of equipment tends to hamper mobility.

Remote Control

17. The advantages of remote control are obvious, in the case of noise problems, as the operator is physically removed from the source of sound.

ILLUMINATION

18. The need for effective illumination has long been recognised in the coal industry. A UK Government report of 1945, called the Reid Report, made recommendations for minimum standards (140). Today, these are incorporated into the legal specification of cap lamps rather than minimum levels of illumination for the general working area.

19. In the USA, regulations have been proposed that require all surfaces in the normal visual field of a miner whilst operating mobile machinery to be illuminated to an intensity of 0.06 ft lamberts (38).

20. Thin seam working imposes additional problems in meeting the legal requirements owing to the generally wide working places and the restricted space available for the fitting of suitable lights on machines.

21. Whilst the normal miner's cap lamp is usually sufficient to eliminate the health hazard caused by excessive eye strain, improved levels of illumination are beneficial in the recognition of potential injury hazards.

CONCLUSIONS

22. Although thin seams have separately identified health hazards, such as arise from working in water, the risks from pneumoconiosis and excessive noise have been reduced in line with the general trend in all seams.

CHAPTER XV

STATE-OF-THE-ART - DESCRIPTION AND COMPARISONS OF MINING SYSTEMS AND WINNING METHODS

GENERAL

1. There are a great many systems available for mining coal, and a wide variety of machines and methods for winning the in-situ coal from the solid.

2. The interface between mining systems and winning methods is portrayed in Table VII. This represents a classification of systems and methods, with the object of identifying feasible designs that may be suitable for US thin seam conditions.

3. Mining systems are sub-divided into room and pillar, room and pillar with pillar extraction, longwall, shortwall and augering.

4. Winning methods are sub-divided into hand got, blasting off the solid, cut and blast, manual control machine mined, remote control machine mined, total remote extraction and hydraulic.

5. A matrix has been formed for combinations of mining systems and winning methods; these have been numbered 1 to 30. Each design has been given a code: C for commonly used, L for limited or historical use and largely discontinued, P for possible use but to date not practised owing to adverse economics or lack of technology, and, finally, U for unlikely or impossible.

MINING SYSTEM

Room and Pillar

6. In the room and pillar system coal is extracted from a series of narrow entries and the strata above the working panel are supported by pillars between the roads.

Pillar Extraction

7. Where pillar extraction is practised, panels are developed using the room and pillar system, and the pillars are extracted systematically, allowing the roof to cave. Support for the pillaring operation is usually in the form of individual timber posts or hydraulic jacks, which provide temporary roof support only. In addition, temporary supports are used to control the waste edge.

8. Frequently, during pillar extraction operations, the roof collapses prematurely causing portions of pillars or whole pillars to be left behind in the waste. In normal conditions, 10% to 15% of the plan area within a panel is lost owing to roof control problems (74).

Longwall

9. In the longwall system coal is extracted from a long face or wall, generally up to about 900 ft in thin seams. The roof adjacent to the wall is supported, but, as the wall is worked, the supports are moved forward and the area behind is allowed to cave. In some special cases, generally to reduce the effect of strata movement or surface subsidence, the waste area may be packed or stowed, but there is still some lowering of the roof.

Shortwall

10. The shortwall system combines many of the features of room and pillar and longwall. The area to be mined is developed by room and pillar equipment, but the blocked-out pillars are larger. The pillars are extracted by taking wide slices off a relatively short wall, the roof being controlled by heavy-duty powered supports of the type commonly used on longwalls. The roof is allowed to cave behind the supports.

11. A major advantage of the system is that the same equipment, with the exception of the powered supports, can be used in both the development and pillar extraction phases of the operation.

12. The length of a shortwall is typically 180 ft (65), so the capital cost of the supports is lower than for a longwall. On shortwalls, the width of the continuous miner requires a wide span of roof to be supported between the face and the powered supports, thus making strong roof conditions a prerequisite for the system.

Augering

13. In this method coal is won by boring either circular holes or narrow slots from a pre-developed roadway. Owing to the very short roof span of the hole or slot, the excavations are self-supporting. Parallel holes or headings are driven from the development entry, leaving ribs of coal between the holes to provide support. The ribs themselves can be designed wide enough to provide general support, or the holes can be bored in groups, leaving wider ribs between the groups of holes, so allowing a measure of crush to occur between individual holes (2).

WINNING METHODS

14. Coal may be broken from the solid in a variety of ways: by hand or with pneumatic picks, blasted from the solid, undercut and blasted, machine mined or hydraulically mined. A variety of equipment is available for winning the coal, from simple ploughs to power loading machines of the continuous miner and shearer loader type.

15. After the coal has been broken it must be loaded on to the haulage system. In the case of hand-got operations the coal is normally loaded by shovels into small cars, or on to a low-capacity portable conveyor.

16. Where the coal has been broken by explosives or has been cut and blasted, it can be loaded by hand, by machine or, in steep gradients, by gravity. Where the coal is wholly machine mined, it is loaded by the machine into cars or on to a conveyor system. In hydraulic mining, the coal is mined by high pressure water jets, then washed from the face into a flume or pipe, which transports it to the pumping station from where it is pumped out of the mine.

COMPARISON OF DESIGN COMBINATIONS

General

17. Most of the combinations of mining systems and winning methods that have been found in the literature survey are included in the classification shown in Table VIII. Omissions have been made where it is considered a system or method is not a possible candidate at this time or where, by definition, it is impossible, such as hand-got augering.

18. Each design combination has been evaluated in accordance with the following criteria: safety, minimum height, normal working gradient, maximum depth, the effect of weak roof and weak floor, methane, spontaneous combustion risk, extraction ratio, productivity, capital cost and operating cost.

19. Every attempt has been made to include quantitative evaluations, but in many instances this has not been possible and in those cases less well defined gradations have been used, such as good, medium, poor, etc.

Safety

20. The safety ratings of the design combinations range from low, in the case of hand-got working, where a miner may be required to work at the coal face with wooden supports, to very good for entirely remote systems. The rating for mechanised longwall workings using shield supports, which provide virtually a continuous steel cover, is considered to be very good. It has been assumed that room and pillar workings are supported by roof bolts and, for this reason, are classified as medium. This may be too low a rating for a very typical US system, but mine visits have highlighted the unpredictable performance of roof bolts in some strata and the roof falls that result.

Minimum Height

21. The minimum heights are dependent on local strata conditions and ground movement. In very good conditions the lower limits quoted may be reduced still further by 1 in or 2 in.

22. Hand-filled room and pillar mining is still carried out in heights of 18 in to 20 in. Where mechanical loading is practised, more height is required as the lowest available gathering arm loaders have a minimum height of approximately 24 in.

23. A minimum working height of 28 in is required for continuous miner room and pillar systems using the auger-type Jeffrey or Wilcox machines, but 30 in is more normal in practice.

24. Hand-loaded longwalls have been worked in the 14-in to 16-in range in the UK, but at very low productivities. A conventional mechanised wall, fully equipped with a shearer, powered supports and armoured face conveyor, normally has a minimum working height of about 30 in. However, there are recorded instances of faces down to 24 in (187).

25. Scraper boxes can work in very low heights, down to 12 in, on an unmanned, unsupported face. The limiting height required for men to set supports on the face is about 16 in to 18 in. Ram ploughs and chain saws, designed for steep workings with no conveyor, again require a similar minimum height for men to set supports but the USSR technique of remote wedge setting could reduce the required height still further.

26. The limiting height for shortwalls is assumed to be in the region of 40 in, owing to the combination of heavy-duty powered supports and continuous miners; however, none are known to be working in this seam height.

27. A simple auger has no lower limit in terms of hole diameter but production and productivity would fix an economic limit.

28. The thinnest extracted section using a Collins miner was 30 in. A machine was proposed for working an 18-in to 20-in seam but the design was abandoned before a machine could be built.

Normal Working Gradient

29. Nearly all mining systems work most efficiently in flat conditions, except where gravity or water flow are used to remove the coal from the face. In general, gradients adversely affect the mobility of equipment and impose additional strain on men who have to move with the operations. This extra effort and the difficulty of handling material tend to reduce productivity, especially in low seams.

30. In room and pillar, pillar extraction and shortwall systems, where free-steered mobile equipment is used, gradients have a severe effect and rule out battery-powered haulage equipment owing to the heavy power demands. The maximum practical gradient for mechanised room and pillar workings is about 10^0, although some equipment will operate in steeper conditions but at a reduced productivity level.

31. Gradient is not so critical with longwall systems as most of the equipment is securely anchored to the powered supports. However, a special design of powered support is required in seams steeper than about 18^0 and with this equipment inclinations of up to 50^0 can be mined successfully.

32. Augers and similar systems are not severely affected, though the thrust and handling devices would require to be specially designed for steep gradients. A theoretical maximum of 90^0 is possible, which amounts to a system of vertical boreholes.

Maximum Depth

33. Strata pressures normally increase directly with depth. This tends to cause the weak strata, found in coal measures, to deform and break up. In room and pillar systems, this effect is minimised by progressively increasing the size of the pillars. A maximum depth, for normal working of this system, is quoted as 1,000 ft, although greater depths are known.

34. In longwall working, extreme depth can exclude retreat mining, owing to deformation of the pre-developed roadways. However, this system has been used successfully in the UK at depths of over 3,000 ft. Advancing longwalls commonly operate at depths of almost 4,000 ft in Europe. The roadways are often protected by destressed strata from previous workings, but maintenance costs can still be heavy.

35. In the case of augers and other partial extraction systems, increasing depth would mean larger ribs and other protective pillars, reducing the extraction ratio.

Effect of Weak Roof

36. Weak roofs tend to add cost and lower the efficiency in all mining operations. Least affected are the remote auger systems, which, owing to very narrow roof spans, do not require support of the production drivages; however, support is required for the access roads.

37. Room and pillar workings can be badly affected by weak tops necessitating a narrowing of the working place in order to maintain the integrity of the roof. As a result, efficiency is reduced because of tramming difficulties and more moves from place to place. In thin seams, a weak roof stratum is often cut out in the mining operation and this can produce coal preparation problems and, consequently, adversely affect the economics of this operation. Longwalls, supported by shields, can frequently operate under a weak roof.

Effect of Weak Floor

38. In room and pillar workings, weak floors have an adverse effect on the mobility of rubber-tyred and cat-mounted equipment, especially in wet conditions. Larger pillars may be required to reduce floor heave. In the case of longwall workings, special care must be taken to select powered supports with a base area large enough to avoid floor penetration. Bottom plated armoured face conveyors are necessary to prevent the bottom strand of the flight conveyor gathering up floor dirt and jamming. Weak floors, especially at depth, can cause a loss of working height owing to floor heave which can produce problems on the wall when

mining operations have been suspended for any length of time, and convergence prevents the passage of the shearer under the powered support beams. Weak floors can also cause severe difficulties in the horizon control of ploughs and scraper boxes.

39. The nature of the roof of the seam can also affect the behaviour of the floor. A sandstone roof tends to produce more serious problems when the floor is weak.

Effect of Methane

40. Strata movements can increase the emission of methane into the mine atmosphere. In longwall and shortwall mining, methane drainage may have to be practised by boring holes up and over the waste and collecting the methane under suction. Another method of methane drainage practised in some parts of the USA is to drain the gas from behind the longwall by means of pre-bored surface holes. In room and pillar workings, ventilation can be more complex and, in consequence, the use of auxiliary fans and other devices is necessary. Ventilation pipes and flexible tubes can be difficult to accommodate in thin seams; however, in general, there tends to be less methane in thin seam mines.

41. Where pillar extraction is carried out, methane is not usually a problem, provided ventilation can be maintained. Much of the methane present in the coal may have migrated into the mine air in the primary mining stage.

Spontaneous Combustion Risk

42. The risk of spontaneous combustion depends largely on the composition of the coal. The method of working does, however, profoundly affect the supply of suitable fractured coal and oxygen, both of which are necessary to promote and sustain combustion.

43. In pillar extraction, abandoned pillars or portions of pillars left behind in the waste can provide the conditions for a heating.

44. The longwall system offers better protection from the risk of spontaneous combustion, as the waste area can be kept virtually free of oxygen. This does not apply to longwalls with bleeders; on the contrary, this method has a high risk in seams liable to spontaneous combustion.

45. Room and pillar systems have a medium risk provided pillar sizes are adequate to prevent crushing. A further advantage of room and pillar working is that the mined-out areas are accessible and hence any problems can be directly dealt with if discovered early enough.

Extraction Ratio

46. The extraction or recovery ratio is particularly poor in augering systems, as it is in room and pillar, when no secondary mining of pillars takes place. Pillar

extraction improves the recovery ratio, but seldom are all the pillars recovered. This can lead to problems in multi-seam working, as randomly abandoned pillars can sterilise other seams.

47. Longwall systems offer the best extraction ratio, particularly a layout that adopts the re-use of gate roads, by extracting contiguous panels. Thin seams often offer the best conditions for this technique.

Productivity

48. Productivity increases with the degree of mechanisation and hence all the hand-got combinations are low in productivity.

49. The highest productivities occur when the operations have a high degree of machine utilisation combined with highly-rated production capacity. Favourable mining conditions are another prerequisite. Good productivities are achieved by continuous miners as they are high-capacity machines, albeit less so in the thinner seams. However, their operation is still of a discontinuous nature compared with the more continuous production cycle of a longwall shearer or plough.

Capital Cost

50. The capital cost varies with the degree of mechanisation. The cost of equipment for a conventional room and pillar section is normally lower than for a continuous miner section. The typical small thin seam US mine can often defray the initial capital costs of new mine development by producing coal early in the project. This is not possible in a European shaft mine; moreover, the capital cost of shaft sinking and equipping is much greater than the minimal surface needs of a small US mine.

51. Longwall and shortwall systems have high capital costs arising from the expensive powered supports and associated equipment. In the USA an extra capital cost can be the equipment necessary for the development of multi entries at the end of the longwall. This operation is less onerous in Western Europe.

Operating Costs

52. Operating costs consist mainly of labour and materials; thus, operating costs for all the hand-got methods are relatively high. Both labour and material costs tend to be higher in thin seams, owing to the lower tonnage of coal extracted per unit of advance.

CONCLUSIONS

53. This section of the study indicates that further work requires to be carried out in Phase II, since the selection of candidate systems for use in the USA will require an in-depth study.

PHASE II

CHAPTER XVI

METHODOLOGY

1. The rationale developed by British Mining to fulfil the requirements of Phase II of this research study has been based on the further examination of candidate mining systems considered suitable for application in US conditions. This section of the report includes detailed evaluations of relevant systems and makes recommendations for further research.

2. The systems have been divided into two main categories: those suitable for seam heights in excess of 30 in and those below 30 in. These are dealt with in Chapters XVII to XXII and Chapters XXIII to XXVIII respectively. The main reason for this categorisation is that seams above 30 in can be mined using similar methods to those employed in seams of medium height, whilst below 30 in different methods are required. Consequently, the approach used in the two seam ranges is different.

3. For the plus-30-in range a number of variations of current systems - room and pillar, longwall and shortwall - are described and evaluated.

4. In carrying out the evaluation of the various systems for the plus-30-in category, consideration was given to conducting site specific examination of selected deposits. However, on reflection, it was decided that a more realistic approach, having regard to the scope of the study and the need to effect comparisons, would be to base the evaluation on a simulated deposit, incorporating typical features of current US thin seam mines.

5. Two terms commonly used in the description of the underground systems and mining methods are layout and design. In this report, layout means the overall geometrical configuration of the mine plan, ie the working panels and network of main roadways. Design refers to the details of the mining system within a particular production panel. The former is often strategic in nature, while the latter can be likened to a tactical concept.

6. One of the problems in preparing the layout and design of mining systems is to decide which comes first. For example, in situations where the layout is confined within limits of geological faulting, its geometry is predetermined before the design stage. In other situations, however, the design of the panels can be the starting point.

7. There is no single set of mining conditions that can be considered typical of US coal mines but the preceding chapters did highlight some parameters which were relatively common in deposits considered for the successful exploitation of thinner seams. These parameters were relative freedom from faults, shallow gradient, low methane emission rates and satisfactory roof and floor.

8. Of the above parameters, roof and floor conditions of the seam probably vary the most. One characteristic of thin seams in the USA is a variation of thickness within a relatively small area which is often associated with variations in the characteristics of the roof and floor. For example, draw slate, which occurs on top of the seam, causes mining problems and, from experience in mines visited in connection with this contract, a soft floor also causes problems.

9. A significant feature of US underground mining is the relative shallowness of depth in comparison with European mines. An average depth of approximately 300 ft was quoted in literature surveyed, although many mines are more than double this depth.

SIMULATED CONDITIONS

10. As stated in paragraph 4, the comparison of candidate mining systems has been based on their application to assumed or simulated mining conditions for a specific type of layout so that the study can concentrate on the more important aspects of the system, ie design and layout, and subsequently evaluate the system.

11. For this reason, the specification of the simulated deposit is arbitrary and simplistic. For example, the natural geological conditions have been considered ideal, to the extent that the floor is not too soft to give mining problems, but soft enough to cut with a continuous miner when necessary. A seam thickness of 33 in has been chosen because it lies midway in the range of 30 in to 36 in, selected as a candidate range for further study.

12. The simulated conditions are as follows:-

(i) Seam thickness: 33 in, free from dirt bands within the seam.

(ii) Roof and floor: satisfactory, ie no draw slate.

(iii) Gradient: 3^0 or 5% dipping from north to south.

(iv) Depth below surface: 300 ft.

(v) Square-shaped deposit, with sides of approximately 6,000 ft.

(vi) Free of faults, water and other natural problems, such as high methane emissions, susceptibility to spontaneous combustion, etc.

13. The quantity of clean coal represented by the imaginary deposit, at slightly over 4 million tons, is considered sufficient to sustain a district of a thin seam mine for several years depending on the rate at which it is extracted. Depending on the percentage of recovery, at an annual output of between 0.35 million and 0.5 million tons it would provide a life of approximately 8 to 12 years, normally considered long enough to defray all or most of the capital cost of modern mining machinery.

14. The block of coal is considered independently of the normal mine infrastructure. It is assumed that all necessary connections and services, such as transportation of personnel, mineral, power and water, would be available at the perimeter of the block. For design consistency, this point has been taken to be the south-east corner.

15. Seven conceptual variations of mining systems in the 30-in to 36-in seam range are discussed under the headings of design and layout (Chapters XVII and XVIII). Chapters XIX to XXII relate to output and productivity, costs, comparison of systems and sensitivity analyses.

16. British Mining have taken the view that in mine planning, which includes the production programme and profit from the operation, besides the layout of the mine workings, the overriding consideration is the design of the chosen mining system. This is normally based on one of the two basic methods of mining, namely room and pillar or longwall. Accordingly, Chapter XVII describes three room and pillar systems, three longwall systems and one shortwall. Where the various system designs have different layouts, reasons for the differences are explained.

17. An outline production programme has been prepared for each of the seven systems chosen for detailed examination; all seven systems being then compared using such parameters as safety, cost of production, etc.

18. Chapter XXII examines the effect of applying sensitivity tests to the prime parameters and observing the subsequent variations on productivity and costs.

19. The candidate systems suggested for seams of less than 30 in are not in current use, except on a limited basis, or have been discontinued. Therefore, they are treated in a descriptive rather than a quantitative manner. Indications are given where it is considered feasible to increase efficiency through advanced technology, possibly to the extent of considering them for the plus-30-in category.

20. Chapters XXIII to XXVII address systems with an application to mining seams below 30 in thick. Many systems in the past have been proposed to mechanise the extraction of such seams, but, historically, the systems yielded results which were only marginally viable from an economic standpoint, and were frequently discontinued.

21. The minus-30-in systems have been classified into the following generic categories: moles, augers, full-face miners, scraper boxes and wide-web cutter loaders. A chapter is devoted to describing each system, its operation and equipment. The major problems experienced by the systems are identified and solutions suggested in order to overcome the problem areas.

22. An assessment is then made of the potential of the systems should the problems be solved, indicating the potential outputs and order of labour complement that would be required to operate the systems. The systems are then considered, together with an indication of the probable research and development effort required to make them competitive with current, medium-thickness, seam-extraction operations.

CHAPTER XVII

DESIGN

GENERAL

1. This chapter is divided into three main parts, each containing conceptual designs for the three most common mining systems: namely, longwall, room and pillar and shortwall. It has not been the intention to enumerate or describe the whole range of design variations possible within these systems, but the reasons for the selection are made clear.

2. Design is discussed under a number of parameters, some of which are unique to a particular system, and some, like ventilation, equipment and legal requirements, are common to all. In this context, design refers to the detailed specification of the mining method to be employed within a typical production section. A starting point in the design procedure is section output required for viable operation of the mine. The arbitrary tonnage range of from 0.35 million to 0.5 million tpa previously chosen translates to a daily tonnage in the range 1,500 tpd to 2,200 tpd of clean coal at a vend of 100%.

LONGWALL

Introduction

3. Highly mechanised longwall mining has been practised in the USA for about 15 years, and is currently used in seams down to 36 in.

4. There is a large measure of commonality amongst the three designs described; the principal common feature being the coal-getting machine and the associated longwall equipment, such as powered supports and an armoured face conveyor.

Common Design Features

5. In thin seam longwalls, coal ploughs currently predominate as the coal-getting machine in the USA. However, the new design of the in-web shearer is likely to be used increasingly in this seam thickness, as current developments increase its reliability. For this reason, this type of machine has been selected for the study despite the time-consuming "shuffle" or tapered cut at both face ends necessary to establish the cutting drum in the new web. This disadvantage has led to a preference for conventional conveyor-mounted shearers which restrict the lower limit of their operating height. However, the in-web shearer is more powerful and has a greater coal-cutting and loading capacity, but in order to maximise these advantages and minimise the proportionate loss of productive time at each face end, it must be used on a wall that is as long as possible. The

maximum length of the face is limited by the chain strength and power of the conveyor drive. Faces in excess of 900 ft are being operated successfully using well-designed armoured face conveyors, so one of the conceptual designs incorporates a face length of 985 ft.

6. Another parameter common to all the longwall designs is the width or depth of the cutting drums, a subject that has received much attention in recent years. Although drum widths in the UK have traditionally been limited to 30 in, research and development indicates that a drum width of 40 in will cut and load satisfactorily, giving an effective width of cutting web of 36 in.

7. Two important features of the designs are concerned with airborne dust and ventilation. Both have clearly-defined requirements laid down in US legislation.

8. In the case of airborne dust, the maximum permissible exposure by a miner - 2 mg/m^3 throughout his working shift - is a stringent standard but capable of achievement by the application of water, through well-designed jets, sprays and venturis.

9. A basic ventilation quantity of 15,000 ft^3/min passing along the face line has been used throughout the designs to ensure maximum dilution of airborne dust without excessive pick-up. This should be more than adequate to keep the face free from methane.

10. US legislation requires the provision of bleeder roadways to ventilate the goaf area. All the designs comply with this requirement. It should be emphasised, however, that the provision of bleeders encourages the migration of air through the goaf, which increases the risk of spontaneous combustion.

Variable Design Features

11. A thin seam longwall should contain the basic design features of a long face line and a minimum amount of entry road development. The three candidate designs have been selected to illustrate varying development requirements, the amount of roadway development for each design being expressed as a ratio of development drivage per unit of face advance.

12. The amount of development or entry drivage required before the longwall can start production is often a critical factor. One system that minimises this requirement is advancing longwall, a system commonly used in the UK. However, the performance of an advancing longwall, in terms of output, is frequently less than a retreating longwall using comparable equipment. One conceptual design incorporates an advance wall initially, and then reverts to a hybrid system, called the z-system, in which each subsequent wall re-uses one of the old gates of the previous wall at one end of the face and forms a new gate at the other end.

13. It should be stressed that the three designs described below attempt to maximise bulk output by incorporating some conceptual ideas which, although not presently practised in their entirety, use existing techniques, exemptions, or variances from the legal code for which precedence exists. The face lengths in the three designs vary according to the overall layout adopted.

Design I - Z-system

14. The design of the longwall for the z-system is illustrated on Plate 1. A headgate is driven in advance of the wall and maintained at a distance of about 60 ft ahead as the wall advances. The headgate would be 25 ft wide x 6 ft high and supported by pairs of 14-ft steel bars set on 6-ft steel posts. The pairs of bars would overlap by 3 ft and be set at 4-ft centres. Excavated material would be mined by a continuous miner loading out on to the gate stage loader. In the headgate, at a position immediately behind the wall, two support walls would be formed using the pump or monolithic pack technique, a 4-ft-wide centre wall pack and a 10-ft-wide roadway support pack. The centre wall pack would divide the headgate into two sections to provide a separate intake ventilation airway and conveyor roadway. The dimensions indicated on Plate 1 are necessary to comply with US ventilation and conveyor clearance regulations. The roadway support pack is incorporated to ensure that the headgate can be re-used as the tailgate for the subsequent longwall, which would be contiguous.

15. Ventilation of the advanced headgate would be by an exhauster scrubber unit which would return clean air along the longwall face.

16. The tailgate of the longwall would be the old headgate of the previous wall. Only the left-hand section would be used for ventilation. On the Plate this air current is shown to divide, some returning behind the longwall, so providing bleeder ventilation of the goaf. If the height of the old headgate was insufficient for the new duty of providing return ventilation, it could be enlarged by using a dinting machine, which is a machine specially designed for this type of work, and is similar to a continuous miner.

17. The first longwall of the series of walls in this z-system design would require the formation of a new tailgate; this would be done using a dinting machine, either in advance of the face or immediately behind it. The excavated material would be stored behind the face to form a roadway support pack. The use of a dinting machine in the subsequent tailgates would depend on the deformation in the gate after mining had taken place.

18. Ventilation of the longwall would be by intake air from the headgate, the diluted gases and polluted air being returned along the tailgate. The conveyor would be in the neutral section of the headgate and air from this section would not contaminate the air on the longwall face.

Design II - Single-entry Retreat

19. Single-entry retreat is popular in Europe. It combines the advantages of retreat with the minimum amount of roadway development. This design is illustrated on Plate 2. It incorporates a 20-ft-wide headgate which is divided by a ventilation partition to allow the conveyor to be sited in a neutral airway. The partition also provides a ventilation air circuit while the entry is being driven at the development stage.

20. The headgate is pre-driven by a continuous miner to a height of 6 ft, whilst the tailgate is formed by re-using one half of the previous headgate. In order to facilitate re-use, the 20-ft-wide headgate would be packed to half its width by one of the support wall pump packing systems now being successfully used in many countries. Steel bars and posts would probably be required for support besides, or in place of, roof bolts.

21. The ventilation circuit is shown to be intake air travelling along both directions in the tailgate and joining on the face line, then returning behind the face via the headgate. Other ventilation configurations are possible and could be designed to suit the overall layout of the mine.

Design III - Three-entry Retreat

22. Multiple-entry retreat is the most common form of longwall mining in the USA and avoids the need to seek variances. A minimum of three entries is required at each end of the longwall to comply with existing legislation. These groups of entries are mined as a room and pillar section, some mines using more than three entries in order to increase production during the development stage.

23. The face design is illustrated on Plate 3. Owing to the formation of the three entries and the overall layout of the series of walls, the face length is significantly reduced compared with the other two designs. The three tailgate entries would be newly driven, while the headgate would be formed from the re-use of the previous three tailgates. The sketch shows all three in use, but strata pressures would be likely to affect the entry adjacent to the previously-mined face.

24. In order to minimise the amount of waste mined in the three entries, their height has been reduced to 4 ft. Each entry is 14 ft wide and driven on 60-ft centres. Additional props and bars would be set as necessary to augment the roof bolts.

ROOM AND PILLAR

Introduction

25. Room and pillar operations currently account for over 90% of the underground bituminous coal output of the USA, nearly two-thirds of this being mined by continuous miners.

26. Despite advances in machine design there has been a substantial decline in productivity in the US coal industry since the end of the 1960s, many views having been expressed as to the reasons for this.

27. One easily identified change that has occurred coincident with the drop in productivity has been the introduction of the new Federal Code of Mining Regulations. The Regulations impose several additional tasks which previously had not been required. These include a rigorous clean-up and rock dusting of the roof sides and floor of the workings requiring extra labour, thereby reducing productivity.

28. Apart from the decline in the output per man, there has also been a decline in the output per machine. This is evidenced by the annual output per continuous miner machine falling from 96,000 tpa in 1972 to 79,000 tpa in 1976. One reason for the decline is the legal requirement for the machine operator always to work under supported roof thereby preventing him from cutting beyond the last line of support. Whilst the use of remote-controlled machines and machines with integral bolters may alleviate the situation, the main effect of the requirement is that the continuous miner must be withdrawn after cutting 15 ft to 20 ft and taken to another working place to allow the last run to be supported.

29. This requirement has a particular effect on thin seam operations as the tonnage of coal cut in a fixed-distance run is directly proportional to the extracted height of the seam. Hence the number of machine moves in a thin seam for a given tonnage is, of necessity, higher than in a thicker seam. It is essential, therefore, in the design of a continuous miner section, to lay out the section to minimise the number of moves, and to ensure that those moves that are necessary are as short as possible, thereby minimising the loss of productive time that occurs during each move.

30. The design of a room and pillar section includes establishing the dimensions for the sizes of roadways and the pillars, planning the shape of the pillars, and deciding on the number of entries to be driven. Considerations of ventilation determine the position of many of the items of equipment and the sequence in which operations may be conducted. It is necessary at the outset to decide if the mining is to include the extraction of pillars on the retreat. If this is to be done, the pillars must be sufficiently large to withstand abutment loads during the subsequent pillar extraction process.

31. British Mining have examined candidate thin seam room and pillar systems by designing and planning three conceptual arrangements. All utilise continuous miners but vary in the method of face haulage and layout. The designs are:-

(i) shuttle cars with no pillar extraction,

(ii) continuous haulage,

(iii) retreat with shuttle cars.

Continuous Miner and Shuttle Car for Room and Pillar System

Width of Entries

32. This is the normal starting point in design, the width being dependent on the characteristics of the roof and the operational requirements of the equipment. The maximum width for safe working is often determined by local experience as some entries will stand at a width of 30 ft whilst others would be regarded as potentially dangerous at a width of 20 ft.

33. For comparative purposes, it is assumed that a width of 20 ft would provide a stable and safe roof. This is a common width of entry and the maximum the Federal Code of Mining Regulations permit to be mined using roof bolts as the sole means of support. Also, at this width there is no problem in the manoeuvring of machinery, particularly shuttle cars.

Pillar Design

34. The purpose of the pillars between entries is to support the superincumbent strata. This support is augmented by panel barrier pillars and other adjacent unworked areas of the seam.

35. In order to achieve stability, it is necessary that individual pillars are sufficiently strong to support the column of strata that rests above them plus half the width of the surrounding excavation, ie the design should ignore any additional support offered by barrier pillars.

36. A number of methods, all empirical, are used to determine pillar size, with varying degrees of success in current designs. In all the methods considered, the actual strengths and weaknesses of the seam must be taken into account, including the presence of weak layers, the frequency of slips and faults and any tendency of the coal to deteriorate at the edges of pillars on exposure to air and moisture.

37. It is proposed to use Salamon's Formula for the pillar design. This takes into account the cross-sectional area of the pillar, the height of the pillar and also the loss in load-bearing capacity per unit area which occurs with increasing size due to the inclusion of more planes of weakness.

38. On the basis of a 20-ft room width and an extracted height of 33 in, a 20-ft square pillar would give a safety factor of 2.03. This is probably more than adequate to ensure the stability of the workings, though the data on which the formula is based were not necessarily typical of the average US coalfields. The size of pillar also allows for minor deviations in dimensions, without undue effect on the stability of the area. These dimensions give 75% of plan area extraction. Plate 4 shows the effect of depth on the pillar size to maintain a constant factor of safety, and the corresponding variation in percentage extraction.

Number of Entries

39. The number of parallel entries which, together with crosscuts, comprise a working section, determines the potential tonnage that can be realised per overall unit of advance of the panel.

40. The maximum number of entries that can be worked by shuttle cars is limited by the amount of cable that can be accommodated on a car's cable reel and by the legal limits of cable length. The effective range of a car is usually 450 ft to 500 ft from the conveyor loading point but this can be increased by back spooling the car's cable, ie anchoring the cable at some point along the haulage path, a practice not to be recommended as the chance of damaging the cables is considerably increased.

41. An increase in the number of entries increases coal output for each increment of advance. Thus the cost of materials and labour used to extend the conveyor, pipes and cables, together with the necessary ventilation stoppings, is spread over a greater tonnage and so reduces the unit cost of these items.

42. It is imperative that all the equipment used is "matched" to ensure compatibility of capacities. For example, it would be wasteful and possibly counter-productive to employ large-capacity shuttle cars in a thin seam if the capacity of the continuous miner was low as this would result in cars waiting at the change point for the continuous miner to complete the loading of the previous car.

43. Any limitations of section width on the loading rate, and hence the section productivity, can be modified by running additional shuttle cars. In the case of electric cable-reel cars, this introduces additional complications as, ideally, each car requires its own individual route to prevent the car's cables from becoming tangled. This would require the ratio feeder to be located a further crosscut away from the last open crosscut, but this would be difficult as the Regulations require that the third crosscut be sealed with stoppings.

44. A nine-entry panel has been chosen to maximise the output potential of an individual section. This is on the basis of a section geometry of 40-ft x 40-ft centres with 20-ft-wide entries and crosscuts, an average machine capacity of 2 tons/min and an expected car factor of 3 tons. This arrangement is comfortably within the reach of cars trailing cable. The effect of the variation in the number of entries on the output of a section is shown on Plate 5.

Ventilation

45. The quality and quantities of air required in a room and pillar section must conform to the requirements of the Federal Code - 30 Part 75, the principal requirements of which are:-

(i) A mean air velocity of 60 ft/min is maintained in any entry where cutting, mining or loading is being carried out.

(ii) A quantity of 3,000 ft^3/min is maintained in any entry where cutting, mining or loading is being carried out.

(iii) A quantity of 9,000 ft^3/min is maintained through the last open crosscut.

(iv) Air from any working face does not return along a road housing a conveyor belt.

(v) Air from a conveyor road is not used to ventilate any working face.

(vi) The velocity of the air current in a trolley road is limited to 250 ft/min.

46. In a 33-in seam with 20-ft-wide entries, the quantity of air necessary to provide a mean velocity of 60 ft/min in a working place, is 3,300 ft^3/min. In order to ensure that this quantity is continuously available, it would be necessary to provide a larger amount. A suggested figure for planning purposes is 5,000 ft^3/min, which could be provided by the use of brattice.

47. In order to provide a minimum of 9,000 ft^3/min through the last open crosscut, the intake air supplied should be in the region of 20,000 ft^3/min, with a nominal quantity of 5,000 ft^3/min/entry where four entries are available as intakes. Of the 20,000 ft^3/min, 4,000 ft^3/min would be used to ventilate the conveyor road and the remaining 16,000 ft^3/min coursed by brattice along the last open crosscut. The provision of this quantity would ensure that 9,000 ft^3/min was maintained along the last open crosscut even when traffic has to pass through curtains in the brattice line.

48. In order to comply with the requirement that the conveyor road must be effectively isolated from the main face air current, the site of the end of the conveyor line, and hence the ratio feeder, must be selected with care.

49. For operational reasons, it is necessary that the feeder be as near as possible to the working faces. If the feeder is situated behind the last through crosscut, then air from the belt road would either enter into or be drawn from the last through crosscut and this would contravene the requirement that the belt road is not used for face ventilation.

50. The most effective manner of isolating the belt line from the last open crosscut and, hence the working faces, is to position the ratio feeder behind the second open crosscut and arrange either for the belt line to bleed directly into the section return or, as is depicted on Plate 6, for the belt road to be ventilated directly from the section intake.

Continuous Miner

51. In order to obtain the optimum level of output within the limitation of a 20-ft-wide entry, it is necessary that the continuous miner should have a high speed of mobility so that at the end of each cutting run it can be moved to the next place with the minimum of delay. Small overall dimensions improve the mobility of

equipment in negotiating turns and reduce the possibility of the machine becoming jammed between roof and floor in areas of seam undulation. A limiting factor in the transfer of equipment from place to place is the handling of the machine's power cable and water hose, which can prevent high tram speeds from being fully utilised.

52. Owing to the limited space available, the motors and transmission components have to be compact in size and this limits the power that can be applied to the cutting mechanism. This, in turn, tends to limit the maximum cutting capacity of the low models. However, within the limitations imposed, it is possible to obtain a machine that will cut and load at an average rate of 2 tons/min including the whole of the face cycle, ie sumping, shearing and trimming the floor to remove any cusps.

53. An analysis of the effect of cutting and loading capacity is shown on Plate 7.

54. It is desirable that the continuous miner should be able to make as long a cutting run as is practical in order that the number of passes, and hence place-to-place moves, are minimised.

55. Remote control has two advantages in thin seams. It improves safety as cabs or canopies are difficult to use in heights of less than 36 in. It also allows longer passes to be made without the operator going beyond the last line of supports. This reduces the number of moves from place to place and, hence, improves productivity. However, in the conceptual designs the calculations of outputs have been based on manual operation of the continuous miner. Plate 8 shows the effect of cutting run length on output.

Shuttle Cars

56. A major disadvantage of shuttle cars in thin seams is the loss of capacity with diminishing height. To compensate for this, manufacturers have produced low seam cars with wide wheel-bases but this limits the car's manoeuvrability. The effect of variation in the capacity and speed of the cars is shown on Plates 9 and 10.

Roof Bolters

57. Roof bolting in thin seams is difficult, due to the problem of drilling a vertical hole and inserting a bolt which is longer than the height of the seam. The process normally used is to drill the bolt holes in short sections using extension drill rods. The time taken for changing rods and adding extensions increases the time required to drill the holes. Likewise, the installation of the bolt, by initially bending the bolt, inserting it in the hole, and straightening it to complete the insertion process, is time-consuming.

58. To speed up the process and reduce the physical effort required, research and development work has been carried out to develop a machine that can drill a hole longer than the seam height and then automatically bend and straighten a bolt as it is inserted into its hole. Whilst several such machines have been constructed and tested, they are still in the prototype stage and cannot at present be specified for a working section.

59. Delays to production during bolting operations tend to preclude the use of bolting machines mounted integrally on the continuous miners. The time delay in the bolting phase of the operation would reduce the section's potential output (see Plate 11 - analysis of output with integral bolting time), thus bolting must continue to be carried out using a separate machine.

Utility Machine

60. In order to maintain the working areas free of accumulations of coal spillage, it is necessary to have a clean-up machine in the section. A scoop is recommended for this purpose as it can also be used as a transport and general-purpose vehicle. Owing to the limited duty the scoop is required to perform, it can be battery-powered with greater mobility than equivalent cable-powered vehicles.

Ratio Feeder

61. The function of the ratio feeder is to allow haulage vehicles to discharge coal at maximum rate and protect the outbye conveyor system by regulating the flow of coal on to the section conveyor. A breaker can be incorporated in the unit to break up large lumps which might otherwise cause disruptions and blockages at subsequent conveyor transfer points.

Tonnage

62. The output from a section designed as described above, is governed by the average cutting and loading rate of the continuous miner, the limitations of the haulage system and the delays caused by moves from place to place. It is important to maximise the time that the continuous miner spends actually cutting and loading. This entails reducing or eliminating, where possible, time delay elements such as stops for support, moves from place to place, excessive breakdowns, cable damage and the delays caused whilst ventilation brattices are being moved or reset.

63. On the basis of an 8-hour shift, with a total travelling time of $1\frac{1}{2}$ hours and $\frac{1}{2}$-hour break for food, there remains 6 hours available for the production of coal. If the continuous miner were to cut and load continuously for this period, then it would produce over 700 tons per shift. In practice, however, the industry statistics show a much reduced figure which indicates that the actual cutting and loading times being achieved are very much lower. Plates 5, 7, 8, 9, 10, and 11 show the effect of various parameters on the potential tonnage output.

Continuous Miner and Continuous Haulage for Room and Pillar System

Continuous Haulage

64. Continuous haulage equipment essentially consists of a series of mobile bridge carriers and bridge "piggyback" conveyors.

65. The mobile bridge carriers are normally light cat-track-mounted chassis carrying a short, approximately 20 ft, conveyor section which can be driven around the section. Mounted between the mobile bridge carrier and the continuous miner is a bridge conveyor approximately 40 ft long. The bridge conveyor is pivoted below the continuous miner's discharge conveyor to receive the output of the miner. The connection of the bridge to the mobile bridge carrier is a short-run dolly, which allows a limited run for the miner without the need to move the mobile bridge conveyor. As the miner cuts its way into an entry, the mobile bridge conveyor is trammed in behind the increment. Plate 12 shows the general disposition of equipment in the design.

66. The use of continuous haulage places a constraint on the width of a section in that the reach of the continuous haulage mobile conveyor units limits the number of entries. The conveyor train must follow the continuous miner, consequently, the crosscuts are normally turned off at approximately 60^0 as opposed to 90^0 in a shuttle car layout.

67. Owing to the restricted width of section, the number of panels required to mine out a block of coal is increased compared with a layout using wider panels with a greater number of entries.

68. The restricted number of entries means that the average advance of the section must increase for a given tonnage output per shift. This, in turn, requires the installation of a greater number of stoppings and more extensions of the conveyor belt and general services.

69. Many of the design criteria and much of the equipment are similar to the systems using shuttle car haulage.

Continuous Miner

70. When used in conjunction with continuous haulage, the continuous miner is free from the constraint of waiting for shuttle cars. As the loading cycle is not interrupted by shuttle car changes, the opportunity for the miner to carry out ancillary tasks, such as floor trimming, whilst waiting for cars, is lost. It is therefore important that the miner is capable of cutting a clean floor in one pass with a loading mechanism which can "sweep" the floor thoroughly. With the improved utilisation of the continuous miner, the output of the system is more sensitive to the continuous miner's capacity than a system using shuttle car haulage. The effect of the continuous miner's capacity on the sectional output is shown on Plate 16.

Width of Entries

71. Although the continuous haulage system eliminates the frequent movement of haulage vehicles, which normally determines the minimum width of entry, the use of a continuous haulage system still requires wide entries to accommodate the longest individual single length of conveyor section. When the crosscut turn-off angle is decreased, a longer conveyor section can be used. In order to accommodate the movement of the bridge conveyor, it is proposed to turn-off the crosscuts at 60^0. One effect of turning off 60^0 crosscuts will be to create larger intersections which could be more difficult to support.

Pillar Design

72. A decreased turn-off angle results in smaller pillars on a given centre distance. Also the shape of the pillars is weaker due to the presence of acute angles. In order to compensate for the loss of area and shape factor, the centre distances have been increased to 50 ft x 50 ft, measured along the line of advance and the crosscuts. The use of an integral value of centre distance is recommended as it reduces the possibility of turns being inadvertently cut at the wrong place, hence reducing the stability of the pillars.

Number of Entries

73. The number of parallel entries possible with a continuous haulage system is dependent on the combined length of the continuous haulage conveyors. To avoid the system becoming over complicated and difficult to handle, the number of mobile carrier units is normally limited to two; consequently, the system can only reach two entries either side of the belt road, giving a total of five entries. A typical layout is shown on Plate 12.

Ventilation

74. The ventilation requirements for the last through crosscut and the individual entries are as previously outlined. As the width of the section is limited to five roads, the control of the total ventilation system is easier, particularly as shuttle cars are not continually passing through brattice curtains.

75. Whilst a flow of 9,000 ft^3/min in the last through crosscut is mandatory, the total quantity needed to achieve this flow is somewhat less than that required in the shuttle car section. A total quantity of 16,000 ft^3/min should be sufficient. With the section having a fewer number of entries, the number of brattices required to control the ventilation would be less, with a consequent improved control of the ventilation.

Static Chain Conveyor

76. The end of the continuous haulage system has to discharge on to the section conveyor. This can most easily be achieved in a thin seam by having the last bridge conveyor discharge on to a static chain conveyor, which, in turn,

discharges on to the main section conveyor. A "dolly", or small-wheeled trolley which runs on the static chain conveyor, connects the bridge to the static chain conveyor.

77. It is necessary for the static chain conveyor to be at least as long as the combined series of mobile conveyors so that the complete system can be retracted when the continuous miner is moved to cut in a different place. In the design, this would entail two entry centres and one crosscut centre, totalling approximately 150 ft. In order that the belt is not extended for each crosscut advance, the static chain should be made 300 ft long so that an extension of section conveyor would be required on every third crosscut intersected.

78. As there is no "storage" in the system, the conveyors must be able to handle the maximum loading rates of the continuous miner.

79. Where the output from several sections is loaded on to the final trunk belt system, it is possible that the convergent streams of coal could each be at a maximum and produce a peak of 18 tons/min - 1,080 tons/hour if the three operating sections use the same belt system.

80. In order to minimise the spillage that this could cause and ensure reliable operation of the conveyor system, either the final trunk conveyor must be of sufficient capacity or small surge bunkers should be incorporated in the system.

Cables

81. A major advantage of the continuous haulage system is the elimination of cable spooling in the haulage process, the supply cables to the various conveyor drives and tram units being mounted along the conveyor booms. The only cable traversing required is where the last bridge section runs along the static conveyor, a simple cable handling situation.

82. A further major advantage of the continuous haulage system is that the power supply and water hose for the continuous miner can be mounted on the haulage system, thereby eliminating cable handling and potential damage to the continuous miner's cable. The elimination of this task eases the duties of the operator and minimises the work of the helper.

Support

83. The choice of support can be either bolting or posts. The advantage of bolting is that once the bolts are installed they cannot be inadvertently knocked out by equipment. This is a desirable feature, especially at intersections where the conveyor sections sweep across the entries.

84. The problem of the installation of roof bolts in thin seams has been outlined previously. With the use of continuous haulage, an added problem is that of having to cross the conveyor line in order to transfer the bolting machine from

one side of the section to the other. This means that for the bolter to cross the line, the continuous miner has to be withdrawn from the face. It is therefore necessary that all the places being mined are cut in a fixed sequence.

85. A further problem with the use of bolts as a support medium is the actual rate of installation. The continuous haulage system has a high production potential and this results in a larger area of roof being exposed per shift. For example, a production level of 300 tons per shift at an entry width of 20 ft entails the cutting and support of approximately 130 ft of linear advance in an equivalent single entry. The use of parallel rows of four bolts at 4-ft centres would entail the installation of one bolt per foot advance cut or 130 bolts. This number could be achieved with one crew and one machine, under ideal conditions, but it is unlikely to be sustained on a continuous basis in a 33-in seam. A realistic approach is to use two machines, each working in separate entries.

Tonnage

86. The main operations in a continuous haulage section are simplified compared with a shuttle car haulage section.

87. The only necessary stoppages in the cutting and tramming cycle of the continuous miner are those for methane checks, required at a maximum time interval of 20 minutes, and those needed to advance the ventilation appliance and to make safety examinations.

88. As there are no stoppages waiting for cars, the tonnage potential of such a system is higher than for an equivalent section using shuttle cars.

89. Since there are no car change delays, the continuous miner is not able to carry out such operations as floor trimming and minor clean up of spillage in the otherwise "dead" time. The net effect of this is to reduce the average cutting and loading rate of the continuous miner as the actual operating time base for the whole operation is increased.

90. It is hence more critical with this type of operation to have a trained and skilled operator who is able to cut the correct floor horizon on the first pass and accurately position the miner for subsequent passes to mine the room out to the required width.

91. The tonnage potential of a five-entry continuous haulage system is shown on Plates 8, 13, 14, 15 and 16 in terms of design variables.

Room and Pillar Retreat Mining

General

92. In retreat mining an area of coal is first cut into pillars which are subsequently extracted on the retreat. The design of the pillars has to ensure that they are stable during the initial mining process and, more particularly, during the

subsequent pillar extraction phase when pillar support for the roof is gradually removed, resulting in the roof subsiding. During this period it is essential that the lowering or caving of the roof is controlled to ensure the safety of personnel. This is achieved by the systematic removal of the pillars leaving behind thin fenders of coal or pillar remnants, which provide a transitory measure of support, eventually crushing when the general line of pillar extraction moves away.

93. Leaving portions of pillars causes a number of problems. The crushing of remnants exposes a large area of fine coal which can, under certain conditions, induce spontaneous combustion. The leaving of whole pillars, because of failure to control the roof, creates a high stress pattern which can result in extremely difficult conditions in adjacent panels or in subsequent seams both above and below.

Width of Rooms

94. The required room width for a retreat panel is determined by the behaviour of the roof, particularly when the pillars are being withdrawn. The width of the roadways has to be sufficient to permit the movement of the equipment yet be small enough to reduce the distance that equipment and operators have to cross from the protection of the solid pillar area to the pillar that is in the process of being removed.

95. During pillar removal, an abutment load is imposed on the barrier pillars of the panel and on the rows of pillars adjacent to the waste. This increased load, together with a general convergence of roof and floor in the waste, causes movement in the roof layers in the entries and crosscuts near to the line of pillars being extracted.

96. The tonnage of coal removed on the primary extraction of formation of the pillars is dependent on the width of the entries and crosscuts and the size of the pillars. Hence, it is desirable to work the widest entries consistent with safety. In the case of retreat working, the purpose of the entries is to form the pillars, provide access to them and obtain extra tonnage as they are withdrawn.

97. The width of room available is dictated by conditions of the seam being worked and the mining method adopted. The optimum room width is found from experience, but, generally speaking, the smaller the width of the roadways, the easier the task of roof control and pillar extraction.

Pillar Design

98. The function of the pillars is to support the roof and strata above, and hence produce a stable working environment underground. Where the pillars are to be extracted, their dimensions must be such that they are strong enough to support the superincumbent strata, or weak enough to yield in a controlled manner. The pillars must be sufficiently strong to accept this load without excessive deformation in order to minimise differential movement in the roof strata.

99. In thin seams, the design of the pillars is not difficult because the large width to height ratios allow relatively small pillars to develop high stresses. These high stresses can be detrimental to a "tender" top, so, for extraction purposes, it is necessary to allow a larger size than is dictated by the design criterion for pillars. Pillar size, like road width, is largely determined by experience in the seam. A commonly-used design parameter in the USA is to extract approximately 50% of the coal on the formation of the pillars.

100. When mining the pillars, it is customary to leave corners and thin portions of the pillar for roof control. The size of these should be sufficiently small to prevent any appreciable build-up of strength when the roof converges during secondary mining. The remnants of the pillar should be strong enough to give temporary support to the immediate roof but be able to fail in a controlled fashion.

101. For a seam 33 in high with a competent roof and floor, square pillars of 42 ft and a room width of 18 ft would give a factor of safety of over five at an initial extraction of 51.1% on the formation of the pillars. This gives a high degree of stability, with a pillar of adequate dimensions that could be split as part of the extraction process.

102. Alternatively, dependent on the techniques adopted on the extraction of the pillars, the shape of the pillars could be made rectangular with the short dimension bounded by the crosscuts. In this case, dimensions of 32 ft x 57 ft pillar sides would give a primary extraction of 51.4% with a satisfactory factor of safety.

Number of Entries

103. Owing to the increased size of pillars required for retreat mining, the number of entries possible within the same width of panel is reduced. Factors such as cable lengths and the reach of the face haulage are used determine the distance that can be mined either side of the conveyor road.

104. The number and width of entries, together with pillar size, determines the width of the panel. The other parameters of depth from surface, strata characteristics and seam thickness, in conjunction with design of the panel, largely determine the height to which the roof should cave on retreat and the degree of subsidence at the surface. In the case of wide panels lying at a shallow depth, caving can continue up to the surface. With comparatively narrow panels in deeper seams, caving does not reach the surface as upper layers of the strata tend to "bridge" across the panel, the load imposed by them being carried by the abutment stresses of adjacent pillars and the surrounding unworked area. The barrier pillars must be sufficiently strong to sustain these abutment pressures and wide enough to protect the current workings from strata movements induced by the adjacent extracted panel. This means that, where retreat mining is practised, barriers left between the panels must be wider than those where the pillars are left intact. In order to minimise the loss of coal in the barriers, the panels should therefore be as wide as possible within the confines of cable length, haulage system capability and productivity.

105. From the foregoing, it is proposed that the number of entries for the retreat layout should be nine, which, within the cable length limitation of 500 ft, allows equipment to reach all necessary areas without resorting to back spooling or similar techniques.

Pillar Extraction

106. The main consideration when extracting pillars is control of the roof. After a few rows of pillars have been extracted, the roof caves and follows as each pillar is drawn. There are three main methods for extracting the pillars:-

(i) Removing a series of parallel slices off the side of the pillar (see Plate 17).

(ii) Driving a series of cuts into the side of the pillar, taking approximately half the pillar, then repeating the process from the other side (see Plate 18).

(iii) Splitting the pillar into two or four portions and robbing the remaining portion (see Plate 19).

107. As the best control of the roof is obtained when the pillars are withdrawn quickly, operations have to be concentrated in an area of one or two pillars.

108. Where pillars are hand-worked or mined conventionally by cutting and shooting, it is necessary to work several faces so that all the required steps of the mining process can proceed simultaneously. With a continuous miner, only one face need be worked at one time, making the continuous miner an excellent device for the removal of pillars.

109. In order to achieve high productivity, it is necessary to minimise the number of place-to-place moves and the time spent with the continuous miner standing idle waiting for roof support operations. Method (ii) results in this situation when the distance of the cutting run is such that the operator does not enter unsupported ground.

110. Where, due to considerations of depth and seam height, large pillars are initially formed, method (ii) cannot be used effectively as the length of the cutting run required to remove half the pillar may exceed the cutting-head-to-operator distance of the continuous miner. This would mean that the cutting operation would have to be stopped to install supports or an excessively large stub would be left in the middle of the original pillar.

111. Where splitting is practised, the length of run needed to produce the initial splits is governed by the cutting-head-to-operator distance of the continuous miner. This means that the machine must be moved to another pillar whilst support work is being carried out, thus reducing productivity at this stage. Once the pillar is split, the remnants can be mined by method (ii), or further extraction may be

terminated leaving a series of small pillars behind. This is termed partial pillar extraction. A problem with partial extraction is that the pillars left may be potentially unstable causing a large area to collapse without warning.

Support

112. During retreat operations, supports are required to control the roof over the working places and form a breaker line to prevent caving of the waste from running into adjacent working areas. The latter is achieved by forming a strong line of support to shear the roof on one side and hence break the continuity of a "running" fall.

113. Support is normally supplied by roof bolting, with posts or jacks to form the breaker lines.

114. Support costs are generally lower in the retreat process as the roof exposed by the extraction of the pillars is allowed to lower or cave. Furthermore, where posts are used, they can subsequently be withdrawn and removed. An advantage of timber supports for thin coal is that costs for short posts are lower and the setting time for timber can be much less than for roof-bolt support.

Pillar Extraction Sequence

115. The removal of a pillar prevents access to the surrounding roadways and the areas they serve. Direction and control ventilation is made difficult by the removal of the pillars and the destruction of stoppings between them.

116. The order of extraction must ensure that the redistributed strata loads do not render adjacent pillars unworkable, and permit access to be maintained to each pillar to be worked.

117. In order to ensure uniform redistribution of loads, the pillars should be pulled or extracted in a line. The preferred stooping or pillar extraction line is shown on Plate 20, where the pillars are pulled in a v shape, thus gaining maximum support from adjacent panel barriers. This method is suitable where pillars are initially split and the remnants worked from the centre of the pillar.

118. Where rectangular pillars are left, two adjacent pillars can be worked from their common entry. This means that to extract all the pillars of a row in the same manner, the panel must be worked in a straight line from rib to rib. Where the initial development was done in a herringbone pattern, adjacent pillars could be worked taking advantage of the v line to distribute evenly the abutment loads.

119. Where pillars are formed over a large area, the average level of stress in the pillars, and in the floor and roof where they rest on the pillars, is increased in proportion to the initial area of the block of coal divided by the area of the coal remaining in the pillars. Thus, with a 75% extraction of the area, the stress in the pillars would, on average, be four times higher than the original levels.

120. Where soft bands or layers exist in either the floor or roof, the effect of the induced high stresses can cause roof convergence or floor lift due to the pillars "punching" into the roof or floor. Provided the extraction of the pillars, once commenced, is done continuously and quickly, this convergence is not normally a problem.

121. In a 33-in seam, however, even a limited movement may inhibit the access of machinery and prevent the extraction of certain pillars. This disrupts the symmetry of the pillaring line causing high stress concentrations resulting in the risk of losing several rows of pillars before pillaring can be resumed. Hence, for success of retreat mining in thin seam coal, conditions must be very good and organisation of operations must be of a high order.

Ventilation

122. The ventilation requirements for a retreat panel differ from those of the same panel when initially developed.

123. Airways are formed around the pillars during their formation and gases present in the coal can gradually migrate, hence presenting a low methane emission on pillar extraction. The pulling of the pillars, however, causes extensive strata movement. This can result in the release of gases, from roof and floor sources, into the waste area, which is difficult to ventilate owing to the destruction of airways.

124. The volume in the goaf available for the accumulation of gases is difficult to assess, as ribs, fenders and occasional pillars prevent good compaction of caved material. An indication of the "free volume" may be obtained by comparing retreat and longwall systems. In longwall mining, where no remnants are left behind the face line and consolidation takes place in a uniform manner, the waste still has an "induced porosity" which amounts to a free space of approximately one tenth of the original volume occupied by the coal. In the case of the irregular waste from a retreat pillaring section, this space would be greater.

125. Control of ventilation is best achieved by using the barrier entries for intake and return and systematically removing stoppings to course the air over the pillar line as it is retreated. Regulations call for a minimum of 9,000 ft^3/min to reach the intake end of a pillar line but the ventilation must also be sufficient in quantity to dilute respirable dust and methane to the prescribed limits.

Continuous Miner

126. The equipment used for the extraction of pillars is normally the same as that used for pillar development. During the retreat phase, the coal in the pillars is under greater stress and, in consequence, tends to break out easier so the instantaneous mining and loading rates can be higher. However, the continuous miner must be sufficiently robust to withstand minor roof falls without damage.

127. The use of remote control in thin seams has much to commend it, as the provision of cabs or canopies in a working height of 33 in is difficult. As the possibility of falls trapping the continuous miner is greater in retreat operations, some means of remotely recovering the machine is desirable. One device available for this purpose is a heavy-duty hydraulic puller that is large enough to drag the miner back, either in the case of a tram malfunction in an inaccessible area, or in the case of a moderate roof fall. The hydraulic puller can be set up under stable ground and attached with a heavy cable or chain to the rear of the continuous miner to drag it free.

Haulage System

128. The haulage system has to be designed to accommodate the larger pillar entries and the higher possible loading rates during the extraction process. The system has to be flexible and require the minimum number of roads to operate in, as the withdrawal of pillars constantly makes roads unavailable.

129. Both shuttle cars and continuous haulage systems can be used. Shuttle cars offer a high degree of flexibility but the increased distances from the change point to the continuous miner, during pillar extraction, tend to limit the potential output.

130. The use of a mobile bridge carrier and bridge conveyor combination gives a reach of approximately 60 ft. This reduces the number of entries that can be mined when the entry centre distance is increased to leave larger pillars for subsequent extraction. Therefore, in the conceptual design for retreat room and pillar, shuttle cars are proposed as the means of haulage.

Ratio Feeder

131. The ratio feeder should be located as far forward as possible to reduce the shuttle car haulage distance, but in a position secure from potential roof control problems emanating from the waste area behind the pillar extraction lines.

Tonnage

132. The potential output of a retreat section is the combination of the tonnage obtained in the development phase and in the subsequent pillar extraction retreat face.

133. As a greater proportion of the coal mined within a panel comes from the primary advancing mining phase, the efficiency of the process is largely determined by the output potential at this stage.

134. The output during retreating is partially dependent on the time delays for support operators in between lifts and the constraints of cutting and haulage capacity. The time delays in moving from place to place are reduced in retreat working as a number of runs are made from essentially the same location.

SHORTWALL MINING

General

135. Shortwall mining is a combination of room and pillar and longwall techniques. Standard room and pillar methods are used to block out a substantial pillar of coal, this single large pillar being mined on the retreat by taking cuts in the form of long runs along one edge. Support is provided by a series of longwall-type powered supports which are advanced after the continuous miner has been backed out from each run.

Length of Wall

136. The length of the wall is one of the major parameters that has to be determined in the design of the shortwall system. The length determines the number of powered supports necessary for the system, so directly affects the initial capital cost of equipment.

137. The length of the wall relative to the width of the development entries determines the proportions of coal obtained from the wall and the development operations.

138. The productivity and cycle of operations are directly influenced by the length of the wall. The longer the wall, the longer the cutting runs of the continuous miner and the higher the tonnage obtained per run across the wall. However, a longer wall results in a greater distance for the coal to be hauled from the continuous miner to the section conveyor. The capacity of the system can therefore be reduced by an excessive length of wall, as there is only a single path along the wall for the haulage system. This is illustrated on Plate 21 which shows the effect of the wall length on the output per shift and the distance advanced/retreated by the wall per shift using shuttle car haulage.

Number of Entries

139. The purpose of the development entries is to form the large pillar for shortwall extraction and to provide the necessary access to the pillar. Where it is proposed to mine a series of adjacent retreat faces, each set of development entries can be used twice. Hence, after the two sets of entries have been driven to establish the first shortwall, only one further set will be required to develop each additional block.

140. In order to maximise the overall recovery of coal from an area, the amount of coal left in the form of chain pillars from the development entries should be minimised. Hence, for both speed of development and a high recovery of reserves, the number of entries in the development stage should be the minimum necessary to comply with the legal requirements of ventilation and access.

141. Shortwall could be worked with a single entry at each side of the block, but this would necessitate the individual development of each block, or the use of a wide entry with an artificial wall down the centre to enable the entry to be used a second time.

142. To operate a single entry would require a number of variances from the mining law, and cause the mine to suffer from periods of low output while the development entries were being driven.

143. Sets of twin entries could be used which would result in one line of pillars being left between blocks. If belt haulage were used in the driving of the development entries, then a variance would still be required to allow the conveyor belt roadway to form part of the ventilation circuit for the entry development.

144. The use of a three-entry system for shortwall development would give a reasonable tonnage during the development stage and require no variance from the mining legislation, in terms of ventilation and conveyor haulage.

145. It would also leave two lines of chain pillars between adjacent shortwall blocks, a major advantage for roof control. By suitable design of the chain pillars, any damaging effect of side abutment pressure could mostly be dissipated in the first row of pillars on the extraction of the shortwall block on the one side of the entries, leaving the entry next to the other block in good condition.

Pillar Design

146. The chain pillars between shortwall blocks would not normally be extracted concurrently with the shortwall extraction. Hence, any coal left in the chain pillars would be lost. The function of the pillars is to provide support for the entries during the development phase and also to withstand any abutment loads when the shortwall pillar is being worked.

147. It is preferable, however, that pillars yield when the blocks on each side of the entries are extracted. This allows a more even subsidence pattern to develop and makes the working of other seams less complex than would be the case if the chains were to attract a very high load.

Crosscuts

148. Crosscuts are necessary for the working of entries during the development phase but they reduce the support for the entries during the working of the shortwall itself. It is therefore desirable that the crosscuts are of minimum width consistent with their function, and are spaced at the maximum distance apart without causing undue delays in the haulage of coal, either in the development or shortwall extraction process.

Width of Entries

149. The width of the entries should be minimised to get the best roof control, particularly when the shortwall is being worked. The width necessary for the entries is determined by the physical size of the equipment to be housed and the space required by the haulage system to get on to the shortwall from the adjacent entry. An entry width of 20 ft is about the maximum that can be allowed to give adequate room for equipment and still provide good roof control.

Dimensional Design and Tonnage Ratio

150. A face length for the shortwall of 150 ft has been selected. This length gives a high level of output for all haulage systems and offers the optimum labour productivity. In the case of shuttle car haulage, bulk output and productivity would drop with a longer length of wall due to an increased single path distance from the continuous miner to the change point. With a continuous haulage arrangement, face lengths in excess of 150 ft would require additional haulage sections, necessitating an increase in labour. Lengthening the wall would also reduce the rate of advance, with a possible deterioration of conditions.

151. For the development of the walls, a three-entry system is proposed on the basis that, with the low height available, it gives the best strata control.

152. Crosscuts, 17-ft wide, are laid out on a 60^0 turn off from the conveyor road to enable continuous haulage to be used in both the development and shortwall extraction. The crosscuts are turned off at 60-ft intervals, measured along the entries. The entries, 20-ft wide, are on 43.3-ft centres giving an apparent centre distance of 50 ft measured along the crosscuts. This arrangement is shown on Plate 22.

Coal Recovery

153. Assuming that only the chain pillars are left after termination of activities, the extraction rate is approximately 88%. This is derived as follows:-

Area of coal in block and development for a 60-ft increment

= (150 + 106.6) x 60 = 15,396

Area of pillars

= (23.3 x 40.4) x 2 = 1,882.6

Area of coal extracted = 13,513.4

Percentage of extraction = $\frac{13,513.4}{15,396}$ = 87.77%

Coal extracted in 60 ft of development is:-

Area of development panel less area of pillars multiplied by the tonnage per square foot of coal

= (106.6 x 60 - 1,882.6) x 0.11583 = 522.8 tons

Coal extracted in 60 ft of wall advance is:-

Area multiplied by tonnage per square foot of coal

= (150 x 60) x 0.11583 = 1,042.5 tons

ie, the ratio of coal produced on the shortwall to each ton of development coal is:-

$$\frac{1,042.5}{522.8} = 1.994 \text{ tons}$$

Ventilation

154. The ventilation requirements for a shortwall layout are in two phases. The first is to ventilate the development entries, and the second to ventilate the wall as retreat operations are conducted. The ventilation along the wall should be arranged to flow from the headgate to the tailgate so that any dust made by the continuous miner does not pass over the equipment operators or other personnel.

155. An air velocity of 500 ft/min would ensure an adequate dust dilution without causing an increase in the dust count due to pick-up. This is equivalent to a quantity of approximately 16,000 ft^3/min, which should be adequate unless methane is a problem, when the quantity would have to be increased.

156. A major benefit of the shortwall and longwall systems, compared to room and pillar, is the ease of ventilation due to the straight nature of the wall.

Width of Cut

157. The width of cut taken by the continuous miner is obviously governed by the width of the miner's cutting head. The width of cut governs the area of roof between the wall and the powered supports. Where there is a tender roof then, to improve control, the distance between the wall and the supports has to be minimised by using a narrow-head machine.

158. On working the wall, there is a general movement of the strata which manifests itself by a convergence between the roof and floor. This convergence may be expressed as an angle of slope, which, amounts to approximately 2.25^0 for a 33-in seam. This means that over the width of a cut that has been taken at a nominal 10 ft, the roof will lower 4.7 in. This is normally not critical in higher workings, but at an initial seam height of 33 in, the reduction in available height to 28.3 in has a negative effect on mobility of personnel.

159. Thus, where shortwall is proposed for thin seams, the width of the continuous miner must be minimal so that the loss of height from convergence does not cause hindrance to the operation.

Sequence of Operations

160. At the commencement of face operations, powered supports are in the forward position with the canopies close to the wall. The continuous miner starts to cut from one end of the wall. Dependent on their forward tip clearances, the powered supports may be partially advanced behind the miner, leaving sufficient space for the miner to be backed out at the end of its cutting runs.

161. After the continuous miner has been backed out, any spillage along the face is cleaned up, using a scoop, and the powered supports are advanced. When supports have been brought up to the face, the operation cycle is repeated.

162. A feature of this operation, which is essentially open-ended pillar extraction, is that no ribs or fenders are left behind the wall. Operationally, a major advantage of shortwall is that support operations do not limit the distance that the continuous miner can cut in one pass, with the attendant delay of moving to another place.

163. The only areas of operational delay are those of the face haulage, the time to retract the continuous miner at the end of the run, and the time necessary to clean up and advance the supports prior to the next run.

Transfer of Equipment

164. The transfer of equipment from panel to panel is more complex for a shortwall installation than for a room and pillar system. This is because the supports have to be disconnected and removed from the wall at the end of the wall's life and then re-installed on the next panel.

165. This does not, however, involve a lengthy stoppage as the continuous miner and haulage equipment can be transferred first and commence development production while the supports are being withdrawn.

Continuous Miner

166. The continuous miner is used as the coal cutting and primary loading machine, both in the development and shortwall extraction phases of the operation. In addition to satisfying the requirements for development, the miner must be compatible with the other shortwall equipment.

167. In order to maintain roof control, the span between the powered supports and the wall should be minimal, requiring a narrow-head miner. The overall height profile of the machine must be low to enable canopy extension bars attached to the powered supports to be advanced over the miner's track and yet allow it to be retracted at the end of its run.

168. Remote control is a distinct advantage as the operator, under the protection of the powered supports, can move forward and be closer to the head of the machine. This is possible due to the ventilation being coursed in the same direction as the machine is advanced.

Shuttle Cars

169. In shortwall operations it is not feasible to increase the carrying capacity of shuttle cars by increasing the width of the car. This is because the width is limited by the distance between the wall and the powered supports. As the continuous miner cuts along the wall, the delay between cars increases as there is only a single tramming path along the wall. This, together with the low car factor, results in a low haulage capacity and, hence, low rate of production.

Continuous Haulage

170. There have been many attempts to develop conveyor systems that can be extended or moved while operating, and moved around a 90^0 corner. These "flexible" systems have many potential advantages but, up to the present, they have not been successfully applied, even in thicker seams where more height is available for the conveyor components.

171. The most successful systems to date have been based on a combination of short fixed-length conveyors with individual drives that are snaked behind the continuous miner. In order to facilitate the movement of the conveyors, every second conveyor is mounted on a mobile chassis, which is steered by an operator. The length of these mobile bridge conveyors is approximately 20 ft, while the length of the connecting bridge conveyors can vary up to 40 ft.

172. In order to get the longest straight element of the system around a 90^0 corner, the length of the conveyor and the dimensions of the entries must be compatible to give operational clearances. With a headgate entry of 20 ft and a

face "entry", eg the distance from the wall to the line of the powered supports, of 10 ft, a straight conveyor section appproximately 35 ft long and 30 in wide can be manipulated around the corner. This would not normally allow any clearance, except that the corner is cut in the form of a curve when the continuous miner commences its cutting run.

173. Hence, with two mobile bridge sections of 20 ft length and three bridge conveyor sections of 35 ft length, a conveyor reach of 145 ft is possible, to which must be added the length of the continuous miner. This would enable a wall length of 150 ft to be worked.

174. With this arrangement, two men would be required to operate the mobile bridge carriers of the haulage system as opposed to two car drivers where shuttle cars were used.

175. In order to achieve smooth operation of the continuous haulage system and to gain maximum reach for a given number of haulage units, the section's conveyor should be located in the entry adjacent to the shortwall pillar. This eliminates the need for cross conveyors and the time delay elements associated with the moving of this equipment as the wall is mined.

176. By using two chain conveyors between the conveyor belt end and the continuous haulage, the need for frequent move-ups is reduced and maximum time is available for production.

Support

177. Support is required during the development phase of the shortwall, and this would be a combination of roof bolts and wooden posts.

178. During the working of the wall, support is provided by a series of hydraulically-powered supports similar to those used on longwalls. Dependent on the type used, these would be spaced on 4-ft to 5-ft centres down the length of the wall and extend across both the tailgate and headgate entries. For a wall 150 ft long and entries at each end of 20 ft width, 38 supports would be required at 5-ft centres. These would be powered by a pump unit in the headgate.

179. The load-bearing capacity of the supports is determined by the height at which the roof is expected to cave and the estimated load caused by the roof when free caving does not take place. When the roof caves into the waste, there is an increase in the specific volume of the roof material. With normal fragmentation, this can amount to a 50% increase compared with the specific volume of the solid material. Where fragmentation is poor, due to the roof breaking into large blocks, the volume increase will be less.

180. Additional support, in the form of cribs, would be required to bridge the chain pillars close to the wall.

Utility Scoop

181. It is important that the track cut by the continuous miner is cleaned before the supports are moved. This reduces the amount of coal lost in the waste, and hence reduces the hazard of spontaneous combustion. Further, having a relatively clean track makes the moving of the powered supports easier as they do not then have to ride over loose coal or "bulldoze" the material forward.

182. The clean-up operation is most conveniently carried out using a scoop. The scoop itself must be able to gain access to the face, and hence must be sufficiently low and narrow to enter the continuous miner's track after it has been withdrawn. Owing to the intermittent duty, the unit can be battery powered.

183. One difficulty is to park the scoop when it is not in use. The machine could be parked in a crosscut near the wall and brought out when required. This would mean moving over the chain conveyor on to which the continuous haulage loads. Alternatively, the chain conveyor could be pulled back and then replaced after the clean-up operation. The chain conveyor could be pulled using a winch attached to one of the powered supports in the headgate.

CHAPTER XVIII

LAYOUT

INTRODUCTION

1. The layout of a thin seam mine follows essentially the same principles as a mine working thicker seams. The factors that must be considered include the depth, the proximate geology of the immediate strata above and below the seam, the general geology of the area including faulting, folding, etc, water, gas and surface restrictions. Surface subsidence is likely to assume an even greater importance in the near future owing to new legislation in the USA.

2. The influence of most of these factors has been minimised for the conceptual designs by assuming idealised mining conditions in the simulated deposit of coal 6,000 ft square and 33 in thick, this deposit being generally level, and free from gas and water.

3. Other factors, which can be referred to as variable parameters, must also be considered in the layout. These include criteria such as the width and length of panels, indirect fracture patterns, zones of high stress, pillar sizes, method of working, unworked seams above and below, the rate of face movement, shape and dimensions of roadways, direction of extraction (rise, dip or strike) and the method and speed of development. The seven candidate layouts described in Chapter XVII represent only a small proportion of the total possible layouts.

LONGWALL

Introduction

4. Possible layouts can range from a simple configuration of advancing panels with adjacent protecting pillars, through combinations of advance and retreat panels, to a series of full retreat panels with no protective pillars between. Some layouts use two or more access roadways from which flank panels are developed at right angles, whilst others use a short advancing longwall to develop the block of coal. Sometimes adjacent panels are operated concurrently, on a stepped basis, but, more frequently, the next panel is started when the previous one has finished.

5. With such a wide selection of layouts, the ratio of development roadway drivage to the actual movement of the longwall varies considerably. In a thin seam, it is imperative that this ratio be as low as possible as roadway development is more costly and often slower, especially if the immediate strata is hard to extract to give sufficient headroom.

6. A major compromise to be made lies in choosing between early production of coal from an advancing face, and the higher production per unit shift from a retreat face, which takes longer to come on-stream. The three candidate layouts demonstrate these criteria.

Layout I - Z-system

7. This layout is illustrated in Plate 23. As the access to the simulated block of coal is at the south-east corner, the quickest way to start full production is with a wide (985 ft) advancing face commencing at the corner and mining to the rise along the eastern flank of the block. Subsequent faces are z-patterned, ie one roadway is made as the face advances whilst the other is pre-formed by using one of the old roadways from a previous face.

8. A face length of 985 ft is considered the longest practical length with modern face-conveyor technology. Using this figure, the block of coal will accommodate six longwall panels.

9. Development of the block is by five entries along the southern edge. The ventilation system is illustrated on Plate 1, and is such that it provides a separate source of fresh air for both the entries and the longwall. The layout provides for a bleeder return airway around the perimeter of the block.

10. On the north flank of the five entries, a protective pillar, 120 ft wide, is formed between the northern-most entry and the longwall start position. The pillars and development entries sterilise 9% of the block. If the coal mined in the entries is added to that extracted by the longwalls, the total percentage of extraction in the block is 92.8%.

11. Two interesting features of the layout are the ratio of development drivage to longwall advance, ie the development footage ratio, and the time to reach full output. The development footage ratio is equivalent to 1.34 ft of entry advance per 1 ft of wall advance, while the time to reach full output is 76 days. The latter is based on a conservative daily development capability of 150 ft of equivalent single-entry development and a total requirement of 7,595 ft of development entry. A further 25 days has been added for equipping and commissioning the wall. The low development footage ratio is an obvious advantage of this layout.

12. Another factor of influence in the formulation of panel layouts is the time taken for the equipment to be transferred from one face to another. If it is intended to extract the complete block with only one set of longwall face equipment, then the transfer time is important. In the z-system layout, this has been estimated at 25 days, which can be a long time without any production and could be considered an undesirable feature of this layout.

Layout II - Single-entry Retreat

13. Plate 24 illustrates this layout. The principal feature is the shorter length of the panels (half the size of the block) and the facility to make easier transfers of the longwall equipment between panels. This necessitates a change in direction of travel of each successive panel and ventilation re-arrangements would be necessary, normally an undesirable feature.

14. The block is developed by driving a series of entries up the centre of the area from south to north. The individual longwall panels are then developed by driving single-entry gates east and west. Despite the use of single entries, the total development footage requirements are high due to the considerable length of primary development.

15. The pillars are similar in size to the z-system layout, but in conjunction with the development roads, they sterilise 20% of the block. The total recovery of coal is consequently less at 87.8%.

16. Both the development footage ratio, at 5.2 ft, and the time required to reach full production, at 165 days, are much greater than in the z-system layout. However, the major advantage of this layout over the z-system is the facility for retreat mining which, in terms of output per machine shift, can give up to 30% more than an advancing face of comparable length.

17. The easier and quicker transfer of equipment from one wall to the next reduces the face transfer downtime to an estimated 19 days.

Layout III - Multiple-entry Retreat

18. A conventional US longwall layout is illustrated on Plate 25. Six panels can be fitted into the block resulting in a wall length of 833 ft (five would have necessitated a longer face than is deemed practicable).

19. Entry pillars and protective pillars are similar in size to the other layouts. Sterilisation by entries and pillars amounts to 26% and the recovery of coal from the block is 82%.

20. The time needed to reach full production is 245 days due to the development of the five main entries along the southern edge and the two groups of panel development entries required for the first wall. The downtime to transfer the equipment between faces has been estimated at 19 days, which is similar to Layout II. The provision of better transport facilities in the three-entry system should ease the transfer problem, despite the fact that the equipment must travel the full length of the block each time. There are also fewer powered supports and other face equipment items to transfer owing to the shorter faceline.

21. Although this multiple-entry layout may be considered to have a high development roadway content, the provision of long panels limits the development footage ratio to 6 ft per 1 ft of wall advance, which is not significantly greater than the ratio for Layout II, although much more than for Layout I.

22. Other detailed arrangements are possible with this system of multiple-entries, but the design illustrates the principal features of a conventional longwall layout in the USA.

ROOM AND PILLAR

Introduction

23. The room and pillar system of mining is characterised by a high degree of flexibility. The system of mining within the panels incorporates crosscuts laid out at right angles to the general direction of advance so additional panels can be conveniently developed off at right angles to the original panel. Where pillars are not extracted, each panel can provide access or development entries to subsequent panels. Thus, there is a minimum of delay in reaching full production.

24. Owing to the inherent mobility of the equipment used in room and pillar operations, production in a new panel can be commenced with the minimum of delay after a previous panel has been worked out.

Objects of the Layout

25. The following is a list of the desirable characteristics of a room and pillar layout:-

(i) The required production rate should be achieved as soon as possible after the commencement of operations.

(ii) The production panels should be located reasonably close together to minimise conveyor belt lines and facilitate supervision.

(iii) The maximum amount of available face room to develop new panels should be maintained during the period of mining of the given area so that other panels can be quickly brought into production should work be suspended for any reason on a working panel, eg poor mining conditions.

(iv) Panel lengths should be chosen for maximum life to minimise section moves and the accompanying construction work.

(v) The moves from panel to panel should be arranged so that distances traversed between old and new working areas are as short as possible.

(vi) Panels should be worked to reach the boundary of the reserves, thereby fully exploring the reserve area prior to full extraction of the area.

Obviously, some of the above requirements conflict, so compromises in the phasing of panels must be made to gain an optimum balance between opposing requirements.

Shape of Reserve Block

26. The shape and size of the block to be developed and mined causes an inter-play between the optimum parameters of panel design and area layout. Where the shape and size of a block permits the mining of a number of adjacent parallel panels, the width of the block determines the number of panels. Where the width of the block is small, it may be necessary to mine slightly wider or narrower panels in order to extract the whole block. Where the block is of irregular shape, the orientation of the panels has to be adjusted to obtain as many "standard" panels from the block as possible, extracting the remaining pockets of coal either by local variations in panel width or additional short panels. In many cases, the demarcation of the boundary is not known precisely because of the presence of faults, etc. Where this type of uncertainty exists, the panels can be directed towards the supposed terminal position and stopped when the boundary limit is determined by underground exploration.

27. In the simulated block, the boundaries have been arbitrarily set at a square with 6,000-ft sides. This largely determines the orientation of the panels, which are laid out parallel to one of the sides of the block.

Layout I
(Plate 26)

28. It is assumed that the orientation of the access to the block is east-west. This allows the initial panel to be driven along the southern edge and worked across the full extent of the block. It is conceptually the simplest. Panels are subsequently driven to the northern boundary to extract the full block, giving long-life panels. While this arrangement gives the minimum number of panels, and consequently panel moves, the length of each panel, at approximately 5,700 ft, means that tandem conveyors would have to be used. In consequence, the total length of conveyors operating within the block is greater than that of the subsequent layouts, and the distance between working panels would make supervision difficult owing to the length of time needed to travel between them.

Layout II
(Plate 27)

29. This layout again uses an initial east-west panel to gain access to the area. A panel is subsequently developed in a northerly direction up the centre of the block. The main body of coal is then extracted by east-west and west-east panels taken off the centre development. A disadvantage of this layout is that it is slow to reach full production, but the production panels, at less than 3,000 ft in length, are convenient for conveying and supervision.

Layout III
(Plate 28)

30. This is a modification of Layout II and has several north development panels which are shown as one development along the eastern boundary, one along the centre and one along the western boundary. The main area is extracted by a series of east-west and west-east panels driven off the main development. This layout has the following features:-

(i) Full production can be achieved in a short time by operating the initial panel, the first north development and several east-west panels simultaneously.

(ii) The individual panel lengths are comparatively short, approximately 1,400 ft.

(iii) The short panel length makes for easy supervision and transport.

(iv) The short panels could be sealed-off, if necessary, without sterilisation of a large block of coal.

(v) Owing to the limited runs, the life-span of the operating panels is short. The layout is suitable for pillar retreat operations as the time between pillar formation and the cessation of pillar extraction reduces the risk of spontaneous combustion.

Herringbone Layout

31. When crosscuts are laid off at an angle other than 90^0 from a central entry in a panel, the resultant arrangement is termed a herringbone layout. It is convenient to drive the subsidiary panels off the development panel at the same angle as the crosscuts. In the case of the crosscuts being laid off at 60^0, as is common where continuous haulage systems are used, subsidiary panels could also be laid off at 60^0. This layout is shown on Plate 29 and has the following features:-

(i) The subsidiary panels are difficult to develop initially.

(ii) At the end of a panel's life, irregular remnants of coal are left against the boundary.

(iii) Panels tend to be of unequal length. If a particular length of run gives the optimum life and convenience, then only a limited number of panels will approximate to this ideal, the remainder being significantly different in length.

Preferred Layout

32. With the absence of steep grades, the preferred layout for the extraction of the simulated block is Layout III (Plate 28). Full production is reached quickly and a reasonable degree of flexibility is provided. If it is decided to retreat the panels, then to obtain a good degree of stability for the initial east-west panel, the adjacent short east-west subsidiary panels would be left unmined to form a barrier until the end of the life of the block.

Number of Panels -
Shuttle Car Haulage

33. The number of panels in each sub area of the block is determined by the width of the individual panels and the panel barrier pillars. Where the dimension of the sub-area is not exactly divisible by the combined width of a panel and barrier, panel width or barrier thickness can be slightly adjusted.

34. If the shuttle car haulage panel design is adopted with a 9-entry layout, each panel will be 340 ft wide, rib to rib. If 60-ft barriers are left between panels, then the total width will be 400 ft. Assuming that the initial development panel has a similar geometry, the remaining 5,600 ft for subsidiary panels, gives 14 subsidiary panels per sub area, ie a total of 4 development and 56 short panels for the whole area (see Plate 30).

Number of Panels -
Continuous Haulage

35. The continuous haulage panels are limited to five entries due to the restricted reach of the haulage system. The 5-entry design with 20-ft entries at 50-ft centres, measured along the line of the crosscuts, gives a panel width of 193.2 ft, barrier to barrier. Allowing 39 ft for the barriers provides for a total of 26 panels and 25 barriers to be laid out within a 6,000-ft block. Hence, there would be a total of 4 development sections and 100 subsidiary panels within the simulated block (see Plate 31).

36. The conveyors of the subsidiary panels must discharge on to the main conveyor system of the development panels. This requires the crosscuts of the development to be in line with conveyor entries of the subsidiary panels, ie they must be at the necessary centre distance and at an angle of 90^0. The crosscut pattern of the development panels, therefore, must be adjusted at each subsidiary breakaway position. This will cause a slight disruption to the working of the main development panels.

Number of Panels - Retreat

37. Retreat working requires not only large initial panel pillars but also substantial inter-panel barrier pillars. This increases the combined width of panel and barrier, thus reducing the number of panels that can be laid out within a block.

38. In the case of a 9-entry shuttle car haulage design with the pillars laid out at 60-ft centres, the panel width becomes 500 ft. This allows a total number of 10 panels, including development, to be laid out within a distance of 6,000 ft with a balance of 1,000 ft for barriers. The barrier pillar dimensions could then be 100 ft between panels and 200 ft between the initial development panel and the subsidiary panels.

39. If retreat working were to be adopted with rectangular pillars of 32 ft x 57 ft, the width of the panel would become 418 ft. This would permit 12 panels, including development, with a total width of barrier pillars of 984 ft.

SHORTWALL

40. Shortwall has much in common with room and pillar. The same general principles apply in the construction of a layout to extract an area of coal. As the extraction is accompanied by the creation of large abutment loads, it is necessary that the shortwall panels are not mined close to the main development access entries until such time as the entries are no longer required.

41. The total width of a shortwall and associated chain pillars is 256.6 ft. This would allow 22 sets of walls and associated development entries.

Protection Pillars

42. As the shortwall is retreated, an abutment load develops on the solid coal ahead of the wall. If the wall is permitted to completely mine out the developed pillar, the abutment load will move over the initial development entries and probably destroy or render them unfit for use.

43. In order to protect the development entries, the walls must be stopped before the abutment loading on the entries reaches a value that can cause damage.

44. If the area is initially pre-developed to the boundaries and the walls taken "on the retreat", then it will not be necessary to leave the protection pillars or pillars as large as would have been necessary had the development been required for continuous access.

Initial Production

45. Full production with the layout depicted on Plate 32 is obtained from three producing sections. In order to achieve a high level of initial production, two sections are employed on main development whilst the third is used for panel development. Actual shortwall extraction is only commenced when the main development reaches the northern boundary. This is to enable the shortwalls to be mined systematically from north to south and hence protect the initial development panel from the effects of abutment loading.

CHAPTER XIX

OUTPUT AND PRODUCTIVITY

INTRODUCTION

1. The principal objective of a coal mining enterprise is to produce coal safely and efficiently in terms of labour and other costs. Key criteria in coal mining are the output achieved by the adopted system and the amount of labour required to operate it. The quotient of output and labour is the productivity of the system, which may be expressed in tons per manshift or in tons per man-hour.

2. As the cost of labour is a significant proportion of the total cost of the mined coal, generally the higher the productivity of a system, the lower the cost per ton mined.

3. In thin coal, where costs, in general, are high, it is of vital importance that the mining system is designed to yield the highest productivity so that the operation can remain competitive with thicker seams and with other sources of fuel.

4. In mechanised mining, the ultimate rate of mining is limited by the rate the mining machine can cut and load the coal or the rate at which the haulage system can remove and transport the product.

5. The ultimate bulk output is the product of the mining rate and the number of hours or minutes the system can be worked per day.

6. A mining enterprise is viable when the revenue from the sale of the product (proceeds) equals or exceeds the total cost of production, including overheads. As many of the production costs are of a fixed nature, there is normally a "break-even" level of production at which mining costs equal the proceeds. Any output above this level yields a profit and the amount of this excess above the break-even point is termed "marginal tonnage". Hence, the object of the layout and design of a mining system is to maximise this marginal tonnage.

LONGWALL

7. Longwall mining has a high output potential.

8. There are two predominant longwall methods, namely, advance or retreat, and two basic types of equipment, ploughs or shearers. The shearer has a wider range of application than the plough. Both plough and shearer longwalls are worked on the same basic principle of taking a web of coal off the whole length of the face in one continuous cutting run. In the case of ploughing, the webs are only a few inches thick and the plough travels at a high speed. With the shearer, the thickness of the web is normally in the range of 1.5 ft to 3 ft and the speed at which the shearer traverses the wall, the "haulage speed", is lower than that of the plough.

9. The operations of the shearer consist of making a cutting run along the wall and, at the gate end, manipulating the machine into a "new" cut ready for the return run.

10. The face can be so arranged that the shearer cuts in one or both directions of travel. With bi-directional shearing, the supports and conveyor can be advanced immediately the shearer has passed on its cutting run. With uni-directional shearing, the conveyor and supports must be left until the shearer has completed its cutting run and has been retracted back along the wall.

11. Bi-directional shearing has the advantage that no time is lost flitting the machine along the "empty" part of the face, and roof control is easier as the supports can be advanced very shortly after the new roof is exposed.

12. At the end of the cutting run, the shearer has to be advanced one web thickness forwards to enable the new run to be taken. This can be done either by running the shearer off the wall into the gate at the end of the wall and hydraulically pushing the shearer forward with the conveyor, or by backing the machine along the track, advancing the conveyor at the end of the face and then cutting back again to the end of the face. In this operation, the shearer cuts on a curved path following the snake of the conveyor. It has the advantage that an open space at the end of the face is not required and, if necessary, the shearer can be made to cut from rib to rib of the whole face.

13. At the end of each complete shear, it is normal to stop cutting in order to check the cutting picks and attend to the lubrication of the machine prior to repeating the cycle.

Potential Output

14. The time to cut from one end of the wall to the other can be calculated and an allowance made for the extra distance cut when moving over into the new track at each end. To this time must be added an allowance for pick changing and servicing the shearer at the end of its run.

15. The number of shears per shift is found by dividing the available shift time by the time per shear.

16. The tonnage per shear is the product of the length cut, height, web thickness and the density of the coal.

17. The tonnage per shift is the product of the number of shears per shift and the tonnage produced per shear. Annual tonnage is obtained by multiplying the shift tonnage by the number of shifts worked per year, taking into account the dead periods for installation and salvage operations.

Actual Output

18. The actual output obtained from longwalls falls short of the potential output for many reasons. Stoppages can be of short or long duration and attributable to a variety of causes. Any stoppage of the conveying system whilst the shearer is on its cutting run, stops the face conveyor and hence the shearer. In a well planned and organised mine, the frequency of conveyor stoppages is low but, even so, there is usually some stoppage of the conveyors during the shift.

19. The shearer and conveyors require a continuous supply of electric power and the shearer also requires water for cooling and dust suppression. Some high-powered conveyors require water for cooling the electric motors and gearboxes. Any disruption to the supply of power and water will, therefore, bring the entire system to a halt.

20. The running of the wall requires the simultaneous availability of all the conveyors and the shearer. As with any other equipment, there is a probability of a breakdown but, with good maintenance, this probability can be kept low. Breakdowns can be of short duration, eg repairs to some electrical component, or can last several shifts, as may occur in the case of failure of a major piece of equipment.

21. When calculating the actual tonnage, an allowance can be added to the actual cutting time to take care of random stops of short duration, and an adjustment made to the available face time to counter stops of long duration.

22. Delays can occur owing to failure to maintain control of the roof. If control of the roof has been lost, the wall must generally be stopped while remedial action is being taken.

23. The output per shift is given by the following equation:-

$$\text{Output per shift} = HWMLR \left((L+C) \left(\frac{1}{S} + D\right) + T \right)^{-1}$$

Where:

H = extracted height

W = width of web

M = operating time per shift
(shift less travel, food and allowance for major downtime)

L = length of face

R = density of coal

C = extra distance travelled or "shuffle"

S = machine cutting speed

D = allowance per unit distance for short duration stops

T = time to change picks and service the shearer at the end of each cut.

24. In cases where the whole seam is extracted, ie no bottom or top coal is left, the parameters H and R are fixed. The relationship between the cutting or haulage speed and the width of web is a function of the design of the cutting drum, the rotational speed of the drum, and the power, haulage force and stability of the shearer.

Effect of Changes in Parameters

25. In the design of a longwall panel, it is possible to specify, within limits, the capability of the mining machine and, in operational management, to control the extent of delay elements in the cycle of operations by the allocation of resources. The effect of changes are shown on Plates 33 to 37.

26. Plate 33 shows the effect of shearer haulage speed at a constant depth of web. It can be seen that increases in the haulage speed show an almost linear increase in the tonnage potential of the face. The major limitations of the haulage speed are the power requirement for cutting and hauling, and the design of the shearer drum. These factors emphasise the advantage of obtaining the highest powered machine available within the limitation of the available height.

27. Plate 34 examines the effect of variations in depths of web cut at a constant shearer output. There is a rapid increase in output as the depth of web is increased, even with a commensurate decrease in the haulage speed. The maximum depth of web is limited by the clearance capacity of the cutting drum, the span of roof that can safely be exposed and the stability of the shearer.

28. With a buttock shearer, the width of the machine determines the minimum depth of web, as the body of the machine must be accommodated within the web.

29. Plate 35 shows the effect of delays during the cutting run along the wall. These are unplanned delays which can be minimised by ensuring good supervision and maintenance of the equipment.

30. Plate 36 shows the effect of varying the shuffle distance per shear, ie the distance that is double-cut on the curve in order to advance the shearer and conveyor into the new track. The plate illustrates that output is not particularly sensitive to this distance. For a shorter face, however, the effect of the shuffle would be more significant.

31. Plate 37 shows the effect on output of a variation in the length of the face. It can be seen that for an increase of length on a relatively short face, there is a substantial increase in tonnage. However, as the face lengthens, the increase in output is less per increment of length.

32. The major technical limitation to face length is the strength of the conveyor flight chain and the available power of the conveyor drives. Where long faces have been laid out with high-capacity shearers, difficulties have arisen in starting the conveyor if it has been stopped when fully laden with coal.

Outputs for Candidate Longwall Designs

Design I - Z-System

33. The data assumed for the calculation are as follows:-

Wall length	-	960 ft
Headgate	-	25 ft
Shuffle distance	-	90 ft
Seam height	-	33 in
Web thickness	-	36 in
Coal density	-	84.24 lb/ft^3
Haulage speed	-	11 ft/min
Delays per shear	-	30 min
Shift time	-	300 min

For practical purposes, the number of decimal places recorded has been restricted and this gives rise to apparent minor discrepancies.

Time/shear	=	$\frac{960 + 90}{11} + 30$	=	125.45 min
Shears/shift	=	$\frac{300}{125.45}$	=	2.39
Output/shear	=	$\frac{985 \times 2.75 \times 3 \times 84.24}{2,000}$	=	342.3 tons
Output/shift	=	2.39×342.3	=	818.5 tons
Advance/shift	=	2.39×3	=	7.17 ft

34. During the extraction of the whole block, six panels will be mined out. This involves a total face advance of approximately 33,300 ft, which will be accomplished in 4,642 coal-producing shifts.

35. In order to establish the necessary access and open the wall lines, a total distance of 44,760 ft of development will be required. Assuming that this development is driven concurrent with the longwall operation, the average rate of development to replace the longwall panels will be 9.64 ft per shift, which, at a width of 15 ft, will yield 16.7 tons of coal.

36. Hence, the average output per shift will be 835.2 tons. In practice, however, the development section would not be operated at such a low average output, but rather at a level of approximately 150 tons per shift, giving an equivalent single-entry advance of approximately 87 ft.

37. When calculating the average annual output, an allowance must be made to account for the time when the longwall equipment is being moved from panel to panel. This move is estimated to require 50 shifts. It is assumed that the transfer operation will be conducted by all the standard production personnel.

Time to mine one panel = $\frac{5,550}{7.17}$ = 774 shifts

Number of shifts to mine and move equipment = 824

Hence, equipment is available for production purposes for $\frac{774}{824}$, ie 94% of the total number of shifts worked.

Assuming double-shift working for 230 days, the average annual output equals:-

$$835.2 \times 460 \times 0.94 = 361,140 \text{ tons}$$

Labour

38. Operating two production shifts and one maintenance shift per day would require the following labour:-

	Shift 1	Shift 2	Shift 3	
Foreman	1	1	1	
Shearer driver	1	1	-	
Chock operators	4	4	-	
Chock maintenance	1	1	-	
Headgate	3	3	-	
Tailgate	2	2	-	
Pump packing	-	-	3	
Mechanics	2	2	4	
	14	14	8	= 36

Development and Other Area Labour

Development and construction - one third of 2 crews per day	6	
Conveyors	6	
Maintenance	3	
Transport	4	
Supervision	1	
		= 20
Total labour		56

Average per production shift = 28

Productivity $= \frac{\text{output}}{\text{labour}} \times 94\% = \frac{835.2}{28} \times 0.94 = 28$ tons per manshift

Design II - Single-entry Retreat

39.

Wall length	=	946 ft excluding developed gates
Time/shear	=	$\frac{946 + 90}{11} + 30 = 124.18$ min
Shears/shift	=	$\frac{300}{124.18} = 2.42$
Output/shear	=	$\frac{946 \times 2.75 \times 3 \times 84.24}{2,000} = 328.7$ tons
Output/shift	=	794.1 tons
Advance/shift	=	7.25 ft
Length of panel	=	2,560 ft
Development footage/panel	=	13,323
For each ft advance of the longwall there is		$\frac{13,323}{2,560} = 5.20$ ft of development
Required development rate/shift	=	37.73 ft
This yields 65.5 tons of coal/shift		
Hence, output/shift	=	859.6 tons
Output/annum	=	859.6 x 460 x 0.88 = 347,966 tons

Labour

40. Longwall crews for 2-shift production:-

	Shift 1	Shift 2	Shift 3	
Foreman	1	1	1	
Shearer driver	1	1	-	
Chock operators	4	4	-	
Chock maintenance	1	1	-	
Headgate	1	1	-	
Tailgate	1	1	-	
Pump packing	-	-	3	
Mechanics	2	2	4	
	11	11	8	= 30

Development and Other Area Labour

Development (single-shift) 1 crew	9	
Construction/utility	4	
Conveyors	6	
Maintenance	3	
Transport	4	
Supervision	1	= 27
Total labour		57

Productivity $= \frac{859.6 \times 2}{57} = 30.16$ tons per manshift

Time to mine one panel $= \frac{2,560}{7.25} = 353$ shifts

Shifts to salvage and re-install equipment = 50

Hence, availability for production $= \frac{353}{353 + 50} = 88\%$

Hence, productivity over full cycle $= 30.16 \times 0.88 = 26.54$ tons per manshift

Design III - Multiple-entry Retreat

41.

Wall length = 831 ft excluding developed gates

Time/shear $= \frac{831 + 90}{11} + 30 = 113.73$ min

Number of shears/shift $= \frac{300}{113.73} = 2.64$

Output/shear $= \frac{831 \times 2.75 \times 3 \times 84.24}{2{,}000} = 288.76$ tons

Output/shift = 761.7 tons

Advance/shift = 7.91 ft

Length of panel = 5,350 ft

Development footage/panel = 28,970

For each ft advance of the longwall there is $\frac{28{,}970}{5{,}350} = 5.41$ ft of development

Required development rate/shift = 42.83 ft

This yields 74.4 tons of coal/shift

Hence, output/shift = 836.1 tons

Output/annum $= 836.1 \times 460 \times 0.93 = 357{,}684$ tons

Labour

42. Longwall crews for 2-shift production:-

	Shift 1	Shift 2	Shift 3	
Foreman	1	1	1	
Shearer driver	1	1	-	
Chock operators	4	4	-	
Chock maintenance	1	1	-	
Headgate	1	1	-	
Tailgate	1	1	-	
Mechanics	2	2	4	
	11	11	5	= 27

Development and Other Area Labour

Development (single-shift) 1 crew	9	
Construction/utility	4	
Conveyors	6	
Maintenance	3	
Transport	4	
Supervision	1	
		= 27
Total labour		54

Productivity $= \frac{836.14}{54} \times 2 = 30.97$ tons per manshift

Time to mine one panel $= \frac{5,350}{7.91} = 676$ shifts

Shifts to salvage and re-install equipment $= 50$

Hence, availability for production $= \frac{676}{676 + 50} = 93\%$

Hence, productivity over full cycle $= 30.97 \times 0.93 = 28.8$ tons per manshift

ROOM AND PILLAR

Potential Output

43. The potential output is the product of the instantaneous output and the time during which this output can be sustained. The cutting and loading rates for machines are normally quoted by the manufacturers in terms of tons per minute, higher and medium seam continuous miners being quoted in the 10-ton-per-minute range and lower seam models being quoted around 5 tons per minute.

44. If these figures were realistic and could be maintained "continuously" then, at a rate of 5 tons per minute, a section working an 8-hour shift should produce 2,400 tons per shift. This level of output is seldom attained even in medium or thick seams where high-capacity equipment is used.

Actual Output

45. The reasons the above output levels are not achieved are twofold. One is that the output or capacity of continuous miners is normally quoted as the highest instantaneous output of the cycle. The other is that the continuous miners cannot be run for the whole of the shift; they have to be stopped while supports are installed and when checks are made for methane, and periodically have to be moved from place to place. The most frequent delay in the activities of the continuous miner is waiting for the haulage system. This is particularly so in thin seams where shuttle car haulage is employed.

46. The normal cutting cycle of a continuous miner consists of the following basic operations:-

(i) the miner is trammed to the face to be cut,

(ii) the cutting head is raised,

(iii) the head is sumped into the seam for a specified distance,

(iv) the head is lowered, "shearing" out coal from the face,

(v) the miner is backed off from the face to trim the floor.

Only operations (iii) and (iv) produce any quantity of coal. The rate of output is governed by the power, weight and traction of the continuous miner.

47. The average cutting rate is a combination of the sumping and shearing rates reduced by the time the machine is engaged on the non-productive elements of the cycle. Since the shearing rate is normally higher than the sumping rate, the combined production rate is lower in thin seams owing to the predominance of sumping.

48. The average rate can be found by the measurement of the actual machine cutting time (as opposed to metered motor time) and the output obtained during that time.

49. For a cutting run comprising a series of sumps and shears, there are further delay elements. The machine has first to be brought to the entry which involves tramming the continuous miner and handling the machine's attendant cable and water hose. Occasionally, where wooden supports are used, some have to be removed and replaced as the machine is moved.

50. On reaching the face, the machine can only cut a limited distance before the ventilation appliance must be extended. In thin seams this is normally a brattice which is run down one side of the entry. The Federal Code requires the ventilating appliance to be within 10 ft of the point of deepest penetration of the face, which limits the possible advance on the initial passes to less than 10 ft. The total distance that an entry can be advanced is further restricted by the legal requirement on support. The total length of the cutting run is limited to the distance the operator can control the machine and still remain under supported roof. For manually-operated machines, as opposed to remotely-controlled machines, this distance is from the head of the machine to the operator's control position. On reaching this position, either the machine must be moved to cut another place, or must be stopped while support work is carried out.

51. While cutting and loading operations are in progress, the continuous miner can only load when the haulage system is available to accept the load. In the case of shuttle cars, the capacity of the system is determined by the cars' useful load and the geometry of the cars' route, which changes for each place cut. Owing to the constantly changing pattern of delays, together with the constraints of cutting run limits, the calculation of time elements to cut every place in a multiple-entry section is long and complex.

52. The following is an example of part of the calculation of the tonnage capacity of a multiple-entry system using two shuttle cars.

Time to fill car = car factor ÷ average loading rate of continuous miner.

For a 3-ton car factor and a miner cutting and loading at an average rate of 2 tons per minute:-

$$\text{Time} = \frac{3}{2} = 1.5 \text{ minutes per car}$$

Average distances travelled, per car, from miner to change point equals distance at beginning of run plus distance at end of run.

$$\underset{\text{(ignoring driver reaction time)}}{\text{Time}} = \frac{\text{Distance}}{\text{Average car speed}}$$

Hence, on a crosscut centre of 60 ft, cutting from 50 ft to 60 ft, the distance is approximately 110 ft.

Tramming time $= \frac{110}{250} = 0.44$ minutes

Where 250 ft/min is taken as the average speed of the car, the average time per car, without bolting delays, equals loading time plus tramming time

$$= 1.5 + 0.44 = 1.94 \text{ minutes}$$

Assuming a 9-entry section with 60-ft centres, the distance from the change point to the ratio feeder will be approximately 300 ft at the barrier entry and the round trip approximately 600 ft. At a discharge time of 30 seconds, the round trip will take:-

$$\frac{600}{250} + 0.5 = 2.9 \text{ minutes}$$

ie, at this position the loading rate is being limited by the change point to feeder time of the car's route. This calculation is repeated for each entry and the total time to advance the section one crosscut centre is evaluated. Allowances are added for the time required to move the continuous miner from place to place and to extend services in the section.

53. The total tonnage mined by the section from an advance of one crosscut, is divided by the total time for the operation. The figure is then multiplied by the available shift time to give an average tonnage per shift.

54. The standard shift time, excluding any overtime, is eight hours from surface to surface. From this time must be subtracted the time required to travel from the surface to the working area. Time is also deducted for a meal break within the shift. In all the calculations, it has been assumed that travelling accounts for 45 minutes per journey and the meal break amounts to 30 minutes. This leaves 6 hours available for production and ancillary tasks.

55. The coal mining process depends on the availability of the various items of equipment used in the section. With increased mechanisation, a situation has arisen whereby if any piece of equipment suffers a malfunction or breakdown, the whole section comes to a halt. An exception to this is where shuttle car haulage is used. Should one car break down then, provided the car did not obstruct access to the continuous miner or the ratio feeder, production at a lower rate could be maintained using the other car in the section.

56. A frequent source of unplanned stoppages in a section is damage to power cables serving the continuous miner and other items of equipment. As repairing a cable can take approximately an hour, this represents a large source of downtime.

57. The handling of cables is naturally more difficult in lower workings and the use of systems that eliminate cable handling have much to commend them, eg a continuous haulage system where the continuous miner cable is carried and protected by the structure of the conveying elements.

58. Delay times per shift from breakdown of equipment can vary from a few minutes, with new equipment and good servicing, to hours with used equipment and indifferent maintenance.

59. Maximum production is achieved when the cumulative delays owing to all sources are minimised. The optimum number of entries should be chosen and the length of cutting runs designed to minimise the unnecessary transfer of equipment from place to place.

Outputs for Candidate Room and Pillar Designs

60. The calculation of output is a complex process and hence use has been made of a computer to derive the output per shift for each design and layout. A continuous miner capacity of 2 tons per minute has been used in each case. The outputs are as follows:-

System	Output/Section/ Shift	Section Shifts/Day	Annual Output
	tons		tons
9-entry room and pillar - shuttle car haulage	252	6	347,760
5-entry room and pillar - continuous haulage	381	4	350,520
9-entry retreat	223	6	307,740

Labour

61. The section crew normally comprises a supervisor, equipment operators, a mechanic and utility personnel. A typical complement is as follows:-

Supervisor	1
Continuous miner operator	1
Continuous miner helper	1
Shuttle car drivers	2
Roof bolter	1
Bolter helper	1
Utility	1
Mechanic	1
	9

In a continuous haulage section, the two shuttle car drivers would be replaced by the drivers of the mobile bridge carriers.

Labour for Three-section, Two-shift Operation

9 Entries - Room and Pillar
(Shuttle Car Haulage)

6 section crews	54
Conveyors	8
Construction (including section stoppings)	6
Maintenance	13
Transport	4
Inspection and supervision	5
	90

$$\text{Productivity} = \frac{\text{output}}{\text{labour}} = \frac{1,512}{90} = 16.80 \text{ tons per manshift}$$

Labour for Two-section, Two-shift Operation

5 Entries - Room and Pillar
(Continuous Haulage)

4 section crews	36
Conveyors	6
Construction	12
Maintenance	9
Transport	6
Inspection and supervision	5
	74

Productivity = $\frac{1,524}{74}$ = 20.59 tons per manshift

Labour for Three-section, Two-shift Operation

9 Entries - Room and Pillar Retreat

6 section crews	54
Conveyors	8
Construction	4
Maintenance	13
Transport	4
Inspection and supervision	6
	89

Productivity = $\frac{1,338}{89}$ = 15.03 tons per manshift

SHORTWALL

Introduction

62. The output and productivity of a shortwall system is the combination of the output and productivities of the development operations and the actual working of the shortwall.

63. The development phase of the operation is equivalent to a standard room and pillar mining system. It differs in that the size and shape of the entries and pillars are designed to suit the requirements of the shortwall, rather than being optimised to give the highest results in themselves. The number of entries is also restricted in order to obtain high development rates and to maximise the total percentage extraction.

Output for Development Entries

64. Plate 38 depicts the sequence of working nine 3-entry development layouts for the shortwall.

Tonnage for 60 ft advance	=	522.8 tons
Cutting time @ 2 tons/min	=	261.5 min
Total transfer distance between cutting runs	=	2,650 ft
Transfer time @ 20 ft/min	=	132.5 min
Allowance for conveyor and power centre extension/60 ft	=	150 min
Total operational time/60 ft	=	544 min
Available time/shift	=	300 min
Average advance/shift	=	$\frac{300}{544} \times 60 = 33.09$ ft
Output/shift	=	$\frac{300}{544} \times 522.8 = 288.3$ tons

Output for Shortwall

65. The basic cycle of operations for the shortwall is that the continuous miner cuts in a continuous fashion from one end of the shortwall through to the other. When the cutting run is completed, the miner and haulage are withdrawn from the wall and the track cleared. The powered supports are then advanced along the wall and the cycle repeated.

66. The cycle of operations is interrupted periodically to pull back the section of conveyor and the power centre. These operations are simplified as the power centre can be located in the entry adjacent to the wall which reduces the connections to be made during a power move.

67. The output for the shortwall is based on the following data, using a continuous haulage arrangement.

Continuous miner average capacity	=	2 tons/min
Miner-haulage transfer rate	=	20 ft/min
Time to move over supports and check picks	=	40 min/strip
Retraction of services	=	2 hours/60 ft = 20 min/strip
Available working time	=	300 min
Tons/strip	=	173.75 tons
Time/strip (cutting time + retraction time (transfer) + operational delays)	=	86.87 + 7.50 + 60 = 154.37 min
Strips/shift	=	$\frac{300}{154.37} = 1.94$
Tonnage/shift	=	1.94 x 173.75 = 337.65 tons
Advance/shift	=	1.94 x 10 = 19.4 ft

Combined Results

68. The ratio of coal produced on the development and on the wall is 1.994 tons from the wall for each ton of development coal.

69. Hence, for mining 1,000 tons, 334 tons would be produced in the development and 666 tons would be produced from the wall. This would require:-

$$\frac{666}{337} = 1.97 \text{ shifts from wall operation and}$$

$$\frac{334}{288} = 1.16 \text{ shifts from entry development}$$

ie 3.13 shifts would be required to produce 1,000 tons, giving an average of 319.5 tons/shift, with a corresponding average "advance" for the shortwall and development of 12.2 ft/shift.

70. In practice, it would be desirable for the continuous miner and haulage to be drawn off the wall at the end of a working day, and the supports moved over. For example, the shift's work would not be limited to 1.94 strips, but would be extended to two whole strips in order to prevent the roof from deteriorating between the powered supports and the wall.

71. The required output could most economically be achieved by working three sections for 2 shifts per day. This would result in a daily output of 1,917 tons, equivalent to an annual output of 440,910 tons.

Labour

72. The section crew for development would be as follows:-

Supervisor	=	1
Continuous miner operator	=	1
Continuous miner helper	=	1
Haulage operators	=	2
Roof bolter	=	1
Bolter helper	=	1
Utility	=	1
Mechanic	=	1
		9

73. During wall operations, the labour requirement would be:-

Supervisor	=	1
Continuous miner operator	=	1
Continuous miner helper	=	1
Chock operators	=	2
Utility	=	1
Haulage operators	=	2
Mechanic	=	1
		9

<u>Area Labour</u>

6 section crews	=	54
Construction	=	6
Conveyors	=	8
Maintenance	=	13
Transport	=	4
Inspectors and supervision	=	5
		90

$$\text{Productivity} = \frac{\text{output}}{\text{labour}} = \frac{1{,}917}{90} = 21.3 \text{ tons per manshift}$$

CHAPTER XX

COSTS

INTRODUCTION

1. For an operation to be profitable, the revenue from the activity must exceed the cost of that activity. Complete control of the revenue is often outside the sphere of the operator as much can depend on the strength of the open market, or in the case of a mine that produces for a captive market, an inter-company transfer price of the coal can be imposed.

2. One area normally under the operator's control is the cost of production; in coal mining there are several elements that contribute to the total cost of production. Unit costs are established when the total cost is divided by the volume of production. If an operation is structured to give the lowest total unit cost, the maximum profit will be attained when the revenue per unit is above this amount.

COST ELEMENTS

3. Mining costs may be classified as follows:-

(i) cost of lease or right to mine,

(ii) cost of exploration and evaluation,

(iii) cost of establishing the surface infrastructure,

(iv) cost of establishing and developing the underground facilities,

(v) cost of capital plant,

(vi) finance charges,

(vii) cost of production.

4. A problem in the establishment of unit costs is that many of the component costs are incurred at different times, which, in the life of a mining operation, can span over many years, the problem being compounded by the recent severe rates of inflation.

CAPITAL COSTS

5. To simplify the cost structure, items (i) to (vi) may be grouped, with their respective finance charges, and allocated to capital costs.

6. Capital costs are usually considered to be incurred up to the time of commencement of production. During the production life of the mine, a charge is levied on each ton of coal produced; this is designed both to pay back the outstanding capital investment and carry the cost of interest (finance) charged.

7. On the assumption that a mine has a fixed asset of coal reserves, a high percentage extraction reduces the unit cost per ton of the initial capital outlay.

8. A high rate of production will lower the interest component of the payback charge and reduce the total payback.

9. In order to illustrate clearly the differences in the candidate mining systems, the finance charges for the mobile mining equipment are separated from the more permanent items of plant, such as the mine fan and trunk conveyors, and included in the operating costs.

OPERATING COSTS

10. Operating costs consist of labour and labour-related charges, materials, power, administration, taxes, levies, royalties and depreciation.

Labour Charges

11. Labour charges cover the direct payments made to employees, the cost of holidays, fringe benefits, training, etc. The total cost of employing a person for one day has been taken as $150, based on 1979 labour rates and the United Mine Workers of America coal industry contract.

Materials

12. The material costs are determined by the requirements of the mining system adopted. In the case of conventional mining, such items as explosives and explosives accessories would be included. In room and pillar mining, using continuous miners, the cost of explosives would be replaced by the cost of cutting tools.

13. Room and pillar support costs would be mainly those pertaining to roof bolts and timber, while for longwall, support costs would be mainly for material to support the gates. The quantity of materials consumed is normally proportional to the tonnage of coal mined.

14. Power charges are a significant part of the overall costs. They are based on the total energy used and the maximum demand, with penalty clauses for such items as power factors below a specified norm.

15. The smaller the disparity between the maximum demand and the average use of power, the lower is the cost of a given amount of energy. Consequently, the

extension of an operation from a single-shift-per-day basis to a double-shift basis would not normally double the power charge as the maximum demand would remain at the same level. Power charges have been directly proportioned to output.

Administration

16. Administration charges comprise the payroll costs and expenses of such departments as Accounts, Purchasing, etc, and are normally considered to be a fixed overhead.

Taxes and Levies

17. These include such items as union dues, property taxes, insurance, Federal levies, etc. The total amount is partially dependent on tonnage although some items are fixed. In general, the 1979 rate per ton was in the region of $3.05.

Royalties

18. Royalty payments are made to the owners of the mineral rights as mining proceeds. If the royalty agreement is based on a fixed payment per ton, then obviously the payment is not affected by the mining system used. If the agreement is based on the area of coal, the unit cost of the royalty diminishes with improved extraction from the in-situ reserves.

Depreciation

19. Most mining equipment has a life that varies from 5 to 10 years. During this period, provision must be made for its replacement. The simplest means of depreciating the equipment is by the straight-line method. If, for example, an item has a useful life of 10 years, its cost would be charged at 10% of its original value for each year of its life. Whichever accounting system is used, the depreciation charge is always dependent on the value of the equipment. Hence, a system utilising high capital cost equipment will carry a higher depreciation charge than one using lower capital cost equipment.

Finance Charges

20. If it is assumed that a mining venture is started on borrowed money, then the mining operation must cover the cost of the initial loan together with the payment of interest charges.

21. If the repayments are made on an equal annual basis over the life of the operation, they must be sufficient to cancel the debt. The ratio of the repayment to the initial sum borrowed is called the capital recovery factor (CRF) and is given by the equation:-

$$CRF = \frac{i\ (1 + i)^n}{(1 + i)^n}$$

where i is the effective rate of interest, and n the number of years.

The use of the CRF is applicable to both depreciation and interest, and represents a reasonably accurate assessment of the cost of finance. This formula can also be applied to items of equipment that comprise the mining sections; this is particularly important in this study as there are large differences in the capital costs of equipment for longwall, room and pillar and shortwall operations.

22. The rate of interest on borrowing money for mining ventures can be higher than for many other commercial transactions owing to the level of uncertainty or risk associated with mining. The risk is somewhat lower when applied to the purchase of equipment for an established mine as many of the uncertainties relating to the mining venture should have been removed by the time the equipment is purchased.

23. For simplicity, the effective life of the mining equipment used in the candidate systems has been taken as eight years. The effect of changes in interest levels is shown on Plate 39 which illustrates the variation in the percentage of the capital sum which has to be repaid per year at various interest rates. At an interest rate of zero, ie an interest-free loan, the rate is 12.5% or one eighth of the capital to be repaid each year. At an interest rate of 10%, the CRF is approximately 18.75%, whilst at a rate of 25%, the CRF increases to approximately 30%.

CANDIDATE SYSTEMS

24. The estimated capital and operating costs for each of the systems are given in Tables IX to XIV and summarised in Table XV. Royalties have been excluded from the totals as they are common to all.

25. The two areas that predominate the cost structure are the cost of capital and the cost of labour.

26. The capital repayment and depreciation form a fixed commitment which must be met irrespective of output tonnage. The unit cost of this element is totally dependent on output and the available market to dispose of the product. It is therefore essential, with a highly capital intensive scheme, to have an assured market for the product.

Effect of Extended Working Hours

27. Tables IX to XV are based on working two shifts per day, five days per week.

28. If the operations were extended to three shifts per day over a seven-day week, the operating time would approximately double, though an allowance would have to be made for the essential maintenance that is normally done on the non-producing shifts.

29. As a result of working 21 shifts per week, as opposed to 10, the output would approximately double. This would result in a reduction in the average working life of the equipment, probably from eight years to five years. (The working life of the equipment is a combination of service and time factor.)

30. The reduction in equipment life from eight to five years would increase the capital recovery factor by approximately 28.3%. This increase would be more than offset by the increase in output which would result in the unit cost of the capital recovery charge reducing to 64.15% of its previous value.

31. On the assumption that the material cost per ton remains static and labour charges are increased by 14% to compensate for working on a seven-day-per-week basis, the unit costs would be as shown in Table XVI. It can been seen that such a working arrangement reduces the total operating costs for all the systems, but has the greatest effect on those using equipment with the highest capital cost.

Effect of Change in Interest Rate

32. The capital recovery factor used in the calculations was based on an interest rate of 20%, which approximates to the prime borrowing rate at the time of the preparation of this report. Many previous exercises have been based on a rate of 10%.

33. If a 10% rate were used on an eight-year repayment, the depreciation/capital repayment charge would fall from approximately 26% of the capital base to about 19%. The cost structure of the system based on an interest rate of 10% is shown in Table XVII.

Effect of Inflation

34. Inflation is constantly increasing costs, though not all costs are affected in the same manner. The effect of inflation on material and labour costs tends to increase these year by year. In the case of equipment purchased on a fixed-term loan, the repayments would be fixed and therefore proofed against inflation. This effectively reduces the rate of interest in present-day monetary terms.

35. The purchase of new equipment would cost more owing to inflation. However, this would be purchased on new loan terms which, again, would show a progressively reducing repayment cost in constant-value terms.

CHAPTER XXI

COMPARISON OF CONCEPTUAL LAYOUTS AND DESIGN

INTRODUCTION

1. The purpose of this chapter is to compare major features of the conceptual layouts and designs and to highlight the differences and advantages each offers. It should be noted that the comparisons, particularly those of a quantitative nature, are those assumed to occur under the "average" conditions simulated in the block of coal. The effect of changing the physical and other parameters is specifically addressed in Chapter XXII.

2. Each system is examined in the following table:-

Criteria	Longwall			Room and Pillar			Shortwall
	Z-System	Single-Entry	Multiple-Entry	9-Entry	5-Entry	9-Entry Retreat	
Health and safety	A	A	B	C	C	D	B
Operating cost, \$/ton	15.90	16.80	13.46	16.99	14.33	18.46	15.35
Capital cost, 10^6\$	10.68	11.18	9.12	6.47	5.03	6.47	9.93
Annual output, 10^3 tons	361	348	358	348	351	308	441
Productivity, tons/manshift	28.00	26.50	28.80	16.80	20.60	15.00	21.30
Time needed to reach full production (1)	4	18	27	5	3	7	3
Continuity of production (2)	94.00	88	93	100	100	100	100
Utilisation of reserves (3)	92.80	87.80	82.00	64.30	59.40	74.40	81.90
Required legal variances	Moderate	Major	-	-	-	-	-
Environmental impact	B	B	C	C	C	D	C

Legend: A, B, C and D represent rank order of desirability where A is the best or most desirable and D the least.

1 = The time in months to reach full output.

2 = Percentage of time the section is available for production as opposed to being moved.

3 = Percentage of coal removed from the in-situ reserve.

3. It should be noted that while the capital cost of the equipment is given, the effect of the different levels of capital required is reflected in the operating cost, which is a combination of labour, material, capital, interest and depreciation charges.

HEALTH AND SAFETY

4. Of the systems evaluated, longwall has a superior safety rating owing to the virtual elimination of roof-fall accidents at the face as a result of the almost complete protection of the operators by the steel canopies of the powered supports. The multiple-entry longwall system is not quite so effective because the higher proportion of multiple-entry development has the same hazard potential as room and pillar working.

5. In the design of the longwall semi-advance layout (Z-system) and the single-entry retreat, steel sets, in conjunction with "pumped packs", provide support and stabilisation for the entries, which considerably enhances protection from falls in these areas. The cost of providing this support is largely responsible for the higher unit operating cost compared with the multiple-entry longwall system.

6. Shortwall occupies an intermediate position. This is because the large proportion of room-and-pillar-type work in this system and the greater prop-free distance, ie the distance from the wall to the powered supports, compared with a longwall, results in a greater area of unsupported roof.

7. Room and pillar retreat working is considered to be the most hazardous. This is because the nature of retreat work causes the roof to move, and there is only a limited space available for escape should roof control be lost. Ventilation of retreat workings tends to be haphazard in practice, which can result in unfavourable dust counts that are injurious to health. Accumulations of methane can occur in the goaf of the pillared area. This system also has a higher risk of spontaneous combustion than the others.

OPERATING COST

8. The operating cost is the sum of the capital repayment and interest charges and material and labour costs. Of the room and pillar systems, the layout employing five-entry, continuous haulage results in the lowest cost. The reason for this is the higher productivity in terms of labour and equipment which reduces both labour and unit capital charges. The material charges for the five-entry system are higher than for the wider, nine-entry system, because more stoppings are required for a given coal output, but this increase in cost is more than offset by the reductions.

9. The cost of the nine-entry retreat system is higher than all the other designs by virtue of the lower productivity of labour and equipment. This is because it has a wider section, designed to maximise utilisation of reserves, which increases the face haulage distance and thus limits the output per shift.

10. The shortwall system has lower material and labour costs than the room and pillar systems, but carries a higher capital recovery charge which makes the total operating cost higher than that of the five-entry, continuous haulage system.

11. The longwall systems have a low material cost and the lowest labour cost of all the systems but the total operating costs are inflated by the high capital recovery component. The material cost arises principally from the support and packing materials required to maintain the stability of the gateroads. In the case of the multiple-entry longwall design, these components are reduced and this provides the system with the lowest material cost and the lowest overall cost.

CAPITAL COST

12. The capital cost is the installed cost of all the required equipment in the producing sections together with the cost of services and haulage for each section.

13. While the cost of equipment for the five-entry, continuous haulage system is marginally higher than for the other room and pillar systems, the higher bulk output means that only two sections are required to achieve the nominal design target of 1,500 tpd. This gives the five-entry, continuous haulage system the lowest capital cost of all the systems considered.

14. The capital costs of the longwall systems vary with differing wall lengths because of the different requirements of powered supports and wall equipment. The semi-advancing Z-system layout has the longest wall length and requires a continuous miner to advance the headgate. An additional machine is also required in the tailgate to dig out floor material to gain adequate height and cross-section for ventilation.

15. The accompanying development section for the Z-system has been costed at one-third of the actual capital cost of the equipment. The reason being that because of the low utilisation of this equipment, one development section could adequately serve three longwalls. In a large operating mine, the equipment would normally be transferred to another area when development was completed.

16. The single-entry system requires services for the wall and for the separate development operation. The increased cost of conveyor, track, cable, etc, more than offsets this layout's shorter wall cost. The single-entry system has the highest overall capital cost.

17. The multiple-entry longwall has a shorter wall length than the other designs owing to the loss of ground caused by the width of the three-entry development gates. As no packs are required for this layout and design, there is no capital cost element for the various items of equipment that make up the pump-packing system. The short length and simpler system give the multiple-entry longwall the lowest capital cost structure of the three longwall designs.

18. The shortwall layout has a relatively high capital cost. Although the cost of a single operating section is moderate, three sections are used in the layout. This gives the shortwall arrangement a higher capital cost than the multiple-entry retreat longwall. However, if the comparison is made on the basis of the capital cost per ton of annual production, the higher output potential of the shortwalls gives a lower value.

ANNUAL OUTPUT

19. A nominal 1,500 tpd of run-of-mine coal was used as one of the design parameters. The actual outputs vary from this amount owing to the interaction of other factors. 1,500 tpd is equivalent to 345,000 tpa for a nominal 230-day working year. The only system that falls short of the target is the room and pillar nine-entry retreat system which employs three sections to achieve 308,000 tpa. It was decided to limit the number of sections in this layout to three otherwise problems in relation to ventilation and control could occur.

20. The shortwall layout, with three producing sections, yields 441,000 tpa as the target output figure could not be achieved with only two sections.

21. The outputs for all the systems are based on working two shifts per day, five days per week. This output could be virtually doubled by working three shifts per day, seven days per week, which would give advantages in terms of cost and revenue.

PRODUCTIVITY

22. The productivity has been measured in tons per manshift. The figures do not include surface workers, clerical staff, or other underground personnel whose work is outside the boundaries of the simulated block.

23. The longwall systems have a productivity of around 28 tons per manshift; the room and pillar systems, using shuttle cars, approximately 16 tons; and the five-entry, continuous haulage and shortwall systems approximately 21 tons.

TIME NEEDED TO REACH FULL PRODUCTION

24. This time is given in months. It is assumed that some output will occur immediately development starts and, as the development proceeds, additional sections will be started. The layouts employing fast-advancing, narrow-width panels open up the block the fastest and, consequently, are able to reach full production in the least time.

25. The longwall multiple-entry retreat system cannot reach full production until the first wall has been completely blocked out. The time of 27 months is based on one set of development equipment being used for 10 shifts per week. This could be shortened by working more shifts per week in the initial development phase of the operations, or by using more equipment at this stage.

26. The start-up time has a substantial effect on the finances of a new mine, particularly a deep mine with a high capital investment for plant and infra-structures; finance charges on capital have to be met while development is in progress during a period with limited output to generate revenue.

27. For example, a capital investment of $10 million up to the time of the commencement of development, with no revenue generated during the development phase until full production was reached, would increase the capital debt from $10 million to $15.1 million at an interest rate of 20% pa over a period of 27 months. At a lower interest rate, eg 10%, the capital debt would still increase by $2.4 million to $12.4 million over the same period.

28. In order to reduce this burden, it is normal to defer expenditure on those items that are not necessary until full production is available. These items may include coal preparation plants and rail-road facilities.

CONTINUITY OF PRODUCTION

29. The seven layouts have different characteristics in terms of their output. The room and pillar systems yield an almost continuous flow of production with a minimum delay as equipment is moved from panel to panel. With adequate preparation the short distances enable a section to be moved from panel to panel in one day. If a move could be scheduled to occur over a weekend, almost full production could be achieved by the first shift of the new week.

30. The shortwall system has two levels of production: that of development and that of wall extraction. The phases occur for each panel roughly in the ratio of 1:2 in time. There is no major loss in production at the end of each panel as the continuous miner and haulage equipment can be moved to a development area and commence production while the powered supports and ancillary equipment are being salvaged and re-installed on a new wall.

31. The longwall system has inherent and long delays in moving from panel to panel. These result in periods of little or no output which continue until the equipment has been salvaged, or reconditioned where necessary, and installed on the next panel. A nominal period of 25 days has been allowed for this move from panel to panel. During this time, it is assumed that all available personnel could be utilised by a re-organisation of shifts.

32. The extent of the continuity is expressed as a percentage of the number of days the equipment works in a panel, compared with the time elapsed between commencement of production of one panel and production of the next panel using the same equipment.

UTILISATION OF RESERVES

33. The total bituminous coal resources of the USA (measured, indicated and inferred) amount to some 679 billion tons (see Chapter I, paragraph 8(v)). While this is a considerable figure, it is finite and, hence, it is desirable that the mining system should maximise the extraction of coal.

34. The figures in the comparison table on page 211 show the proportion of coal extracted from the total reserve of the area. No attention has been paid to the effect of mining on any other seams above or below.

35. The room and pillar five-entry system has the lowest utilisation in the seam at 59.4% extraction. The use of the wider panels with proportionally less barrier pillars increases the extraction to 64.3%. Retreat mining increases the extraction to approximately 74.4%, assuming a good measure of pillar extraction.

36. The use of retreat, however, is damaging to other seams above and below the workings and may sterilise adjacent seams due to the irregular and high local stresses generated in the strata by the failure to extract certain pillars or portions of pillars.

37. The shortwall layout gives an extraction of 81.9%. The use of this system could also have an adverse effect on other seams, but not to the same extent as retreat working owing to the complete extraction over the width of the shortwall, which results in a more ordered distribution of stresses.

38. The longwall system achieves a 100% extraction within the panels, but the overall utilisation is somewhat less as barrier pillars are left to protect the main development entries and to isolate specific areas within the block should this be necessary.

39. The Z-system and the single-entry retreat leave no barrier pillars between individual panels and, hence, achieve the highest percentage extraction. The multiple-entry retreat layout leaves a series of chain pillars between the panels and achieves a lower percentage extraction of 82%. The longwall layouts, particularly where no pillars are left between panels, have only a limited effect on other coal seams which may overlay or underlay the workings.

40. The amount of coal extracted can have two impacts on the cost structure of a mine. Where the reserves are limited, a reduction in the utilisation leads either to a reduced annual output or a reduction in the life of a mine.

41. The reserves of the simulated block are 4.17 million tons of coal. If a 100% utilisation could be achieved, then at 1,500 tpd, the block would last 12.1 years. The life of the block for the seven layouts, at full production, is given below:-

System	Annual Output	Utilisation	Production Life in Years	CRF @ 20% Interest
	(10^3 tons)	(%)		
Z-system advance	361	92.8	10.7	0.2331
Single-entry retreat	348	87.8	10.5	0.2346
Multiple-entry retreat	358	82.0	9.6	0.2420
9-entry, shuttle cars	348	64.3	7.7	0.2651
5-entry, continuous haulage	351	59.4	7.1	0.2755
9-entry retreat	308	74.4	10.1	0.2377
Shortwall	441	81.9	7.7	0.2651

42. If the block represented an entire mine, it would have to carry the whole mine cost. Assuming a total capital cost of the mine and equipment of $20 million and a zero value of assets at the end of its life, then extension of the life from 7.1 years to 10.7 years would reduce the overall cost of production by:-

$$\frac{\$20 \times 10^6\,(0.2755 - 0.2331)}{361{,}000} = \$2.35/\text{ton}$$

43. Royalties can be negotiated on a tonnage-mined basis or on a coal-area basis. If the basis is that of coal area, the greater the utilisation, the lower the royalty cost.

LEGAL VARIANCES

44. The US coal mining regulations are based mainly on room and pillar mining; longwall mining, in terms of the Federal Code of Regulations, being considered a modified form of pillar extraction.

45. In order to operate the semi-advancing Z-system, several variances from the law may be required. The concurrent use of a continuous miner and shearer on the longwall could be interpreted as the operation of two "sections" as defined in CFR 30-75.319.1 and would, hence, require separate ventilation splits. The use of the single headgate partitioned by a packwall would also require approval.

46. The operation of the single-entry retreat system, variations of which are frequently used in Europe, would require variations in the development stage and during the wall operation. In the development stage, approval would have to be obtained to ventilate the entry using a partition to divide the entry, one side being an intake with the other a return, and subsequently operate a conveyor haulage in the return side. A variation would also be needed in terms of the requirements on escapeways (CFR 30-75.1704).

47. During the operation of the wall, it would be useful to use the conveyor road as part of the wall's ventilation circuit. This would again require a variance.

48. The multiple-entry layout has been designed to comply with the existing requirements of the Federal Code of Regulations.

ENVIRONMENTAL IMPACT

49. The working of coal underground normally has some effect on the surface. The extent of the effect, and the use to which the surface can be put, is dependent on the type and extent of the underground workings.

50. With room and pillar workings, where the pillars are adequately sized and left intact, there is the minimum disturbance of the surface. There is always a possibility that the pillars might fail some time in the future, owing to time-dependent failure, so, under normal circumstances, land that overlies the workings should not be used for building purposes unless the workings are backfilled; an expensive procedure.

51. In the case of the room and pillar retreat layout, there would be extensive distortion of the surface, though the land should still be fit for agricultural use. However, the land would be unsafe for building purposes as some of the remaining pillars could fail at any time in the future. Stabilisation of the workings would be an extremely difficult and costly procedure owing to the lack of access into the workings caused by the mining method.

52. With longwall, there would be an immediate surface effect which could cause damage to any existing buildings. An advantage of longwall is that once the surface has been lowered, there should be no further movement. The effect is an overall lowering of the surface by approximately 60% of the seam height. In the case of a 33-in extraction, this results in a lowering of approximately 20 in. After the cessation of mining activities, the land may be used for any purpose, unless the general lowering has materially affected the drainage pattern of the surface.

CHAPTER XXII

SENSITIVITIES

INTRODUCTION

1. The preceding chapters have considered a discrete block of coal under ideal conditions. Further, the size of the simulated block was such as to constrain some of the dimensions used in the panel design so that the optimum values may not have been realised in all cases. The purpose of this chapter is to indicate the effect of change in variables and to establish the effect of the changes on the matters of prime importance to the industry: production and operating costs.

DEPTH OF COVER

2. The depth of cover is of major significance in the choice of mining system and the economics of the operation. In the case of the room and pillar system, it is the major factor used in the design of the size of pillar necessary to ensure the stability of the mining system. At increasing depths, the pillars have to be made larger to accommodate the increasing load of the superincumbent strata.

3. An increase in the size of pillar for a given width of entry causes a decrease in the recovery of coal and utilisation of the coal lease area. The effect of the depth is shown on Plate 4.

4. As pillars become larger there is an increase in the haulage distance within the producing sections. This has a bearing on output, productivity and cost of production. The effect of increased centre distances is shown on Plate 40, which illustrates the falling off in the output that occurs. As the number of persons in the section remains constant, the effect of diminishing output is to reduce productivity and to cause an increase in unit cost.

5. Owing to the increase in pillar size with depth, room and pillar operations become both wasteful of reserves and unproductive at depth. It is not possible to state a precise cut-off depth at which the employment of room and pillar should be precluded, but its use at depths exceeding 1,500 ft must be carefully evaluated.

6. Depth also has an effect on longwall, but to a lesser extent than with other systems. When barrier or protection pillars are used, these become larger, marginally reducing the utilisation of the area.

7. The increased strata pressures at depth increase the support costs for roadways and the amount of roadway or entry maintenance work required. In the case of advancing longwalls, the gates are situated in the mined-out area and are, to an extent, protected from the strata pressures as the mined-out area has been stress-relieved.

FAULTING

8. Small faults can cause operational difficulties and, if the displacement is high enough, can form physical limits to the extent to which an area can be mined.

9. The presence of a physical barrier can either limit the width of a panel or terminate a panel's advance.

10. As previously indicated, it is desirable that the length of panel run be maximised to save time on panel-to-panel moves. This affects longwalls to a greater extent than room and pillar or shortwall as a longwall face requires a considerable time to be salvaged and re-installed.

ROOF

11. In room and pillar, a competent roof allows wide rooms to be excavated with the minimum of support. As the quality of the roof deteriorates, the width of the entries and rooms has to be reduced and the density of artificial support increased. Decreasing room width reduces output and productivity owing to the reduced tonnage cut per foot of advance. Plate 14 illustrates the effect of room width on output for a continuous haulage section. It will be noted that the relationship between output and room width is not a smooth function. This is due to the width making discrete changes in the number of cutting runs necessary to complete the crosscuts.

COAL IMPURITIES

12. If the run-of-mine coal is unsuitable for the market, then it must be processed. This is normally a combination of screening into various size ranges and the removal of waste or inferior material. The effect of coal preparation is to reduce the volume of coal available for disposal compared with the volume mined. The ratio of disposable coal to mined coal is a "yield" or "vend" which can vary from almost 100%, where the only preparation is screening, to below 50% where coal is prepared for a metallurgical market.

13. The yield has a major effect on the cost structure as, when the yield is low, the mining cost is multiplied by the reciprocal of the yield.

14. The mining process of cutting coal off the solid is influenced by the hardness of the coal and the presence of impurities. Where there are hard bands of waste material within the coal seam, the wear and damage to the tools can be increased. This increases the cost of bit replacements and reduces output owing to "slow cutting" and delays to production occasioned by changing bits.

15. Reduced output due to hard coal or impurities in the seam, increases material costs and the unit cost of labour and overheads.

16. When the cutting tools on a continuous miner or a longwall shearer strike hard materials, sparking can occur. The heat and intensity of the sparks is a

function of the type and composition of the material being cut, and the cutting speed of the tool. Where the sparks are hot enough to ignite any methane that might be present, there is an obvious safety hazard.

17. Where the nature of the coal has a high sparking probability or incendive index, special care has to be taken to ensure that there is no build-up of methane around the cutting elements of the mining machine. This necessitates a high standard of general ventilation and applying local ventilation to the immediate area of the cutting tools. The maintenance of good ventilation is more easily achieved using the longwall system of mining. The relatively simple arrangement of a shearer permits devices such as venturi air movers to be incorporated within the cutting-drum drive shaft (hollow shaft ventilation) to provide ventilation right at the area of cutting. The tool speed can be reduced by gearing down the cutter-drive train, a limitation to the lower speed being the strength of components to withstand the higher torques and forces which are then generated.

FLOOR QUALITY

18. Soft floors can present difficulties to all mining systems. In room and pillar, a soft floor may yield under the coal pillars, necessitating the employment of larger pillars. When the floor yields, it moves into the working area. This can be a major problem in a low seam as the loss of a few inches can cause equipment to become trapped and reduce the mobility of persons. A soft floor can become broken by the repeated passage of mobile equipment. This can result in a reduction of the face haulage capacity of a section, loss of output and consequent increase in cost.

19. During the cutting process, the driver of the cutting machine has to control the elevation of the lower contact of the cutting mechanism. Where the floor is soft, some of the floor may be cut out with the seam, which increases the available working height but dilutes the coal product. Where a very hard and abrasive floor is present, contact of the cutting drum and the floor may cause heavy wear and damage to picks or bits and, in room and pillar, to the grousers on the cat tracks of continuous miners.

EFFECT OF PANEL WIDTH

Room and Pillar

Shuttle Cars

20. The width of a panel is governed by the entry centre distances and the number of entries. The wider the panel, the more coal is extracted per foot of overall panel advance.

21. Where the main coal haulage system employs belts, the conveyor belt must periodically be extended and items such as power centres brought forward. If this work of extension is done by the regular production crew, there is no coal production during the time of the extension.

22. A wide section allows more time for production than a narrower section. As section width continues to increase, however, the average face haulage distance increases which reduces the haulage capacity and the output.

23. There is, therefore, an optimum width of section which will give the highest average output for a given set of equipment. Plate 5 shows the effect of increasing the number of entries in the panel at a constant centre distance for entries and crosscuts of 40 ft. The maximum number of entries that can be reached using two shuttle cars, without back spooling, would be 19.

24. It can be seen from the plate that output and productivity increase with the number of entries up to nine. Eleven entries show a negligible increase per section shift and after eleven entries, output and productivity generally decline.

25. The lowest unit cost occurs at eleven entries, which corresponds with the highest productivity and bulk output. However, by reducing the extension delay time (normally taken at one shift), the maximum output at lowest cost would occur at nine entries.

26. The use of equipment with different capacities affects the output and also the width of section that would yield the maximum output, eg if a shuttle car with a faster rate of travel were to be used, the highest output would be obtained from a greater number of entries. The use of higher-rated continuous miners would expose the limitations of the haulage system and the maximum output would be obtained from a lesser number of entries.

Continuous Haulage

27. The use of continuous haulage, together with a continuous miner, is a powerful production combination. The system does not, however, have a very extensive reach and, as a result, the number of entries would be normally limited to five. The reach of the system could be extended by using additional mobile bridge carriers and bridge conveyors. The effect of this would be to increase the capital cost and the number of persons required to operate the system.

28. An advantage of using a wider section would be a greater utilisation of the coal reserves, as the number of inter-panel barrier pillars would be reduced.

Shortwall

29. With the shortwall system, the ratio of coal won from the development phase of operations compared with the coal from the wall, is a function of the length of the wall. The longer the wall, the higher the proportion of coal that will come from the wall relative to development coal, and the higher will be the utilisation of the coal reserves.

30. Where shuttle car haulage is used on a shortwall, the output from the wall is limited by the haulage capacity of the cars. When the length of the wall is

increased, there is a decrease in the time delay element per ton of coal caused by track cleaning and support move-over operations.

31. The use of shuttle cars along the length of the wall means that only one car can operate at one time as the wall can be considered a "single entry". The effect on the output is shown on Plate 21.

32. One of the main capital costs with a shortwall installation is that of the powered supports. This cost increases directly with the length of the wall and, depending on the type of support used, the cost of supports per foot of face will be in the region of $6,000. At a 20% interest rate and a life of eight years, $6,000 would generate a payback charge of $1,564 pa. Hence, to show a lower unit cost by increasing wall length, the improvement in productivity must save more than $1,564/ft of wall length.

Longwall

33. The length of a longwall does not materially change the number of persons required to operate the wall as the labour requirements of the wall-end operations and the shearer are independent of the wall length.

34. Support of the head and tailgates constitutes the major material cost area. The capital cost, however, is a combination of fixed amounts for items such as the shearer, stage loader and other off-wall equipment, together with the cost of the powered supports and armoured-face conveyor, which is proportional to the length of the wall. This is a simplification of the physical situation because as the wall becomes longer, more powerful conveyor motors are necessary and a heavier duty conveyor chain is required to handle the higher drive power.

35. The productive potential of a longwall is shown on Plate 37, which illustrates that as face length increases, the output increases owing to the diminishing effect of delays and operations at the end of the wall. Plate 41 shows the effect of length variation on output, capital cost and unit cost of a longwall. The range of lengths has been extended to 2,000 ft in order to illustrate theoretically how the unit cost would decrease then increase. However, a length of 2,000 ft is unrealistic in terms of present conveyor technology.

EFFECT OF VARIATION IN OUTPUT

36. For any given system, daily variations in performance occur that may be related to local geological conditions, breakdowns or other problems.

37. The unit cost of production is affected by the level of output. The effect differs between the various systems owing to their different cost structures.

38. In general, the cost of labour remains constant as does the capital cost repayments. Material costs are considered to be directly proportional to output and this cost would be expected to decline with a poor output and increase with output. There are, of course, exceptions to the above. A typical example would be

a major breakdown that would halt production, involve a heavy expenditure in spare parts to effect repairs, and considerable overtime payments to get back into production.

39. The effect of changes in output level on unit costs is depicted on Plates 42, 43 and 44. The variation in output is expressed as a percentage, 100% representing the "standard" output for each of the layouts and systems previously discussed.

40. It can be seen that the general shape of the curves is similar for all the systems.

41. The 50% of standard would represent a section operating under severe difficulties, while the 130% situation implies excellent conditions combined with a reduction in operational delays.

42. It will be seen that the variation in output has a substantial effect on unit cost. Hence, if the revenue accrued per unit of sales remains a constant, a reduction in output would reduce the profit margin and possibly move the operation into a loss situation.

CHAPTER XXIII

MOLE MINING

INTRODUCTION

1. The term mole mining is used to describe a system where coal is obtained from a remotely-operated narrow face machine that drives a long "blind" entry.

2. The system has several component sub-systems; these being haulage to remove the coal, ventilation, monitoring, control and a "pushing" system whereby the cutting machine is forced into the face.

3. Mole-type miners have been produced in the USA, the UK and the USSR. The only systems operating at present are those of the USSR but the results obtained do not indicate that the systems are outstandingly successful.

4. The concept of the mole miner has obvious attraction for work in thin seams as, apart from the design of the machine and supporting sub-systems, height has little effect on productivity as men do not enter the mole's hole.

5. The earliest mole mining system was developed in the USA by the Union Carbide Company. It was designed to be operated from the surface to mine into coal seams exposed in the high walls of strip operations where stripping was no longer economic. The Carbide miner was self-propelled, towing its own haulage system behind. Control of horizon was by means of pick force sensing which was interpreted on an oscilloscope. The system's cutting height was confined to a narrow range around 5 ft.

6. The available information indicates that mole mining systems were introduced in the USSR in the late 1960s and early 1970s. These have involved a variety of cutting mechanisms, drums, scrolls and augers. The most recent application of mole mining systems in the USSR has been in steep seams where the cut coal does not require a separate conveying system. The other applications in level conditions appear to have been abandoned, an example being the Tentek machine, which used a scroll as a conveyor, drilled in a narrow-faced entry and then reamed out to a larger width as the machine was withdrawn. This system failed owing to high frictional losses along the scroll which was also used to transmit mechanical power to the cutters.

THE COLLINS MINER

7. The Collins miner is an example of a mole mining machine that was developed in the UK specifically to mine thin seams. Several machines were constructed and given full-scale trials at underground test sites. Whilst initially these were not confined to the thinnest seams, the results obtained were promising and it is difficult to judge why further development was not pursued.

Description of Equipment and Operation

8. The Collins miner was designed to cut an entry approximately 6 ft 3 in wide, 300 ft long from a pre-developed access road 12 ft wide by 7 ft 6 in high. Production was to have been obtained from the consecutive driving of a series of parallel entries leaving rib pillars between the entries for support. The width of the rib pillars determined the overall proportion of coal extracted to the total reserve and the stability of the workings. The layout of the access roads and miner stalls is shown on Plate 45.

9. The system consisted of an electrically-driven cutting machine and a series of connected pushing rods which were propelled by hydraulic rams from a launching platform located in the access road. The cut coal was conveyed out of the hole using a belt conveyor, the push rods acting as conveyor structure. All control operations were carried out from a control module located near the mouth of the hole.

The Miner

10. The cutting mechanism consisted of three overlapping auger heads driven from a water-cooled gearbox. As each of the augers cut a circle, the profile of the face cut by the heads was three overlapping circles. The shape was then "squared" out by removing the two bottom and two top cusps lying between the circles. This was done by static (unpowered) cutter blades mounted on the main frame of the machine.

11. The coal cut by the revolving heads and the static blades was conveyed to the back of the machine by a short integral flight conveyor powered by the machine. This gathered the coal directly behind the cutter heads and discharged it on to the belt conveyor at the back of the machine.

12. The machine was driven by a single, water-cooled, 120-hp, electric motor. Compressed air turbines had been used in earlier machines but were found lacking in power.

13. The machine was mounted on skid plate bases connected to the main frame by hinges at the front and via lifting jacks at the rear. The jacks, when extended, lifted the rear of the machine causing the auger heads to cut down and when retracted for the machine to cut upwards.

14. The machine also had side jacks to enable steering to be carried out in the horizontal plane by reacting on the sides of the hole. Provision was made in the base plate to accommodate nucleonic steering detectors for measuring the thickness of coal left on the floor under the machine.

The Launching Platform

15. Plate 46 shows the launching platform. The platform was rail-mounted and carried the miner from hole to hole. Its function, in addition to transport, was to align the miner to the direction of the new hole by adjusting jacks on the floor.

The platform housed a pair of pushing cylinders and a pawl mechanism for propelling the push rods and hence the miner. The reaction from the pushing cylinders was taken by horizontal jacks which rested rib side of the access road opposite the mouth of the hole.

16. Two versions of pushing arrangement were tried: single cylinders with pawls and a continuous pushing platform. Better results were obtained with the single-action system owing to mechanical problems with the continuous-action platform.

Push Rods

17. The functions of the push rods were to propel the machine forwards in the hole and to supply the necessary cutting thrust to both the rotating auger heads and the static blade cusp cutters. The rod structure also carried the conveyor.

18. The rod sections were 3 ft 4 in long and connected together with cross braces to form a rigid structure. Vertical members were attached at each joint which extended up to the roof. The purpose of the vertical members was to carry the conveyor idlers and prevent the structure from being bent against the roof by the pushing forces.

19. The push rods were stored in supply cars adjacent to the mouth of the hole. After each section of the push-rod structure had been pushed into the hole, additional push rods were manually placed on to the launching platform and attached to the previous section which was entering the hole. On the completion of the hole, the push rods were used to retract the machine and were then stacked back into the supply cars ready for the next hole.

The Conveyor System

20. The conveyor system employed a single conveyor which ran in the access road and then passed over a 45^0 deflection pulley which deflected the belt 90^0 into the hole. A return pulley was mounted on the back of the miner.

21. As the miner progressed into its hole, the return pulley pulled the conveyor into the hole. Slack was made available from an automatic take-up arrangement at the end of the main access road. Idlers were attached to the push-rod structure to carry the top strand of the belt which was used to convey the cut coal. In order to provide a strong belt, all the joints were vulcanised.

22. As the belt travelled over the 45^0 angle station, the coal was tipped off and then reloaded on to the same belt after the belt had been turned throughout 90^0 and then reversed around a return pulley.

Ventilation System

23. Provision was made for ventilating the hole cut by the miner by dragging in a ventilation tube which was connected to a fan and scrubber arrangement. It was found that a ventilation quantity of 2,000 ft^3/min was more than adequate for maintaining the hole free of methane but insufficient to allow a clear view of the machine to be maintained at all times.

24. The fan and scrubber unit were rail-mounted in the train of ancillary equipment in the access roadway. The ventilation tubing was handled by allowing a long loop to form in the access roadway and controlled by a restraining pulley in the eye of the loop. All the power and monitoring cables, control cables and hoses were attached to the ventilation tubing and pulled into the hole as one unit.

Monitoring Systems

25. As the miner cutting in the hole was remote from the operators, it was necessary to remotely monitor the action of the miner, its position and the environment. To this end the miner was fitted with sampling heads for methane, temperature, sensors and a nucleonic sensor for measuring the thickness of coal under the machine. The deviation of the machine from its planned horizontal cutting line was measured optically using survey-type instruments from the mouth of the hole.

26. The current drawn by the miner's motor and the thrust absorbed by the miner and push-rod assembly were measured externally.

27. All the instrument readings were made from the controller's cabin where the operator could observe the steering information and make the necessary adjustments.

Problems with Collins Miner

28. There were many initial problems with the design of the machine, but most of these were eliminated by redesign of components between trials. Some of the initial problems are enumerated below.

Control of Cutting Horizon

29. It was found that the machine tended to climb in the hole, even when the rotating cutters were undercutting the base plate of the machine. This problem was overcome to a large degree by repositioning the bottom static blade cutter and the geometry of the lifting jacks.

Conveyor Belt Reliability

30. The power drawn by the conveyor was in excess of the calculated values. This was mainly owing to coal being carried back to the machine on the bottom return strand of the conveyor and was largely overcome by incorporating scraper

ploughs on the structure to clean the bottom strand. The problem was magnified by minor roof falls which landed over the top strand causing excessive spillage which was then carried back.

Jamming of the Machine

31. In the initial trials there was a build-up of coal and stone on the top of the machine which caused it to jam in the hole even with a thrust of 33 tons being applied to the push rods. This was overcome by fitting a false top over the machine which presented a smooth surface between the machine and the roof.

Roof Falls

32. Several minor falls of roof occurred during the trials. The causes were identified as:-

(i) poor trimming of the top cusps by the top static blades,

(ii) the cutting into the roof by the vertical sections of the push-rod assembly (the rod assembly was modified to eliminate part of this difficulty),

(iii) poor (highly-stressed) geological conditions that were a feature of the early test sites.

Power Consumption

33. The miner's motor was rated at 120 hp. For long periods it was found that the motor was drawing approximately 150 hp and, consequently, was overloaded. Some improvement was achieved by redesigning the internal loading and conveying mechanism.

Absorbtion of Thrust

34. Thrust was required to supply the necessary cutting forces to the rotating heads, to provide the cutting forces to the static blade cutters and to overcome the frictional forces on the machine and push-rod structure. Thrust was also required to balance the combined tension of the top and bottom strand of the conveyor belt on the conveyor's return pulley at the back of the machine. It was found that the applied thrust was insufficient for long holes, which resulted in the rate of advance being reduced at greater depths of penetration.

Access Development

35. The access roads used for the original trials were 12 ft wide and approximately 7 ft 6 in high. These roads were driven by drilling, shooting and loading out by hand. This is a slow and, therefore, expensive method of development and hence placed a heavy charge on each ton of coal produced. The amount of development required per ton of coal produced could be reduced by extending the length of the miner's holes.

Results Achieved

36. During a series of underground trials at three different test sites, a total of 68,900 tons were mined. The highest performance figure was achieved during the last trial at Rothwell Colliery in 1966, where, in one week, 1,762 tons were mined working three shifts per day.

37. Numerous time studies were carried out during the project. A "standard" time for mining one hole and then moving on to the next hole was measured and calculated:-

Dimensions of hole	
Height	33 in
Width	78 in
Length	300 ft
Tonnage	200 tons (approximately)

Activity	Time (min)	Percentage of Total
Launching	18.35	6.4
Cutting	159.56	55.9
Reversing pawls	1.62	0.6
Withdrawing machine	67.88	23.8
Servicing and repositioning	38.06	13.3
Total	285.47	100.0

38. A potential output of 255 tons per shift was derived from these figures for a $7\frac{1}{4}$-hour shift.

39. It can be seen that during the operating cycle the actual cutting time accounted for only 55.9% of the total time. The other major user of time was the withdrawal of the machine, which accounted for 23.8% of the total.

40. During the cutting of a hole, it was found that the rate of advance of the machine was high at the beginning of a hole and then diminished with loss of thrust as the hole became deeper. During the earlier part of the cutting, advance rates of up to 6.9 ft/min were recorded. This is equivalent to a mining rate of 4.5 tons/min. The average mining rate for a hole was 1.25 tons/min during cutting.

Reasons for Abandoning Work

41. Whilst promising results were achieved, the project must be viewed against the economic environment of the UK coal industry in the late 1960s. This was the time when large quantities of natural gas had been discovered in the UK and, hence, coal's role was expected to contract. High-cost operations were closed down on a wholesale scale, many of them thin seam mines. There was a cut back in administrative staff and a general reduction of expenditure on research and development. Work on many projects that had not reached the revenue-earning stage were stopped, the Collins miner being included in the list of economies.

Unsolved Problems and Possible Solutions

Vertical Steering

42. Considerable difficulties were experienced in controlling the vertical steering as local changes in the hardness of the floor would, in some instances, cause the bottom blade to "ride up" over the cusps.

43. The use of a powered cutting element in place of the static cutter blade would solve this difficulty and also reduce the thrust required to propel the machine forward in the hole.

Vertical Monitoring

44. The machine was equipped with nucleonic monitors which did not give reliable indication of the miner's position in the seam. In the last trial, an attempt was made to steer the miner by observing the colour of the product for out-of-seam contamination.

45. Since the time of the trials, which were concluded in 1966, there has been a vast improvement in horizon-sensing systems. Nucleonic steering is now a well-proven technique. Other techniques are also available such as natural radiation measurement, seismic and pick force sensing. The most appropriate technique would depend on the conditions where a machine would be expected to work and, in order to ensure success, several types of indicator could be used to monitor both the roof and floor horizons.

46. The solution to the problem of vertical steering is not seen to be a major difficulty.

Horizontal Steering

47. The system used depended on an optical system whereby the machine could be viewed from the mouth of the hole. This gave good results in terms of measured deviations but depended on a seam free of undulations so that the machine would remain visible during the whole of the cut. Operations had to be suspended periodically to allow sightings to be made due to dust in the hole.

48. The use of a laser would probably reduce time lost waiting for dust to be removed to enable a line of sight to be established. This would not, however, overcome the problems of undulations, particularly if a lower model machine, say 20 in high, were constructed and longer holes were attempted.

49. There have been considerable advances in the use of electronics in the art of surveying. Possible solutions could be the use of remote survey stations sited in the hole or the use of a repeating gyroscope-type instrument mounted on the machine. Another suggestion would be the remote measurement of angular deflection along the length of either the push-rod structure of any other structure within the hole.

Minor Roof Falls

50. The whole concept of the machine was that the holes would be self-supporting. This was to be achieved by making the width of the opening narrow, approximately 6 ft 6 in. The roof falls were thought to be due to:-

(i) some of the ground being disturbed by previous mining activity in the underlying seams,

(ii) the push rod structure catching and gouging into the roof,

(iii) the inadequate action of the top static blade cusp cutters leaving behind an unstable cross-section of roof which had been weakened by the revolving heads cutting through the cross bedding of the seam.

51. The use of a positive-action, powered, top-trimming mechanism would cut a superior top profile which would leave a much more stable roof.

52. Modifications were made to the push-rod structure during the development to prevent cutting into the roof. These could be revised and further modified if necessary.

53. It is not considered that inter-seam interactions would be a problem under typical US conditions.

54. Another solution to the problem of a weak roof would be to make the initial hole even narrower than the 6 ft 6 in of the Collins design and then back-ream the hole as the machine is retracted. This technique has been tried in the USSR but failed for mechanical reasons not roof control.

55. An advantage of this arrangement is that the retraction phase of the operation becomes productive.

Conveying

56. The Collins miner requires a haulage system that can enter the hole after the miner and automatically extend as the miner progresses into the hole.

57. The solution used in the prototypes was a belt conveyor that was deflected over a pair of 45^0 inclined rollers for both the top and bottom strands of the belt. As the miner progressed into the hole, idlers were attached to the push-rod structure which supported the top load-carrying strand of the belt. In order to supply additional belt for the increasing length of run, belt was progressively released from the belt storage system of the drive unit.

58. The conveyor system had two problems:-

(i) any falls of roof fell on to the top strand of the belt, causing spillage within the hole. The bottom strand of the belt carried back fines and spillage into the hole which tended to jam the return pulley behind the miner.

(ii) the 45^0 deflection pulley placed a considerable strain on the belt and belt joints.

59. An improvement in the cutting action to give a more competent roof would solve most of the conveyor problems. A small elevating device could be installed at the return pulley to lift any returned material and load it on to the outgoing top strand.

60. Two other approaches to the conveying problem are offered:-

(i) The belt conveyor could be replaced by a scroll or screw conveyor similar to an auger. This could run in a trough which could form the thrusting structure in place of the push rods.

(ii) A form of continuous haulage could be used with haulage units of about 20 ft in length forming a train.

61. The use of a screw conveyor or continuous haulage unit would solve the carry-back problem and any falls would be less likely to cause spillage than with a belt unit.

Thrust

62. The provision of adequate thrust was a problem that was never completely resolved. There are a number of avenues by which the problem could be tackled:-

(i) The thrust requirement of the miner could be reduced by the elimination of the static blades and their replacement by powered cutters.

(ii) The push-rod assembly could be strengthened and more powerful pushing cylinders used on the launching platform.

(iii) The machine could be made to be self-propelled either by using pushing rams reacting on the sides of the hole or by mounting the machine on "cat tracks", although this solution would increase the height of the machine.

63. The most likely solution would contain several of the above suggestions.

Push Rod Handling

64. The provision of 300 ft of push-rod structure occupied a considerable amount of space and required the services of two men to handle the rods and assemble them on the launching platform. The crew for operating the whole system was seven men, which included a mechanic and supervisor.

65. If an automatic handling system could be devised to unload and assemble the push rods or equivalent structure, then, on the same basis of manning, the crew would be reduced from seven to five, which would increase the productivity of the system by 40%, assuming the operation was carried out at the same rate.

Power Requirements

66. The machine was fitted with a 120-hp electric motor. During the trials it was found that the motor was frequently overloaded, drawing 150 hp for extended periods. If additional powered cutting elements were fitted and a higher rate of advance achieved, then even more power would be required.

67. The problem of power could be overcome either by uprating the motor or by using additional motors. The design of water-cooled motors has progressed and it is now possible to design higher powers into the same physical frame size of older, lower-powered motors.

POTENTIAL OF SYSTEM

68. The mole-type miner would appear to have a high production potential if satisfactory solutions to the preceding technical problems are developed.

69. Assuming that an advance rate of 10 ft/min could be achieved in a 30-in seam with a 6-ft 6-in wide head, then the cross section of profile cut would be 14.91 ft^2, which would yield approximately 180 tons for a 300-ft hole:-

	Minutes
Time to set up and launch	20
Time to cut hole 300 ÷ 10	30
Time to withdraw	30
Time to move to next hole and service	30
Total	110

70. This would indicate that two or three holes could be mined in a shift, giving a production of the order of 350 tons to 550 tons per shift with a labour crew of five.

71. Should the machine be made double-acting, ie have the facility to ream the hole out on the retreat, then, as the launching and moving activities would presumably remain the same in terms of time, there would be an increase in output.

72. In order to maintain the stability of the principal access road, the holes would not be reamed all the way back to the mouth but the reaming operation would be stopped to leave a larger rib pillar adjacent to the access road.

Economic Assessment

73. Whilst it would appear technically feasible to solve the remaining problems of the mole miner, the decision to proceed must either be dictated by economics or by some other necessity to mine thin coal.

Productivity

74. The productivity potential of the mole miner working in a 30-in seam yielding, say, 450 tons per shift with a crew of five, would be 90 tons per manshift. This productivity does not take into consideration the labour expended in the development of the access roads.

75. It is likely that in a 30-in seam, on a three-entry development layout, with 300-ft holes bored at 10-ft centres in each barrier rib side, some 9.4 tons of coal would be produced for each 1 ft of development driven. This would require at least 50 ft of development per shift to replace access roads. If, say, nine men were involved in the development, the total labour requirement would rise to 14 men, whilst the development, say 16 ft wide and taking out 4 ft of floor to give a height of 6 ft 6 in, would yield a further 80 tons of coal and approximately 250 tons of rock. This would give a clean coal total output of 530 tons and a productivity of 37.8 tons per manshift.

Capital Costs

76. After suitable research and development work, it is likely that the cost of manufacturing a mole miner and ancillary equipment would be comparable with a continuous miner and its ancillary equipment. To the capital cost of the mole miner must be added the cost of equipment for driving the access roads. The total cost of the installation would also include all the services necessary for the driving of the access and the mining of the holes, eg conveyors, cable, pipes and track.

77. At a first approximation, the total cost would be in the same order as the establishment of two room and pillar continuous miner sections.

Operating Costs

78. The material cost for the operation of the mole miner would be low as no support material costs would be incurred within the holes. The support cost of the access entries would be comparable with normal room and pillar costs as no caving or major strata movement is planned with the mole system. Labour costs would be relatively low with the mole miner in line with the high overall productivity.

79. The capital recovery charge per unit of output is likely to be relatively low as whilst the total capital cost would be fairly high, the bulk output would also be high.

80. It is likely, therefore, that the mole miner would be capable of producing coal at a cheaper rate from a 30-in seam than would a room and pillar system working a 36-in seam.

Summary

81. The mole miner would seem to meet most of the requirements of a "new" thin seam system. It would appear to offer the prospect of mining thin coal in a more productive and economical manner than the presently-used systems in the USA and yet provide a safer environment than is currently available.

82. The system would appear to be able to mine coal as thin as 20-in (plans were being made to construct a 20-in Collins model in the 1960s). In the thinner coal, it is probable that productivity would fall and unit costs would rise but the system would still provide a method to mine safely such reserves.

83. One criticism that could be levelled at the mole miner is that it is wasteful of reserves. This is not necessarily true as to be included in reserves, coal must be "economically mineable". If the mole miner makes coal economically mineable which was previously uneconomic, then, far from wasting reserves, it would increase them.

CHAPTER XXIV

AUGERS

INTRODUCTION

1. Auger mining first became significant in the USA in connection with surface mining operations in the early 1950s. Following success on the surface, many attempts were made in the USA and UK to take the technique underground. Since the 1960s, some underground augering has been practised in the USSR but with limited success.

2. Augering has been suggested as a means of extracting pillars left underground and of increasing the total extraction by boring into the rib side of mined-out panels.

DESCRIPTION OF EQUIPMENT AND OPERATION

3. The auger system of mining is probably one of the simplest systems conceivable. A hole is drilled into an exposed coal face and the "drill" screws the coal out of the hole. At the mouth of the hole, the coal is gathered and hauled away.

4. On the completion of one hole, the equipment is moved over and another hole drilled adjacent to the first.

5. The distance between adjacent holes governs the extraction ratio of coal removed to coal in situ and the stability of the ground after augering operations have been completed.

6. During the drilling cycle, the equipment has to be set up, the hole collared and then, as drilling proceeds, extra scroll sections are added.

7. When a hole reaches its pre-determined length or is terminated for any other reason, the rods have to be uncoupled, withdrawn and stored ready for the next hole.

8. The withdrawal is a time-consuming process and storage space is at a premium in confined workings. A significant improvement to the system is to arrange for the drilling of one hole and the retraction of scrolls from an adjacent drilled hole to proceed simultaneously so as scroll sections are required for the hole being drilled, they are extracted from the completed hole. This saves time and minimises the requirement for storage.

9. The system is completed by a mechanism that collects the coal as it drops from the mouth of the hole and either elevates it for loading into cars or loads it on to the first stage of a conveyor belt haulage system.

10. The general arrangement of the equipment is shown on Plate 47.

The Head

11. The simplest auger system consists of a boring head that cuts the coal at the front of the hole. The head is fitted with a number of cutting tools, some being similar to the bits used on other coal mining machines whilst others resemble the roller bits used in drilling operations.

Scrolls

12. The head is given a circular motion and forced into the end of the hole by a series of scroll sections that also act as a screw conveyor. An important function of the scroll is to impart stability to the head and to ensure the head stays on its required course.

Platform

13. The platform is situated next to the coal face to be bored. It houses the mechanism to turn the scrolls and thrust and retract them into and out of the hole. Other equipment mounted on the platform includes the devices for handling the scrolls to transfer them from a completed hole to the hole currently being drilled. The platform is moved from hole to hole by having it mounted on rails, wheels or walking legs, dependent on design. Arrangements for hole alignment and direction are built into the platform. The reaction forces of drilling are accommodated by hydraulic rams that bear on the opposite rib side of the entry to the hole being bored.

Power and Ancillaries

14. The power source for surface augers is normally a diesel engine. For underground use, electric power is more convenient. The power source may be mounted on the platform but in underground applications the power centre would be in fresh air some distance away from the site of the hole.

Coal Clearance

15. The coal is screwed out of the hole by the scroll sections. At the mouth of the hole the coal falls down and is collected by a short elevating conveyor, normally a scraper conveyor. This conveyor either loads the coal directly into shuttle cars or on to a conveyor system.

16. As the gathering conveyor is required to be below the mouth of the hole, the mouth must be approximately 10 in above the floor. This means that either the floor must be removed adjacent to the mouth of the hole or the hole must be started above the floor of the seam - a somewhat wasteful procedure, particularly in a thin seam.

Controls

17. Control over the auger is exercised by the initial position and direction of the auger when a hole is commenced and by the speed of rotation and applied thrust as the hole is being drilled. The control of the direction and horizon drilled is one of the main problems associated with augering, if not the greatest. The result of applying greater or lesser amounts of thrust depend on the bedding of the seam and the roof and floor. The other controls are to manipulate the transfer of scrolls from one hole to the next and to couple new scroll sections as required.

Ventilation

18. Whenever coal is cut there is some liberation of gases and make of dust. In order to minimise the risks attendant with inflammable gases, the holes are ventilated as they are drilled. The ventilation pipe must, of necessity, be carried within the scroll, either wrapped around the centre shaft or, in the modern designs, carried within a hollow-centre shaft. The ventilation will carry some dust out of the hole. This dust-ladened air must be either diluted or collected and the dust removed.

PROBLEMS WITH AUGERS

19. There are two main areas of technical problems with augers in thin seams: directional control and low bulk output.

Deviation

20. As augering proceeds there is a tendency for the holes to wander off the original starting line. Movement in the plane of the seam is not of high significance particularly if all the holes of a series have the same trend of deviation. If, however, the horizontal deviations are of a random nature, then the space left between consecutive holes has to be increased, reducing the overall recovery, in order to prevent adjacent holes from breaking into each other.

21. The problem of vertical deviation is more serious as, when the hole moves out of the seam into the floor or roof, the product becomes contaminated and when the hole is completely out of the seam there is no point in continuing as no coal is being produced.

22. The problem of deviation has been experienced in all countries where underground augering has been attempted, and the effect of this deviation has been to limit the length of the holes drilled.

23. In thin seams, the problem of vertical steering is more serious. This is due to the small diameter and, therefore, less rigid scroll sections that are used which have a lower stabilising influence on the direction of the hole and the lesser distance a hole must penetrate into the roof or floor before the contamination reaches an unacceptable level.

24. In the USSR, holes are seldom augered for more than about 110 ft owing to guidance problems, and a figure of 80 ft would appear to be the average length of hole attempted.

25. Initial experience in the UK in the 1950s indicated a maximum hole length of about 90 ft with a probable optimum length of 50 ft.

Coal Extraction

26. The maximum extraction possible with an auger boring a hole equal in diameter to the thickness of a seam is approximately 78% when the holes "touch" each other.

27. In practice, it is necessary that the holes are spaced so they cannot break into each other and the rib of coal between the holes acts as a support for the mine roof. Even where the rib pillars are designed to fail after a number of holes have been bored, the overall extraction will not rise much above 50%.

Output

28. The bulk output obtained by augers is relatively low. This problem becomes more acute with augers in thin seams than with other systems as the cross-section cut by the auger is proportional to the square of the diameter bored and not the seam height alone, as with most other systems.

Hole Length

29. The length of the hole drilled directly affects output as it determines the number of hole-to-hole moves that have to be made to obtain a given tonnage. A further and highly significant feature of the hole length is the amount of access development required to expose rib sides for augering. The amount of development is decreased with increasing hole length. This is of vital importance in thin seams as the coal produced in the driving of the access roads decreases with seam thickness, which effectively increases the cost of driving the access entries.

30. In order to increase the length of the holes, it is necessary that some means be employed to eliminate deviation or that some means be incorporated in the equipment to monitor deviation and then correct the deviation without having to stop the process and withdraw the auger from the hole.

SUGGESTED SOLUTION TO PROBLEMS

Deviation

31. Surface operations have been able to consistently drill longer, more accurate holes than underground. The question arises - what is the difference between the underground and surface equipment and the drilling techniques? Part of the answer is that on the surface, where space is not the same constraint as in the underground environment, longer lengths of scroll can be used. This reduces

the number of joints in the string and, hence, must give a more rigid support to the cutter head.

32. It is suggested, therefore, that means be found of using longer auger scrolls underground and to increase the rigidity of the scrolls and joints. A second solution would be to case the scrolls in a steel tube, either rotating with the scroll, or non-rotating. If the casing were able to rotate, it could be made integral with the scroll and, hence, produce a far more rigid scroll section than the present centre-shaft types. In order to clear any fines on the outside of the casing, a rudimentary external scroll could be wound around the casing that could remove any fines between the casing and the hole.

33. If the casing were non-rotating, then a bearing assembly could be interposed between the casing and the cutter head. The casing could then be made to carry the thrust, with the scroll providing only the cutting torque. The use of a head supported independently of the scroll could form the basis of a steering system.

Deviation Sensing

34. Where there is a significant difference in the composition of the roof and floor, an analysis of the contaminants coming out of the hole with the coal would reveal in which direction the auger had deviated from the horizontal or plane of the seam. Knowing the deviation, it would then be necessary to make a steering correction. This could be done by varying the thrust and/or speed of rotation of the auger, dependent on the boring characteristics of the seam and surrounding strata, or by inducing a counter-deflection in the orientation of the augers head.

35. Another technique that could be applied to monitoring the position of the auger in the seam is pick-force sensing, ie mounting a sensing tool on the outer periphery of the cutter head and measuring the cutting force as the head rotates.

36. With increasing hole length the frictional losses owing to the conveying action of the scroll and friction between the scroll and the side of the hole will increase. The amount of power available and the torsional strength of the scroll section will ultimately limit the length of the hole that can be drilled, but this limit will have to be found after the problem of deviation has been solved.

Coal Extraction

37. A maximum recovery of 78% is inherent boring adjacent circular holes. This is further reduced by leaving ribs between adjacent holes to cater for hole deviation and to give support to the mine roof.

38. Recovery could be improved by the reduction in the size of the inter-hole ribs, by improved directional control and by allowing the holes to crush after augering.

39. By changing the shape of hole bored from a circle to a rectangle, the percentage extraction would be increased. However, this would require a variable-geometry cutting head which would detract from the simplicity of the auger.

Output

40. Improvements to output per shift would result from longer holes which would reduce the time lost in moving from one hole to the next. Further increase in output might be possible after the solution of the problem of deviation by increasing the power and the thrusts available on augering equipment.

RESULTS OBTAINED

41. The present outputs of underground augers are comparatively low, especially when compared with similar auger installations operating on the surface, where three-man crews have produced 600 tons per shift to 700 tons per shift from 28-in seams.

42. The results obtained underground in the USA indicate a maximum output of around 200 tons per shift for a 42-in-diameter auger. For a smaller diameter, the output would be considerably less. In the USSR it is reported that augers produce approximately 80 tons per shift, in sections of 24-in to 39-in, but in order to get higher outputs, a number of augers are worked in the same vicinity. Reported productivity with augers is in the range of 15 tons per manshift to 17 tons per manshift, but it is not stated whether this includes labour for the necessary access development work.

POTENTIAL OF AUGERS

43. The potential output of augers may possibly be improved by increasing the size and power of the drive mechanism, by reducing the time delays in adding and removing scroll sections and by drilling longer holes to reduce the relative time delay in moving per foot of hole drilled.

44. A 30-in-diameter auger, if advanced at 10 ft/min, would yield approximately 2 tons/min. If the time spent on actual boring could be brought up to 50% of the available shift time, then outputs of 300 tons per shift would appear feasible.

Economic Assessment

45. If augers are to produce coal economically from thin seams as a principal system of extraction, then the coal produced by the augers, together with the coal produced from access development operations, must be able to cover the cost of both augering and development in terms of capital repayment, labour and materials.

46. Owing to the low tonnage likely to be produced per foot of development, it would appear that to be successful, augering requires an inexpensive companion technique for producing the necessary access entries. It is hence more likely that

augering would be used as a supplementary extraction technique to increase output from entries that have to be driven to serve other purposes.

Summary

47. Augering is a currently used and accepted technique for surface mining. Only limited success has been achieved with underground augering. The main problem that would appear to limit the further application of augering is the universal difficulty that has been experienced with directional control. This has limited the depth to which holes can be bored.

48. The limitation on the depth of hole increases the development footage required to sustain an auger "section".

49. Should means be found of drilling longer holes, say 300 ft as opposed to the current 50 ft to 100 ft, then augering would appear to have an improved potential application.

CHAPTER XXV

FULL-FACE MINERS

INTRODUCTION

1. The term full-face miners applies to systems where the principal mining machine spans the full width of the face and the face is advanced or retreated along its whole width simultaneously. The term could be applied to the "borer type" continuous miner if worked on a single entry, but is normally applied to much wider faces as typified by longwalls.

2. Because the full face miners tend to be long and for most of their length they are confined to the cut or extracted width, there is a considerable amount of mechanical and structural equipment "permanently" confined to low sections where, in the case of thin seams, the extracted height is limited. The presence of equipment on the face or wall demands access for men, if only for maintenance purposes. This limits the systems to seams in excess of 16 in.

3. A well-known full-face system is the coal plough. Other systems less well known are the Yarmak miner, the in-seam miner and the miniwall. All share a similar cutting mechanism, breaking coal of the solid by means of relatively-slow-moving cutters traversing the whole face in a lateral manner.

COAL PLOUGHS

Description of Equipment and Operation

Conveyor

4. Plate 48 shows a plough arrangement for a seam height of 20 in to 24 in.

5. Ploughs are used extensively as a method of working thin seams and are currently being used in seams down to 18 in thick. Below 30 in, however, there is generally insufficient space to employ powered supports and hence the productivity and potential of the system are greatly reduced.

6. There has been much research and development carried out on coal ploughs. The essential mechanism has, however, remained much the same. The main component of a plough system is the armoured face conveyor. The conveyor has two principal functions: to convey broken coal off the longwall and to serve as a guide for the plough unit as it traverses the wall, cutting coal from the face.

Plough

7. The plough also has two functions: to cut coal from the solid and to load it on to the conveyor. It serves as a mobile tool holder containing a series of heavy-duty cutter blades that cover a large portion of the extracted section of the seam. In order to assist in the control of the cut, some of the blades are adjusted

in height. Thus, if there is a general tendency for the plough to cut into the floor, the bottom blades are raised and vice versa.

8. The depth of cut of the blades in some models is adjustable to take advantage of soft horizons in the seam. The blade or blades spanning the softer horizon are adjusted to pre-cut that section, the blades being given a deeper penetration than other blades on the plough. The initial cutting out of a soft section then allows subsequent blades to attack harder sections, when there is a second "free face" for the coal to fracture to.

9. Where there is a good parting between the coal and the roof, and the top portion of the seam falls freely, it is not necessary for the blades on the plough to cut right up to the roof.

10. The plough body normally contains guide shoes that run on guides attached to the armoured face conveyor. The guides are designed to transmit the necessary forces from the conveyor to cause the plough to cut into the face and give a measure of stability to the body.

11. To increase the stability of the plough there is either a "sword" that runs under the conveyor or a stabiliser arm that runs over the top of the conveyor. This engages on a second guide on the goaf side of the conveyor, thus giving a good measure of rigidity to the plough.

12. The plough is hauled by a heavy-duty round-link chain that is joined to the plough and hauled by driving sprockets at each end of the wall. The return chain between the driving sprockets travels either through a guide tube or a separate compartment within the ramp plates or goaf side trappings dependent on the design.

13. In some variations of plough design the hauling chains are carried on the goaf side of the conveyor and attached to the stabilising sword that runs under the conveyor. In other models the chains run on the face side of the conveyor. This gives a pull that is more in line with the reaction of the cutting forces and hence produces less friction on the trappings and guides owing to the reduced turning moments on the plough body.

Drive Units

14. Each drive unit consists of a combination of motor, fluid coupling and gear box and chain-drive sprocket. The drive sprocket normally incorporates a shear pin that is designed to shear when an overload develops, thus reducing the instances of chain breakage and protecting the drive unit from damage.

15. The drive units are designed to operate in two directions and the motors are rated to accept frequent stops and changes of direction.

Pushing Cylinders

16. The plough is forced into the face by the conveyor which, in turn, is pushed by cylinders. The cylinders are attached to the conveyor at one end and anchored at the rear. In the early models, the cylinders were actuated by compressed air and in the later models by hydraulic fluid.

17. The anchoring device for the cylinders is either a sprag between the roof and floor - a "stelling unit" - or, in latter models, the cylinders are attached to the powered supports.

Supports

18. The supports must perform two principal functions: stabilise the roof over the conveyor and travelling tracks, and cause the roof to break or shear at the goaf side.

19. The design of the supports must enable a clear, and as far as possible unobstructed, path to be maintained for men to travel.

20. Hydraulic powered supports provide the strongest and most secure form of support. They also require the least labour and effort to move forward with the advance of the wall. The lowest powered support currently manufactured has a closed height of approximately 20 in, which limits their application to extracted heights of approximately 26 in and above. This is to allow sufficient clearance for convergence and for the support to be lowered from the roof so that it can be moved forwards.

21. Below this height, ploughs are used with prop-and-bar systems which allow more mobility of personnel but require considerably more labour to set and withdraw as the wall advances. At heights lower than 18 in to 20 in, where the use of roof bars or beams would unduly restrict travelling and working, the bars are dispensed with and the support is provided by single props.

Operation

22. The various types of plough differ in the way they are mounted and guided, the manner in which they are pushed into the wall and the speed at which they are hauled along the wall.

23. Usually the pushing cylinders can be set to exert a constant force. As the plough traverses the face, the blades slice off a fairly constant depth web of coal. Where the plough meets a hard section of coal, the normal forces exceed the pushing forces which causes the pushing cylinders to retract. A different approach is to advance the conveyor a fixed amount after the passage of its plough and then the hydraulic cylinder is "locked". This causes the plough to cut at a uniform penetration on its traverses and assists in maintaining wall alignment.

24. The power transmitted to the plough is the product of the hauling force and the haulage speed. The haulage force is limited to the tensile strength of the haulage chain, which is set by the size of the chain.

25. In order to get more coal for a fixed haulage force and depth of cut, the plough must traverse the wall quicker.

26. When the plough is cutting and loading in the same direction as the conveyor and the speed of the plough approaches the conveyor speed, the conveyor is then slow moving, relative to the plough. This can be clearly seen when the plough speed is equal to the conveyor speed. As the plough moves, cutting and loading coal, then the conveyor scraper bars will always be level with the plough. This is equivalent to loading coal on to a stationary conveyor. The only result is spillage.

27. In order to overcome this problem and yet achieve higher cutting and loading rates, a number of variations in design have been tried. These include varying the speed of the conveyor according to the direction of travel of the plough and even reversing the conveyor. The most recent designs of ploughs operate at a much higher speed than the conveyor, which is termed "overtaking" speed operation. Where the plough's average speed is twice that of the conveyor, each flight or scraper bar is loaded twice on its passage along the wall. This gives a fairly constant loading of the conveyor without the complications of conveyor speed changing.

Results Obtained

28. The best results from ploughs with powered supports, mining at a height of 1 m (39 in) in Germany, are in the region of 2,500 tons per day at a productivity of approximately 35 tons per manshift, excluding development of entries.

29. Current results in the lower end of the thin seam range are not so readily available as the use of ploughs in Western Europe for mining seams of less than 39 in has greatly diminished.

30. In the 1960s, outputs in the region of 300 tons per shift were obtained from UK seams working in a height of 24 in. The rate of output was restricted owing to advancing the gate roads as opposed to the main line of the longwall. This resulted in an OMS of some 5 tons inclusive of development.

31. It would appear quite feasible that a modern plough installation, working at a height of 24 in, using powered supports, would give advance rates of 10 ft to 15 ft per shift, yielding 500 to 700 tons per shift on a retreat basis with a 600-ft wall in good working conditions. This would yield a production of between 30 and 50 tons per manshift with wall labour only, and 20 to 30 tons per manshift including development.

Problems with Ploughs

32. The state-of-the-art in the design and manufacture of ploughs has attained the situation where a considerable range of coal hardnesses can be cut. The equipment, owing to its simple concept, is reliable but there remain a number of difficulties that are discussed below.

Horizon Control

33. With soft floors it is very difficult to maintain the desired horizon. The general tendency is for the plough to cut into the floor. If the bottom blades are raised, particularly if the seam has a strong lower section, the plough will then climb into the seam. In a thin seam this reduces working height and makes operations difficult.

34. Where there are local variations in the coal and floor strength, the plough will tend to dig into the floor in some parts of the face and climb in others. The local use of jacks and wedges to tilt the conveyor can, to some extent, overcome this problem but lifting the conveyor presents its own difficulties in thin seams, due to the restricted available heights.

Roof Cutting

35. The plough operates most efficiently where there is a good parting at the top of the seam and the coal, when undercut, drops freely. Where the top coal does not part readily from the roof, it prevents the supports from being brought forwards. Frequently, the only recourse is to bring the top coal down by hand, a time-consuming and physically arduous task in a thin seam.

36. The cutting range of the plough can be increased by fitting top blades but this does not solve the problem where there are variations in the height of the seam over the length of the wall.

Coal Strength

37. The early ploughs were limited in their range of application and could not effectively mine many of the harder coal seams. Improvements in design have extended their range of application, but a problem arises where there is a variation of the coal cuttability along the length of a longwall. In the softer sections the plough will tend to take thicker cuts and when the plough reaches a hard section an overload situation will arise with frequent stops from shear-pin failure. Hardness or resistance to cutting can arise from local increases in in-situ stress on the face or from discontinuous waste inclusions.

Support

38. In suitable conditions, ploughs have the capacity to advance several feet per hour of operation. Where props and bars are used as the means of support, these have to be manually set and withdrawn, which is physically heavy work, requiring considerable labour. The ability to advance the supports could limit the possible advance of the face.

Faults

39. The main object of plough design has been to produce a machine that will cut coal but not penetrate into the floor. Where there are faults along the line of a wall, the plough has considerable difficulty in grading a suitable line through a faulted region. Frequently, the production cycle has to be stopped and the ground on each side of the fault prepared for the plough.

Stableholes

40. The plough is not able to overcut the end of the conveyor. This means that the conveyor and plough drive units must be housed off the wall either in the pre-developed gates of retreat layout or in specially-prepared stables at the ends of the wall in an advancing layout. Whilst this is not a significant limitation in retreat working, it can cause problems in advance working. A separate system must be used to advance the wall ends that requires special support arrangements and additional labour.

Safety and Health

41. Ploughs are a relatively safe system to work, particularly where there is sufficient height to enable modern shield-type powered supports to be used.

42. Dust suppression presents some problems on plough faces because of the difficulty of supplying suppression water to the rapidly moving plough body and directing it on to the cutting faces of the blades. A partial solution of the problem has been the installation of water sprays attached to the goaf side of the conveyor that spray the coal face. In order to prevent the wall from becoming flooded, the sprays are sequentially operated so they begin to spray as the plough approaches their position and are turned off after the plough has past. This reduces dust entrainment around the plough but the problem of dust from the falling of top coal remains. The top coal does not necessarily fall at the same time as the plough undercuts a specific portion of the wall but may fall a few moments later, when the sprays have stopped, thus releasing dust into the ventilation stream.

Suggested Solutions to Problems

Horizon Control and Roof Cutting

43. Most of the development work to date has been directed to the guiding of the plough on the conveyor or attachments and the tilting of the conveyor to suit the required horizon. A possible solution would be to vary the geometry of the lower cutter blades of the plough as it moves along the face. This would require some method of actuating the movement of the lower blades. The problem would be to supply power to the plough to cause movement without the complication of cables or hydraulic hoses. One solution here would be an on-board power source with commands to raise or lower being transmitted by radio control. A similar solution could be used to vary the height of top blades thereby solving the problem of top coal that sticks to the roof.

Support

44. To realise the full potential of ploughs in thin seams, more development work would have to be carried out in the design of powered supports. It is possible that the closed height could be further reduced below the current 20 in. A major difficulty would be to reduce the height and still provide reasonable travelling room for operators. The use of "batch-control-type" techniques for remotely actuating the supports would reduce the requirement for movement of operators on the face.

Dust

45. The further suppression of dust on plough installations is a major problem. The wider use of top cutting blades together with sequential sprays would, however, reduce the possibility of top coal dropping after the line sprays had ceased to operate.

POTENTIAL OF PLOUGHS

46. The range of applications of the system would be increased by further development on the plough blade geometry. The output of the system would increase in thinner coal sections with the design of suitable powered supports. It is feasible that output of the order of 700 to 1,000 tons per shift could be obtained in heights of 24 in on the basis of a plough travelling at 300 ft/min taking a web of 1 in to 2 in. An output of 1,000 tons per shift would require a face advance of approximately 20 ft in a 24-in seam on a face length of 600 ft.

Economic Assessment of Ploughs

Capital Costs

47. The capital costs of a plough installation are comparable with any other form of conventional mechanised longwall. A major cost item is the provision of the powered supports.

48. The length of the wall directly influences the installed capital cost of equipment as additional wall length requires more conveyor support equipment. Increasing the length of the wall, however, reduces the cost of development and a compromise must be reached between the access development cost and the cost of the installation.

49. The strength of the conveyor chains to haul the product off the wall is the major physical limitation of wall length.

50. During the 1960s, plough faces of over 900 ft in thin seams were used in the UK. Whilst these had a low gate-formation cost, they were not particularly successful owing to their relatively slow advance causing a general deterioration in mining conditions.

Operating Costs

51. The operating costs are modest except when powered supports cannot be used, in which case the operation of support installation becomes labour intensive, and expensive, accompanied by a reduction of the total output. The consumption of materials is low, the major item of cost being the driving of the necessary wall entries of gates.

Summary

52. Ploughs have been, and still are, a major producer in thin seams.

53. The range of application of the plough is limited by geological conditions, which have a major bearing on horizon control, the ease in which the top of the seam parts from the roof, and the hardness of the coal.

54. High productivities are only possible where there is sufficient height to operate powered supports, this at present being in the plus-26-in range of seams.

THE YARMAK MINER

Description of Equipment and Operation

55. The Yarmak miner was designed specifically to mine thin seams. A prototype was surface tested in 1966, modifications carried out and an underground trial made. Owing to the severe operational problems encountered, the trials and further development work were suspended.

56. The system was designed to operate as a semi-remote-control longwall, men being required on the wall for maintenance and to make adjustments to the steering controls. The control of the cutting and advance was to be made from the gate roads.

57. The system consisted of a heavy-duty scraper-chain conveyor aligned to the face with the return strand being mounted at roof level and carried in a separate guide to the bottom load-carrying strand. The bottom and top conveyor guides were connected by joints which allowed a degree of freedom in the vertical plane but no movement in the horizontal plane. Hence, the conveyor advanced as one unit over the whole face.

58. Attached to the flight bars of the conveyor were a series of heavy-duty cutter picks which continuously cut the full height of the seam over the length of the face. The picks of the lower strand cut the bottom half of the seam and the picks attached to the top strand cut the top half (see Plate 49).

59. Steering of the machine was carried out by tilting the conveyor sections independently at roof and floor level. This operation was controlled by an operator travelling along the wall at the back of the support system.

60. The equipment consisted of an integrated cutting and conveying chain supported in special conveyor sections at roof and floor level. The conveyor sections were mounted on beams which formed an integral part of the self-advancing roof-support system. At the rear of the supports, shields were mounted which were attached to the rear of the top conveyor support beams and moved forward with them.

Chains and Picks

61. An endless, single-strand, round-link chain was used for the cutting and conveying operation. Four short bracing chains arranged in parallelogram were used to give stability to the flight bars and resist the reaction forces of the cutter picks. Special heavy-duty picks were mounted on the ends of the flight bars, these being arranged to cut at different heights to cover the whole of the face.

Conveyor Sections

62. The conveyor sections, 2 ft wide by 8 ft long, were of single-deck construction with rigid butt joints. They were designed to have sufficient flexibility in the vertical plane to follow undulations in the seam along the wall but were ridged in the horizontal plane and, thus, kept the cutter sections against the coal along the full length of the wall. The conveyor pan and guide sections were attached by hinges and hydraulic cylinders to the roof beams of the powered supports.

Roof Supports

63. The support units were spaced at 4-ft intervals along the wall. A support unit consisted of two support frames each with two hydraulic legs. The frames were positioned on each side of the floor and roof beams to which the conveyor sections were attached. The hydraulic controls of the frames were so arranged that no frame could be released before the full setting pressure was reached in both legs of the adjacent support frame.

Drive Units

64. The conveyor cutting chain was driven by two 65-hp drive units, one at each end of the wall. The drive units were standard plough drives consisting of an electric motor, fluid coupling and reduction gearbox driving the chain sprocket. The drives were sequentially started and gave a chain speed of 92 ft/min. The drives were mounted on anchor-type supports located at the end of the wall.

Wall Manriding Equipment

65. To reduce the physical effort of travelling along the wall, 0.5-hp winches were mounted at each end of the wall. The winches hauled a 15-ft long "carpet" made from conveyor belting along the access track. Control of the winches was via pull wires along the track. Lock-out switches were provided, as a safeguard for the operator, to prevent inadvertent operation of the winches and all other functions.

Horizon Control

66. The section cut by the highest and lowest picks, and the height of the clearance picks was varied by tilting the conveyor pans and top guides. The tilting action was performed by hydraulic cylinders and follower valves which were mounted on brackets on the goaf side of the conveyor sections. The follower valves were operated, via a linkage, from the goaf side of the powered support legs. The main function of the valves was to enable the correct adjustment of piston rod extension to be selected and maintained.

Other Equipment

67. The conveyor was fitted with hinged ramp plates at the front to assist in the loading of the coal.

68. The whole length of the wall was provided with lights attached to the conveyor in front of each support.

69. Loudspeaker telephones, with built-in signalling and lock-out facilities, were provided throughout the length of the wall. The system contained a pre-start warning device and was interlocked with the stage loader.

Problems with the Yarmak Miner

Cutting and Loading

70. The machine and system were found to cut very well and were capable of rates of advance/retreat of the order of 2 in/min.

Horizon Control

71. Steering of the equipment proved to be very difficult and, for this reason, the trial was stopped. Steering difficulties arose as the operator could not adequately see the area being cut and there was insufficient adjustment in the equipment to correct misalignment.

72. The steering problem was aggravated by mechanical problems with the ramp plates being torn-off their mounting and wedging against the floor.

Advancing Mechanism

73. Many difficulties were experienced with advancing the conveyor and supports because of problems with the hydraulic control circuits and the high forces caused by jamming of ramp plates. The high forces then caused further damage to the conveyor frames.

Suggested Solutions
to Problems

74. The problems of steering, advancing and mechanical strength could be solved if the system were redesigned. A major problem was the hinged ramp plates. The use of rigid ramp plates in place of the hinged design would simplify the front-end structure and also increase its rigidity.

75. A system of trailing tapes could be employed to check that the unit was advancing evenly, and hence reduce the stresses in the structure by keeping a straight face. The tapes, stored on reels along the wall, could be anchored on the floor and paid out into the goaf as the system advanced. The amount paid out by each tape would provide a direct indication of the advance of each portion of the wall.

76. The problem of the operator having to observe deviations from the horizon could be overcome by the use of TV cameras along the wall. These could be monitored from the end of the wall and adjustments made to the steering rams by remote control. Access to the wall would then only be required for maintenance and inspections.

POTENTIAL OF SYSTEM

77. The theoretical maximum carrying capacity of the conveyor unit of the Yarmak system is approximately 270 tph. To utilise this capacity, the whole unit would have to advance at the rate of 11.25 ft/h with a 300-ft length of wall, a 24-in seam, and coal having a density of 80 lb/ft^3. This would appear feasible as advance rates of 10 ft/h were indicated for short periods in the surface trials. On the basis of six hours' continuous operation per shift, the machine would yield 1,620 tons with an advance of 67.5 ft/shift. An output of 1,000 tons/shift with a corresponding advance of approximately 41.7 ft could be considered reasonable for purposes of production planning.

78. The drivage of access or gate roads would still remain a problem to be solved. If a multi-entry development system were adopted, with a three-entry set serving as headgate and tailgate for adjacent panels, then about 150 ft of drivage would be required per shift in order to maintain sufficient development to replace the entries and their crosscuts.

79. The amount of development required would be reduced if longer walls could be used. Two factors would mitigate against longer lengths, namely chain strength and cost. At present, the strength and power requirements of the armoured face conveyor limit the length of conventional mechanised longwalls to a maximum of about 1,000 ft. The chain in the Yarmak system has two functions, one of supplying the cutting forces and the other of conveying the coal. Each would reduce the theoretical maximum length of chain.

Economic Assessment of the Yarmak Miner

Productivity

80. A full-face miner based on the Yarmak principle and producing 1,000 tons per shift from a 300-ft wall would probably require two operators on the wall per shift to control the horizon. An operator would be required at each end of the wall to control the setting of the power unit and gate supports.

81. The crew would be completed with a console operator or mechanic and a foreman, giving a total crew of seven persons per shift.

82. This would give a wall productivity of about 143 tons per manshift. This productivity would be considerably reduced by development workers as three development crews would probably be required to complement each wall crew. This would involve a further 27 men, bringing the total labour force to 34. Neglecting the coal produced by the development, the productivity would then be in the region of 29 tons to 30 tons per manshift.

Capital Costs

83. The capital costs of the system are likely to be higher than a comparable longwall system, such as a plough. This is due to the more complicated construction of the conveyor pans and the powered supports. Remote sensing and control systems would further add to the capital cost.

Operating Costs

84. The material costs are not likely to be high as no consumable support material is required. Material would be required principally for support in the development heading and for pipe track, conveyors and stoppings. Labour costs would be low due to the high potential productivity giving a low overall operating cost.

Summary

85. The prototype machine failed owing to inadequate design, particularly of the steering mechanism. Further research and development were stopped because of the problems and the economic climate of the UK coal industry at the time. Should the design problems be overcome, the machine would be capable of achieving very high outputs out of seams down to 24 in.

86. The capital costs of the installation would be high but this would be compensated by high output and productivity.

87. The major problem remaining would be to provide the necessary development required, some 150 ft per shift.

88. The system would be potentially safe as the men on the wall would be fully protected whilst those in the gates would have a comfortable working height.

THE IN-SEAM MINER

89. The in-seam miner was originally designed to advance independently the ends of an advancing longwall where the main wall machine could not overcut the conveyor drives.

90. It has been extended in length and a 60-ft-long model is used for driving a dirt absorbing heading. The function of the miner in this case is to mine a 60-ft-wide wall or face in conjunction with the driving of a roadway higher than the coal seam. The stone roof mined in the formation of the roadway is then packed into the waste area formerly occupied by the coal. The object of this arrangement is to reduce the amount of stone sent out of the mine and to obtain some revenue from the coal mined to offset the cost of the drivage.

91. In-seam miners are currently being produced to mine down to 34 in. It is possible that a machine with a range down to 24 in could be produced if the market is present.

Description of Equipment and Operation

92. Plate 50 shows a general arrangement of an in-seam miner.

93. Coal is cut from the whole of the face by cutterpicks which traverse the face from end to end. The picks are mounted on pick carrying plates which, in turn, are mounted on a chain. The chain runs in a rigid jib guide which extends the full width of the face and carries the bottom and top race for the chain. At each end of the jib, the chain is hauled around a toothed sprocket which can be either electrically or hydraulically powered.

94. Loading buckets, attached to the chain, pick up the cut coal when travelling along the bottom race of the jib. The coal is discharged via a chute on to a delivery conveyor when the buckets have been elevated around the delivery-end drive sprocket. The machine is similar to the Yarmak miner in that a single chain and drive assembly has the dual function of conveying and cutting.

95. The chain jib is pivot-mounted on to a base frame which allows the jib to be raised or lowered hydraulically to control the vertical cutting horizon of the machine. The mechanism is effective in vertical steering.

96. The machine is advanced into the face by hydraulic pushing rams attached to the base frame and staked on to anchor units.

97. When used as a stablehole machine, support behind the miner is normally supplied by a prop-and-bar system. There is no reason why a series of powered supports could not be installed behind an in-seam miner to establish the machine as an independent coal producer.

Problems with In-seam Miner

98. The in-seam miner would have the following problems as a production machine.

Bulk Output

99. The bulk output of the in-seam miner would be limited by the conveying capacity of the bucket system. This is adequate for short-length machines but would become a bottleneck for a long production model.

Short Length

100. The longest machine constructed to date had a length of 60 ft. It is unlikely that machines much over 100 ft in length could be produced without considerable modification to the chain haulage mechanism in order for it to be able to carry the higher loads generated by the increased length. With the present short length machines there is no problem with undulations in the seam. With increased lengths and a rigid frame structure, problems of maintaining an acceptable horizon could arise in an undulating seam.

101. In a thin seam, a short length would be a serious economic problem owing to the amount of development that would be necessary to service a short length of wall.

Results Obtained

102. The average results obtained, using in-seam miners as stablehole machines, are not representative of the potential output of the machine and system owing to the machines being constrained by support requirements to stay in step with the longwall.

103. Where the machines have been used independently on such tasks as opening out new longwalls and in conjunction with tunnel driving work, they have achieved much better results. Advances of 2 in/min have been obtained on a 34-ft-long unit, in a 42-in seam, giving an output of 45 tph to 50 tph.

104. This would give 250 tons to 300 tons per shift if operations were arranged so that continuous cutting could be achieved over the duration of the shift.

Suggested Solutions to the Problems

105. The main drawback of the machine is its relatively low conveying capacity. In order to increase the capacity, it would be necessary to increase the size and number of conveying buckets on the machine. The present chain speed of 200 ft/min could be increased slightly but it is unlikely that this would appreciably increase the capacity.

106. Increasing the length of the machine by adding further jib sections would increase the number of picks cutting at the face but the strength of the chain and applied power to the sprockets would have to be commensurately increased.

107. An increase in length would probably require an increase in the strength and rigidity of the main frame and jib frame to ensure that the machine would advance in a uniform manner without causing structural failure.

POTENTIAL OF SYSTEM

108. The present potential of the system is about 250 tons to 300 tons per shift. If the conveying capacity could be increased, then the potential output from the system would be correspondingly increased. Assuming a doubling of the conveyor capacity, an output of 400 tons to 500 tons per shift should be possible from the system.

109. An output of 500 tons per shift from a face 200 ft long, 24 in high, at a coal density of 80 lb/ft^3, would result in an advance/retreat rate of approximately 31 ft per shift. This would be required at each end of the face giving a total advance of 62 ft per shift.

110. The in-seam miner can easily be operated as an advancing unit, as its original design was to advance as a stablehole machine. If the machine was operated as an advancing unit the development entry requirements would not be severe as, with a suitable floor, continuous miners could be used at both ends of the panel to rip out floor to create the necessary access roads. Special dust suppression precautions would have to be taken during such an operation and legal variances obtained so that the advancing of the wall and the making of the access roads could proceed simultaneously.

111. The material cut from the floor to create the gates could, in the case of the headgate, be loaded out with the coal. In the case of the tailgate, the material would have to be stowed in the waste or a transport-haulage system introduced to dispose of the dirt to gain additional height.

Economic Assessment of In-seam Miner

Productivity

112. If the capacity could be raised to 400 tons per shift and the wall was 100 ft long, the operation of the in-seam miner would require approximately three operators, a mechanic and a foreman. In order to advance the gates and install supports, a further six men would be required, giving a total crew of 10. This would give a section productivity of approximately 40 tons per manshift.

Capital Costs

113. The capital costs of the system would be that of the in-seam miner, powered supports and equipment for producing access entries at each end of the face. The total cost would be moderate owing to the relatively short length of face.

Operating Costs

114. Operating costs would be moderate. The overall cost of labour would be low, but expenditure on materials for support of the entries would be high owing to the low output of coal per foot of entry caused by the short face length.

Summary

115. The in-seam miner requires some research and development work in order to increase its conveying capacity.

116. Only relatively short faces would be possible with this type of machine.

117. The machine will mine effectively in an advancing layout and could be used to reduce the cost of access entry development. Access roads could be economically formed by removing floor in the waste behind the advancing wall. To obtain an efficient layout with only two access roads, major variances would be required from the current US mining legislation on escapeways and ventilation.

118. Outputs of 400 tons per shift may be possible at a sectional productivity of 40 tons per manshift.

119. The system should be relatively free from hazards as good support would be achieved from the powered supports. The make of dust is minimal owing to the low cutting speeds, and the dust is effectively kept out of the main airstream by the jib frame.

THE MINIWALL SYSTEM

Introduction

120. The miniwall system is intended for use on wall lengths of 100 ft to 300 ft and in seam heights of up to 10 ft.

121. Conceptually, the system could probably be adapted to work seams of less than 30 in, but this would make access along the length of the wall, for maintenance purposes, very difficult due to the limited lateral and vertical space.

Description of Equipment and Operation

122. The miniwall system is designed to cut a parabolic-shaped face with a series of closely-spaced reciprocating cutters linked together with a cable or chain. The cutters are activitated from the ends of the wall by hydraulic cylinders situated at each end. The stroke of the cylinders gives effective movement to the cutters of about 6 ft. Plate 51 illustrates a general arrangement of the miniwall system.

123. The cutters are guided by a single centre-strand armoured face conveyor which conforms to the shape of the wall (a single centre-strand AFC will operate on an uncompensated curve).

124. The cutters and conveyor are protected from roof falls by a series of canopies which form a shield, with their tips in close proximity to the face.

125. The roof shields and conveyor are advanced by means of actuators situated in the entries adjacent to the block of coal being mined.

126. The operators are stationed behind the cutter actuators a safe distance away from the face.

Problems with the Miniwall System

127. As the system is untried, the problems mentioned above are hypothetical and their magnitude could only be ascertained by a number of trial installations.

Vertical Steering

128. Vertical steering is a problem that has plagued most plough-type mechanisms, except those working with very hard floor and weak coals. As the miniwall system is unmanned, it would be difficult to determine where along the wall a deviation in horizon had taken place. Access to that particular section would be required to change the attitude of the cutters so that the correct horizon was cut.

Roof Control

129. The purpose of the roof shields is to prevent material from the roof falling on the conveyor or cutter mechanism rather than to hold the roof adjacent to the wall and to cause it to cave behind the shields. Two sources of difficulty arise from this arrangement:-

(i) pieces of roof material could jam between the shields and the face;

(ii) the weight of caved roof resting on the shields could obstruct the movement of the shields.

Potential of System

130. Assuming that ideal conditions could be found, ie a soft coal, hard floor and a roof that did not break off right up against the face when unsupported, then the system might work.

131. The capacity of the system has been suggested as 1,000 tons per day per foot of face height. The actual rate would more likely be limited to the conveying capacity of the armoured face conveyor. An output of 1,500 tpd to 2,000 tpd is conceivable in a 30-in seam with an operating crew of possibly four persons per shift.

132. The operation, though producing coal at this rate, would advance at a rate of over 50 ft/day (300 ft length, 30 in seam, coal density 80 lb/ft^3). This would require several room and pillar development sections to block out the necessary ground which would have to be taken into account when the productivity of the overall operation was evaluated.

Capital Costs

133. The capital costs of the system would be somewhat lower than for a comparable length longwall as the shield supports would not be as complex. If, for reasons of roof control, it became necessary to use a support that was positively set against the roof and was then individually advanced by remote control, the system would become more complex than a comparable longwall and would have a considerably higher capital cost.

Operating Costs

134. The system would appear to offer a low operating cost owing to the small consumption of materials and the high productivity. The majority of the cost would be in the supporting development sections, and the total cost would be considerably less than a room and pillar system operating alone.

Summary

135. In concept, the miniwall is a simple system for coal extraction.

136. It is likely that in operation several major problems would occur which could result in the loss of the wall and in the equipment being abandoned.

137. If the potential problems of roof control and horizon steering could be overcome, then the system could produce very economically.

CHAPTER XXVI

SCRAPER BOXES

INTRODUCTION

1. Scraper boxes are probably one of the simplest systems of longwall mining.

2. The scraper box was initially used as a haulage device on hand-worked longwalls in thin seams. The coal was broken from the coal face and loaded manually into the scraper track. The scrapers consisted of a "box" open at the top, bottom and front. The rear of the box contained a hinged scraper blade. When the box was drawn forwards, the blade assumed a closed position preventing the contents of the box from escaping. When the box was reversed, the blade swung on its hinge and opened, allowing the box to ride over any material in its track.

3. In order to extend the reach of the system, additional scrapers were operated on the same wall, the wall being divided into equal sections with one scraper per section. Each scraper carried its load along its section and left it on the floor. The next box collected the coal and took it along the wall to be collected by the next scraper. The last scraper took the material and tipped it into the headgate of the wall. By adding cutter blades to the scrapers, it was found that the units could cut, load and haul the coal off the wall.

4. One of the earliest successful scraper box systems was the Haarman scraper box. In this system the scraper boxes were forced up against the coal face by means of a heavy skid board. This caused the blades to take a shallow cut off the face each time the box passed.

5. In the later models the skid board was replaced by a heavy-duty chain which spanned the entire length of the longwall. The ends of the longwall were kept in advance of the centre allowing the wall to assume a bow shape. The chain was kept under tension which forced the boxes up against the solid coal. This arrangement became known as the "chain tension scraper box".

6. A major advantage of the chain tension scraper box was the elimination of the skid board on the wall and therefore the requirement of personnel to continuously move the board forward as each pass of the scraper boxes was made.

7. This meant that the only task on the wall was to install roof supports to prevent the roof from caving close to the wall and, hence, closing the scraper track.

THE HAARMAN SCRAPER BOX

Description of Equipment and Operation

8. Dr. Haarman, in 1947, combined the pneumatic pushing cylinder, as applied to the plough system, with the scraper box and fitted static cutter blades to each box. The pushing cylinders were staked between the roof and floor and pushed a skid board towards the face. The skid board then forced the scraper boxes against the face.

9. Labour was only required on the wall for the setting of roof supports, advancing the pushing cylinder stells and setting waste-edge supports. The boxes were hauled by rope using a double-drum winch in the headgate and a return wheel at the tailgate end of the wall.

Problems With Haarman System

10. The Haarman system - whilst a large improvement on hand-breaking coal from the face - still required personnel on the wall. This limited its application to seam heights which were accessible to men, ie seams of a thickness greater than approximately 16 in.

11. The pulling forces required on the boxes were high owing to each box being forced into the face irrespective of the local changes in the coal hardness, and to the frictional forces that developed between the scraper boxes and the guiding skid boards. These high forces required a large winch.

12. The skid boards, approximately 14 in high, also prevented personnel obtaining easy access to the scraper boxes to make any adjustments to the cutter blades.

Results Obtained

13. In the soft thin seams of the Ruhr in Germany, the Haarman system worked well. Although the bulk tonnage achieved was modest, productivity was much higher than with the hand-won systems and, hence, costs were lower.

14. The following figures were achieved in the early 1950s:-

Wall length	155 m	509 ft
Seam height	0.60 m	24 in
Extracted height	0.75 m	30 in
Daily advance	2.60 m	8.5 ft
Daily output	340 tonnes	375 tons
Labour	68 persons (3 shifts)	

15. Coal production was on two shifts, with maintenance, the moving of sheave-wheel anchor beams and water infusion being carried out on the third shift.

16. This arrangement gave a section productivity of 5 tonnes per manshift compared with an output of 2.1 tonnes per manshift for a similar section where the coal was broken from the solid using hand-held pneumatic picks.

CHAIN TENSION SCRAPER BOX

Description of Equipment and Operation

17. In the chain tension scraper box arrangement the skid boards of the Haarman system were replaced with a round-linked chain that spanned the wall. The boxes were attached to the tension chain by means of trapping guides on the goaf side of the boxes. Plate 52 gives a general arrangement of a chain tension scraper box set for double-unit operation.

18. The boxes were hauled by wire ropes actuated by a double-drum scraper winch. The ropes were deflected around sheave wheels at each end of the face. The sheave wheels were attached to a beam to facilitate the adjustment of the position of the sheaves with the advance of the face.

19. Tension was applied to the chain by means of a mechanical puller situated in one of the tailgates (a double-unit face has a centre headgate and a tailgate at each end). The tensions used varied from 4,000 lb to approximately 8,000 lb, with a low tension of 500 lb being used on the first few passes of the boxes to ensure that the track was clear and the boxes would not jam.

Results Obtained

20. The following results were achieved in 1960 in the 20-in to 22-in Harvey Seam in Durham, UK:-

Wall length	420 ft
Layout	"Double unit", ie two simultaneously-advancing panels sharing a common headgate
Output (10 months)	27,059 tons
Labour (10 months)	5,998 shifts
Productivity	4.5 tons per manshift

21. The productivity covered all operations associated with the operation of the face, including the driving of advance entries to house the sheave beams and the packing of all the stone from the headings into the waste area of the faces.

22. An average of 31 men were employed per day on all operations. Of these, only 11 men were directly involved with the wall operation, the remainder being employed on the driving of headings and the packing of stone.

23. Of the 11 men involved on the wall, 7 men were employed on timbering and support work, the remaining 4 operated the scraper equipment.

Remote Operation

24. The chain tension scraper was used in the UK in seams down to 16 in with men on the wall to set supports.

25. In view of the considerable high-quality reserves that existed in the Durham coalfield in seams less than 14 in thick, several attempts were made to "remotely" mine these seams using scraper boxes.

26. A 2-in-diameter hole was drilled across the proposed wall line from one gate to another, a distance of approximately 120 ft. The hole was then progressively reamed out using rope-drawn tools until it was large enough to allow a scraper box to pass through.

27. The scraper box and tension chain were then hauled through the hole and scraper operation commenced.

28. It was found that all went well for an advance of approximately 27 ft to 45 ft until the roof caved, closing the scraper track against the wall. In some instances, it was possible to extract the boxes, install temporary narrow boxes and recommence scraping until the next roof failure occurred.

29. Where scraping could not be directly resumed after a fall of roof, a new hole was bored and reamed out to establish a new face line.

30. The best cycle of operations was to mine for 45 ft at the rate of 6 ft per shift, taking eight shifts, and then to re-establish the face in another 6 shifts of boring and reaming.

31. This whole cycle, in an average seam height of 13 in, yielded approximately 235 tons of coal for an expenditure of approximately 44 manshifts, giving a productivity of 5.3 tons per manshift. This excludes the labour expended in the driving of the entries.

Problems with Chain
Tension Scraper Boxes

32. There were a number of problems with the chain tension arrangement for scraper boxes.

Normal Cutting Force

33. The use of a chain as a means of pushing the boxes into the face did not provide a sufficiently positive normal force. This resulted in slow rates of advance, although rates of 1 ft per hour were recorded.

Wall Shape

34. The chain meant that the wall had to be in the shape of a bow. This was found, in some instances, to cause a hardening of the coal in the middle of the length of the wall, presumably due to the lagging portion attracting extra stress from the roof, ie a "weighting effect".

Operational Delays

35. Time was lost in the anchoring of the sheave beams at each end of the wall and the moving of the sheave wheels. This was mainly due to the operations being performed by hand.

Wall Length

36. The inadequacies of the equipment, and the opening-out techniques to establish new walls in the lower seam heights, limited the possible wall lengths. A consequence of the limited wall lengths was the high ratio of stone mined in the access roads per ton of coal extracted by the scraper boxes. This had an adverse effect on the economics of the system.

Roof Control

37. Roof control in seam heights accessible to men was effective though expensive in labour. In seam heights of less than 20 in, single wooden props were used without roof beams. These did not provide the best security for the operating personnel. The development of powered supports has not been pursued in these low conditions.

38. The control of the roof in the lower sections inaccessible to men is a major problem that has to be resolved. The early trials in the UK were terminated because of the inability to mine more than about 50 ft without losing the wall owing to roof falls.

39. Experience in the USSR, using ram ploughs without support, again indicates that a distance in the order of 50 ft is mineable before roof falls stop operations.

Floor Dilution

40. Scraper boxes can only be worked where the floor is considerably harder than the coal. Attempts to "steer" the boxes have been made in thicker seams by adjusting the position of the cutter blades but, in the main, where soft floors have been encountered, the action of the scrapers running over the floor has caused the floor to be scraped out with the coal.

Conveying Capacity

41. The maximum conveying capacity of a scraper box system in level conditions is the loaded capacity of one of the boxes multiplied by the number of passes the system of boxes can make per unit of time.

42. The number of passes is dependent on the speed at which the boxes travel and their distance apart.

43. In thin seams, the capacity of the boxes is limited by the available height for the coal inside the box to heap up.

44. The previously-used systems employed rope speeds in the region of 120 ft/min with boxes spaced some 30 ft to 50 ft apart. At a spacing of 30 ft, with an overlap distance of 10 ft, the average distance travelled by the box per cycle was 80 ft. Hence, at 120 ft/min average speed, the cycle would take $60 \times 80 \div 120 =$ 40 seconds. With a load of 0.6 tons, this would give a capacity of $60 \times 60/40 \times 0.6 =$ 54 tph.

Reasons for Diminishing Use

45. Scraper boxes, in their present form, are still a relatively low-capacity system, though extremely useful in thin seams.

46. Owing to the economic pressures in the 1960s and early 1970s, the use of scraper boxes diminished with the closure of many of Europe's very thin seam operations with which they were associated.

47. In the ultra thin, minus-15-in sections, the problems of roof control were not solved and the time taken to re-open faces, together with the discontinuity of production whilst this was taking place, caused the system to be abandoned.

Suggested Solutions to the Problems

Wall Shape

48. Where the normal cutting force is supplied by a rope or chain in tension, there must be an angular displacement between adjacent boxes. For relatively small angles, the normal force is directly proportional to the tension in the rope or chain and to the deflection in the tension member as it passes around the box.

49. In order to get a straight wall and yet maintain or increase the force pushing the boxes into the face, it would be necessary to increase the tension forces from the 2-to-4-ton range to, possibly, the 30-to-40-ton range. This could be done by using the large cross-section high-tensile chains that are now commercially available.

50. The tension force could be augmented by using the "return" haulage chain as a tension chain.

Operational Delays

51. Many of the previous delays of moving up the sheave beams could be eliminated by housing the beams (or direct-haulage motorised pullies) in self-advancing anchor units which would lock themselves between the roof and floor at the ends of the wall.

52. The general use of powered equipment would speed-up the ancillary operations, reduce the labour requirements and improve the efficiency of the whole operation.

Wall Length

53. The economics of the operation require the minimum of stone to be mined per ton of coal produced.

54. Increasing the wall length would achieve this objective. An increase in wall length would require more boxes and a greater output per unit of advance. This would then require additional conveying capacity.

55. A longer wall would also increase the tension forces required to keep the boxes hard up against the wall. This could be achieved by the use of heavy-duty equipment.

56. The increase in wall length would require a longer initial hole for the establishment of the face line. The problem of deviation of the hole line with increasing length would then become a limiting factor in the ultra-thin seams of less than 15 in. The problem could be partially alleviated by the use of large-diameter drills to open the line. The same constraints that apply to augers would not necessarily apply to the boring of new face lines. Extra time could be spent on

the initial alignment of the drill and the rate of feed could be adjusted to maintain alignment, rather than the maximisation of coal output from the drilling of the hole.

57. The maximum length of face line that could be opened up would depend on local seam conditions. With primitive equipment drilling 2-in-diameter holes, wall lengths of 240 ft were opened up in the late 1950s and, hence, with modern equipment, lengths in excess of 300 ft should be practicable.

Roof Control

58. In seam heights of 20 in to 30 in, ie below the range that can be effectively mined by the plus-30-in systems, but high enough to afford a reasonable height for working and travelling, it would be possible to adopt or develop powered supports to operate behind the scraper track. These could be attached to a common guide rail which could be used to force the boxes into the face and dispense with the tension chain. This, in fact, would be a return to the Haarman design.

59. Powered supports, with a suitably-designed travelling way and the use of sequential or batch control of the movement of the supports, would minimise the labour required on the wall and, at the same time, provide safer conditions for the operators.

60. In the seam heights below which men can effectively operate, roof control determines whether scraper boxes can be used. If a roof could be found that would bend and close in the goaf without breaking, then there would not be a problem. However, it is not anticipated that such roof conditions would frequently be found in the USA and hence other solutions must be sought.

61. Two possibilities exist. One is to accept that failure of the roof will occur and determine what distance can be mined before the probability of failure of the roof is indicated. Concurrent with the mining of a wall, a new line could be opened up. Prior to the incipient failure, the equipment could be transferred to the new wall and the cycle repeated.

62. A second line of attack would be to develop a remote support system. This could be apparatus for the remote siting of permanent support elements in the waste, such as wedges or props, or the pneumatic injection of waste material into the goaf. This could be done from each gate side to form a loose pack in the waste that would give sufficient support to the roof to prevent fracture.

63. The use of inflated tubes in the waste has been suggested. A number of these could be used. They could be sited immediately behind the scraper track and inflated tight between the roof and floor. After the wall and scraper had advanced sufficiently, the tube at the back, nearest the waste, would be deflated, hauled out of the waste and re-installed at the front.

64. On some occasions, it is likely the roof would lower as the tubes were deflated, trapping the tubes and damaging them as they were withdrawn. This would necessitate the tubes to be of an inexpensive construction.

65. Whatever support system is adopted and developed, it must be able to operate remotely. It must not require maintenance to be carried out along the line of the wall in very low conditions and it must be inexpensive.

Floor Dilution

66. There would not appear to be any simple solution to weak floors and the inadvertent mining of a soft bottom. Hence, any further development of the scraper box should be directed to hard bottoms, leaving the weak-bottomed seams to other "steerable" systems.

Conveying Capacity

67. The conveying capacity in thin seams is constrained by the height available for the box. In order to obtain the largest individual load possible, the boxes should be as wide and as long as is practicable.

68. The cycle time, ie the time for a complete backwards and forwards cycle of the string of boxes, could be shortened by using more boxes and, hence, reducing the distance each box has to travel and increasing the speed at which the boxes traverse the face.

69. A greater number of boxes on a length of wall would require a greater total force to operate the cutter blades and a greater haulage force to cause the boxes to reciprocate. Higher speeds and higher forces would require stronger equipment and higher powers.

70. In order to accommodate the haulage forces, the haulage ropes could be replaced by round-link chains of heavy section and these could be directly driven by motorised sprockets at each end of the wall.

71. The use of heavy-duty chains for haulage could serve the dual purpose of providing the haulage forces and the reaction or normal force to cause the boxes to ride up against the coal face.

Summary of Solutions

72. The problems associated with scraper boxes may be summarised as equipment capability and roof control.

73. The length of face and output could be improved by using stronger and higher-powered equipment, both to increase the conveying capacity and the rate at which coal is cut.

74. With the necessary research and development work, it would appear that the problem of roof control could be solved.

POTENTIAL OF SYSTEM

75. Previously-obtained results have shown that outputs of 60 tph could be sustained for a few hours per shift.

76. Improved equipment should ensure that this figure would become the average output in seams of approximately 18 in.

77. Using this as a base figure, the following results would be obtained for a 400-ft-long single panel:-

Seam height	18 in
Output per hour	60 tons
Output per shift	300 tons
Advance per shift	12.5 ft
Crew	3 to 5 persons

78. The labour requirement would be an operator at each end of the wall to control the advance of the anchor stations and to operate the remote support-installing equipment, together with a mechanic and supervisor.

79. The operation of the boxes could be automatic with reversing and limit switches to maintain the reciprocating motion.

80. The above figures do not include the necessary labour for developing the greater-than-seam-height entries that could be either pre-developed or produced concurrently with an advancing layout. The use of a double-unit, single-entry layout would reduce the amount of development required, but this would require considerable variations from the present US mining legislation on escape ways and ventilation.

Safety and Health

81. With no men working in the extremely low environment near to where the coal is being cut from the face, the system would have a high inherent degree of safety. No men would be under newly-exposed roof and, by suitable control of the ventilation and the judicious use of water sprays attached to each box, very little dust would be produced by the system.

Economic Assessment

Productivity

82. Productivity in the region of 60 tons to 100 tons per manshift in seams less than 24 in thick would be possible with an improved system. However, this productivity would be at least halved if the development labour to produce the entries were considered.

Capital Costs

83. The capital costs would be relatively low, the equipment consisting of two anchor units with one or two drives and a tensioning arrangement. The boxes themselves would be inexpensive and, likewise, the connecting haulage and tension chain.

84. As no powered supports would be used on the main run of the wall, the total cost of the equipment, excluding the developing equipment, would be far less than the cost of equipping a room and pillar section.

85. The low capital cost, and hence the finance and depreciation charges, would place only a low capital recovery charge on each ton of coal produced.

Operating Costs

86. The labour content of the operating costs would be relatively low. Material consumption would largely depend on the system adopted for roof control along the length of the wall.

87. If spaced-out wooden wedges provided the necessary degree of roof control, then the costs would be relatively low. If it were necessary to resort to filling the waste area with a weak cementacious material constrained in bags or tubes, then the costs would be high, possibly prohibitively so.

Summary

88. Scraper boxes are a proven system of thin seam mining.

89. The output and productivity potential could be improved at low cost by up-rating the individual items within the system.

90. The system is capable of mining down to heights of 12 in or even less but with the reduction in height, the capacity of the system would fall off and costs would inevitably increase.

91. The cost of a research and development project to up-rate the equipment and form a demonstration would be relatively low.

92. The critical technical problem to be overcome would be the control of the roof in seams too low for human access.

CHAPTER XXVII

WIDE-WEB CUTTER LOADERS

INTRODUCTION

1. Wide-web cutter loaders were used in the mechanised working of thin seams in the UK, Germany and in the USSR. Numerous machines and models have been produced but, apart from the USSR and in a few isolated instances in the UK, the use of web cutters (as opposed to wide-web shearers) has ceased. Machines of the wide-web class include the "multi-jib", the "back-to-back", the "midget", the Korfmann and the Russian UKT. Wide-web cutter loaders have an application down to approximately 16 in.

DESCRIPTION OF THE EQUIPMENT AND OPERATION

2. In the wide-web system a strip of coal from 4 ft to 6 ft wide is cut off a longwall face. The coal is cut by a "wide-web machine" and is loaded on to a fixed-position conveyor.

3. As new roof is exposed, it is supported by means of props installed between the conveyor and the wall. At the end of the cutting run, the machine is turned around to cut back along the face. The conveyor is then dismantled and re-installed in a new track adjacent to the face whilst the supports are withdrawn at the rear allowing the waste to cave.

4. The cutting, moving over and withdrawal of supports are separate operations and are normally carried out by different crews on different shifts. This causes the operation to be of a cyclic nature.

5. At the ends of the wall there has to be sufficient space available to turn the cutter around and to accommodate the drive unit of the face conveyor. This means that for advancing faces, large stableholes are required at the ends of the walls which are normally worked by separate crews.

6. The system is completed by ripping or excavation of roof to make the gate roads. The material blasted out of the roof is then hand-packed into the waste to form packs at the side of the gate roads.

Cutter Loaders

7. All the machines used were floor mounted. Various cutting mechanisms were incorporated and, on some of the machines, a combination of mechanisms was used. Haulage of the machine was normally by a rope which was paid out in the front of the machine and staked at the end by a sprag between the roof and the floor. The machines normally had on-board winches which hauled in the rope dragging the machine forward. In some instances, haulage was achieved by a static

rope along the length of the wall which was wrapped around a capstan wheel on the machine. When the wheel was turned, the machine was hauled along by the frictional forces. The machines were normally powered by air-cooled electric motors. The available powers were low due to the small frame sizes that could be accommodated within the height available. Plates 53 to 55 show the arrangement of cutters for the multi-jib, "back-to-back" and midget miner.

8. With the back-to-back arrangement, two machines were coupled together, the front machine acting as a haulage and doing a limited amount of cutting whilst the back machine completed the cutting and loaded out the bulk of the coal.

9. In order to apply more power to the cutters from the on-board motor, remote haulage systems have been used. These consist of winches at the ends of the wall which haul the cutting machines through the face. These are frequently used in the USSR on the UKT machines.

Cutting Mechanisms

10. The cutting mechanisms used were mostly frame jibs with the cutter picks being carried on chains. In the case of the multi-jib machine, a series of jibs were used, one above the other, to cut out the required seam height (see Chapter VIII, Cutting Mechanisms).

11. In the case of the midget miner, the web was cut by a series of rotating auger heads and the cusps between the heads trimmed by a top and bottom chain jib. The Korfmann machine also used augers and trimmed the roof with a chain jib. A further vertical jib was used to square out the back of the cut.

12. The Russian machines use a combination of shearer-type drums and jibs. These machines are normally used in steeply-inclined seams where the cut material falls away from the back of the machine.

Conveyors

13. The earliest of the wide-web cutters used bottom belt conveyors. These consisted of a powered drive unit anchored at one end of the wall with a return pulley at the other end. The loaded strand of the conveyor rubbed directly on the floor of the seam whilst the return strand ran on "hangers" suspended close to the roof. The system was simple and cheap as the conveyor belting used was frequently belting discarded from the main belt haulage system of the mine.

14. The absence of conveyor structure simplified the moving of the belt from one track to the next as it was merely a matter of removing the pins of the mechanical belt joints, and moving the individual lengths of belt into the new track then coupling up after the drive and return pulley had been moved over.

Support

15. In the very low sections worked by men on the wall, support was normally provided by either wooden props with headboards or with thin corrugated steel

straps. The props were set by hand but as the cutter normally cut a fixed height of extraction, the props could be supplied pre-cut to the correct height, which facilitated easy and quick installation.

16. Support of the waste edge, the breaker line, was normally supplemented by cogs incorporating half bricks in the structure to facilitate their subsequent withdrawal. Alternatively, quick-release mechanisms were used.

17. On completion of the cutting run, the cogs were dismantled and re-erected at the new waste edge. The intervening props were then withdrawn allowing the waste to cave.

Floor Cutting

18. With the "back-to-back" arrangement of machines, the leading machine was used to pre-cut the seam at the bottom, the remainder of the cutting and the loading function being carried out by the rear machine. In very low seams of about 12 in to 14 in, where there was a weak floor, the arrangement was slightly changed. The leading machine was used to undercut the seam in the floor and the cuttings from the floor were mechanically cast into the goaf using a "gummer stower". The rear machine then cut and loaded out the seam. This arrangement had a number of advantages:-

(i) The working height for personnel was increased.

(ii) The material cut from the floor gave a measure of support to the roof when cast into the goaf.

(iii) Although out-of-seam material was being cut concurrently with the coal, only a clean coal product was actually loaded out.

RESULTS OBTAINED

19. The results achieved in terms of tonnage and productivity were low, mainly due to the cyclic nature of the series of operations and the labour required for the tasks of moving forward the conveyor and supports. The tonnage mined per web advance of the face was dependent on the web thickness, the length of the face and, above all, on the extracted height. Hence, in comparing the tonnage and productivity, the height must be taken into consideration.

20. The following are examples of the results achieved by wide-web machines in the UK in the early 1960s:-

System	Extracted Height	Web	Output Per Day	Productivity Per Manshift
	in	ft in	tons	tons
Multi-jib	21	4 6	300*	5.8+
Back-to-back	17½	4 6	178	3.4
Midget	30	4 9	164	7.0⁰

* Double unit face; length 450 ft per unit

\+ Including all face labour and labour to advance gates

⁰ A retreat longwall development; labour excluded

21. The above results may appear low but these machines and systems were frequently used in conditions where other systems had failed owing to weak tops or coal too hard to be mined either by plough or by scraper boxes.

PROBLEMS WITH WIDE-WEB CUTTER LOADER SYSTEM

Equipment

22. The wide-web equipment was developed mainly from existing longwall coal cutters. The longwall cutters had been designed principally to make a slot in a coal face approximately 5 in wide to assist in subsequent blasting operations. Their new duty, when modified, was to cut out the whole section of a seam using the same basic motor, haulage mechanism and cutting gear train. Many complaints were made against the reliability of the equipment but the basic problem was the imposition of a duty in excess of the original design.

Cyclic Nature of Operations

23. The wide-web cutter loader, operating in thin seams, does not readily lend itself to a prop-free face layout but, owing to its width, requires support to be installed between the face and the conveyor. This means that either the support has to be removed to advance the conveyor or that the conveyor has to be removed and then re-installed in front of the lines of support. This precludes the use of conventional powered supports and, of necessity, makes the operation both expensive in labour and cyclic in nature.

Safety and Health

24. A major problem with the early wide-web machines was the large percentage of fine coal produced and the large amounts of dust, including respirable dust, that were released into the atmosphere.

25. The system of support using wooden props could not be adequately pre-loaded, which resulted in the hazard of possible roof falls.

SUGGESTED SOLUTIONS TO THE PROBLEMS

26. The problem of limited bulk output could be approached as follows.

Increased Output Per Web

27. Assuming that the operation is accepted as being of a cyclic nature and that one cut or web would be obtained in one shift, then, in order to increase the bulk output, the volume of coal cut in one pass would have to be increased. This could be done by making the wall longer and increasing the width of the web cut.

Improved Machines

28. Increased length implies that in order to complete a web in a shift, the machine must travel or cut at a higher haulage speed. A higher haulage speed and a wider web would require higher machine power and a stronger, heavier haulage and transmission to handle the power.

29. The higher powers could be provided by the use of water-cooled motors. The water could subsequently be used for dust suppression.

30. The cutting mechanisms would have to be examined to ensure that at the higher rates of advance and haulage speed, the coal output from the increased cutting capacity could be cleared from the region of the cutting picks or tools.

Wall Conveyors

31. The increase in length of wall cut would require a longer conveyor and the increased web and haulage speed of the cutting machine would mean that the capacity of the conveyor would have to be commensurately increased. This should be possible with the improved, synthetic, solid-woven, conveyor belts now available.

Support

32. The setting and withdrawing of timber supports is time-consuming and labour intensive. In order to improve the standard of support and, hence, the integrity of the working place and to reduce the task of erecting and withdrawing the supports, the timber could be replaced with light-weight hydraulic supports. These could be set using a "Wander" hose, ie be pumped up with external hydraulic power but without using permanently-installed and interconnected support hydraulics.

33. The breaker cribs could be replaced by custom-designed self-advancing hydraulic supports which, again, could be powered by a Wander hose.

Elimination of Delays

34. A frequent delay element in the operation of wide-web machines was the operation of the machine's haulage. This was normally in the form of a winch drum on the machine containing approximately 60 ft of wire rope. This was paid out in front of the machine and attached to a shoe which was staked to the floor using a sprag between roof and floor. After the machine had hauled itself forwards and exhausted the rope, the whole operation had to be stopped whilst the rope was drawn out of the machine, manually pulled out in front of the machine and then re-staked in position. This operation added a delay of approximately 10 minutes for each 50 ft of wall length cut.

35. The use of a chain haulage along the full length of the wall would eliminate this element of delay.

Multi-Shift Production

36. The traditional system of operation of the wide-web machines was to take one web per 24 hours and use the balance of the time available to carry out all the other operations such as moving the conveyor forwards, moving the supports and advancing the rippings to produce the access gates.

37. The adaption of using pre-formed gate entries, ie a retreat system, combined with an improved support system, would permit one or more webs to be taken per 24 hours, probably one web per shift.

38. A convenient approach would be to arrange the web thickness and face length to permit one web to be taken and the necessary preparation work to be completed in one shift. Two production shifts could be worked in 24 hours and the swing shift used for maintenance and other ancillary tasks.

Dust Suppression

39. The problem of dust suppression could be approached in two ways: the use of sprays within the cutting mechanism and the use of cowls and shrouds to prevent any dust entering the main ventilating airstream. This is seen as an engineering problem which is not unique to wide-web cutters but requires attention and development in almost all forms of underground coal cutting.

REMOTE OPERATION

40. The wide-web cutters have been suggested in the USSR as a remotely-operated system in very low seams. The area of application has been in highly-inclined seams where there is no need for a coal transport system on the wall, but the coal would slide away down the floor of the seam under the influence of gravity. In these layouts the cutter would be remotely hauled from the top end of the wall.

41. The supports would be remotely installed using a series of wedges transported and tightened by a hydraulic wedge setter. This unit would be lowered

down the face and the wedges tightened at the appropriate position. The unit would then be withdrawn, new wedges loaded in and the unit lowered again to set the next support. The setting of the supports would be carried out concurrently with the operation of the cutting machine.

42. The system, though of interest, could only be applied, in its present form, to steep seams, as in less steep areas a conveying device would be required. The concept of a wedge setter could, with suitable modifications, be adapted to other remotely-operated thin seam systems.

ECONOMIC ASSESSMENT

43. Assuming the method of extraction were applied to a retreat layout operating on a 600-ft-wide longwall, taking a 7-ft-wide web, this would yield, in a 24-in seam, approximately 336 tons of coal per web. This would diminish to approximately 252 tons in an 18-in seam. If two webs were cut per day, this would yield 672 tons and 504 tons per day in the 24-in and 18-in seams respectively.

44. The use of a chain haulage would reduce the machine crew to three, one driver and two operators following the machine installing the supports. Possibly a further five men would be required per shift for moving supports and other operations in moving over into the new track.

45. This would give a productivity in the region of 30 tons to 40 tons per manshift, dependent on the seam section. This does not include labour required for development purposes, which would require possibly a doubling of the total labour force, reducing the overall productivity to 15 tons to 20 tons per manshift.

46. Wide-web cutter loaders would be inexpensive in terms of capital required and, hence, capital charges.

47. The potential productivity is low owing to the labour-intensive nature of the process, mainly the erection and withdrawal of supports on the wall.

48. The relatively long length of wall possible would reduce the cost of development per ton of coal produced.

49. Material consumption would be low if hydraulic supports were used in place of timber.

SUMMARY

50. The wide-web system is a proven method of coal production and has produced where ploughs and scraper boxes have failed.

51. The range of application is basically limited by the minimum height at which man can operate on faces, though seams as thin as 12 in can be mined where it is possible to cut out floor material.

52. The operation is of a cyclic nature, normally one cut being taken per 24 hours. The output could be increased by using more powerful equipment, capable of wider webs and longer lengths of cut, and increasing the number of cuts per day.

53. The system depends on personnel operating within the seam height. Although much can be done to limit the heavy physical labour required, the system is intrinsically labour intensive and, hence, the cost of operations is likely to be relatively high.

CHAPTER XXVIII

COMPARISON OF MINUS-30-IN SYSTEMS

INTRODUCTION

1. Many systems have been developed and experimental trials conducted in an attempt to mine seams less than 30 in thick both safely and efficiently. Most of the systems were proven to produce coal but, owing to general economic pressures, have been partially or wholly abandoned with the general reduction of thin-seam working. The systems discussed in the previous chapters are summarised below with a subjective indication of the amount of research and development work that would be necessary to make them viable:-

System	Minimum Operating Height	Potential* Output Per Shift	Potential** Productivity	Safety and Health	Operating Costs	Required Research and Development
	in	tons	tons/manshift			
Mole	20	400	30-40	Good	Low	High
Augers	12-15	200	10-15	Good	High	Moderate
Full*** face	16-24	1,000	30-40	Moderate /good	Low/ moderate	High
Scraper boxes	-12	400	30-50	Good	Low	Low
Wide web	16-18	330	15-20	Moderate	Moderate	Moderate

* At a common seam reference height of 24 in
** Including an estimate for access development
*** Including ploughs without powered supports

2. The comparative table is based on many assumptions: principally, that satisfactory solutions would be found to solve the past technical problems.

3. The suggested outputs and productivities should not in any way be regarded as absolute figures but rather as a general guide to those systems that might be worth further development.

MINIMUM OPERATING HEIGHT

4. The heights recorded indicate the minimum at which the systems have been operated in the past or for which it was planned to produce equipment prior to funding being cut off. The minimum range suggested would vary with the precise unit chosen within the system and the general seam conditions in which the equipment would be required to operate.

5. In the case of augers, 12 in to 15 in is suggested as an economic cut off rather than a technical limitation, as it is quite possible to bore a 2-in-diameter hole in coal for long distances and produce some small output in the process.

OUTPUT

6. The outputs of all the systems, except the full-face miner, are based on the reasonable expectation of what would be produced if improvements were made to proven systems. The full-face miners of the Yarmak type were never proven, although indications were gained of their potential during abortive trials. The figure of 1,000 tons per shift is therefore very dependent on a successful outcome of research and development work.

PRODUCTIVITY

7. All the systems examined showed good potential productivities on the actual coal extraction process.

8. It would appear that the major bottleneck would be the driving of access entries for the housing of equipment and the provision of the normal services to the specific area of coal being worked. The driving of entries would not yield much output owing to the thin coal but would require a high complement of labour.

9. The labour required for driving entries would probably exceed the labour requirements of the coal extraction operation. Consequently, productivity would be governed more by the ratio of entry drivage per ton of coal extracted than by the actual labour productivity of panel extraction.

10. Systems that lend themselves to wide panels can be more competitive than others owing to the lower development requirements. Augers, if used as a primary coal extraction process, have high development requirements owing to the short lengths of holes that can be bored and the low utilisation of the reserves.

SAFETY AND HEALTH

11. With the exception of the wide-web system and low plough layouts, operating personnel would not be required to enter under the low roof or, if they were, as in the case of the full-face systems, they would be well protected from hazardous situations.

12. The wide-web and low plough layouts would not provide the same level of protection owing to support being provided by individual prop-type supports.

13. In the remote systems, such as augers and mole miners, very good control of the ventilation could be exercised and any dust-laden air from the cutting process could be directed away from the operators or treated in large scrubbers to remove particles potentially injurious to health.

OPERATING COSTS

14. A major element in the operating cost structure of all the systems would be the cost of driving the access entries.

15. The use of wide panels and obtaining variances from the mining law to reduce the development footage would be essential to reduce the cost of mining and make the systems economically competitive.

16. Most of the systems, although developed for thin coal, could be applied to seams of medium thickness.

17. In thicker seams, the cost of operations would be vastly reduced as the cost of access would drop to zero (the entries would be self-financing) and the capacities of the systems would be increased.

REQUIRED RESEARCH AND DEVELOPMENT

Mole

18. In the case of the mole miner, the cost of R & D would be high owing to design work being required on all parts of the system. However, the system has been proven and, whilst a heavy expenditure would be required, the probability of the project being successful would be high.

Augers

19. The main problem with augers is the short lengths of hole they are able to bore. R & D would be necessary to make the augers stay in the seam or correct deviations. As this is one specific problem, the cost of a programme to overcome the difficulty might not be high but the probability of finding an effective solution, if a solution is, in fact, to be found, is by no means certain.

Full Face

20. Considerable R & D work would be required to make the full-face systems viable. A complete engineering redesign would be necessary to simplify and strengthen the structure and to overcome the problems with the steering mechanism.

21. Whilst the work would be straightforward, the cost of making and testing prototypes would be extensive. The ultimate potential of the system would, however, make it an attractive proposition.

Scraper Boxes

22. The R & D cost for a trial scraper-box system would be low. The scraper boxes themselves are of relatively simple construction. The drive and tension equipment could be made from existing equipment. An example of this would be the drive units: these could well incorporate presently available plough drive units.

23. The solution to the problem of roof control would be more problematic and, whilst the cost of R & D work on alternative solutions would not be high, the probability of finding an economical solution would be limited.

Wide Web

24. The wide-web machines have produced large outputs at relatively low rates. To develop the system further, the basic cutter units would have to be redesigned to extend their capabilities and improve their reliability.

25. If a self-advancing support system could be devised that would work in conjunction with the other elements of the system, the potential output and productivity would be considerably improved.

APPENDIX "A"

LIST OF PUBLICATIONS ANALYSED

No	Title	Author and Reference
1	Ultra-Deep is Routine at Dutch Mine	Robert Harrold, Coal Age, May 1979
2	Experience with Collins Miner at Rothwell Colliery	W.J. Charlton & M. Riddell & J. Nixon, Mining Engineer, September 1967
3	Mechanisation of 12 in to 24 in Seams	A. Williams, Institution Mining Engineer, March 1962
4	The Development of the Midget Miner at New Lount Colliery	L.J. Mills, Midland Counties Institute, March 1959
5	Power Loading in Thin Seams in the Barnsley Area of the NCB	A.W. Tuke, Mining Engineer, May 1977
6	Quality Control at the Face. Symposium on Mining Methods	J.B. Keirs, Institution of Mining Engineers, 1974
7	Some Aspects of Thin Seam Mining in North Derbyshire	A.F. Deakin, Mining Engineer, May 1977
8	Thin Seam Mining, Part 2, at Annesley Colliery	A.J. Wardle, Mining Engineer, May 1977
9	Equipment for Continuously Monitoring Methane at the Face. The T4 Head	L.R. Cooper, Mining Engineer, January 1969
10	Thin Seam Shearing at Barrow	Anon, Colliery Guardian, February 1974
11	Thin Seam Mining and Dust Suppression, Langworth Colliery	Ted Watson, Colliery Guardian, December 1975
12	Support at the Face. Symposium on Mining Methods	A. Wheeler, Institution of Mining Engineers, 1974
13	Stable Hole Mechanisation with NCB Dosco In Seam Miner	M. Goldsby, Mining Engineer, July 1978
14	Combatting Dust by Water Infusion	J.C. Deprez, Mining Engineer, July 1978

No	Title	Author and Reference
15	Dirt Adsorbing Heading at Whitwick Colliery	J.W. Ash, Mining Engineer, July 1977
16	Control Equipment for a Remote Controlled Mole Mine - the Collins Miner	V.M. Thomas & P.J. Beque, Mining Engineer, June 1963
17	The Development of the Collins Miner	R.F. Lansdown & G.B. Dawson, Mining Engineer, September 1963
18	A System for the Extraction of Coal Without Supports	Dosco Overseas Engineering, 1977 (manufacturer's literature)
19	Development and Underground Trials of the MRDE Conveyor Miner	NCB Internal Report, 1972 (No 72/33)
20	New Method Group Report No 1 (Thin Seam Mining Without Supports)	R.L.J. MacRae, NCB Internal Report, Bretby, undated
21	Summarised Description of Mechanical Coal Winning Using Scraper Boxes	Anon, NCB Internal Translation
22	Slurry Pumped 3,000 ft Vertically	Jiro Wakabayashi, Coal Age, June 1979
23	The Geological Mining Appraisal of the Major Coal Basins in the USSR	IIASA, 1977
24	Cutting Coal Winning in the Federal Republic of Germany	Dr. Hans-Rolf Sander, Gluckauf, August 1972, p664 to 676
25	Wide Web Working with the In-Web Shearer at Florence Colliery	H.C. Evans, Coll & Mining Engineer, April 1978, p527 to 536
26	Economic Commission for Europe "Methods of Working Thin Seams"	July, 1967 ST/ECE/COAL 30
27	15th Report of Mines Safety and Health Commission	Commission European Community, 1977
28	A Review of Thin Seam Mechanisation	H.G. Cook, Mining Engineer, March 1966, p391-405

No	Title	Author and Reference
29	Growth of Longwall Technologies in USA	SME-AIME, February 1979
30	Making Mining Safer Yet	J.L. Collinson, Mining Engineer, November 1976, p83 to 92
31	The Law Relating to Safety and Health	UK Stationery Office
32	The Noise Environment of the Underground Coal Mine	MESA IR 1034
33	Pyro Mine Sticks with Conventional	Robert Harrold, Coal Age, May 1979, p78 to 81
34	Thin Seams Yield High Outputs	Harold Davis, Coal Age, July 1979, p104 to 129
35	Impact of Changing Technology on the Demand for Metallurgical Coal and Coke Produced in USA to 1985	USBM
36	Projects to Expand Fuel Sources in Eastern States	USBM IC 8725
37	Roof Fall re Support Accidents re Study	USBM IC 8723
38	Coal Mine Illumination	USBM IC 8709
39	Continuity Raises Thin Seam Output	Harold Davis, Coal Age, November 1978, p58 to 59
40	NCB Accident Report Forms	
41	1979 R. & D. More Mine Related	Coal Age, February 1979, p74 to 75
42	Underground Auger Mine Pillars	Nicholas P. Chironis, Coal Age, April 1979, p103 to 106
43	External Problems Continue to Hinder Remote Control Usage in Mines	Coal Age, May 1975

No	Title	Author and Reference
44	The Drive for Safety in the Mines	Coal Age, July 1973
45	Coal Exploration	Miller Freeman Publications, May 1976
46	Managing Health and Safety in a Period of Change	J.L. Collinson, Mining Engineer, November 1976, Paper No 3, Harrogate
47	Noise Control of an Underground Continuous Miner Auger Type	MESA IR1056
48	Coal Mine Injury and Employment Experience by Occupation 1972/75	MESA IR1065
49	Injury Experience in Coal Mining 1976/74/73/72 (four volumes)	IR1097, IR1076, IR1075, IR1074
50	A Flexible Helical Rock Bolt	USBM RI8300
51	Longer Than Seam Height Drill Development Program	USBM RI8273
52	Single Entry Development for Longwall Mining	USBM RI8252
53	Dust Control on a Longwall Face with a Shearer Mounted Dust Collector	USBM RI8248
54	Reduction of Dust and Energy During Coal Cutting Using Point Attack Picks	USBM RI8185
55	Laser Alignment Sensor for Continuous Mining Machines	USBM RI8134
56	Cutting Experiments Using a Rotating Water Jet in a Borehole	USBM RI8095
57	An Analysis of the National Coal Economy and a Marginal Producing Region 1976/90	Iowa State University Card Report 68
58	Technological Innovations Abound in Coal Mountains of Appalachia	Coal Age, May 1975

No	Title	Author and Reference
59	Coal Mining	SME Mining Engineering Handbook, Volume 1, 1973
60	A Review of Spontaneous Combustion Problems and Controls with Application to USA Coal Mines, 1978	PD-NCB/USA Department of Energy
61	Injury Experience in Coal Mining 1977	MSLIA (MSHA)IR1108
62	Advanced Mining Systems Emerging Trigged by Heavy Federal Funding	Coal Age, February 1977
63	Health and Safety Get Big Boost from Longwall	Coal Age, January 1977
64	Coal Facts, 1979	West Virginia Coal Association
65	Shortwall Mining at Helen Mining Company	Mining Congress Journal, August 1977
66	Coal Facts	National Coal Association
67	The Impact of Overmining and Undermining on the Eastern Underground Coal Reserve Base	US Department Commerce PB262519
68	Conceptual Design of an Automated Longwall Mining System, Volume 1 Final Report	USBM PB263213
69	Accident Cost Indicator Model to Estimate Costs to Industry and Society from Work-Related Injuries and Deaths in Underground Coal Mining	USBM PB264438
70	Conceptual Design of an Automated Longwall System, Phase 1 Survey of Operating Longwalls	USBM PB263211
71	Thickness of Bituminous Coal and Lignite Seams Mined in the USA in 1945	-

No	Title	Author and Reference
72	Basic Estimated Capital Investment and Operating Cost for Underground Bituminous Coal Mines, 1975	USBM IC8689
73	Improvements in and Relating to Mining Machines (Star Wheel Disc Cutter)	British Patents Office
74	Comparative Shortwall and Room and Pillar Mining Costs	USBM IC8757
75	Thickness of Bituminous Coal and Lignite Seams Mined in 1965	USBM IC8345
76	Coal Recovery from Underground Bituminous Coal Mines in the USA, Leg Mining Method	US Department of Interior IC8785
77	An Approach to Automated Longwall Mining	AIAA - E. Palowitch, P. Broussard, February 1979
78	Mini Symposium Longwall Underground Mining	SME-AIME, New Orleans, February 1979
79	Shortwall Mining Experience in the USA	AIME - E. Palowitch, F.R.Zachar, February 1976, Las Vegas
80	Coal Mine Mechanisation and Accident Frequency Rates of Hand and Mechanised Loading in Illinois	USBM IC7063, April 1939
81	Projects to Expand Fuel Sources in Eastern States, and Updated Information Circular	IC8725, IC8765 US Department of Interior
82	Coal Data	National Coal Association, 1977
83	Horizon Control of Shearer Power Loaders	G.E. Green, R.J. Dalton, Colliery Guardian, May 1971, p228 to 231
84	The Revitalised Coal Industries	H.E. Collins, Colliery Guardian, September 1974 to June 1975

No	Title	Author and Reference
85	The Development of Low Seam Mining in the USA	Jeffrey Mining Machinery
86	Plough Extraction of Steep Protective Seams (0.2 m to 0.45 m) (Russian and English text)	Donetz Coal Research Institute UGOL
87	Research and Application of Augering in Thin Seams (Russian and English text)	Karaganda Coal Research Institute UGOL T. Tokmagambetov, P. Varekha & E.A. Sudarikov
88	Mechanisation of Unmanned Room Extraction of Thin Seams in Weak Strata	Karaganda Coal Research Institute, UGOL
89	US2U Integrated Scraper Ram Plough	UGOL
90	Achieving a 1,000-tonne Longwall Load in a Thin Level Seam	V.M. Ivashin, UGOL
91	Soviet Equipment Projection (table)	UGOL
92	The Working and Total Mechanisation of Thin Coal Seams	A.V. Dokukin, UGOL
93	A Century of Coal Face Mechanisation	Forrest S. Anderson & Robert H. Thorpe, Mining Engineer, August 1967, p775 to 785
94	Coal Cutting Machines - Transport of	Institution of Mining Engineers, 1901
95	Mining Methods in Europe	Hill Publishing Co, 1909
96	Jeffrey Continuous Haulage Systems	Jeffrey Mining Machinery
97	Jeffrey 404L Ram Car	Jeffrey Mining Machinery
98	Impact of the Metal/Non-Metal Special Accident Prevention Programme for Fiscal Year 1976	IR1105-1979, MSHA

No	Title	Author and Reference
99	Coal Miners' Attendance at Work	NCB Medical Service
100	Annual Report and Statistics 1978	National Coal Board
101	Bureau of Mines Minerals Availability System and Resources Classification	BOM Circular IC8654
102	The Development of the Vertical Drum Shearer Loader (Dranyam)	Midland Counties Institute, December 1960
103	The Reserve Base of US Coals by Sulphur Content, Eastern States	BOM Circular IC8680, 1975
104	The Reserve Base of Coal for Underground Mining, Western States	BOM Circular IC8678, 1975
105	Soviet Coal Through 1980	Mining Congress Journal, October 1974, Strishkov, Markon, Murphy
106	Monthly Statistics Bulletin, United Nations	Vol XXXIII No 6, June 1979
107	Mechanisation Profile No 20	National Coal Board, 1978
108	Cannelton, Pittston Try New Miner (Wilcox)	Coal Age, August 1979, Robert Harrold
109	World Coal Resources and Production	World Coal Industry Report and Directory, 1977
110	Dawson Miller (Development)	Dawson & Mills, The Mining Engineer, May 1962
111	The Machine for New Mining Techniques	F.S. Mall & J.H.R. Cope, Mining Engineer, April 1971, p451 to 461
112	Mechanised Mining in Czechoslovakia	Colliery Guardian, September 1970, p440 to 444
113		MRDE
114	Comparison of Injury Hazards in Different Coal Seam Heights	Shirley Hudson, MSHA, Denver, Colorado

No	Title	Author and Reference
115	Mechanisation of the Narrowest On Seams	Coal, Gold and Base Minerals - S. Africa, May 1979, p19 to 24
116	Advances in Noise Control of Underground Coal Mining Equipment	J.Alton Burke, Coal Conference and Expo III, 1976
117	Extinguishing Coal Mine Fires	Alex Bacho, David R. Forshey, Adi Guzdar, David Monaghan. Coal Conference and Expo III, 1976
118	Design of Mine Layouts	Working Party Report, 1972, NCB Mining Department
119	Comparative Costs of Collins Miner and Longwall Operations	R.L.J. MacRae, Report No 48 25/11/63, NCB MRDE
120	NCB/Dosco In-seam Miner Range of Machines	TU (79) 5, NCB MRDE
121	Development and Trials of the 860 - 1100 mm In-seam Miner	Internal Report No 78/32, NCB MRDE
122	Dust and its Effect on Longwall Mining	Eugene H. Jones, Joseph Kuti, Mining Congress Journal, August 1979, p47 to 53
123	The Selby Coalfield	L.J. Mills, The Mining Engineer, October 1975
124	Diagram of Twin Auger - Longwall Coal Cutter (text in Russian)	UGOL No 7 1970, Anon
125	Vertical Drum Machine (text in Russian)	UGOL - Ukrainy No 3, 1973, Anon
126	Vertical Drum Machine (text in Russian)	UGOL - Ukrainy No 2, 1972, Anon
127	45^0 Angle Shearer (text in Russian)	UGOL, No 11, 1958
128	Vertical Drum "mole" Miner Raise Boring System (text in Russian)	UGOL - Ukrainy No 3, 1974
129	Steep Seam Cutter Loader (text in Russian)	UGOL - Ukrainy No 5, 1977

No	Title	Author and Reference
130	Short Face or Stable Hole Machine (text in Russian)	UGOL - Ukrainy No 10, 1977
131	Review of Mechanisation, Conveyors and Machines (text in Russian)	UGOL - No 4, 1956
132	Chain Saw Plough (text in Russian)	UGOL - Ukrainy No 11, 1977
133	Chain Saw Plough (text in Russian)	UGOL - Ukrainy No 6, 1978
134	Mini Mining System	Montgomery Mining Machinery Mfg Co, 1979
135	The Breakage of Coal by Wedge Action	C.D. Pomeroy, Colliery Guardian, November 1963, p642 to 648, p672 to 677
136	Dust Control in Coal Mining, MRDE Handbook No 15	NCB, September 1978
137	Institute of Occupational Medicine Report, 1972 to 1975	National Coal Board
138	Thin Seam Coal Mining in the US	Bruce Nelson and B.V. Johnson, USBM Twin Cities Mining Research Centre
139	Underground Thin Seam Mining	Bruce Nelson, USBM Twin Cities Mining Research Centre
140	Coal Mining Report of Technical Advisory Committee	UK Stationery Office, March 1945, C.C. Reid and others
141	Methane Prediction in Coal Mines	IEA Report, ICT IS/TR04, December 1978
142	Estimating Methane Content of Bituminous Coalbeds from Adsorbtion Data	USBM, RI 8245, 1977
143	Analysis of US Underground Thin Seam Mining Potential	Ketron, Inc, September 1978
144	Coal in America	McGraw Hill

No	Title	Author and Reference
145	Physical Environment, Productivity and Injuries in Underground Coal Mines	C.L. Christenson and W.H. Andrews, Journal of Economics and Business, Vol 26, No 3, p182 to 190
146	Design Tables for Bord and Pillar Workings in Coal Mines	Chamber of Mines, Anon, South Africa, 1972
147	How a Small Operator Gets Big Productivity	Richard H. Mason, Coal Mining & Processing, July 1978
148	Retreat Mining Systems	R. Rawlinson, Symposium on Productivity Through Technology, Paper 9, October 1978, Harrogate
149	Rock Mechanics in Coal Mining	M.D.G. Salamon and K.I. Oravecz, Chamber of Mines, South Africa
150	Coal and Energy Quarterly	NCB, No 22, Autumn 1979
151	Collins Miner - Manor Colliery Retreating Longwall Operations	J.T. Walton, Mechanisation Report, 1968
152	Thin Seam Shearing at Barrow	Anon, Colliery Guardian, February 1974, p53 to 55
153	Hansa Hydromine Goes on Stream	Paul C. Merritt, Coal Age, 1978, p135 to 137
154	Thin Seam Mining - Remote Erection of Thin Seam Support (text in Russian)	UGOL, May 1978
155	Review of Partial Extraction System (text in Russian)	UGOL, June 1976
156	Layout of Steep Seam Cutter Loader (text in Polish)	Przeglad Gorniczy, No 6, (934), 1976
157	Thin Seam Mining - Vane Tempest Colliery	E.P. Farrange, The Mining Engineer, August 1979, p141 to 156
158	US Coal Mining Accidents and Seam Thickness	D.P. Schlick, R.G. Peluso and K. Thirumalai, Australian IMM, September 1976
159	Copy of Roof Control Plan	April 10 1979

No	Title	Author and Reference
160	Project Register 3-79 Coal Winning Underground	International Energy Authority, Mining Technology Clearing House
161	Project Register 003/77 Monitoring and Control of Machines and Mining Systems	International Energy Authority, Mining Technology Clearing House
162	Korfmann - Thin Seam Power Loader Seam 0.6 to 1.3 m	Anon
163	Safety Statistics, Transvaal, OFS and Natal Mine	The Reef, November 1978, p83
164	Retreat Mining	C.T. Massey, The Mining Engineer, October 1977, p39 to 43
165	Modern Augering	Anon, Coal Age, January 1971, p66 to 69
166	Mechanised Longwall (Russian UKT Miner)	Karl Hartland - Consultant to Moroccan Ministry of Power
167	Miniature Equipment Boosts Production in Low Coal Mines	Anon, Coal Age, January 1959, p87 to 88
168	Joy CU43 Continuous Miner	Leaflet
169	Mobile Coal Cutter	Coal Age, June 1959
170	Underground Auger	Coal Age, April 1959
171	Low Height Shovel	Coal Age, April 1959
172	Duckbill Layout Pillar Retreat	Coal Age, April 1960
173	Underground Auger	Coal Age, November 1959
174	Joy Low Seam Equipment	Coal Age, July 1959
175	Low Seam Shuttle Car	Coal Age, April 1959
176	Thin Seam	Coal Age, February 1960, p98 to 102
177	Thin Seam Continuous Miner	Coal Age, May 1961, p90 to 93
178	Low Seam Miner	Coal Age, July 1959

No	Title	Author and Reference
179	Extra-low Personnel Carrier	Coal Age, April 1959, p150
180	Joy CU42 Compton Miner and Low Seam Equipment	Coal Age
181	Joy Rotobuster	Coal Age, June 1959
182	Lee-Norse Miner for Low Coal	Coal Age, July 1957, p108
183	Rolf 4 at Woolley Colliery	C.L. Round, Colliery Guardian, January 20th 1967
184	Priorities of Development in the Extraction of Thin Seams (text in in German and English)	W. Knissel, G. Lange, H. Kundel, Gluckauf, 18th September 1975, Year 111, No 18, p853 to 859
185	NCB Safety Statistics	Figures from sample mines
186	Coal Cutting Machines	Transactions of the Institution of Mining Engineers, Vol 23, 1901 to 1902, p328 to 341
187	Progress with Thin Seam Mechanisation	C.H. Davis and J.H. Paterson, The Mining Engineer, October 1965, p33 to 53
188	Remote Control Mining Comes of Age	George C. Lindsay, Coal Mining and Processing, September 1973, p30 to 41
189	Jewell Ridge Coal Corporation Applies Laser Instrumentation to an Underground Mine	Anon, Coal Age, June 1974, p72 to 75
190	Gas Composition Calculation for the In-situ Gasification of Thin Seams and the Approach to Modelling	S. Debrand and O.J. Hahn, Institute for Mining and Minerals Research, University of Kentucky, Lexington, Kentucky 40506
191	Progress Report, Important Future Mining Technology	Anon, Mining Equipment International, p9 to 17
192	Results of Installation of Underground Mine Lighting Systems	Robert L. Vines, Coal Convention, AMC, Pittsburgh, May 1st to 4th
193	La Gazeification Souterraine du Charbon (text in French)	P. Ledent, Industrie Mineral, February 1977, p81 to 91

No	Title	Author and Reference
194	Foraction Avec Marteaux Fond de Trou Utilisation Dans Les Houillerres	J. Decherf, Industrie Mineral Mine, March 1977, p10 to 15
195	Improvements of Designs and Techniques of Winning with Ploughs	Ing Tegethoff, Project No 2 593-001, Ruhrkohle AG Essen
196	MRE Auto Percussive Coal Plough, NCB	F.E.W. Marsh, Report No 2070, 1957, Research Programme No 1.3
197	Poland Develops its Own Mining Equipment	R.K. Singhal, Coal Mining and Processing, May 1969, p30 to 33
198	Continuous Belt Haulage Makes Mark Underground	D.C. Torre, Coal Mining and Processing, March 1974, p32 to 35
199	Longwall in Thin Seams	George W. Sanders, Mining Congress Journal, March 1970, p57 to 64
200	Huwood Slicer Loader	NCB Bulletin 58/195
201	Long Hole Blasting	NCB Bulletin 58/203
202	Flight Loading	NCB Bulletin 56/164
203	Self Employment Works for C S & S	Robert Harrold, Coal Age, April 1979, p58 to 61
204	New Possibilities and Current Development of Hydraulic Transport in Mining Industry of Federal Republic of Germany	M. Kuhn, Fourth International Conference on the Hydraulic Transport of Solids in Pipes, Paper E4, 18th to 21st May, Banff Springs, Alberta, Canada
205	Hydromechanical Coal Winning and Hydraulic Transport	Dr. Hans Maurer, Paper presented at Transmatik 76, Karlsruhe, March 1976
206	Mining Problems Encountered in the Application of Hydrotechniques at Hansa	DPL Ing Hilgenstock, second meeting of MTCH Technical Working Group on Hydro-transport, September 1978
207	Hansa Hydraulic Mine	NCB MRDE Visit Report
208	S & B V Activities Related to Hydro-mining and Hydrotransport	Dr. H. Maurer, 1978

No	Title	Author and Reference
209	Supervision and Control of the Hydraulic Conveyance of Raw Coal by Modern Measuring Equipment at the Hansa Hydromine	D. Jordan and R. Wagner, Fifth International Conference on the Hydraulic Transport of Solids in Pipes, May 8th to 11th 1978, Hannover, Germany
210	Thin Seam Working with Hydraulic Monitors	Z.B. Susloukh, UGOL - Ukrainy Vol 16, No 8, August 1972, p9 to 10

TABLE I

COAL QUALITY AND RESERVE STATISTICS FOR THIN SEAMS, BY STATE

State	Coal Region	Coalfield	Seam	Average Thickness	Coal Quality							Reserves	Resources
					M	V	A	FC	CV	S	Coking		
				in	%	%	%	%	Btu/lb	%		10^6 tons	10^6 tons
Alabama	S Appalachian	Coosa	Fairview	28	1.5	26.7	13.9	57.9	13,010a	4.1			
		Plateau	Underwood	27	0.9	24.6	15.8	58.7	12,720a	3.0	Good		
		Northern	Cobb	2 x 24									
Arkansas	W Interior		U Hartshorne	14 - 34									
Georgia	S Appalachian		Aetna	24	3.49	28.90	2.45	65.16	14,628	0.79	Good		
			Dade	24	3.07	27.60	4.70	64.63	14,398	0.76	Good		
Illinois	E Interior		Colchester 2	18 - 40	13 - 19	31 - 45	3 - 11	35 - 48	10,400-11,700a	1 - 5			
Indiana	E Interior		Blue Creek	24 - 36			low-med			low-med			
			Mariah Hill	24 - 36			med			med			
			Dale	24 - 30			med			low-med			
Iowa	W Interior		Nodaway	0 - 36								> 326.18	
			Mystic	30								> 1,038.62	
			Whitebreast	14								> 256.71	
Kansas	W Interior	South-east	Thayer	10 - 24	6		12		14,800b	2			
			Mulberry	26	10		23		14,732b	5			
			Mulky	12 - 22	2.8		9.3		15,130b	3.9			
			Bevier	15	6.5		24		15,077b	2.6			
			Croweburg	14			26		9,800a	2.5			
			Fleming	12 - 15	5		24		15,120b	5			
			Mineral	15 - 24	5		21		15,029b	4			
			Dry Wood	10 - 14									
			Rowe	10 - 20	6.5		26		10,100a	4.7			
		East-central	Nodaway	> 24	10.2		10		11,093a	7.6			
		North-east	Cherokee	14 - 36									
Kentucky	C Appalachian	Eastern	L Elkhorn	up to 32	2.1	36.1	4.2	57.6	13,790	0.7			
			U Elkhorn 1	28			4.6		14,200	0.7			
			U Elkhorn 2	24 - 36									
			Hazard 7	30 - 32	4.2-4.8	35-57.6	5.4-6.5	51.1-54.7	13,100-13,290	0.6-0.9			
			Lily	25 - 40	4	40.0	10.9	49.1	12,620	3.3			
			Princess 5	28									
			Princess 7	35									
Missouri	W Interior		Rowe	18	11		12		11,260a	3.9			
			Drywood	14									
			Weir-Pittsburg	24 - 30	8		8.7		12,348a	3.6			
			Tebo	18 - 30	10.4		12.4		11,700a	3.9			
			Mineral	12 - 24	8		14.8		11,380a	5.3			
			Fleming	18 - 24	7.2		15.8		11,225a	4.6			
			Croweburg	11 - 36	14.1		8.9		11,051a	4.3			
			Mulky	12 - 28	10.6		9.4		11,450a	4.7			
			Summit	18 - 24									
			Lexington	20 - 28	14.5		11.7		11,549a	3.4			
			Mulberry	12 - 42	8		18.1		10,973a	3.4			
Oklahoma	W Interior	North-east	Rowe	10 - 36									9c
			Weir-Pittsburg	13 - 36						> 3		Large	
			Mineral	14 - 22						3 - 6			> 27c
			Croweburg	12 - 41						< 1			679c
			Iron Post	11 - 17						3.5-5.0			65-137c
			Dawson	14 - 30						>3			183c
		Arkoma	Stigler	12 - 32							Yes		533c
			Cavanal	24									159c
			Secor	18 - 52			high			high			436c
Pennsylvania	N Appalachian											Recoverable reserves estimated at 13,000 million tons in 24-in to 28-in seams; 10,000 million tons in 28-in to 36-in seams	
Tennessee	S Appalachian		Aetna	24	2.0-5.7	24.6-39.4	2.1-23.8	46.4-67.4	10,940-14,680	0.4-5.1	Good		
			Blue Gem	20 - 36	3.4-5.7	36.8-39.8	1.2-5.2	51.1-56.3	13,480-14,300	0.7-2.2			
			Dade	24	2.8-6.3	23.2-29.3	4.7-13.4	60.6-65.4	12,830-14,398	0.4-1.4	Good		
			Glen Mary	24 - 40							Yes		
			Jellico		2.4-3.8	34.3-36.1	4.3-7.4	54.5-58.1	13,260-13,930	1.7-1.9			
Texas	North-central		Strawn	28	3.6	33.9	15.2	47.4	11,650a	2.4		117	
			Bridgeport	20	11.9	32.6	13.5	42.2	9,370a	2.1		29	701
			Cisco	6 - 30			>13		<10,000a	> 3			50
	Rio Grande		Santo Tomas Cannel	24 - 36	3.2	45.6	14.7	36.4	11,660a	2.2			115d
Virginia	N Appalachian		Hagy	20 - 32									
			Lyons	30 - 32									
			Taggart Marker	18 - 40									
W Virginia	C Appalachian		Uniontown	24 - 36									600e
			Harlem	24									270e
			Brush Creek	30									90e
			Little Eagle	30									900e
			Cedar Coal	30									250e
			Douglas	30									415e
			L Douglas	24									100e
			Sewell B	30									120e
			Pocahontas 2	0 - 36									285e
			Bens Creek	30									250e

Source: Data abstracted from 1978 Keystone Coal Manual

Notes:
a = as received basis;
b = dry, ash-free basis;
c = total remaining resources;
d = original inferred resources;
e = original mineable tonnage.

TABLE II

COAL QUALITY FOR COUNTIES IN WHICH MEASURED, INDICATED AND INFERRED RESOURCES PER SEAM LISTED EXCEED 60 MILLION SHORT TONS

(Seam Thickness = 14 in to 28 in)

State	Seam	County	Coal Quality	
			S	CV
			%	Btu/lb
ALABAMA	American	Jefferson, Walker	1.2-1.8	12,700-14,740
	Black Creek	Jefferson, Walker	0.8-1.4	14,030-14,510
	Cobb	Tuscaloosa	1.2	13,300
	Fire Clay	Jefferson	2.0	13,560
	Jefferson	Jefferson, Walker	2.7	14,060
	Johnson	Tuscaloosa	1.2	14,320
	Mary Lee	Fayette, Jefferson, Tuscaloosa, Walker	0.7-1.0	12,570-13,400
	Pratt	Fayette, Jefferson, Tuscaloosa, Walker	1.8-2.2	13,260-13,940
KENTUCKY	Amburgy	Knott, Leslie, Letcher, Perry	3.4	13,150
	Auxier	Pike	0.8	12,530
	Bingham	Pike	1.4	14,140
	Blue Gem	Knox, Whitley	1.2-1.3	13,840-13,980
	Collier	Harlan		
	Elkhorn Leader	Letcher		
	Fire Clay	Breathitt, Clay, Knott, Lawrence, Leslie, Magoffin, Perry	0.8-1.0	12,763-14,180
	Fire Clay Rider	Breathitt, Leslie		
	Gun Creek	Magoffin		
	Haddix	Breathitt	1.3	13,680
	Harlan	Harlan	0.9	14,120
	Hazard	Breathitt, Leslie, Perry	0.7-1.6	12,190-14,050
	Hazard 7	Breathitt	0.8	13,220
	Jellico	Knox		
	Lily	Clay, Whitley	1.4-3.6	13,750-13,900
	L. Whitesburg	Breathitt		
	Moss	Knox		
	Peach Orchard	Lawrence	0.8	12,763
	Tom Cooper	Magoffin, Morgan		
	U. Elkhorn 1	Floyd, Knott, Pike	0.7-0.9	13,880
	U. Elkhorn 2	Floyd, Knott, Letcher, Pike	0.8-1.0	14,050-14,100
	U. Elkhorn 3	Breathitt, Floyd, Johnson, Knott, Pike	0.7-1.1	13,880-14,560
MISSOURI	Bevier	Buchanan, Charlton, Howard, Lafayette, Macon	4.7-5.0	10,930-12,070
	Bevier Wheeler	Buchanan, Harrison, Johnson, Linn, Putnam, Sullivan	5.6	12,560
	Croweburg	Adair, Buchanan, Charlton, Randolph, Vernon, Henry, Howard, Johnson, Linn, Macon	4.3-5.6	10,870-12,640
	Fleming	Henry		
	Lexington	Ray	5.1	12,040
	Mineral	Bates, Vernon	3.2-6.8	12,480-13,080
	Mulberry	Bates	2.9-5.1	10,640-12,460
	Mulky	Johnson, Lafayette, Macon, Randolph	3.8	11,570
	Rowe	Henry, Johnson, Lafayette, Vernon		
	Summit	Randolph		
	Tebo	Bates, Buchanan, Henry, Johnson	2.7-4.7	11,860-12,860
	Weir-Pittsburgh	Macon		

continued

TABLE II (ii)

(continued)

State	Seam	County	Coal Quality	
			S	CV
			%	Btu/lb
OHIO	Clarion	Vinton	3.6	12,940
	Fishpot	Belmont		
	L. Freeport	Carroll, Columbiana, Jefferson	1.9-3.0	13,450-13,530
	L. Kittanning	Athens, Carroll, Columbiana, Lawrence, Meigs, Morgan, Muskingham, Tuscarawas, Vinton, Gallia, Guernsey, Harrison, Jefferson	2.6-7.3	11,630-13,470
	Meigs Creek	Belmont	3.1	12,710
	Meigs Creek L.	Monroe, Washington	5.8	11,770
	Meigs Creek U.	Washington	5.8	11,770
	M. Kittanning	Athens, Carroll, Columbiana, Gallia, Tuscarawas, Guernsey, Harrison, Lawrence, Muskingham	1.6-4.9	12,490-13,490
	Pittsburgh	Washington		
	Restone	Meigs	3.2	12,500
	Uniontown	Belmont		
	U. Freeport	Carroll, Columbiana, Lawrence	2.3-4.7	12,410-13,210
	Waynesburg	Belmont		
PENNSYLVANIA	Brookville	Clearfield	2.5-2.7	12,926-12,960
	Clarion	Butler	3.8	12,110
	Homewood	Lawrence		
	L. Clarion	Clarion	4.0	12,850
	L. Freeport	Armstrong, Beaver, Butler, Cambria, Clearfield, Indiana, Jefferson, Somerset	1.3-3.0	12,410-14,160
	L. Kittanning	Beaver, Butler, Clearfield, Indiana, Lawrence, Somerset	1.8-2.9	12,830-13,850
	M. Kittanning	Beaver, Butler, Cambria, Clearfield, Lawrence, Somerset	0.9-3.7	13,040-13,790
	U. Clarion	Clarion		
	U. Freeport	Armstrong, Clearfield, Indiana, Somerset	1.3-2.4	13,067-14,020
	U. Kittanning	Cambria, Somerset	1.1-2.1	13,800-14,320
VIRGINIA	Blair	Wise	0.9	14,400
	Dorchester	Buchanan, Wise	1.5	13,680
	Hagy	Buchanan	0.8	14,460
	Jawbone	Buchanan, Dickenson	0.7	13,778
	Kennedy	Buchanan, Dickenson	0.8	14,790
	L. Banner	Buchanan, Dickenson	0.8-0.9	14,110-14,370
	Lyons	Wise	1.6	14,540
	Norton	Wise	1.1	14,010
	Raven	Buchanan, Dickenson	0.6	14,780
	Splash Dam	Buchanan, Dickenson	0.7-0.8	13,780-14,370
	U. Banner	Dickenson	0.7	14,310
WEST VIRGINIA	Alma	Logan, Mingo	0.6-1.6	13,800-14,370
	Beckley	Wyoming	1.0	14,730
	Buffalo	Logan		
	Campbell Creek 2	Logan	0.7	14,290
	Cedar Grove	Logan, Mingo	1.2	13,910-13,970
	Chilton	Logan	0.6	13,870
	Chilton A	Logan		
	Coalburg	Logan		
	Hernshaw	Logan		
	Pocahontas 3	McDowell	0.6	14,760
	Redstone	Monongalia	2.0	13,120
	Sewell	McDowell, Wyoming	0.5-0.6	14,370-14,700
	Welch	McDowell	0.6	14,450
	Williamson	Logan, Mingo		
	Winifrede	Logan	1.4	13,420

Source: Data abstracted from Ketron, 1978 Report.

TABLE III

COAL QUALITY FOR COUNTIES IN WHICH MEASURED, INDICATED AND INFERRED RESOURCES PER SEAM LISTED EXCEED 60 MILLION SHORT TONS

(Seam Thickness = 28 in to 42 in)

State	Seam	County	Coal Quality	
			S	CV
			%	Btu/lb
ALABAMA	American	Walker	1.8	12,700
	Black Creek	Walker	1.4	14,030
	Mary Lee	Fayette, Jefferson, Tuscaloosa, Walker	0.7-1.0	12,570-13,400
	Pratt	Fayette, Jefferson, Tuscaloosa	1.8-2.2	13,260-13,940
KENTUCKY	Amburgy	Knott, Leslie, Letcher, Perry	3.4	13,150
	Barren Fork	McCreary	4.0	13,120
	Bingham	Pike	1.4	14,140
	Collier	Harlan		
	Cornet and Hazard	Harlan	0.7	13,980
	D.	Harlan		
	Darby	Harlan	1.0	14,000
	Fire Clay	Floyd, Knott, Leslie, Letcher, Martin, Perry	0.7-0.9	13,800-14,240
	Fire Clay Rider	Brackitt, Leslie		
	Haddix	Martin		
	Harlan	Harlan	0.9	14,120
	Hazard 7	Knott		
	Imboden	Harlan		
	Lily	Clay	1.4	13,900
	Peach Orchard	Martin		
	U. Elkhorn 1	Floyd	0.9	13,880
	U. Elkhorn 2	Floyd, Knott	1.0	14,100
	U. Elkhorn 3	Floyd, Johnson, Knott, Letcher	0.7-1.4	13,910-14,560
MISSOURI	Bevier	Adair, Howard, Macon	4.7-5.1	10,930-12,070
	Bevier Wheeler	Harrison, Sullivan		
	Croweburg	Charlton, Linn		
	Lexington	Putnam, Ray, Sullivan	3.1-5.1	10,200-12,040
	Mineral	Bates, Vernon	3.2-6.8	12,480-13,080
	Mulberry	Bates	2.9-5.1	10,640-12,460
	Rowe	Henry, Harrison		
	Tebo	Henry	2.7-3.9	11,860-12,860
	Weir-Pittsburgh	Charlton, Linn, Macon		

continued

TABLE III (ii)
(continued)

State	Seam	County	Coal Quality	
			S	CV
			%	Btu/lb
OHIO	Clarion	Vinton	3.6	12,940
	Fishpot	Belmont		
	L. Freeport	Jefferson	3.0	13,530
	L. Kittanning	Carroll, Columbiana, Guernsey, Jefferson, Lawrence, Tuscaloosa, Belmont	2.6-7.3	12,480-13,470
	Meigs Creek	Belmont	3.1	12,710
	Meigs Creek L	Morgan, Washington	5.6-5.8	11,770-12,100
	M. Kittanning	Athens, Carroll, Columbiana, Gallia, Guernsey, Harrison, Lawrence, Morgan, Muskingham, Tuscarawas	1.6-4.9	12,490-13,180
	Pittsburgh	Monroe		
	Restone	Meigs	3.2	12,500
	Uniontown	Belmont		
	U. Freeport	Carroll, Columbiana Guernsey, Harrison, Jefferson	2.3-3.0	12,560-13,210
	Washington	Belmont		
	Waynesburg	Belmont		
PENNSYLVANIA	L. Clarion	Clarion	4.0	12,850
	L. Freeport	Armstrong, Beaver, Butler, Cambria, Clearfield, Indiana, Somerset	1.3-3.0	12,410-14,160
	L. Kittanning	Armstrong, Butler, Cambria, Clarion, Clearfield, Indiana, Jefferson, Somerset	1.4-3.6	12,490-14,380
	M. Kittanning	Butler, Clearfield, Lawrence, Somerset	2.0-3.7	13,040-13,790
	U. Clarion	Clarion		
	U. Freeport	Beaver, Butler, Cambria, Indiana, Somerset	1.3-2.0	12,329-14,020
	U. Kittanning	Cambria, Clearfield Somerset	1.1-2.1	13,750-14,320
VIRGINIA	Eagle	Buchanan	1.0	14,340
	Hagy	Buchanan	0.8	14,460
	Imboden	Wise	0.7	13,500
	Jawbone	Buchanan, Dickenson, Wise	0.6-0.7	12,760-13,778
	Kennedy	Buchanan, Dickenson, Russell	0.8	14,790
	Raven	Dickenson		
	Splash Dam	Buchanan	0.8	14,370
	U. Banner	Dickenson	0.7	14,310
WEST VIRGINIA	Alma	Logan, Mingo	0.6-1.6	13,800-14,370
	Beckley	McDowell, Raleigh, Wyoming	0.7-1.0	14,110-14,730
	Campbell Creek 2	Logan, Mingo	0.7-1.0	13,890-14,290
	Cedar Grove	Logan, Mingo	1.2	13,910-13,970
	Chilton	Logan	0.6	13,870
	Hernshaw	Logan		
	L. Cedar Grove	Logan	0.6	14,520
	Pocahontas 3	Raleigh, Wyoming	0.6-0.7	14,510-14,610
	Pocahontas 4	Wyoming	0.8-0.9	14,490-14,790
	Sewell	Wyoming	0.5	14,370
	U. Freeport	Monongalia	1.8	13,620
	Waynesburg	Marion, Marshall		
	Williamson	Logan, Mingo		
	Winifrede	Logan	1.4	13,420

Source: Data abstracted from Ketron, 1978 Report

TABLE IV

US AND UK THIN SEAM ACCIDENT TRENDS

US Code	US Description	UK Description	Fatal and Serious Accidents			All Accidents		
			US Disabling	British Reportable		US None Disabling	British	
				Thin Seam Sample	All Coal Mines		Thin Seam Sample	All Coal Mines
01	Falls	Falls of roof at face.	L > M > H	0.27	0.23	H > M	16.0	14.0
02	Pressure bumps					H > M,L		
03	Falls of face	Falls of roof elsewhere underground.		0.04	0.04		2.4	3.6
05	Sliding of falling materials/ objects	Falling objects.	L,M > H	0.04	0.09	H > M > L	9.7	12.9
11	Haulage	Haulage and transport at face	L > M > H	0.04	0.41	H M,L	1.5	0.7
		Haulage and transport elsewhere underground		0.23			6.6	6.2
				0.27	0.41			
15	Machinery	Machinery at face.	L > M > H	0.11	0.13	H,L > M	2.8	2.7
		Machinery elsewhere underground		0.00			0.8	1.1
08	Hand tools	Use of tools and appliances	L > M > H	0.00	0.04	H,L > M	8.9	7.2
07	Handling material	Handling supplies	L > M > H	0.04	0.04	H > M,L	22.2	23.3
06	Slips and falls	Stumbling, falling, slipping.	M > L,H	0.27	0.16	H > M > L	32.4	29.8
18	Miscellaneous	Other causes	L,M > H	0.00	0.06	H > M	15.3	12.4
Total			L > M > H	1.04	1.20	H > M,L	118.6	113.9

Source: US data abstracted from "Comparison of Injury Hazards in Different Coal Seams Heights" (114), British statistics courtesy of the NCB

Notes:

1. All accident frequency rates given in accidents per 100,000 shifts.
2. L, M and H refer to accident frequency rates for low, medium and high seams respectively.
3. H > M,L means that both M and L are significantly less than H. However, L and M are not significantly different from each other.

TABLE V

WORLD-WIDE THIN SEAM COAL DEPOSITS

Geological Parameters	USSR	USA	Spain
Definition of thin seams	1.2 m (48 in)	-	-
Seam dip	Gentle to very steep	Mainly flat	0 - 90°
Seam depth	300 m - 1,100 m (984 ft - 3,609 ft)	Reserves calculated to a depth of 1,000 ft	500 m (1,640 ft)
Coal strength	Variable	Variable	-
Roof	6.4% sandstone 8.0% limestone rest shale	Generally good and strong. Frequent draw slate	Strong, variable
Floor	Mostly clay shales	Medium	Strong, variable
Extent of seams	Donetz and Lvov-Volynsky	Wide areas but thickness in seams varies over area	Fragmented
Water	Mostly dry, but some very wet	Fairly dry when worked above drainage table	Variable
Faults	Normally undisturbed	Normally undisturbed	Highly disturbed
Cleat	Mostly well defined	Not generally well defined	-
Spontaneous combustion	Variable risk	Variable risk	-
Methane	Variable emission	Generally low emission	Low emission
Quality	Coking coal	Often coking and low sulphur	-

continued

TABLE V (ii)
(continued)

Geological Parameters	United Kingdom	Czechoslovakia	Poland
Definition of thin seams	0.91 m (36 in)	1.0 m (39 in)	1.0 m (39 in)
Seam dip	0 - 45° Mostly 0 - 6°	0 - 16° 51% 16° - 36° 34% + 36° 15%	0 - 10° 39% 10° - 45° 54% + 45° 7%
Seam depth	1,100 m (3,609 ft)	400 m - 600 m (1,312 ft - 1,968 ft) Some at 1,000 m (3,281 ft)	0 - 800 m (0 - 2,625 ft)
Coal strength	Hard	Hard	-
Roof	Shale	-	Sandstone, silts and conglomerates
Floor	Mostly clays	-	Sandstone, silts and conglomerates
Extent of seams	Northumberland, Durham Yorkshire and Derbyshire	Ostrava Karvina and Eastern Bohemia	-
Water	Mostly dry	-	-
Faults	Mainly undisturbed except Wales and Scotland	Highly disturbed	-
Cleat	Mostly well defined	-	Mostly well defined
Spontaneous combustion	Low risk in thin seams	-	-
Methane	Mainly gassy	All gassy	Mainly gassy
Quality	Often coking	High quality coking coal	-

continued

TABLE V (iii)
(continued)

Geological Parameters	Colombia	France	Belgium
Definition of thin seams	-	1.0 m (39 in)	0.6 m (24 in)
Seam dip	Flat to steep	$0^\circ - 20^\circ$ 47% $20^\circ - 45^\circ$ 46% $+ 45^\circ$ 7%	$0 - 45^\circ$ mostly $0 - 30^\circ$
Seam depth	-	-	275 m - 1,160 m (902 ft - 3,806 ft)
Coal strength	-	-	Variable
Roof	Variable	-	Competent shale and sandstone
Floor	-	-	Good shale and sandstone
Extent of seams	-	Nord, Pas de Calais, Cevennes	Charleroi-Namur, Liege
Water	-	-	-
Faults	Disturbed	-	Undisturbed
Cleat	Variable	Mostly well defined	Variable
Spontaneous combustion	Low risk	-	-
Methane	Mainly non-gassy	Mainly gassy	-
Quality	-	-	Anthracite

continued

TABLE V (iv)
(continued)

Geological Parameters	Germany	China	Bulgaria	Romania
Definition of thin seams	0.7 m (28 in)	-	1.3 m (51 in)	-
Seam dip	$0 - 10^{o}$ 63% $10^{o} - 20^{o}$ 9.5% $+ 20^{o}$ 27.5%	flat or slight 69.7% $10^{o} - 25^{o}$ 22.4% $+ 25^{o}$ 7.9%	$10^{o} - 90^{o}$ mostly -45^{o}	$5^{o} \div 70^{o}$
Seam depth	Maximum 1,200 m (3,937 ft)	Mostly less than 200 m (656 ft)	150 m - 300 m (492 ft - 984 ft)	-
Coal strength	Soft but hard in the Saar	-	-	-
Roof	Shale, sandy shale in thin seams	-	Hard sandy shales	-
Floor	Shales, sandy shales	-	-	-
Extent of seams	Aachen, and lower Saxony	Widely distributed	Svoge Basin and Balkan field	Valea - Jiului and Anina
Water	Mostly dry	-	-	Dry
Faults	Mainly undisturbed	Undisturbed	Highly disturbed	Highly disturbed
Cleat	Mostly well defined	-	-	Not generally well defined
Spontaneous combustion	Variable risk	-	Low risk	-
Methane	Low emission	Some gassy	Some gassy	Mainly gassy
Quality	Coking coal	-	Anthracite	Coking coal

TABLE VI

HAULAGE UNITS IN USE IN UNDERGROUND MINES

Units	1971	1972	1973	1974	1975
Locomotives:					
Trolley	3,068	3,107	2,937	2,850	2,966
Battery	351	305	339	245	461
All other	179	-	-	-	-
Total	3,598	3,412	3,276	3,095	3,427
Tractors, rubber-tyred	1,853	1,937	1,980	2,265	2,388
Trailers, rubber-tyred	2,095	2,522	2,396	1,714	1,708
Mine cars	55,836	51,174	46,790	46,800	43,921
Shuttle cars:					
Cable reel	6,175	5,797	5,507	6,241	6,372
Battery	401	570	666	650	698
Total	6,576	6,367	6,173	6,891	7,070
Shuttle buggies	254	256	329	386	228
Gathering and haulage conveyors	3,437	3,776	3,902	4,892	5,187

TABLE VII

DESIGN COMBINATION

Mining System	Hand Got and Blasting Off Solid		Cut and Blast		Manual Control Machine Mined		Remote Control Machine Mined		Total Remote Extraction		Hydraulic	
	No	Code	No	Code	No	Code	No	Code	No	Code	No	Code
Room and pillar	1	L	2	C	3	C	4	C	5	P	6	P
Room and pillar with pillar extraction	7	L	8	C	9	C	10	C	11	U	12	P
Longwall	13	C	14	L	15	C	16	L	17	L	18	U
Shortwall	19	P	20	P	21	C	22	P	23	U	24	P
Augering	25	U	26	U	27	U	28	U	29	C	30	P

Note:

Each design combination has been allocated a number and a code letter: C, L, P or U.

C - commonly used
L - limited or historical
P - possible with future technology
U - unlikely or impossible

TABLE VIII

COMPARISONS OF DESIGN COMBINATIONS

Design Combination	Safety	Minimum Height (in)	Normal Working Gradient (degrees)	Maximum Depth (ft)	Weak Roof	Weak Floor
1 Room and pillar, hand got and blasted off solid	Low	18	0 - 25	Normal maximum 1,000	Intermediate	Negligible
2 Room and pillar, cut and blast	Medium	20	0 - 25			Difficult Tramming
3 Room and pillar, manned machine	Medium	28	0 - 10			
4 Room and pillar, remote control	Good	28	0 - 10			
5 Pillar extraction, hand or solid blasted	Low	36	0 - 25	Normal maximum 1,000	Severe	Excessive Convergence and Difficult Tramming
6 Pillar extraction, cut and blast	Low/ Medium	36	0 - 25			
7 Pillar extraction, manned machine	Medium	32	0 - 10			
8 Pillar extraction, remote control	Medium/ Good	32	-			
9 Longwall, hand got and blasted off solid	Low	16	0 - 70	Presently up to nearly 4,000	Severe	Negligible
10 Longwall, cut and blast	Low	16	0 - 50			Excessive Convergence
11 Longwall, manned machine	Very Good	24	0 - 50		Negligible	
12 Longwall, remote control	Very Good	30	-			
13 Longwall, total remote extraction	Very Good	12	0 - 90		Severe	
14 Shortwall, manned machine	Medium	40	0 - 10	Normal maximum 1,500	Severe	Excessive Convergence
15 Shortwall, remote control	Good	40	-			
16 Augering, remote extraction	Very Good	12	0 - 90	Presently up to 2000	Negligible	Negligible
17 Augering and hydraulic		12	-			

TABLE VIII (ii)
(continued)

Design Combination	Methane	Spontaneous Combustion Risk	Extraction Ratio	Productivity	Capital Cost	Operating Cost
1 Room and pillar, hand got and blasted off solid	Severe	Medium	Poor	Low	Low	High
2 Room and pillar, cut and blast				Fair	Intermediate	Intermediate
3 Room and pillar, manned machine				Good	Intermediate	Low
4 Room and pillar, remote control				Good	Intermediate	Low
5 Pillar extraction, hand or solid blasted	Intermediate	Severe	Intermediate	Low	Low	High
6 Pillar extraction, cut and blast				Low to Fair	Low to Intermediate	High
7 Pillar extraction, manned machine				Fair to Good	Intermediate	Intermediate
8 Pillar extraction, remote control				Fair to Good	Intermediate	Intermediate
9 Longwall, hand got and blasted off solid	Negligible when drainage practised	Normally low (but high with bleeders)	Good	Low	Low	High
10 Longwall, cut and blast				Low	Low	High
11 Longwall, manned machine				Good	High	Intermediate
12 Longwall, remote control				Good	High	Low
13 Longwall, total remote extraction				Low	Low to Intermediate	Intermediate
14 Shortwall, manned machine	Intermediate	Medium	Intermediate	Good	High	Intermediate
15 Shortwall, remote control				Good	High	Intermediate
16 Augering, remote extraction	Negligible	Intermediate	Poor	Fair to Good	Intermediate	Low
17 Augering and hydraulic					Intermediate	Low

TABLE IX

CAPITAL COSTS - LONGWALL

Item	Z-System Advance	Single-Entry Retreat	Multiple-Entry Retreat
Number of longwalls	1	1	1
Number of development sections	1/3	1	1
Annual output including development (tons)	361,140	347,966	357,684
Length of wall (ft)	960	946	831
Longwall:	$	$	$
Buttock shearer	580,000	580,000	580,000
Armoured face conveyor	884,300	872,500	776,700
Stage loader	77,000	77,000	77,000
Power centre	22,000	22,000	22,000
Switchgear	100,000	100,000	100,000
Powered supports	4,010,160	3,948,780	3,478,200
Hydraulic pump	55,000	55,000	55,000
Signals	57,000	57,000	55,000
Man trip	25,000	25,000	25,000
Continuous miners	794,000	-	-
Pump pack equipment	308,000	308,000	-
Spares	1,336,000	936,200	778,200
Sub-totals	8,248,460	6,981,480	5,947,100
Development:			
Continuous miner	397,000	397,000	397,000
Two shuttle cars	178,000	178,000	178,000
Ratio feeder	67,000	67,000	67,000
Roof bolter	70,000	70,000	70,000
Power centre	22,000	22,000	22,000
Rock duster	4,000	4,000	4,000
Man trip	25,000	25,000	25,000
Utility scoop	68,000	68,000	68,000
Spares	277,000	277,000	277,000
Sub-totals	1,108,000	1,108,000	1,108,000
Chargeable	369,333	1,108,000	1,108,000
Conveyors, cables, pipes and track	2,060,000	3,090,000	2,060,000
Totals	10,677,793	11,179,480	9,115,100

TABLE X

OPERATING COSTS - LONGWALL

Item	Z-System Advance		Single-Entry Retreat		Multiple-Entry Retreat	
	Cost ($)	$/Ton	Cost ($)	$/Ton	Cost ($)	$/Ton
Capital recovery charge 8 years, 20% interest.	2,782,733	7.71	2,913,478	8.37	2,375,481	6.64
Materials and power:						
Power	72,660	0.20	71,039	0.20	71,630	0.20
Equipment parts	254,309	0.70	248,636	0.72	250,703	0.70
Lubricants and hydraulic fluid	138,000	0.38	134,921	0.39	136,043	0.38
Brattice	3,633	0.01	3,552	0.01	3,581	0.01
Roof bolts and steel beams	229,423	0.64	157,028	0.45	41,243	0.11
Stoppings	1,400	-	2,000	-	2,700	0.01
Cutting bits	65,394	0.18	63,935	0.18	64,467	0.18
Packing material	260,500	0.72	282,900	0.82	-	-
Rock dust	3,633	0.01	3,552	0.01	7,163	0.02
Sub-totals	1,028,952	2.84	967,563	2.78	577,530	1.61
Labour:						
Number of persons	56		57		54	
Payroll costs	1,932,000	5.35	1,966,500	5.65	1,863,000	5.21
Totals	5,743,685	15.90	5,847,541	16.80	4,816,011	13.46

TABLE XI

CAPITAL COSTS - ROOM AND PILLAR

Item	9-Entry Shuttle Cars	5-Entry Continuous Haulage	9-Entry Retreat
Number of sections	3	2	3
Annual output (tons)	347,760	350,520	307,740
	$	$	$
Continuous miner	397,000	397,000	397,000
Shuttle cars	178,000	-	178,000
Ratio feeder	67,000	-	67,000
Continuous haulage	-	295,000	-
Roof bolter	70,000	95,000	70,000
Power centre	22,000	22,000	22,000
Rock duster	4,000	4,000	4,000
Man trip	25,000	25,000	25,000
Utility scoop	68,000	68,000	68,000
Miscellaneous	15,000	15,000	15,000
Spares	282,000	307,000	282,000
Sub-totals	1,128,000	1,228,000	1,128,000
All sections	3,384,000	2,456,000	3,384,000
Conveyors, cables, pipes and track	3,090,000	2,575,000	3,090,000
Totals	6,474,000	5,031,000	6,474,000

TABLE XII

OPERATING COSTS - ROOM AND PILLAR

Item	9-Entry Shuttle Cars		5-Entry Continuous Haulage		9-Entry Retreat	
	Cost ($)	$/Ton	Cost ($)	$/Ton	Cost ($)	$/Ton
Capital recovery charge 8 years, 20% interest	1,687,185	4.85	1,311,126	3.74	1,687,185	5.48
Materials and power:						
Power	69,552	0.20	70,104	0.20	61,548	0.20
Equipment parts	330,372	0.95	332,994	0.95	292,353	0.95
Lubricants and hydraulic fluid	173,880	0.50	175,260	0.50	153,870	0.50
Brattice	10,432	0.03	10,516	0.03	6,155	0.02
Roof bolts	375,580	1.08	378,561	1.08	234,606	0.76
Timber	-	-	-	-	60,824	0.20
Stoppings	41,731	0.12	77,114	0.22	21,542	0.07
Cutting bits	62,591	0.18	63,094	0.18	55,393	0.18
Rock dust	52,164	0.15	52,578	0.15	36,929	0.12
Sub-totals	1,116,302	3.21	1,160,221	3.31	923,220	3.00
Labour:						
Number of persons	90		74		89	
Payroll costs	3,105,000	8.93	2,553,000	7.28	3,070,500	9.98
Totals	5,908,487	16.99	5,024,347	14.33	5,680,905	18.46

TABLE XIII

CAPITAL COSTS - SHORTWALL

Number of sections	3
Annual output (tons)	440,910
	$
Continuous miner	397,000
Continuous haulage	295,000
Roof bolter	95,000
Power centre	22,000
Rock duster	4,000
Man trip	25,000
Powered supports (38)	1,071,000
Hydraulic power pack	55,000
Spares	487,000
Sub-total	2,451,000
All sections	7,353,000
Conveyors, cables, pipes and track	2,575,000
Total	9,928,000

TABLE XIV

OPERATING COSTS - SHORTWALL

Item	Cost ($)	Ton/$
Capital recovery charge 8 years, 20% interest	2,587,330	5.87
Materials and power:		
Power	88,182	0.20
Equipment and parts	418,865	0.95
Lubricants and hydraulic fluid	220,455	0.50
Brattice	4,409	0.01
Roof bolts	174,600	0.40
Timber	4,409	0.01
Stoppings	61,727	0.14
Cutting bits	79,363	0.18
Rock dust	22,045	0.05
Sub-totals	1,074,055	2.44
Labour:		
Number of persons	90	
Payroll costs	3,105,000	7.04
Totals	6,766,385	15.35

TABLE XV

TOTAL MINING COSTS

System	Annual Output (Tons)	Capital Cost ($)	Capital Recovery ($/ton)	Materials and Power ($/ton)	Labour ($/ton)	Total ($/ton)
Longwall - Z-system Advance	361,140	10,677,793	7.71	2.84	5.35	15.90
Longwall - Single-entry Retreat	347,966	11,179,480	8.37	2.78	5.65	16.80
Longwall - Multiple-entry Retreat	357,684	9,115,100	6.64	1.61	5.21	13.46
Room and pillar - 9-entry Shuttle cars	347,760	6,474,000	4.85	3.21	8.93	16.99
Room and pillar 5-entry Continuous haulage	350,520	5,031,000	3.74	3.31	7.28	14.33
Room and pillar 9-entry Retreat	307,740	6,474,000	5.48	3.00	9.98	18.46
Shortwall	440,910	9,928,000	5.87	2.44	7.04	15.35

TABLE XVI

TOTAL MINING COSTS
BASED ON WORKING
21 SHIFTS PER WEEK

System	Annual Output (Tons)	Capital Cost ($)	Capital Recovery ($/ton)	Materials and Power ($/ton)	Labour ($/ton)	Total ($/ton)
Longwall - Z-system Advance	722,280	10,677,793	4.95	2.84	6.10	13.89
Longwall - Single-entry Retreat	695,932	11,179,480	5.37	2.78	6.44	14.59
Longwall - Multiple-entry Retreat	715,368	9,115,100	4.26	1.61	5.94	11.81
Room and pillar - 9-entry Shuttle cars	695,520	6,474,000	3.11	3.21	10.18	16.50
Room and pillar - 5-entry Continuous haulage	701,040	5,031,000	2.40	3.31	8.30	14.01
Room and pillar 9-entry Retreat	615,480	6,474,000	3.52	3.00	11.38	17.90
Shortwall	881,820	9,928,000	3.77	2.44	8.03	14.24

TABLE XVII

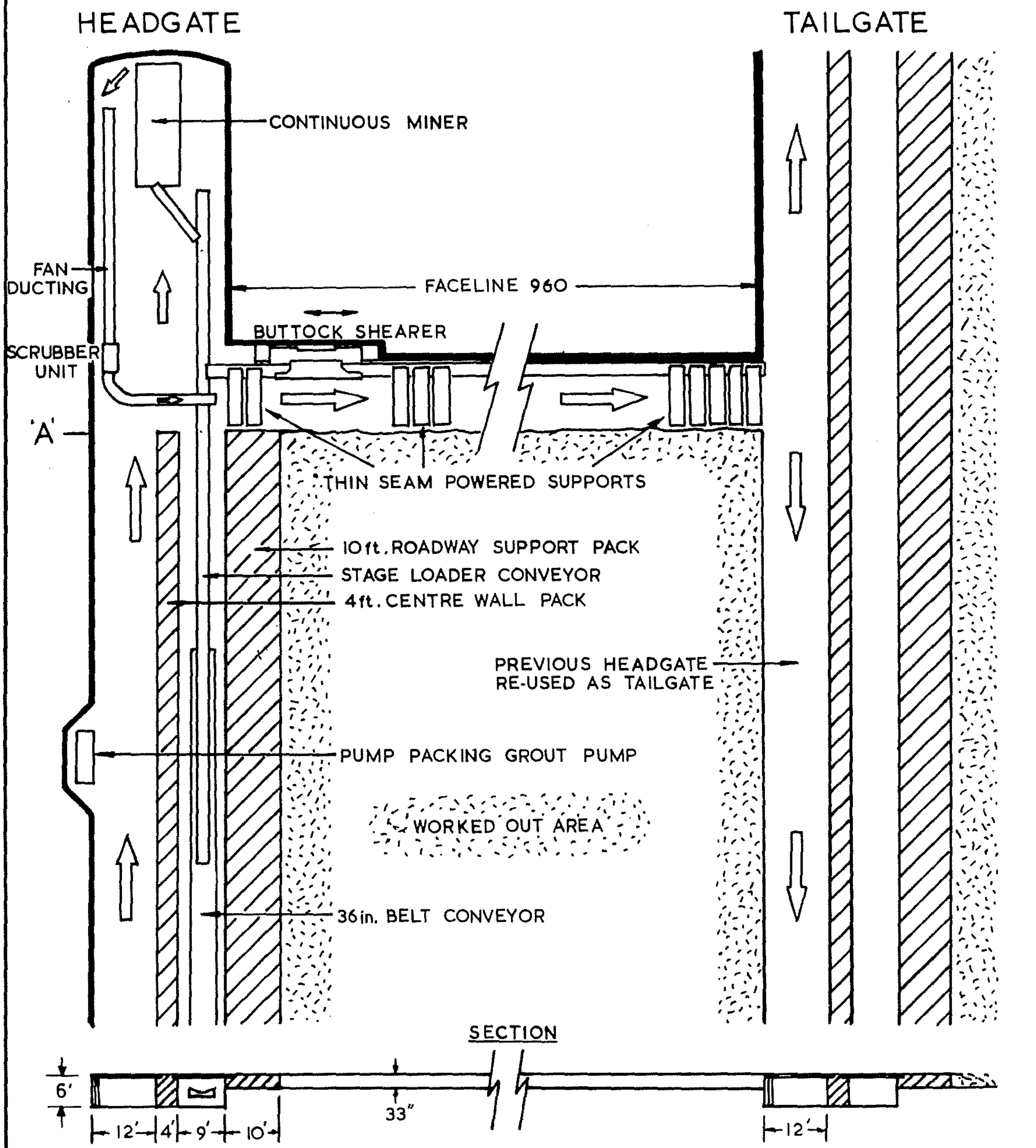
TOTAL MINING COSTS
BASED ON AN INTEREST
RATE OF 10%

System	Annual Output (Tons)	Capital Cost ($)	Capital Recovery ($/ton)	Materials and Power ($/ton)	Labour ($/ton)	Total ($/ton)
Longwall - Z-system Advance	361,140	10,677,793	5.54	2.84	5.35	13.73
Longwall - Single-entry Retreat	347,966	11,179,480	6.02	2.78	5.65	14.45
Longwall - Multiple-entry Retreat	357,684	9,115,100	4.78	1.61	5.21	11.60
Room and pillar - 9-entry Shuttle cars	347,760	6,474,000	3.49	3.21	8.93	15.63
Room and pillar 5-entry Continuous haulage	350,520	5,031,000	2.69	3.31	7.28	13.28
Room and pillar 9-entry Retreat	307,740	6,474,000	3.94	3.00	9.98	16.92
Shortwall	440,910	9,928,000	4.22	2.44	7.04	13.70

PLATE 1

DIAGRAM ILLUSTRATING ROADHEAD SYSTEM FOR
LONGWALL ADVANCE LAYOUT ('Z' SYSTEM)

0 5 10 20 30 40 50 feet



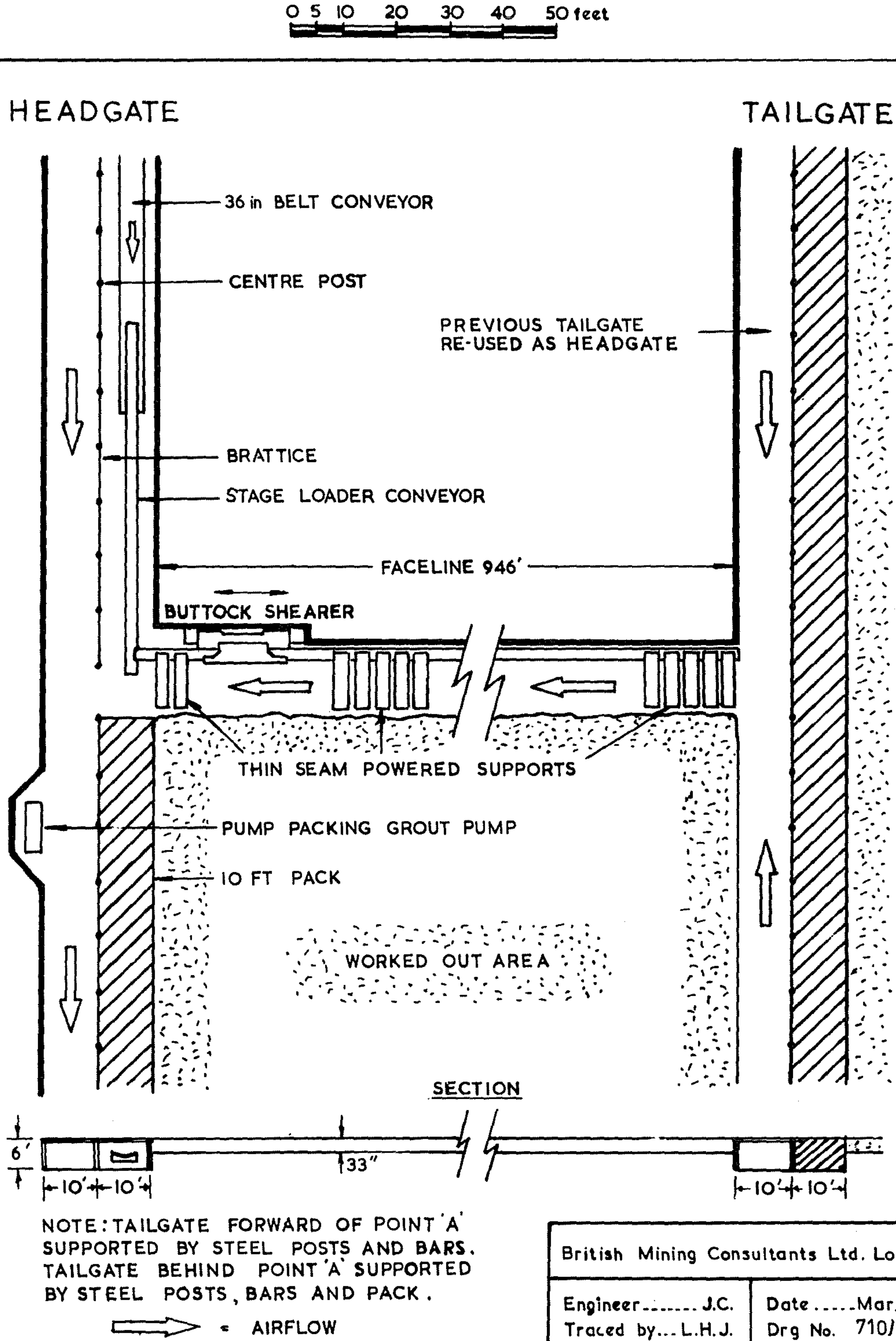
NOTE :- HEADGATE FORWARD OF POINT 'A' SUPPORTED BY STEEL POSTS AND BARS. HEADGATE BEHIND POINT 'A' SUPPORTED BY STEEL POSTS, BARS AND PACK SYSTEM.

⇨ = AIRFLOW

British Mining Consultants Ltd. London	
Engineer .... J.C.	Date .... Mar. 80
Traced by ..... L.H.J.	Drg No 710/1

PLATE 2

DIAGRAM ILLUSTRATING ROADHEAD SYSTEM FOR LONGWALL RETREAT SINGLE-ENTRY LAYOUT



NOTE: TAILGATE FORWARD OF POINT 'A' SUPPORTED BY STEEL POSTS AND BARS. TAILGATE BEHIND POINT 'A' SUPPORTED BY STEEL POSTS, BARS AND PACK.

British Mining Consultants Ltd. London	
Engineer....... J.C.	Date ......Mar. 80
Traced by... L.H.J.	Drg No. 710/2

PLATE 3

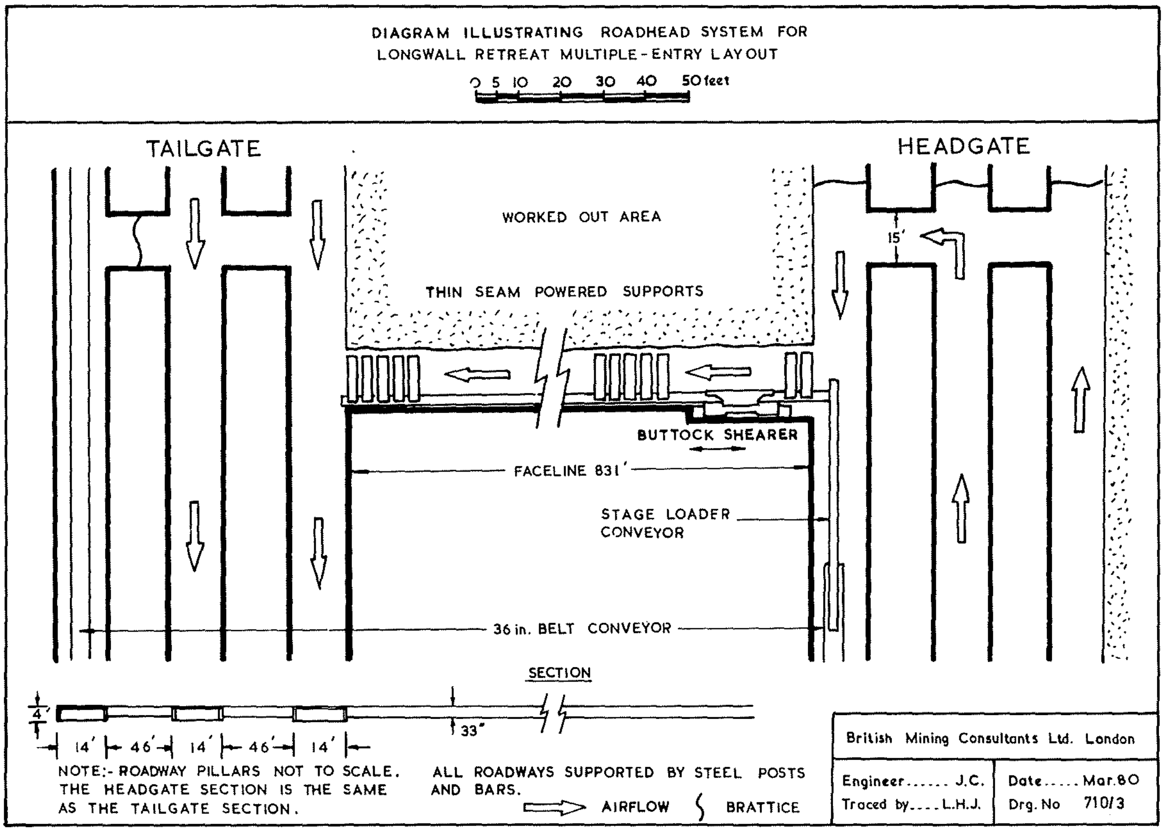
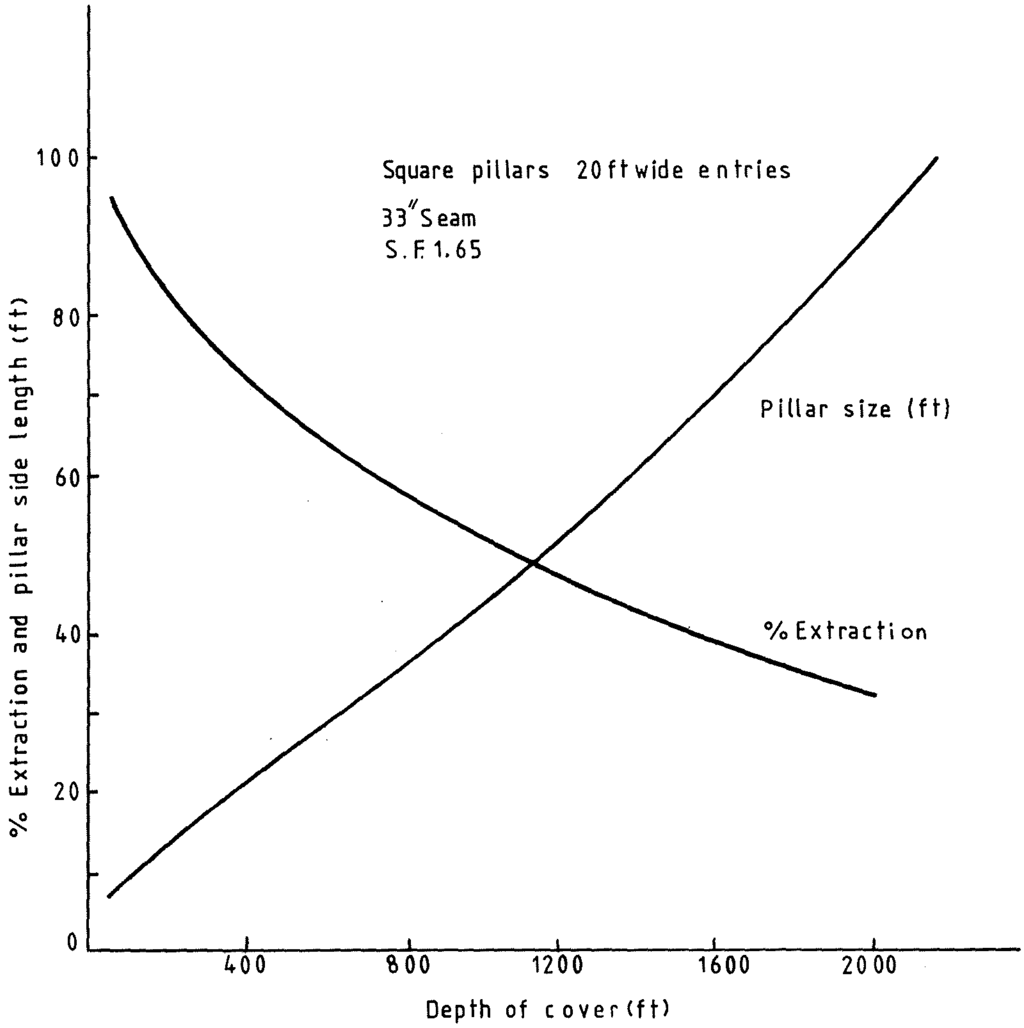


DIAGRAM ILLUSTRATING ROADHEAD SYSTEM FOR
LONGWALL RETREAT MULTIPLE-ENTRY LAYOUT
0 5 10 20 30 40 50 feet
TAILGATE
HEADGATE
WORKED OUT AREA
THIN SEAM POWERED SUPPORTS
15'
BUTTOCK SHEARER
FACELINE 831'
STAGE LOADER
CONVEYOR
36 in. BELT CONVEYOR
SECTION
4'
33"
14' 46' 14' 46' 14'
NOTE:- ROADWAY PILLARS NOT TO SCALE.
THE HEADGATE SECTION IS THE SAME
AS THE TAILGATE SECTION.
ALL ROADWAYS SUPPORTED BY STEEL POSTS
AND BARS.
AIRFLOW
BRATTICE
British Mining Consultants Ltd. London
Engineer...... J.C.
Traced by.... L.H.J.
Date..... Mar.80
Drg. No 710/3

PLATE 4

EFFECT OF DEPTH ON PILLAR
SIZES AND EXTRACTION

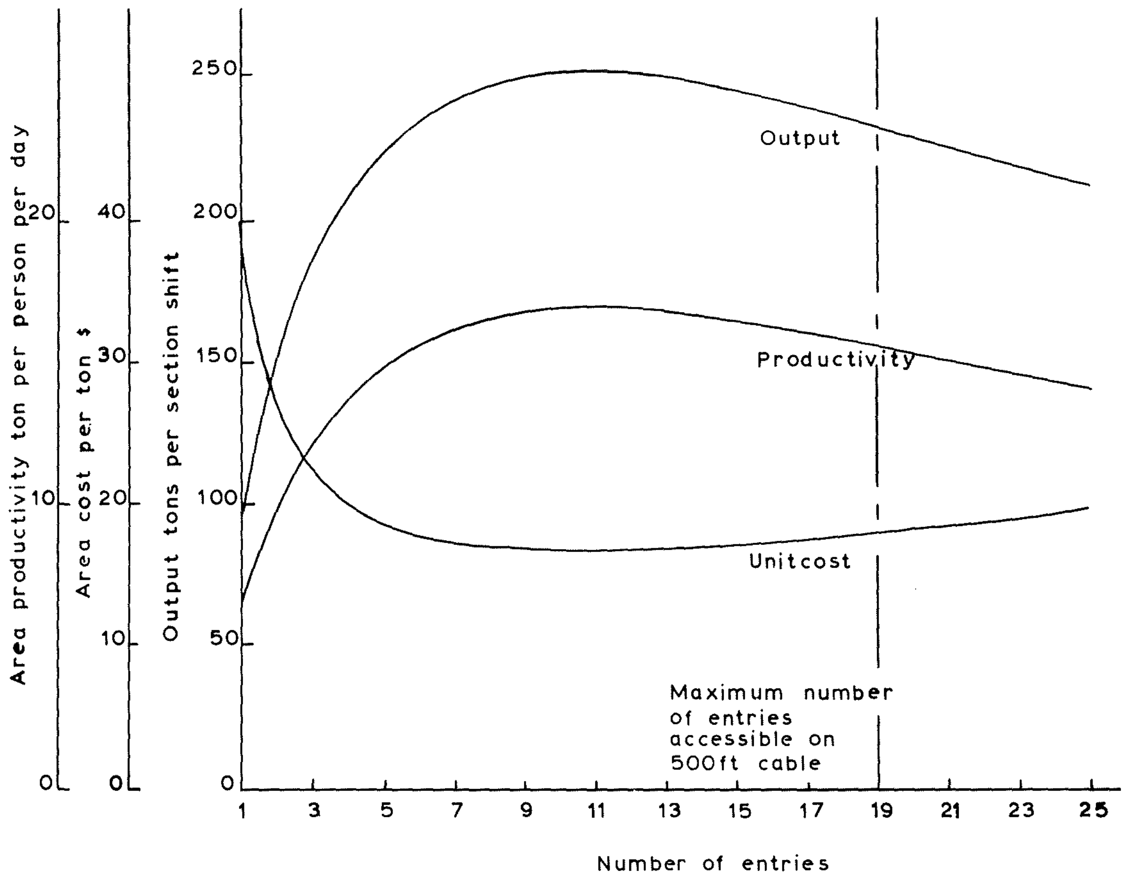


British Mining Consultants Ltd London	
Engineer...... J.H.C.	Date...... MAY 80
Traced by...... J.W.	Drg No......710/4

PLATE 5

ROOM AND PILLAR SHUTTLE CAR HAULAGE

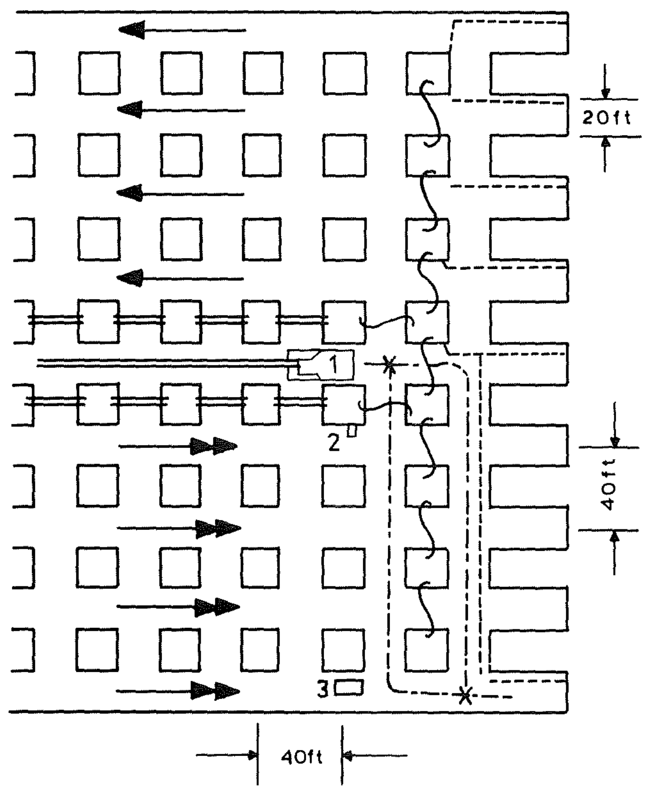
EFFECT OF VARIATION IN NUMBER OF ENTRIES



British Mining Consultants Ltd, London.	
Engineer........ J.H.C	Date...... May 80
Traced by...... N.M	Drg No...... 710/5

PLATE 6

ROOM AND PILLAR
GENERAL ARRANGEMENT - 9-ENTRY SECTION



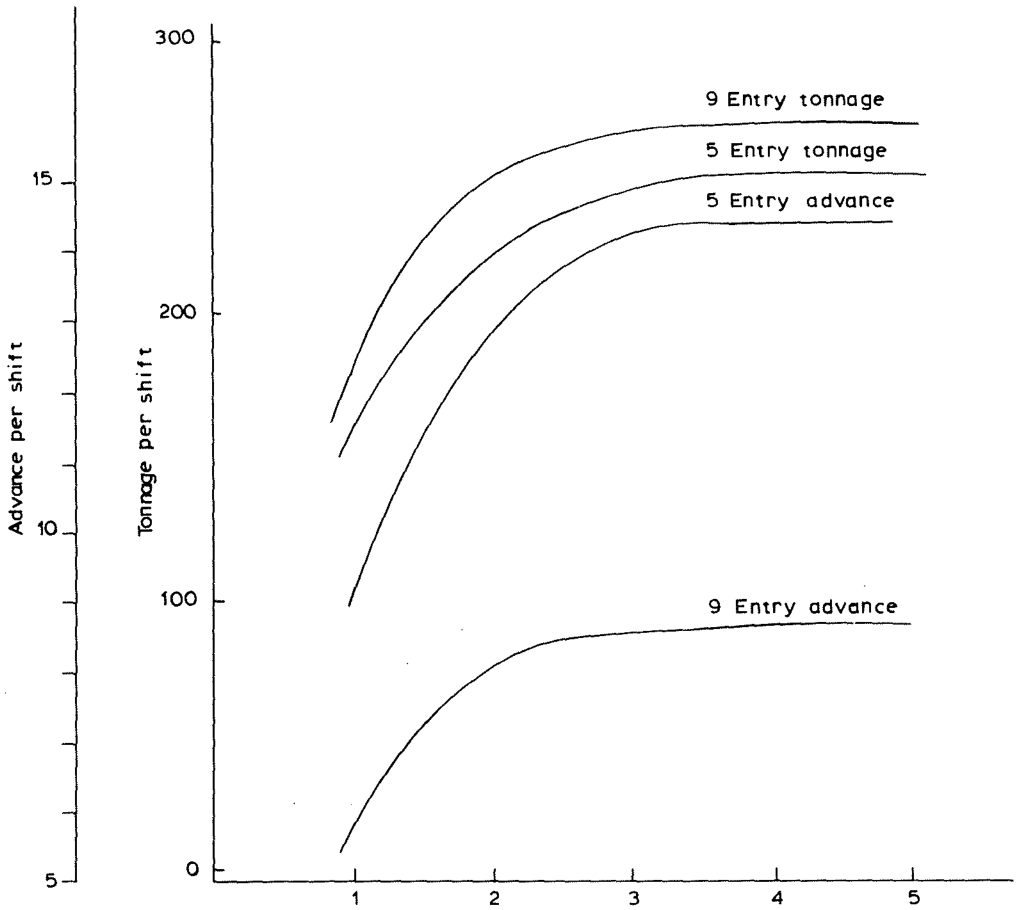
LEGEND

.........Ventilation stoppings
..........Entry brattices
...................Brattice curtains
...........Intake air
...........Return air
1.................Ratio feeder
2.................Power distribution box
3.................Power centre
..........Conveyor
...........Shuttlecar routes
X.............Shuttle car change points

British Mining Consultants Ltd London	
Engineer.........J.H.C.	Date........June 80
Traced by......J.M.E	Drg No......710/6

PLATE 7

ROOM AND PILLAR
SHUTTLE CAR HAULAGE
EFFECT OF VARIATION IN CONTINUOUS MINER CAPACITY

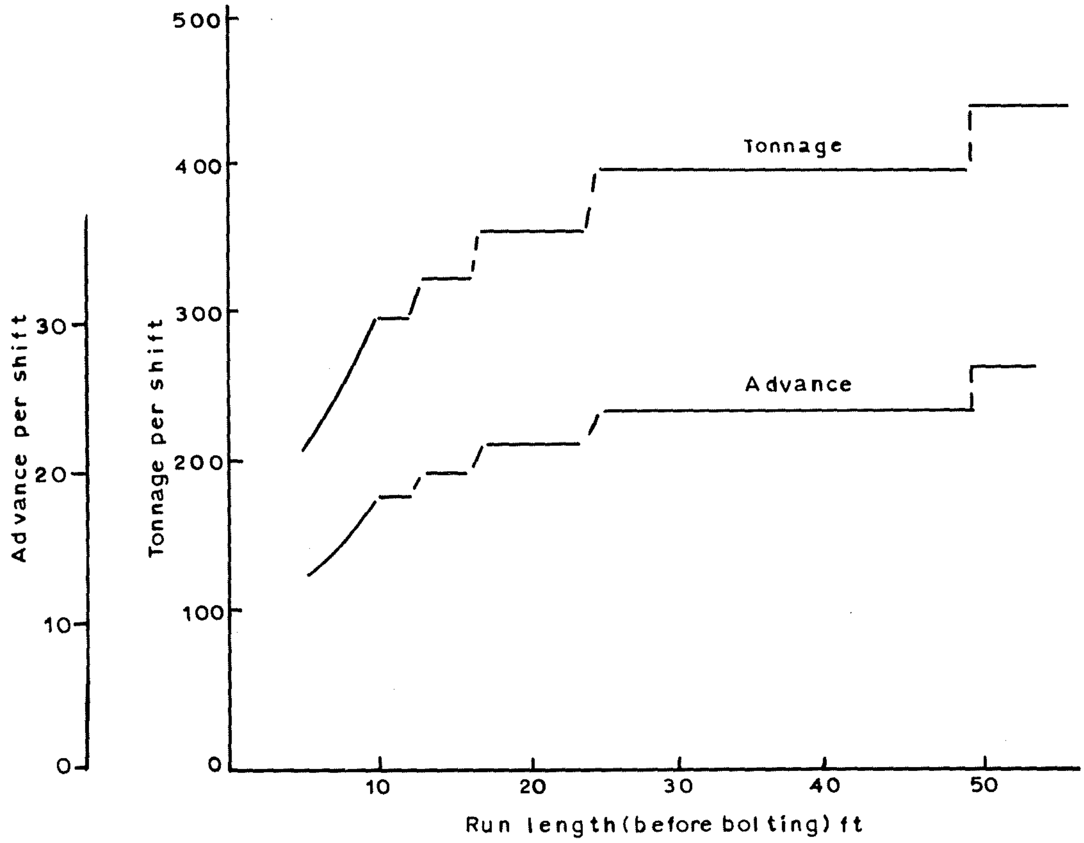


Continuous miner tramming speed	30ft/minute
Length of run before support	17ft
Shuttle car factor	3 tons/car
Shuttle car average tram speed	250ft/minute
Shuttle car discharge time	1 minute
Room width	20ft
Available shift time	300 minutes
Centre distance crosscut and entries	40ft
Seam height	33"
Services and conveyor belt extension	1shift/crosscut
Coal density	84.24lbs/ft^3

British Mining Consultants Ltd London	
Engineer............ J.H.C.	Date................May 80
Traced by...........J.M.E	Drg. No...........710/7

PLATE 8

ROOM AND PILLAR CONTINUOUS HAULAGE
EFFECT OF VARIATION IN RUN LENGTH



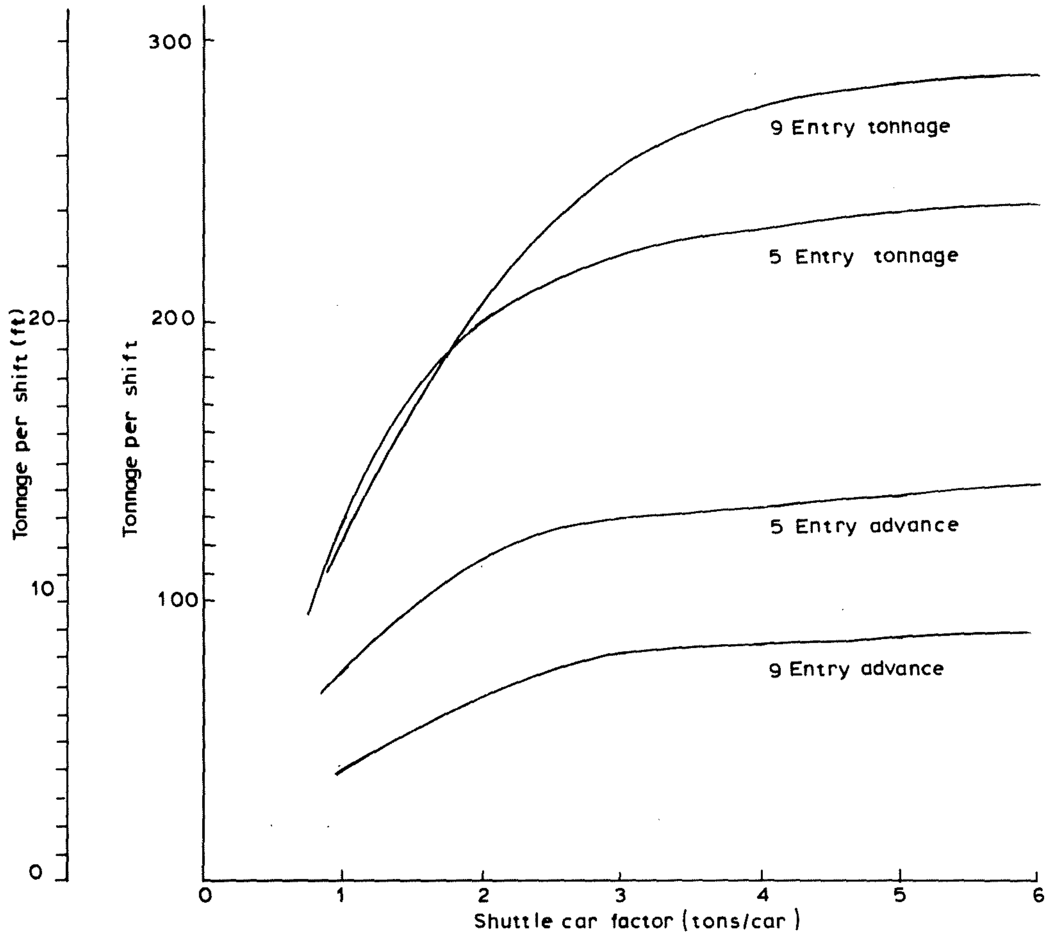
Continuous miner - average capacity	2 tons / minute
Miner-haulage transfer speed	10 ft / minute
Cross cut turn off angle	60°
Centres cross cut and entry	50 ft
Room width	20 ft
Seam height	33"
Coal density	84·24 lbs / ft^3
Available shift time	300 minutes

British Mining Consultants Ltd London

Engineer...... JHC	Date....... May 80
Traced by....... J. W.	Drg No....... 710/8

PLATE 9

ROOM AND PILLAR
SHUTTLE CAR HAULAGE
EFFECT OF VARIATION IN SHUTTLE CAR FACTOR



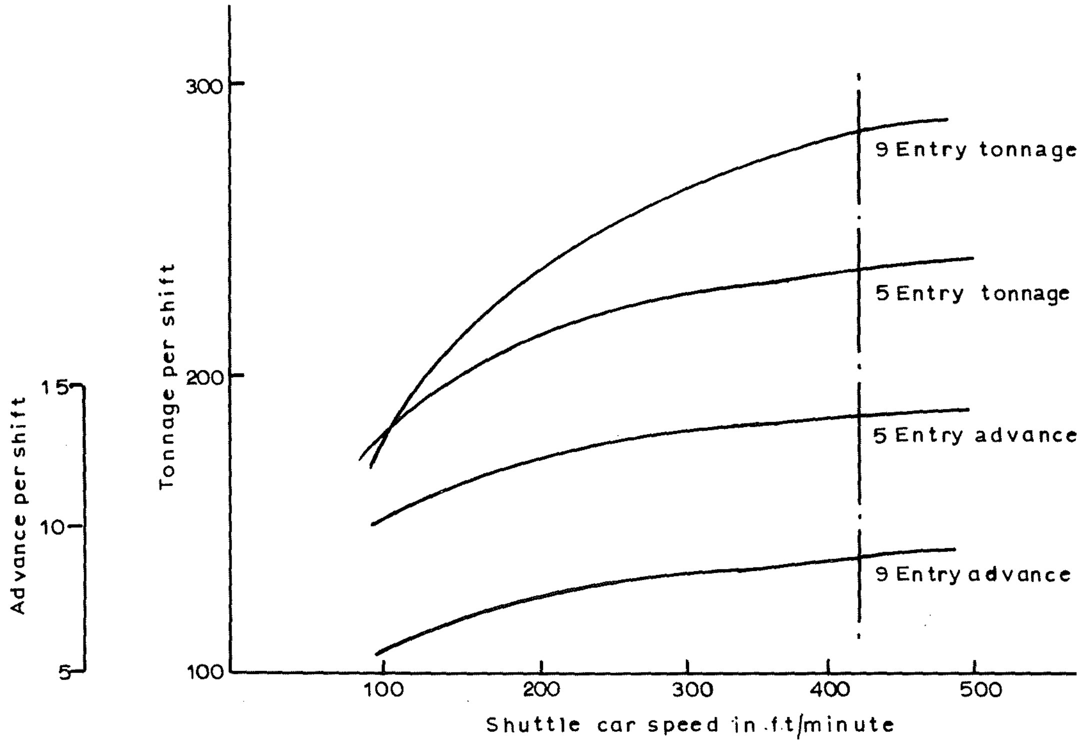
Continuous miner average capacity	2 tons/minute
Continuous miner tram rate	30ft/minute
Length of run before support	17ft
Shuttle car average tram speed	250ft/minute
Car discharge time	1 minute
Room width	20ft
Available shift time	300minutes
Centre distance crosscuts and entries	40ft
Seam height	33"
Services and belt extension	1shift/crosscut
Coal density	84.24lbs/ft^3

British Mining Consultants Ltd London

Engineer.......... J.H.C.	Date.......... May 80
Traced by.......... J.M.E.	Drg No........ 710/9

PLATE 10

ROOM AND PILLAR
SHUTTLE CAR HAULAGE
EFFECT OF VARIATION IN SHUTTLE CAR SPEED

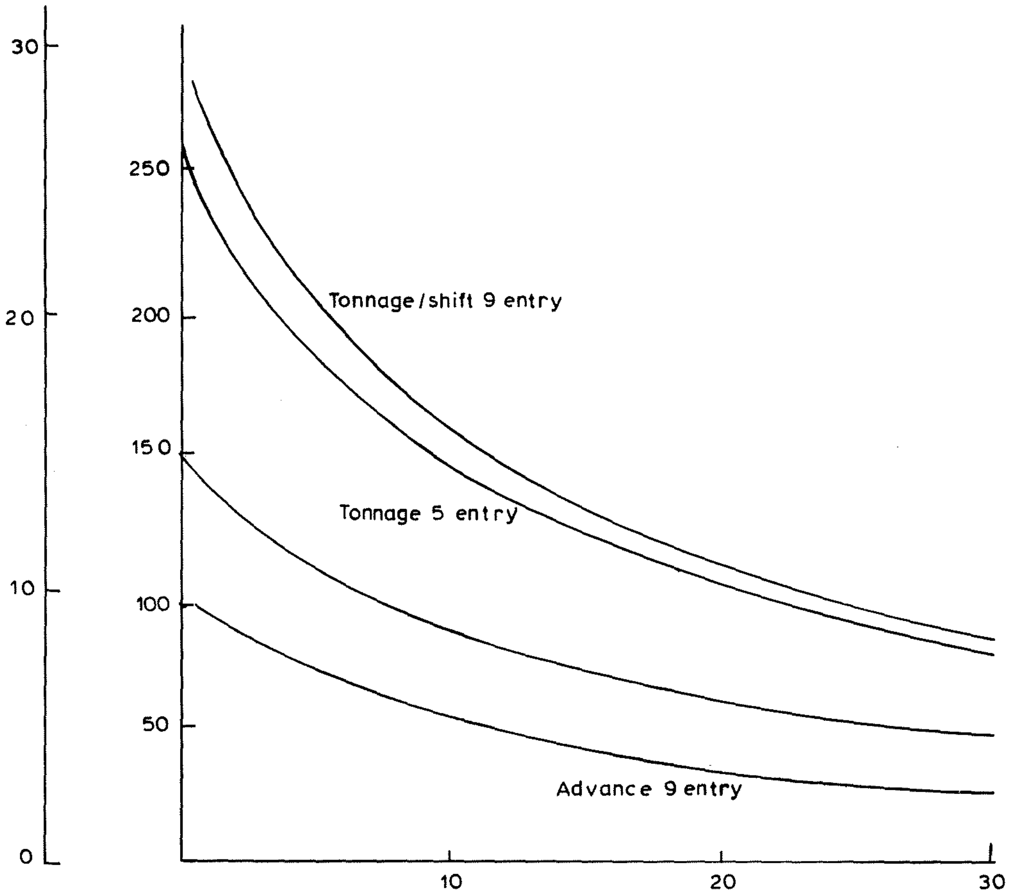


Continuous miner average capacity	2 tons/minute
Continuous miner tramming speed	30 ft / minute
Shuttle car factor	3 tons/car
Shuttle car discharge time	1 minute
Room width	20 ft
Centre distance crosscuts and entries	40 ft
Seam height	33 "
Available shift times	300 minutes
Services and conveyor extension	1 Shift/crosscut
Coal density	84·24 lbs/ft^3

British Mining Consultants Ltd London	
Engineers...... J. H. C.	Date...... May 80
Traced by...... J. W.	Drg No...... 710/10

PLATE 11

ROOM AND PILLAR
SHUTTLE CAR HAULAGE
INTEGRAL BOLTING SYSTEM



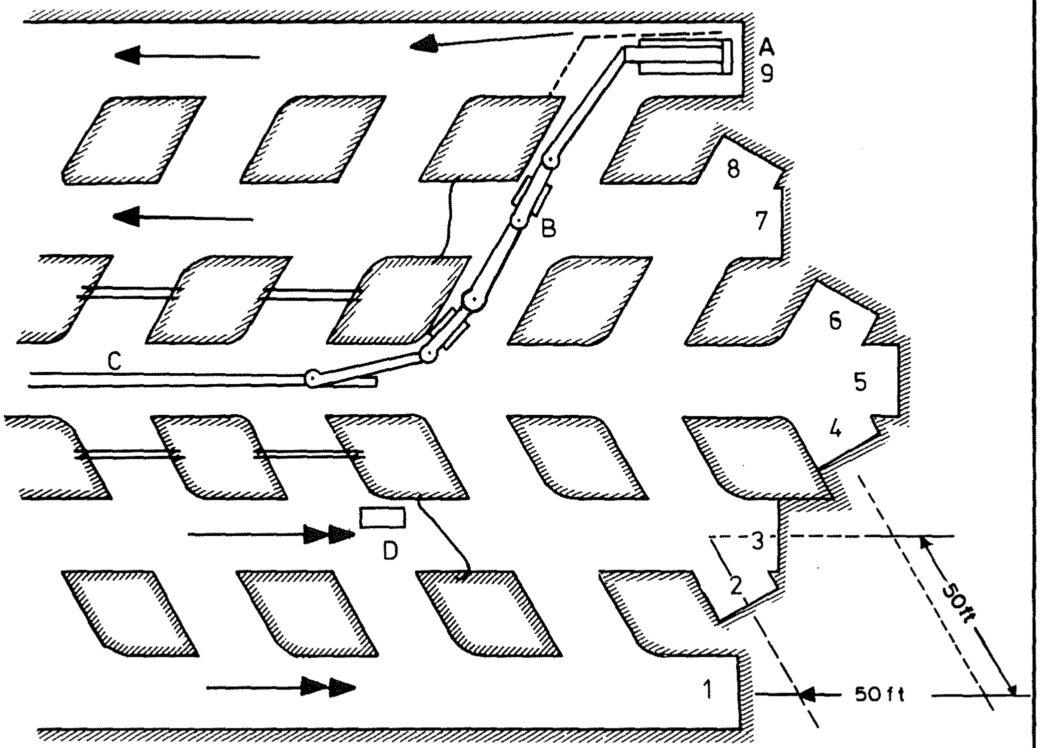
Continuous miner capacity average	2 tons/minute
Miner tram rate	30ft
Distance between stops for support	variable
Time to install support	variable
Shuttle car factor	3 tons/car
Shuttle car average speed	250ft/min
Discharge time	1 minute
Room width	20ft
Centre distances entries	40ft
Centre distances cross cuts	40ft
Seam width	33 inches
Coal density	84.24 lbs/ft^3

British Mining Consultants Ltd London

Engineer....... J.H.C.	Date............ June 80
Traced by..... J M E	Drg No......... 710/ 11

PLATE 12

ROOM AND PILLAR
5 ENTRY CONTINUOUS HAULAGE PANEL DESIGN
SHOWING CUTTING SEQUENCE AND VENTILATION



LEGEND

........Ventilation brattice

........Entry brattice

........Permanent stoppings

........Return airflow

........Intake airflow

1, 2 -- 8,9 ....Cutting sequence

A............Continuous miner

B............Continuous haulage

C............Static chain

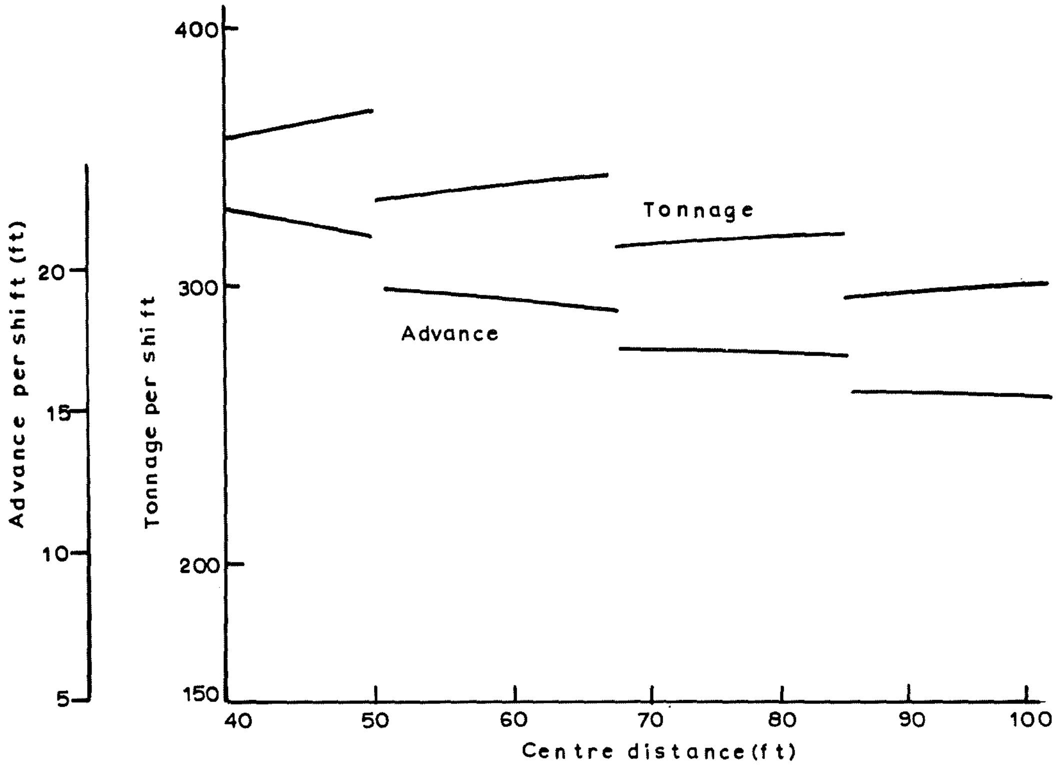
D............Power centre

APPROX SCALE 1" = 50'

British Mining Consultants Ltd London	
Engineer........J.H.C.	Date........June 80
Traced by......J.M.E	Drg No.....710/12

PLATE 13

ROOM AND PILLAR-5 ENTRY CONTINUOUS HAULAGE-
EFFECT OF VARIATION IN CENTRE DISTANCE

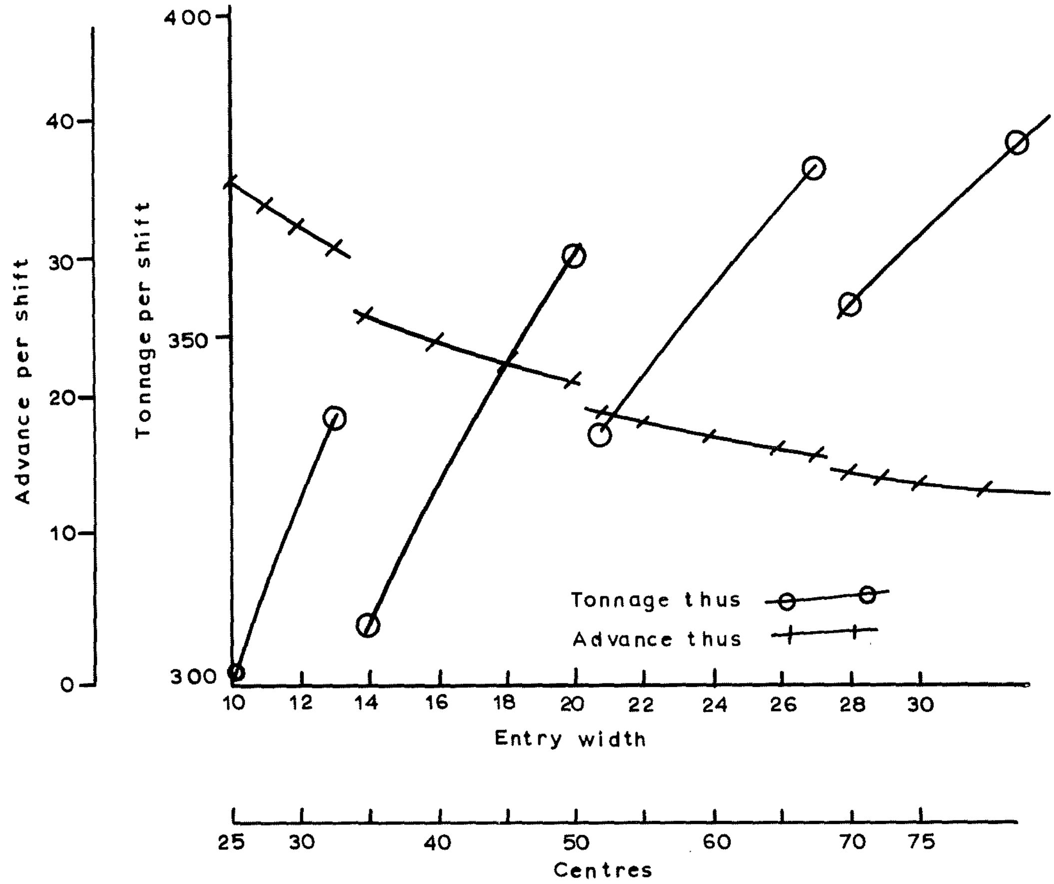


Continuous miner average capacity	2 tons/minute
Miner-haulage transfer rate	10 ft/minute
Length of run before bolting	17 ft
Cross cut turn off angle	60°
Room width	20 ft
Seam width	33"
Coal density	84 ·24 lb/ft
Available shift time	300 minutes

British Mining Consultants Ltd London	
Engineer...... J. H. C	Date....... June 80
Traced by....... J. W.	Drg No....... 710/13

PLATE 14

ROOM AND PILLAR-5 ENTRY CONTINUOUS HAULAGE - EFFECT OF VARIATION ON ENTRY WIDTH AND CENTRE DISTANCE



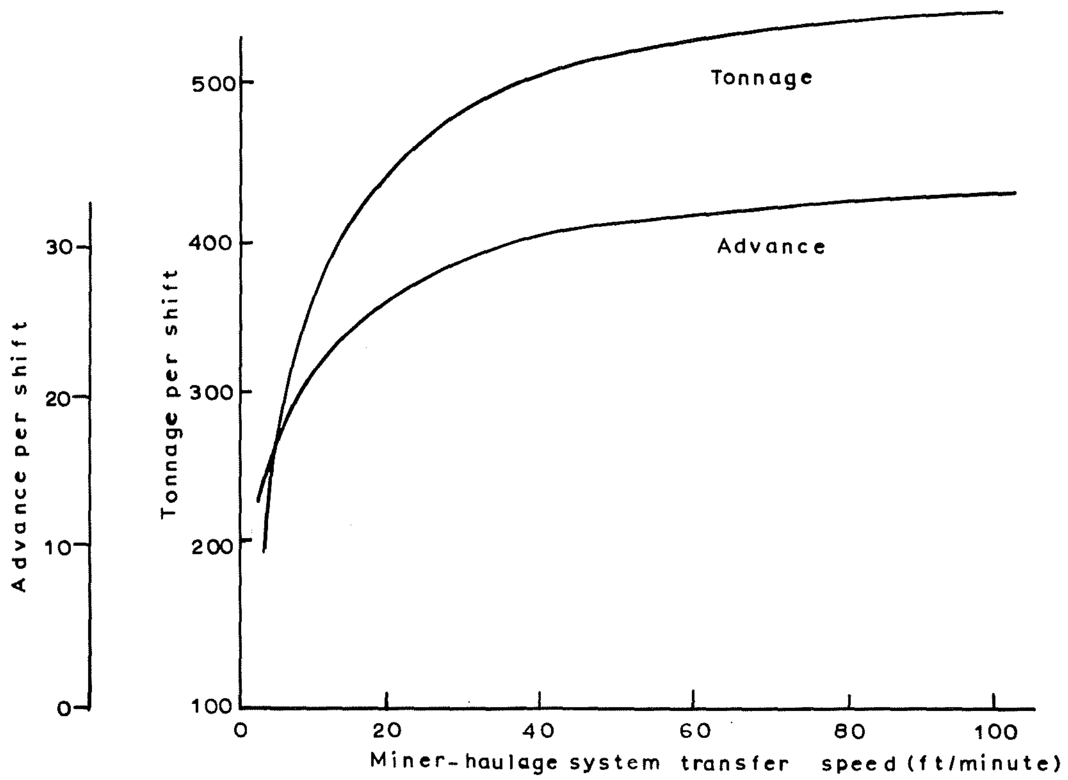
Continuous miner average capacity	2 tons/minute
Miner-haulage transfer speed	10 ft/minute
Length of run before bolting	17 ft
Cross cut turn off angle	60°
Seam height	33"
Coal density	84·24 lb/ft
Available shift time	300 minutes

British Mining Consultants Ltd London

Engineer...... J.H.C.	Date....... May 80
Traced by...... J.W.	Drg No 710/14

PLATE 15

ROOM AND PILLAR - 5 ENTRY CONTINUOUS HAULAGE - EFFECT OF VARIATION IN MINER - HAULAGE TRANSFER SPEED

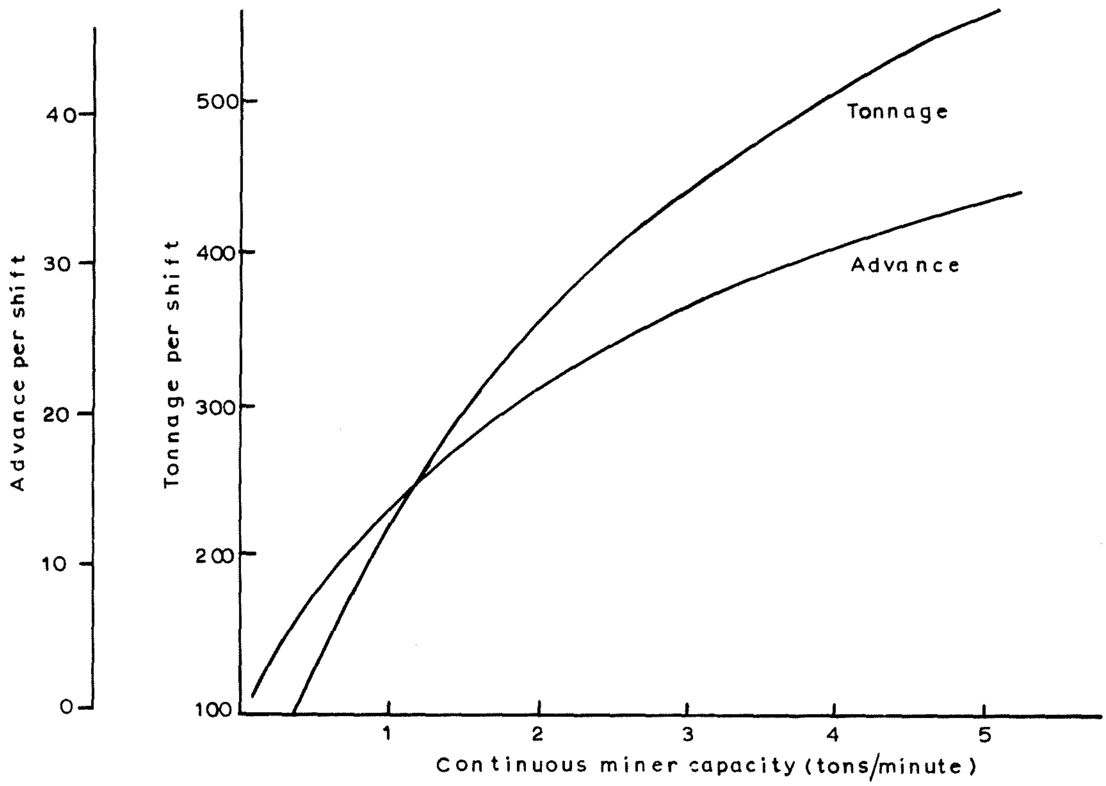


Continuous miner average capacity	2 tons/minute
Length of run before bolting	17 ft
Cross cut turn off angle	60°
Room width	20 ft
Seam width	33"
Coal density	84·24 lbs / ft
Available shift time	300 minutes
Centre distance crosscuts and entries	50 ft

British Mining Consultants Ltd London	
Engineer...... J H C	Date...... June 80
Traced by...... J W	Drg No...... 710/15

PLATE 16

ROOM AND PILLAR-5 ENTRY CONTINUOUS HAULAGE-
EFFECT OF VARIATION IN CONTINUOUS MINER CAPACITY

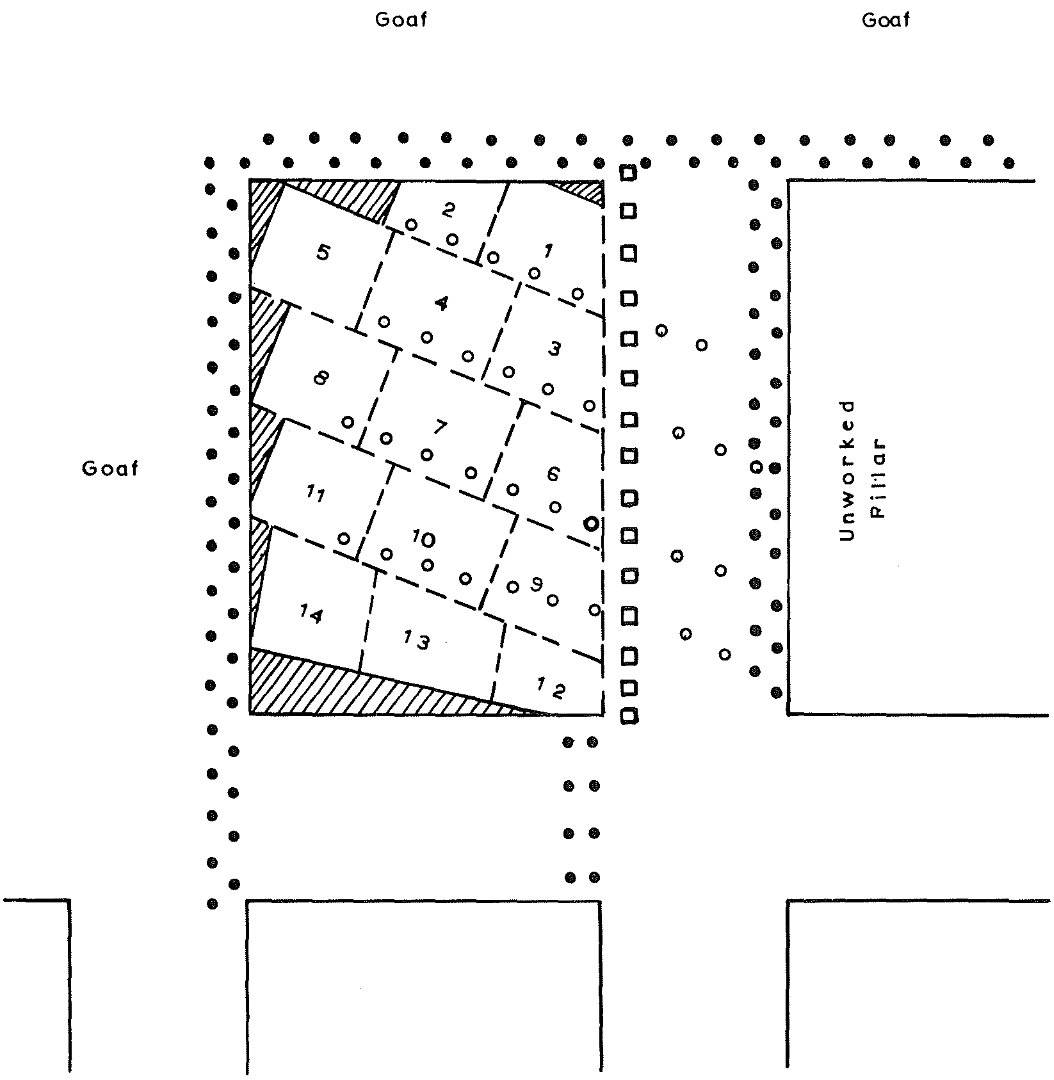


Miner -haulage transfer speed	10 ft/minute
Run length before bolting	17 ft
Cross cut turn off angle	60°
Centres, crosscut and entry	50 ft
Room width	20 ft
Seam height	33"
Coal density	84·24 lbs/ft
Available shift time	300 minutes

British Mining Consultant Ltd London	
Engineers...... J.H.C.	Date...... June 80
Traced by...... J.W.	Drg No......710/16

PLATE 17

ROOM AND PILLAR RETREAT
SINGLE PILLAR EXTRACTION



Legend

- ▫ Roof bolt
- • Breaker post
- ◦ Post
- Remnant stub or fender

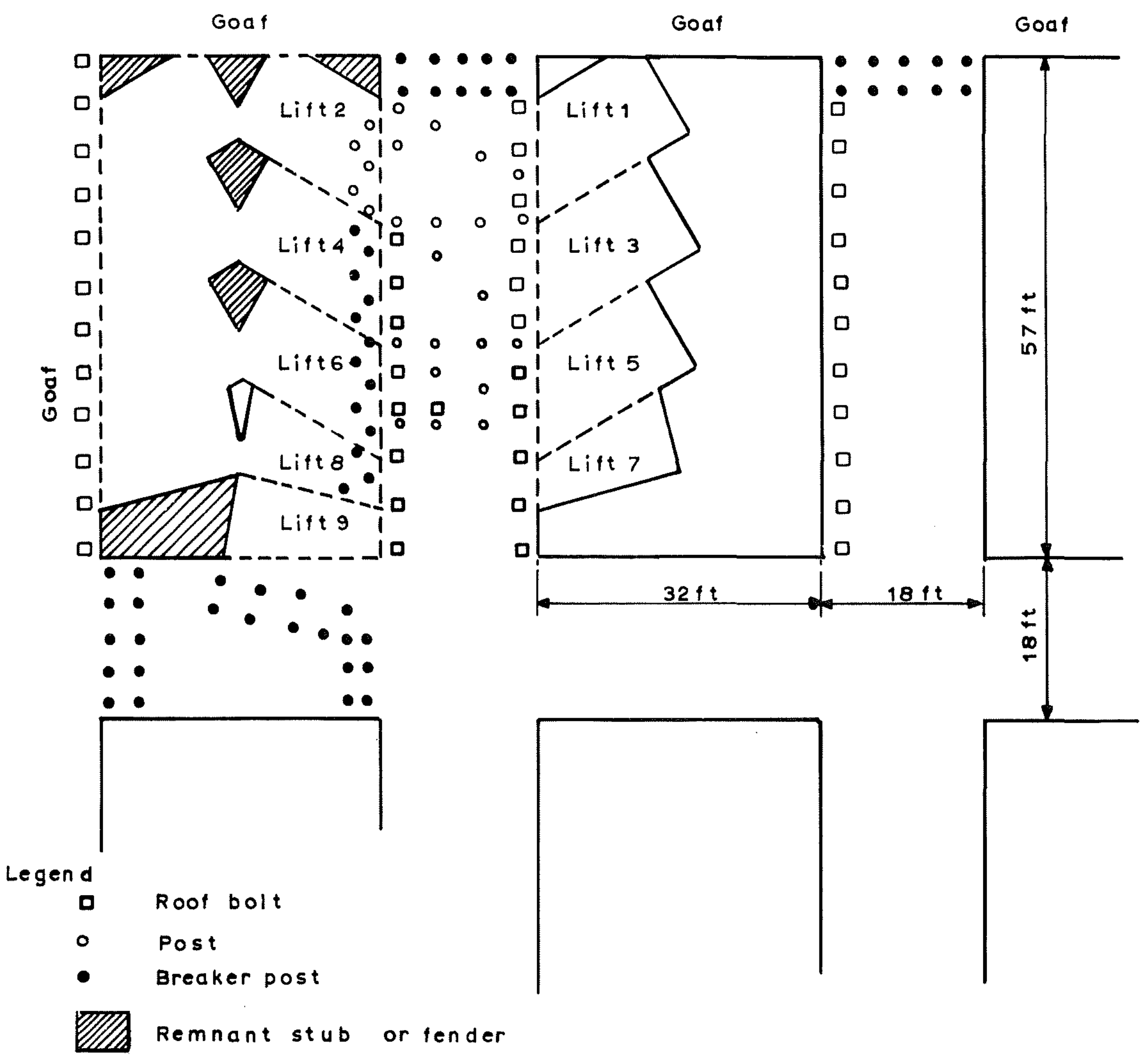
British Mining Consultants Ltd London

Engineer...... J H C	Date...... June 80
Traced by...... J W	Drg No......710 /17

PLATE 18

ROOM AND PILLAR RETREAT

SIMULTANEOUS EXTRACTION OF TWO PILLARS FROM COMMON ENTRY

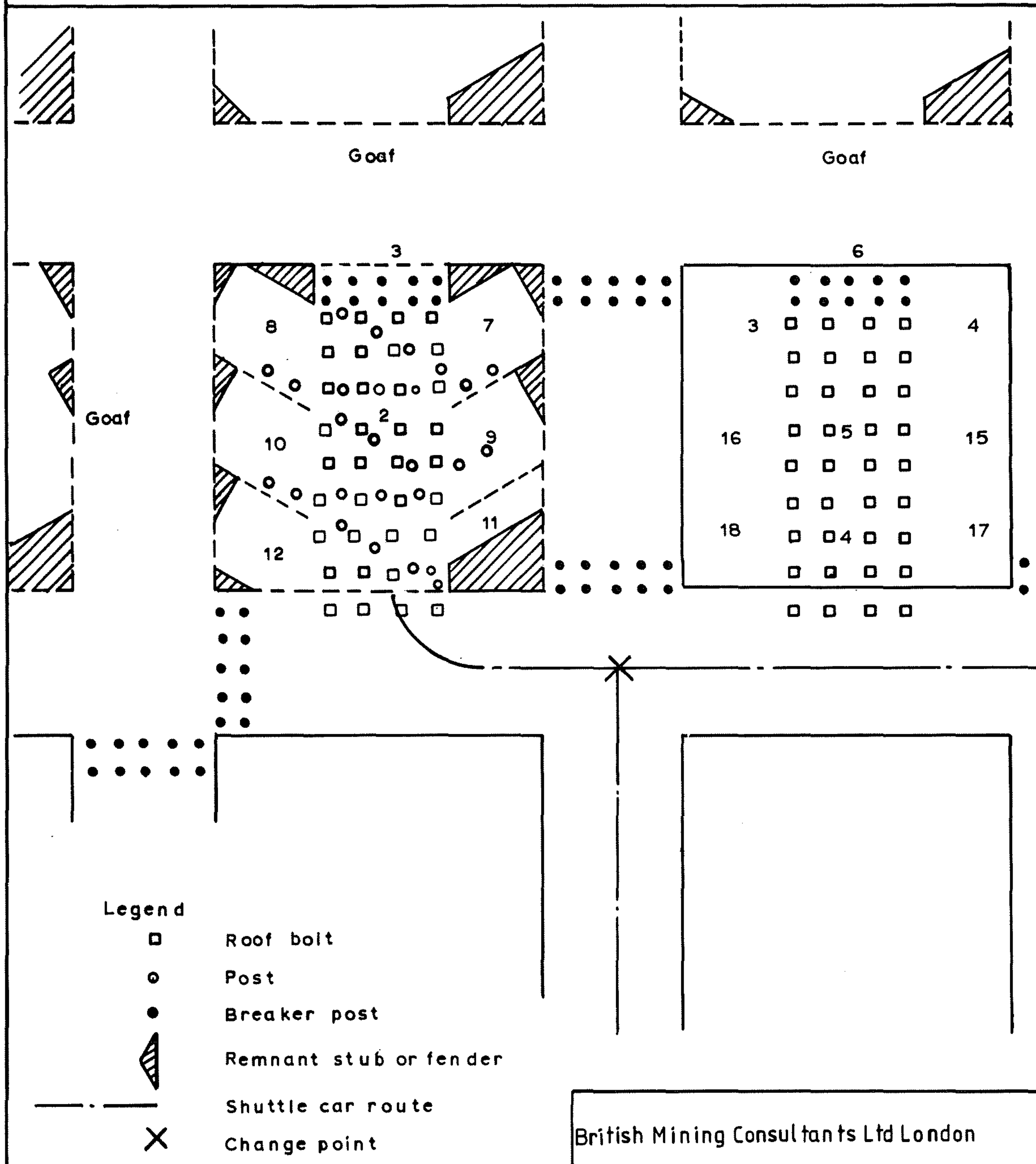


British Mining Consultants Ltd London

Engineer....... J H C	Date....... June 80
Traced by....... J W	Drg No....... 710 / 18

PLATE 19

ROOM AND PILLAR RETREAT
PILLAR EXTRACTION BY SPLITTING

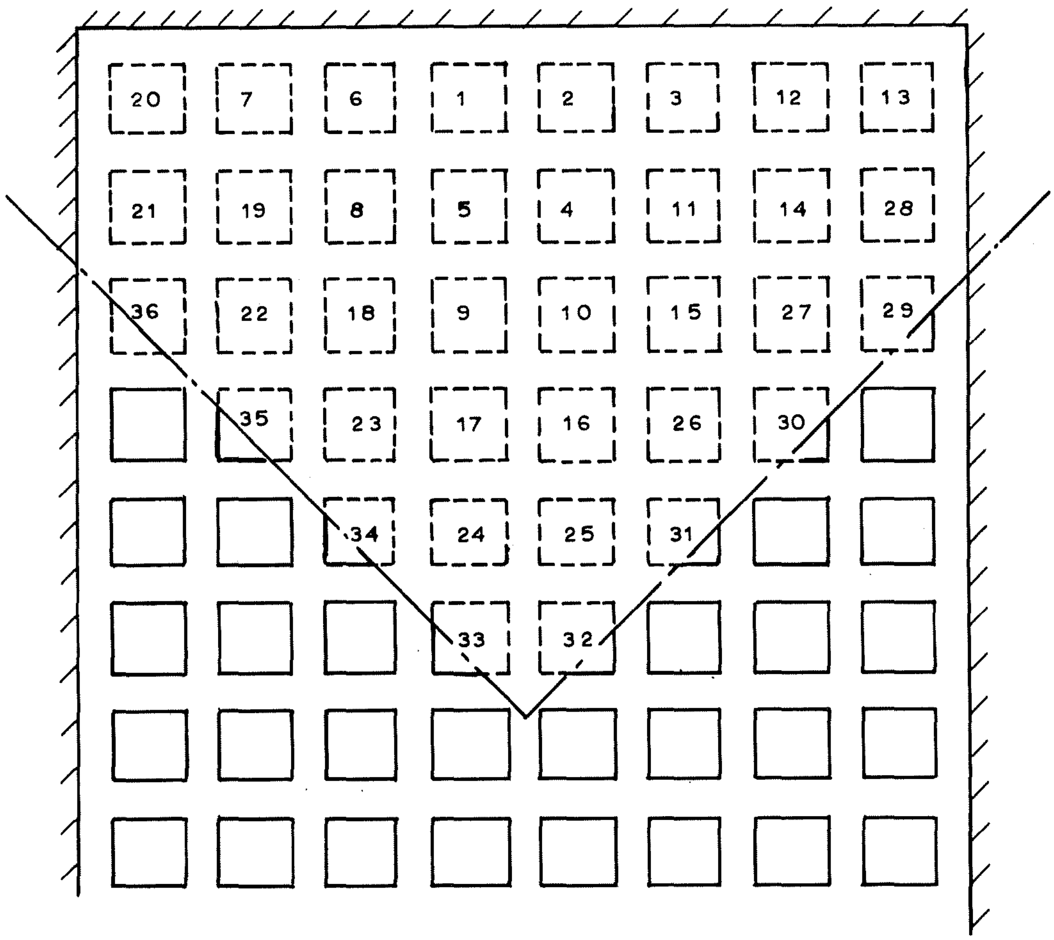


British Mining Consultants Ltd London

Engineer...... J H C	Date...... JUNE 80
Traced by...... J W	Drg No...... 710/19

PLATE 20

ROOM AND PILLAR RETREAT
SEQUENCE OF PILLAR EXTRACTION
(BLEEDERS NOT SHOWN)



Legend

Unworked pillar

Extracted pillar

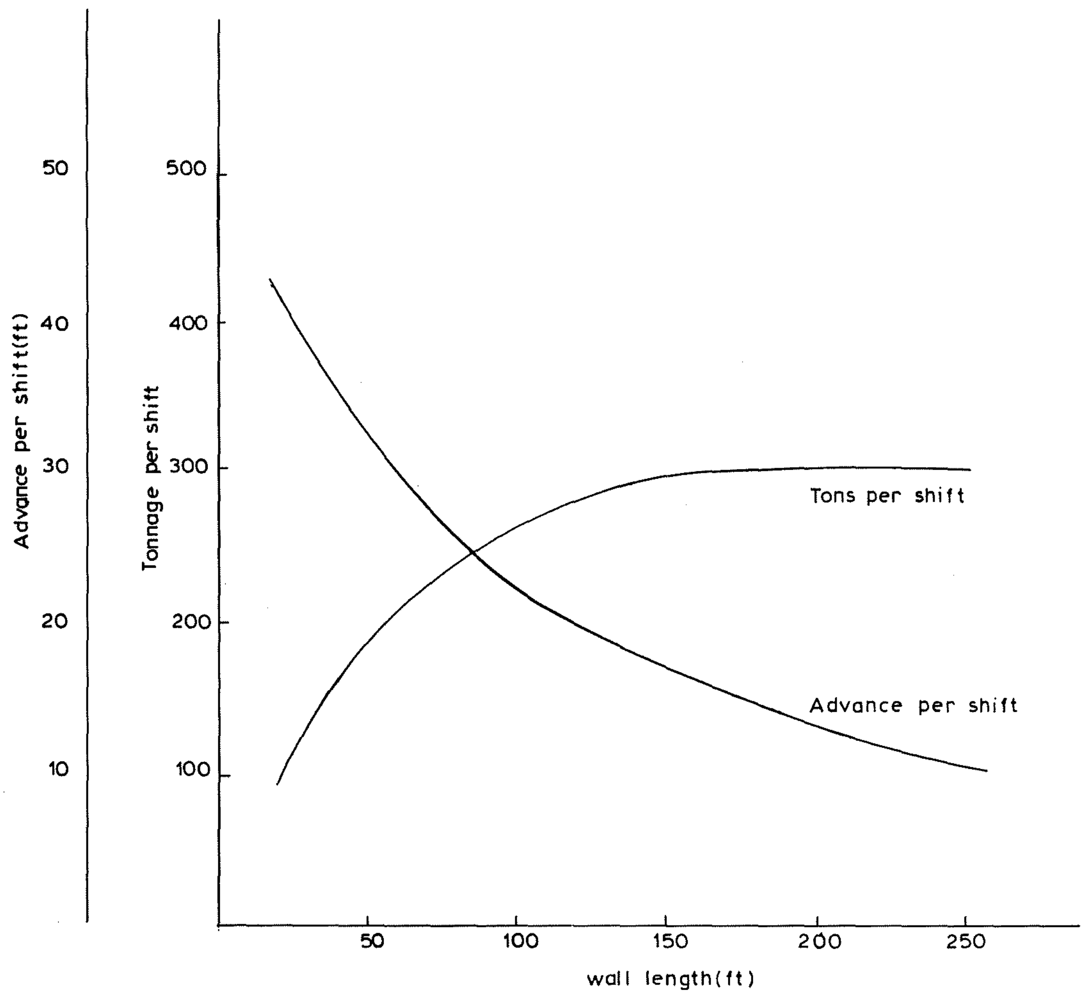
Working line

British Mining Consultants Ltd London

Engineer...... J H C	Date...... June 80
Traced by...... J W	Drg No...... 710/20

PLATE 21

SHORTWALL
SHUTTLE CAR HAULAGE
EFFECT OF VARIATION OF WALL LENGTH



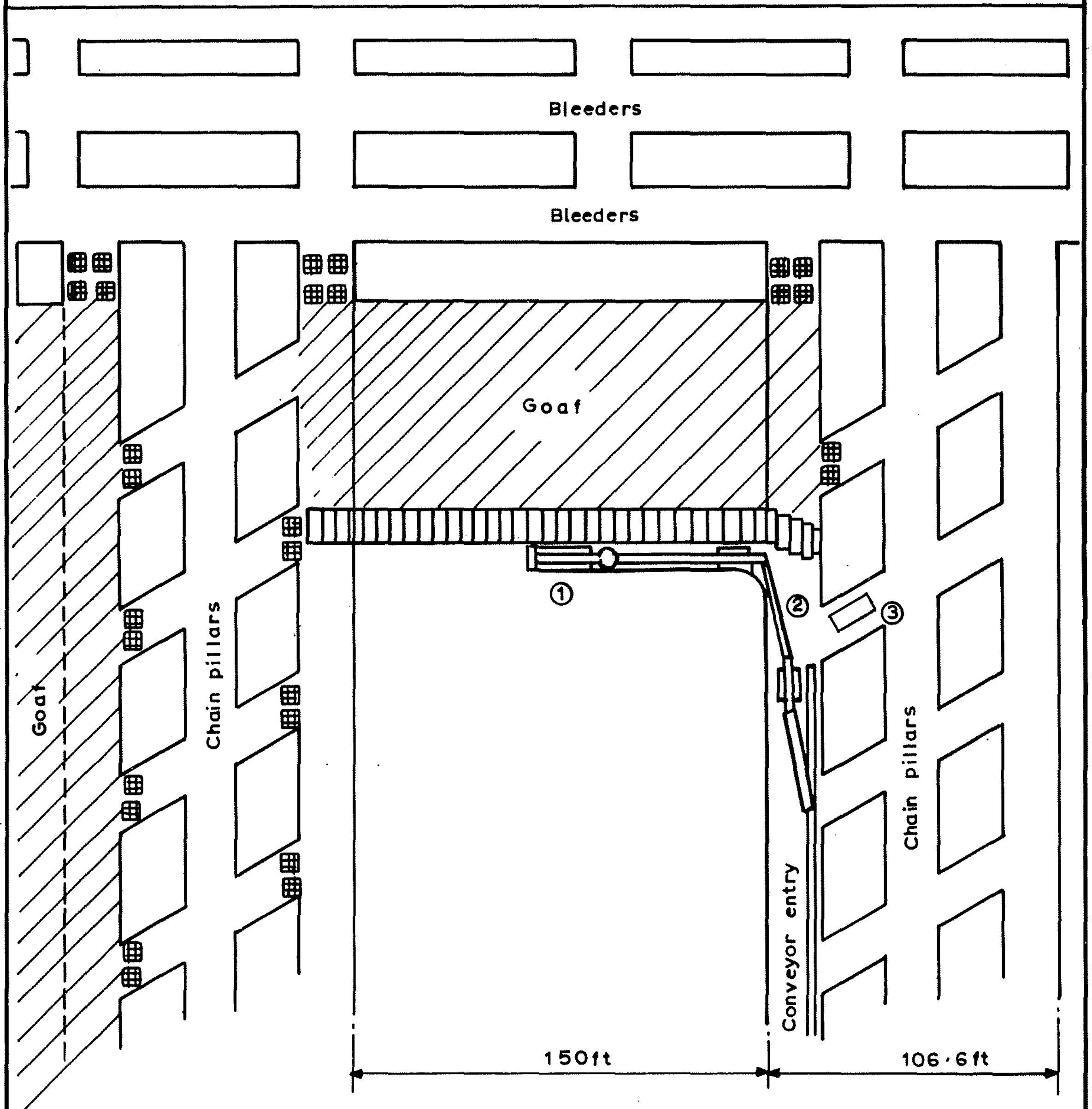
Continuous miner average capacity	2ton/minute
Shuttle car factor	3 tons/car
Shuttle car average speed	250 ft/minute
No of shuttle cars	2
Width of strip	10ft
Seam height	33 inches
Coal SG	1.35
Available face time	360 minutes/shift
Car discharge time	1 minute
Room width	20 ft

British Mining Consultants Ltd London

Engineer.......... J.H.C.	Date .... May 80
Traced by.......... J.M.E	Drg. No........... 710/21

PLATE 22

SHORTWALL
GENERAL ARRANGEMENT



Legend

① Continuous miners

② Continuous haulage

③ Clean up scoop

▦ Cribs

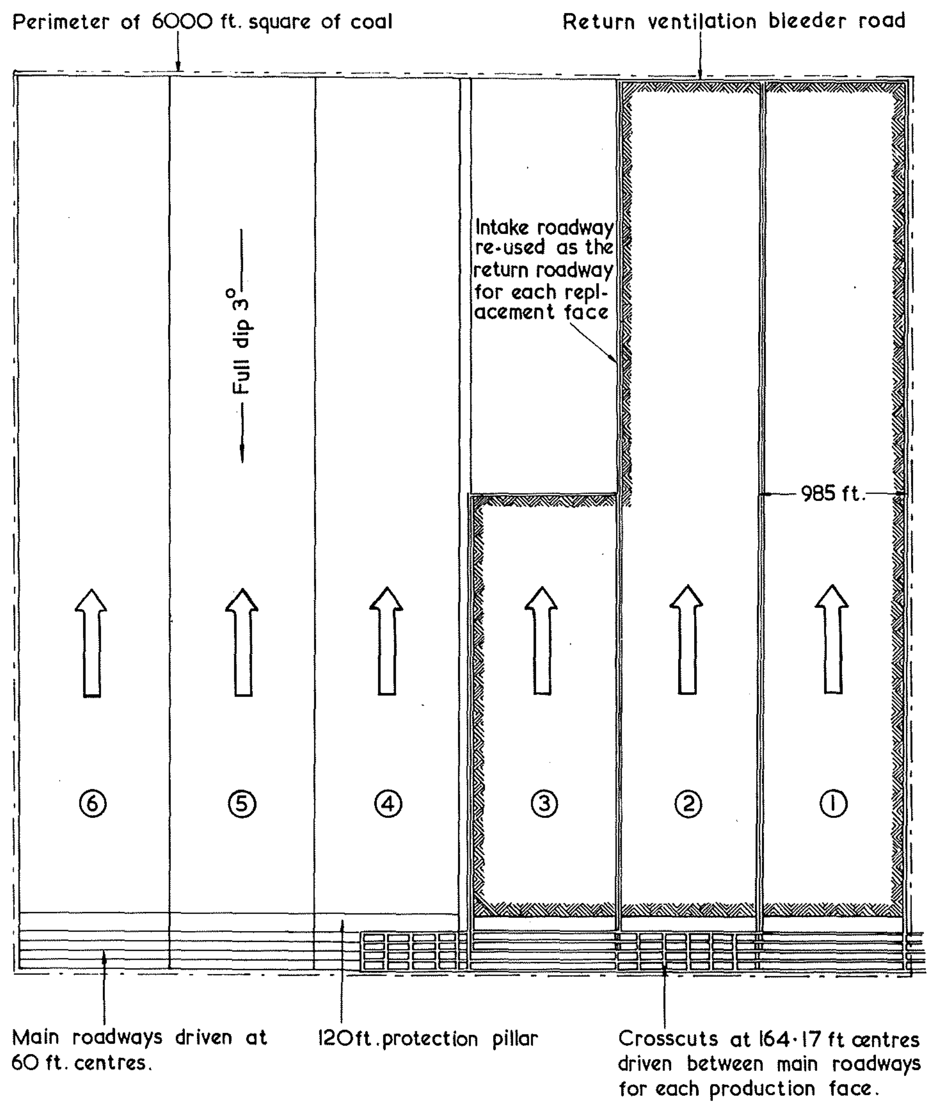
British Mining Consultants Ltd London

Engineer...... J H C	Date....... June 80
Traced by...... J W	Drg No......710/22

PLATE 23

DIAGRAM ILLUSTRATING LONGWALL ADVANCING
LAYOUT ('Z' SYSTEM)

0 500 1000 2000 3000 feet



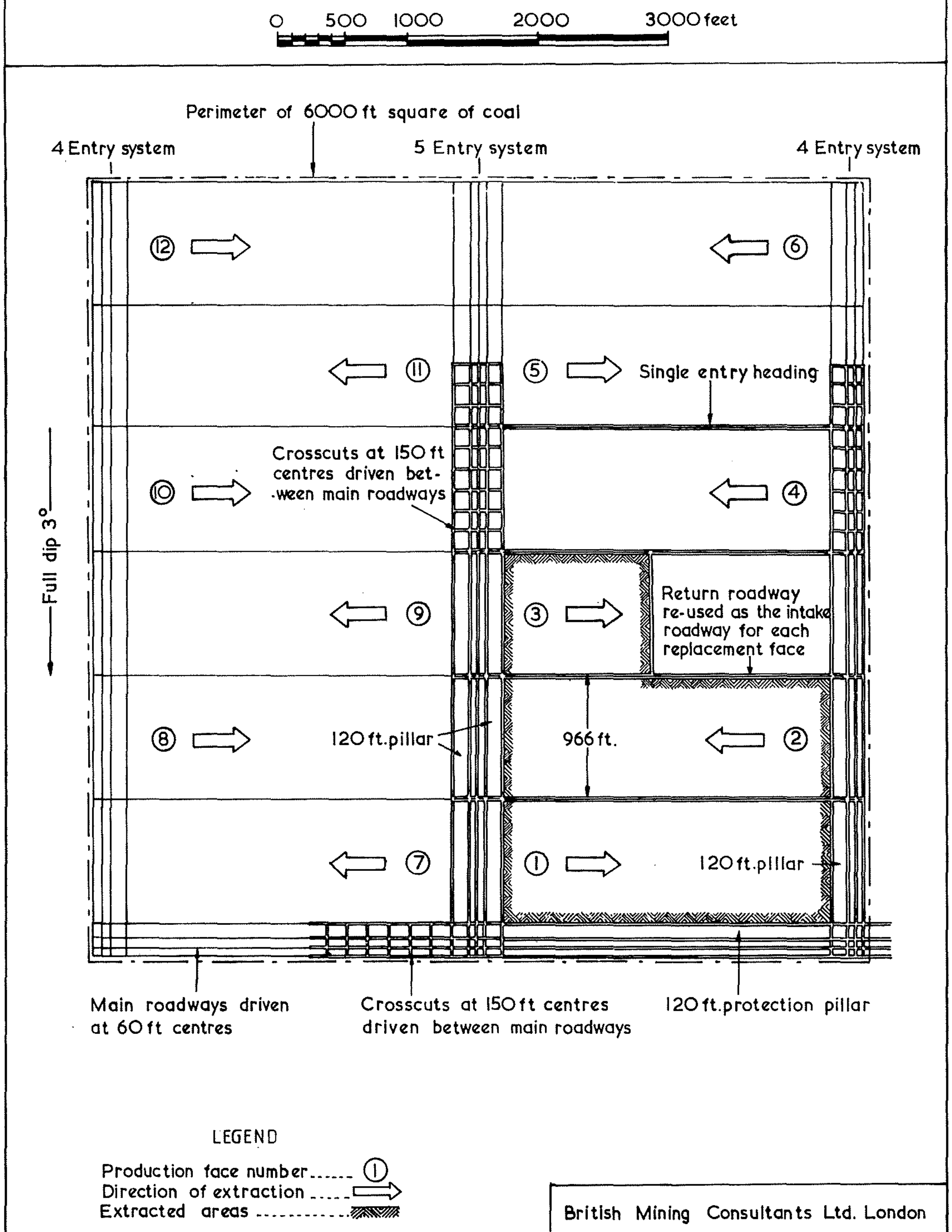
LEGEND

Production face number ...... ①
Direction of extraction ..... ⇨
Extracted areas ....

British Mining Consultants Ltd. London	
Engineer .... J.C.	Date ..... Mar 80
Traced by ..... L.H.J.	Drg. No 710/23

PLATE 24

DIAGRAM ILLUSTRATING LONGWALL RETREAT LAYOUT (SINGLE ENTRY)

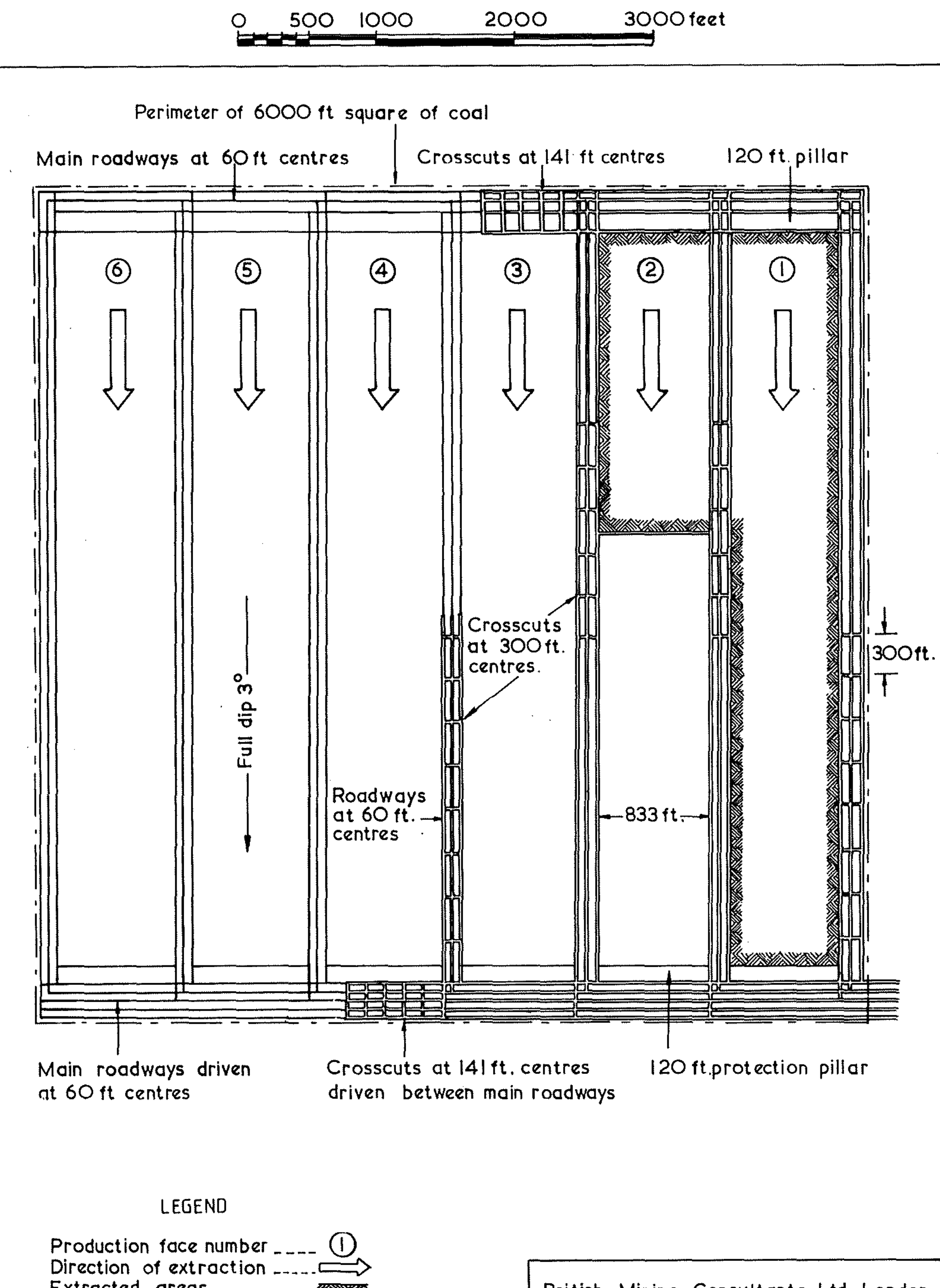


British Mining Consultants Ltd. London

Engineer .... J.C.	Date ..... Mar 80
Traced by ..... L.H.J.	Drg. No 710/24

PLATE 25

DIAGRAM ILLUSTRATING LONGWALL RETREAT LAYOUT (MULTIPLE ENTRY)



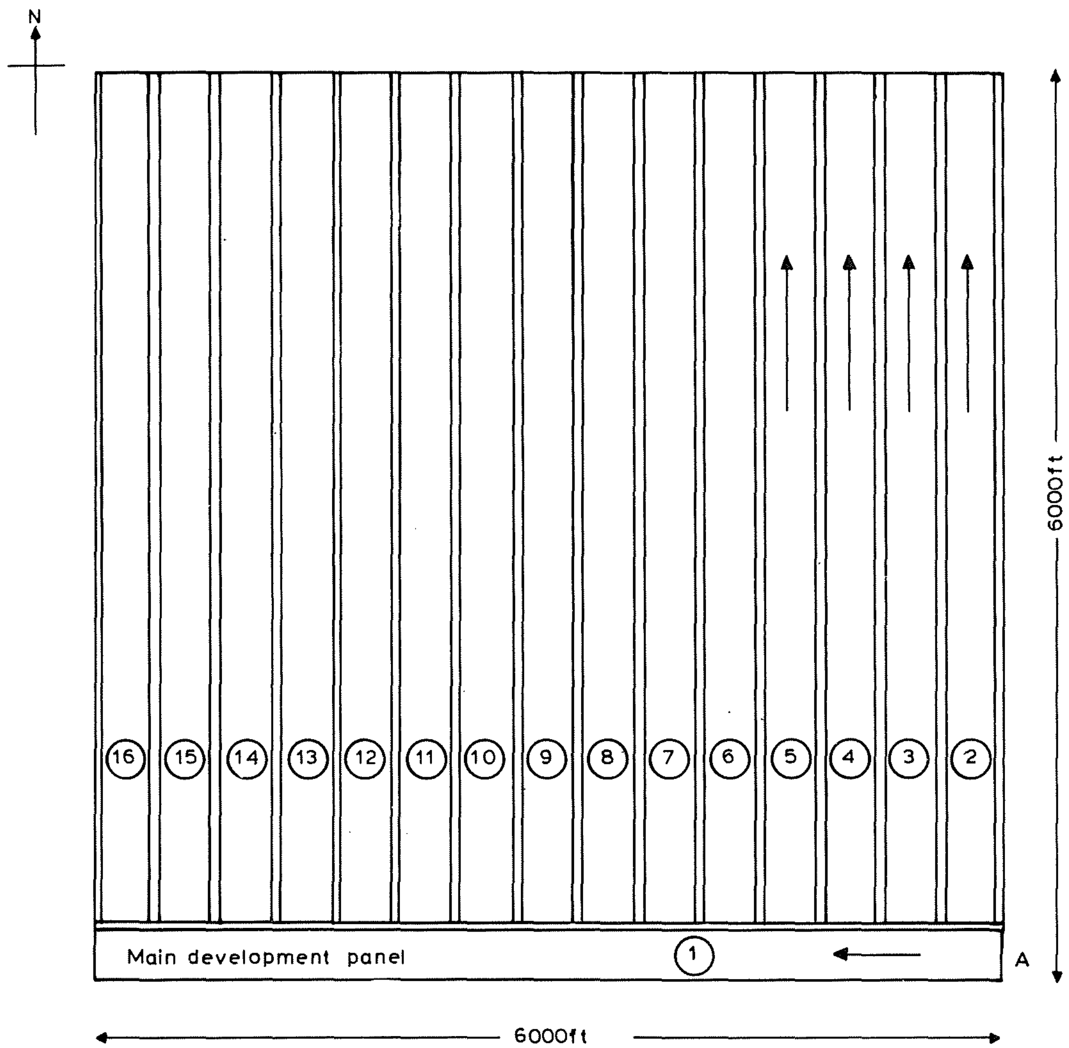
British Mining Consultants Ltd. London

Engineer .... J.C.	Date ...... Mar. 80
Traced by ..... L.H.J.	Drg. No. 710/25

PLATE 26

ROOM AND PILLAR

CONCEPTUAL LAYOUT I



LEGEND

1, 2, 3 ..... 16 ....Panel numbers and sequence of working

.........Barrier pillars

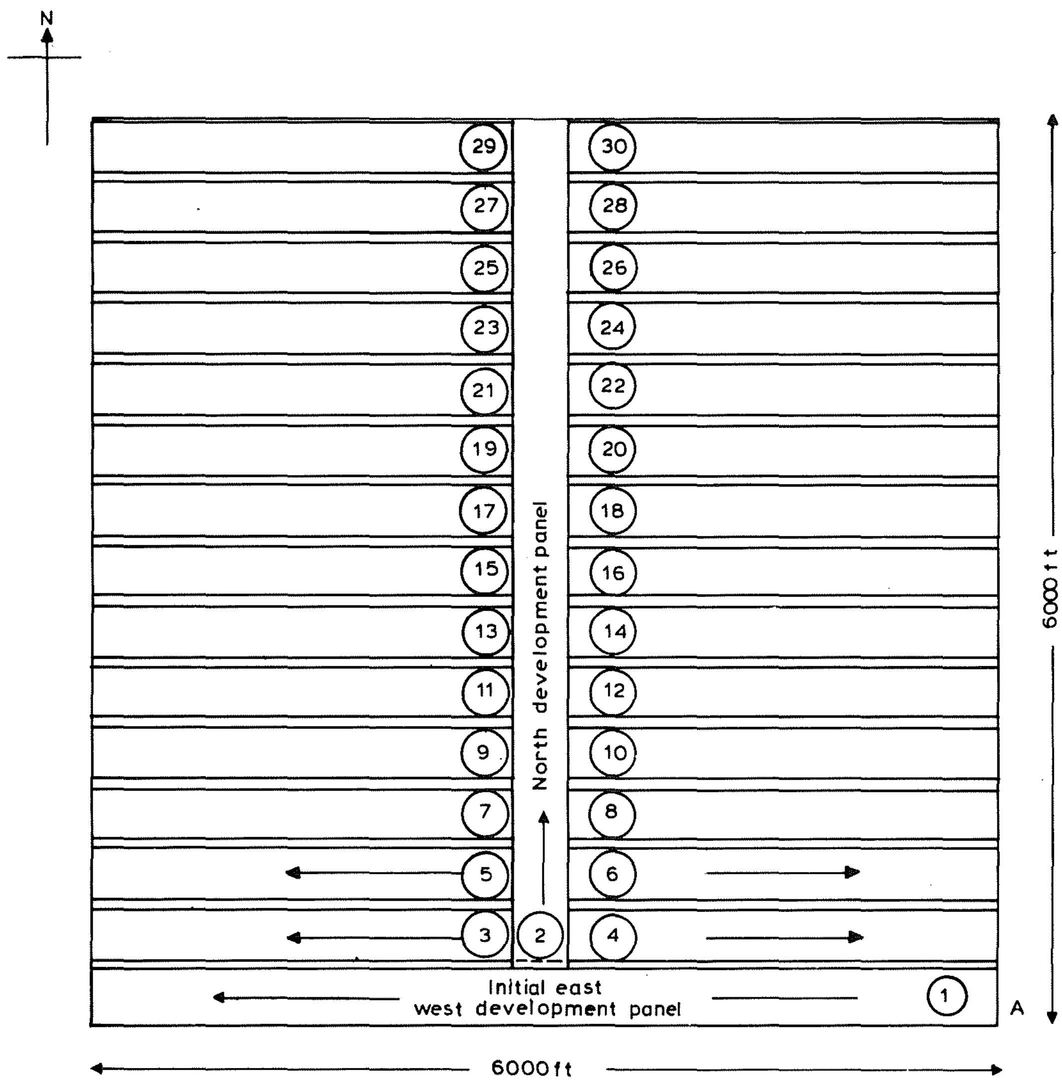
A..............................Entry point to area

..............Direction of advance

British Mining Consultants Ltd London	
Engineer......... J.H.C.	Date....... June 80
Traced by..... J.M.E	Drg.No.... 710/26

PLATE 27

ROOM AND PILLAR.
CONCEPTUAL LAYOUT II



LEGEND

1 2 3 .... 30 .... Panel numbers and sequence of working

.... Barrier pillars

.... Direction of advance

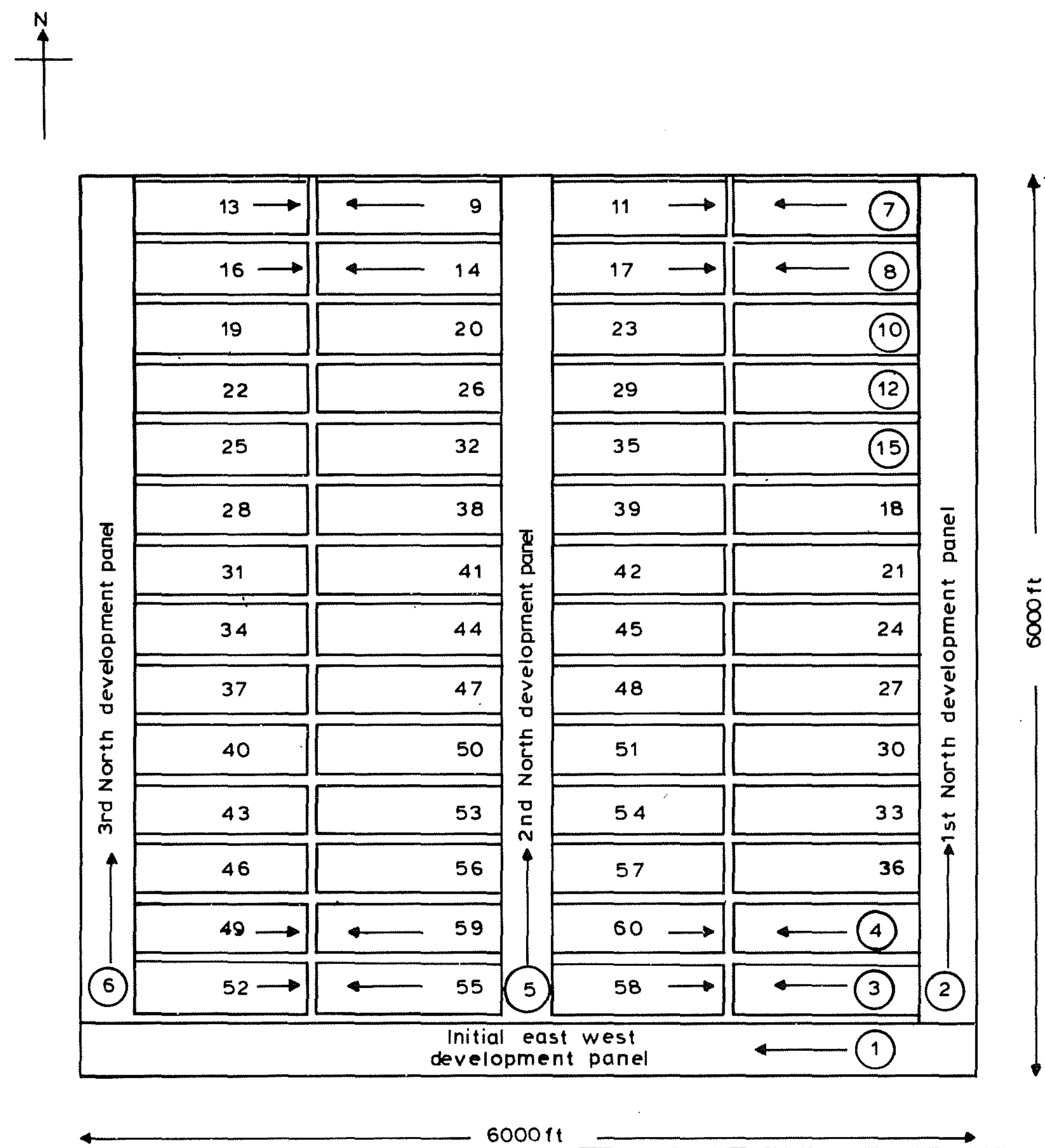
A .... Point of entry to area

British Mining Consultants Ltd London	
Engineer....... J.H.C	Date....... June 80
Traced by..... J.M.E	Drg No.....710/27

PLATE 28

ROOM AND PILLAR

CONCEPTUAL LAYOUT III



(1) (2) (3)......(60).......... Sequence of working panels

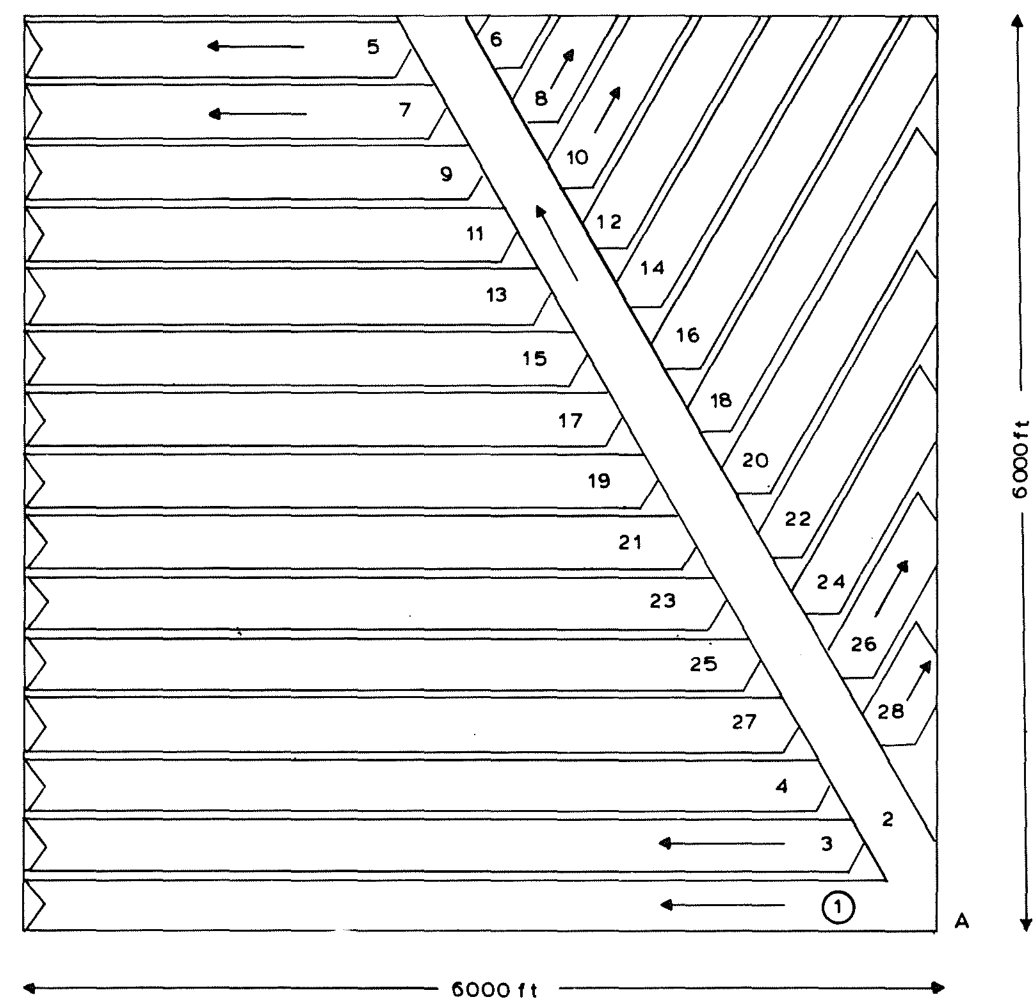
.... Panel barriers

British Mining Consultants Ltd London	
Engineer........ J.H.C. Traced by......J.M.E	Date.........June 80 Drg No......710/ 28

PLATE 29

ROOM AND PILLAR

60° LAYOUT (HERRING BONE)



LEGEND

1 2 3 _ _ _ _ _ 28 .... Sequence of working panels

.... Barrier pillars

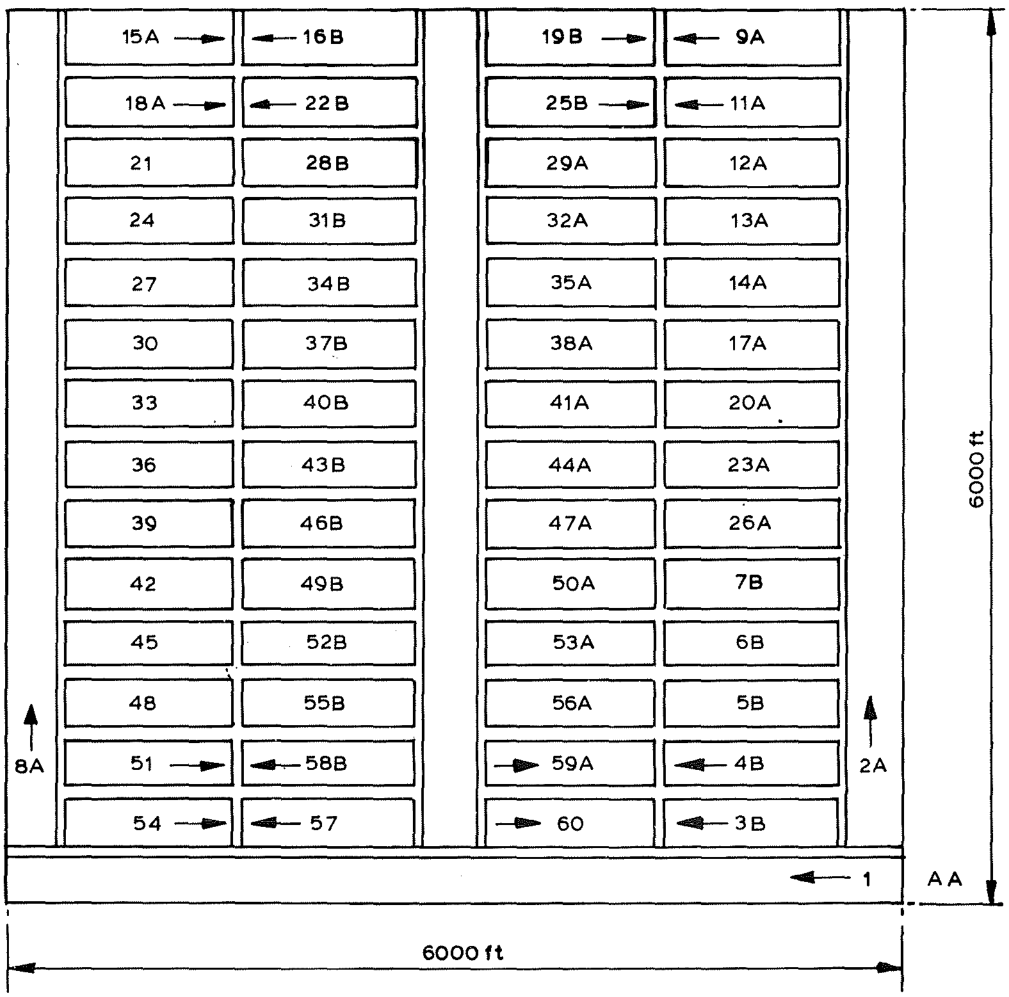
.... Direction of advance

A .... Point of entry to area

British Mining Consultants Ltd London	
Engineer........ J.H.C. Traced by..... J.M.E	Date....... June 80 Drg No... 710/29

PLATE 30

ROOM AND PILLAR
9 - ENTRY LAYOUT
EXTRACTION SEQUENCE USING 3 SECTIONS

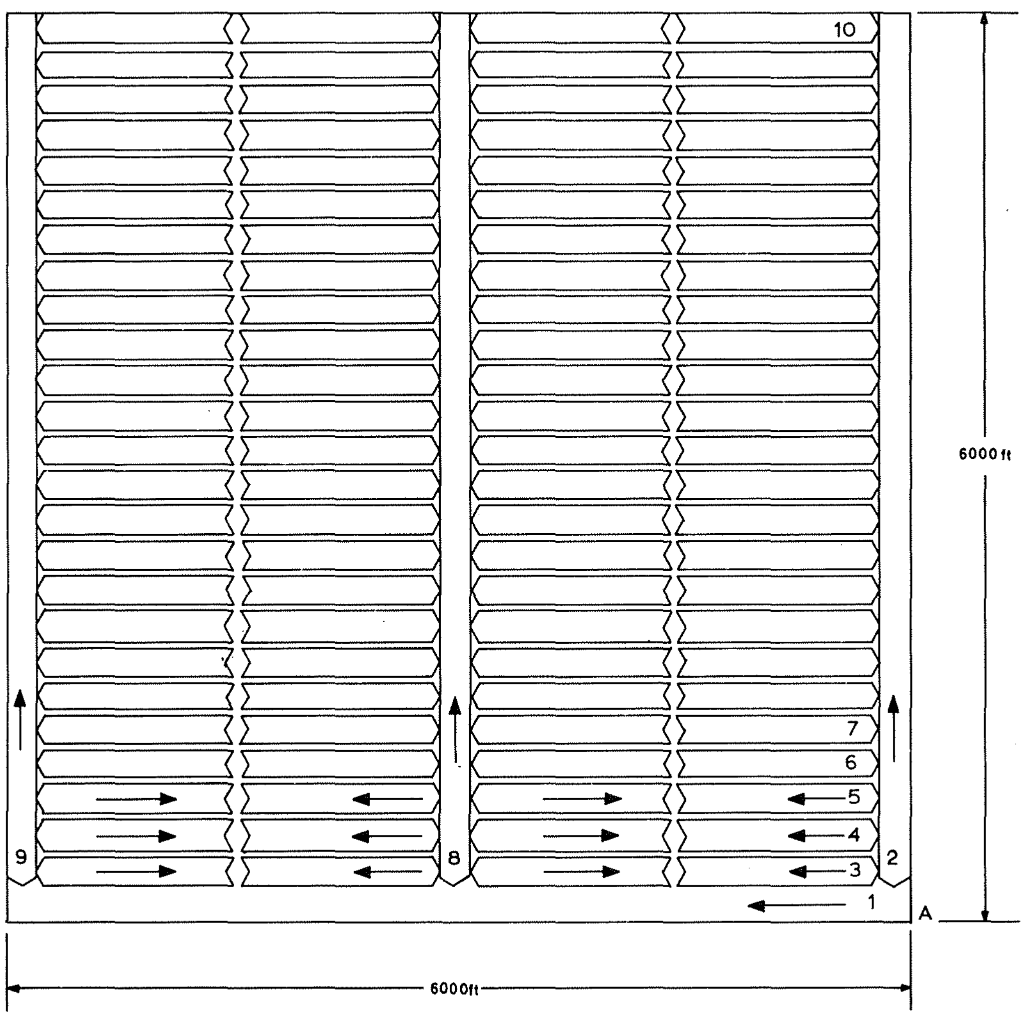


Legend
Initial sequence of operations :-
60 panels extracted by 1st section......1,8,15 etc.
59 panels extracted by 2nd section.....2A,9A,11A etc.
58 panels extracted by 3rd section.....3B,4B,5B etc.
→ Direction of advance
═ Barrier pillars
AA Point of entry

British Mining Consultants Ltd. London	
Engineer..........J.H.C.	Date........ June 80
Traced by......E.W.	Drg. No.....710 / 30

PLATE 31

ROOM AND PILLAR
LAYOUT FOR CONTINUOUS HAULAGE
(Narrow panels, 5 entry)



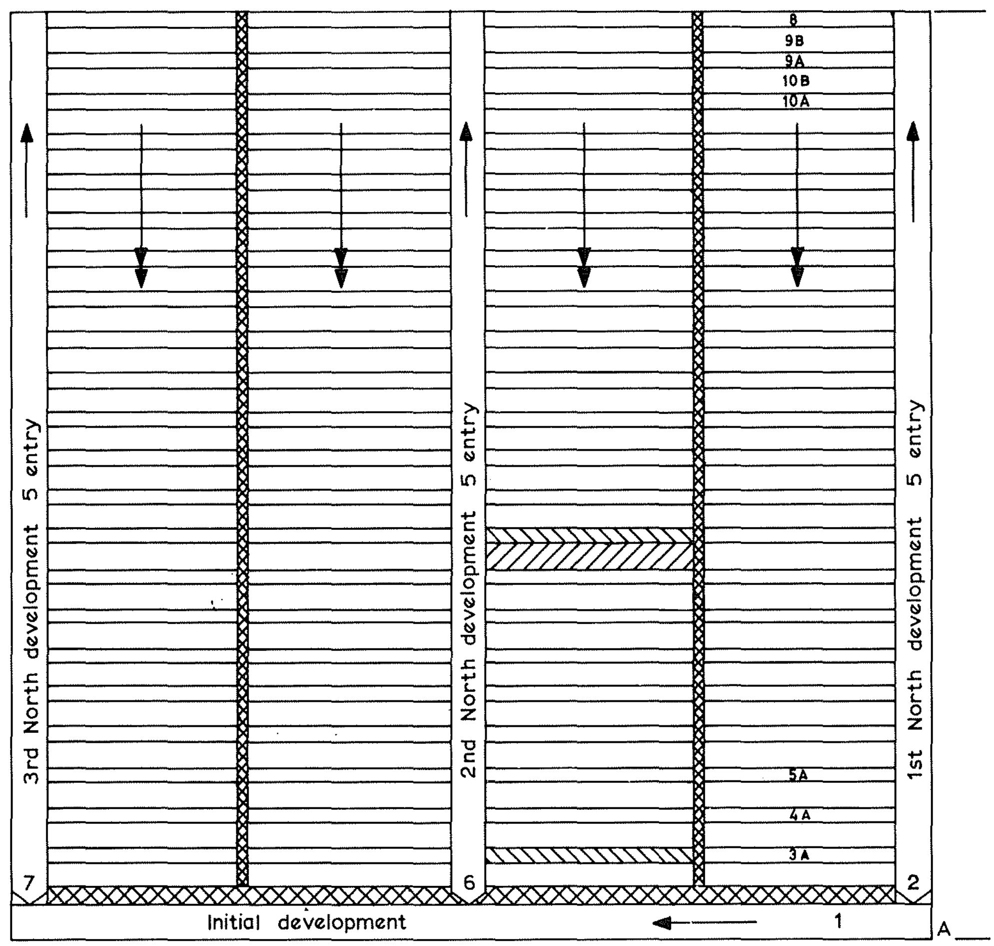
LEGEND

Initial development sequence............1,2,3......10
Direction of advance........................ →
Barrier pillars.................................. ═
Point of entry to area................................A

British Mining Consultants Ltd, London	
Engineer..........J.H.C.	Date..........June 80
Traced..................J.S.	Drg. No.......710/31

PLATE 32

SHORTWALL
GENERAL LAYOUT



LEGEND

Shortwall development (3 entry)

Shortwall panels

Barrier pillars

Direction of development advance

Sequence of shortwall extraction

Initial sequence of operations:-

Main development 1,2,6,7etc

Panel development .... 3A, 4A, etc

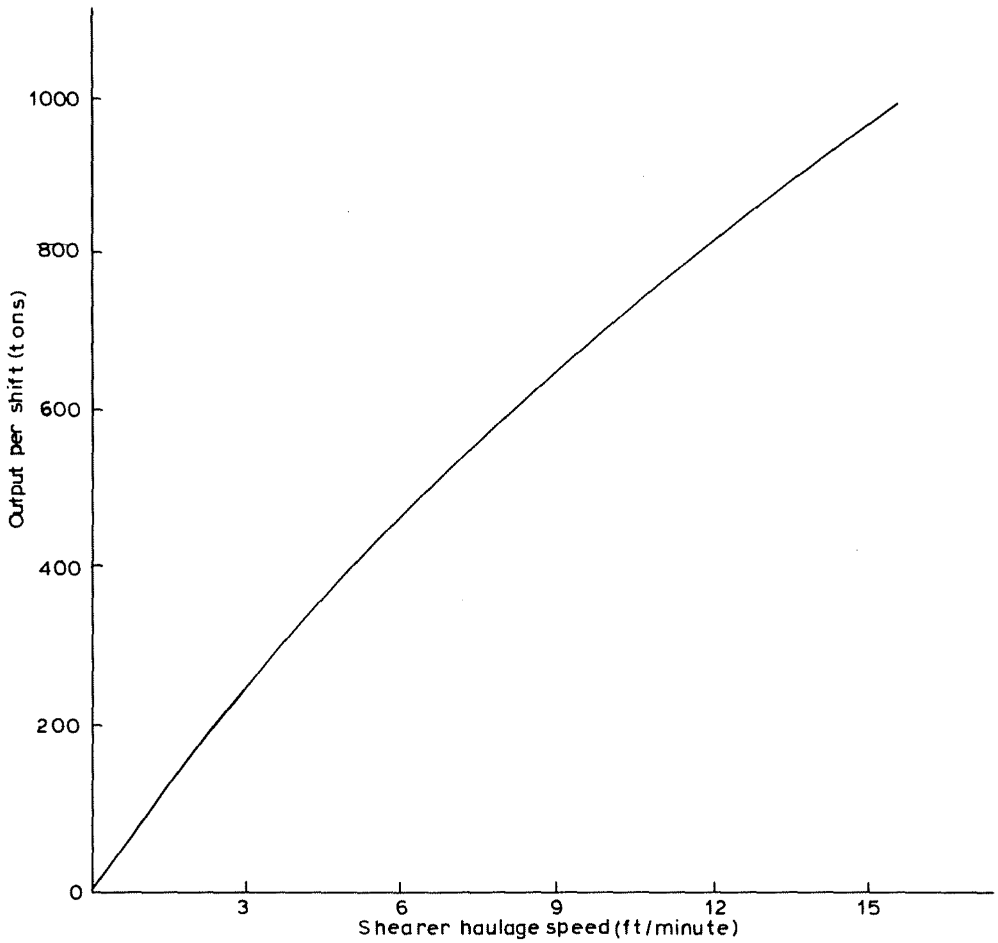
Shortwall extraction .... 9B, 10B, etc

British Mining Consultants Ltd, London

Engineer .... J.H.C.	Date .... June 80
Traced .... J.S.	Drg. No. 710/32

PLATE 33

LONGWALL
EFFECT OF HAULAGE SPEED
ON OUTPUT- FIXED WEB THICKNESS



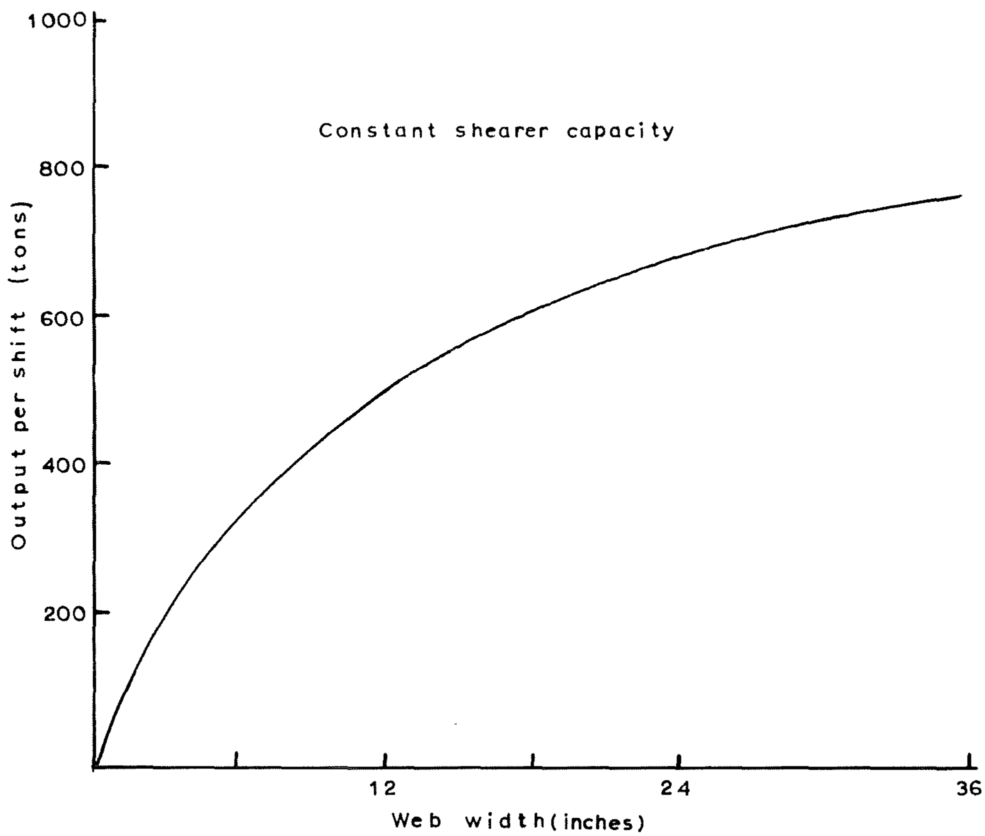
Wall length	960 ft
Headgate	25 ft
Shuffle distance	90 ft
Seam height	33 in
Web thickness	36 in
Coal density	84.24 lb/ft^3
Delays per shear	30 min
Shift time	300 min

British Mining Consultants Ltd London

Engineer.............. J.H.C.	Date................ May 80
Traced by............ J. M.E	Drg No............ 710/33

PLATE 34

LONGWALL
EFFECT OF WEB THICKNESS
ON OUTPUT PER SHIFT



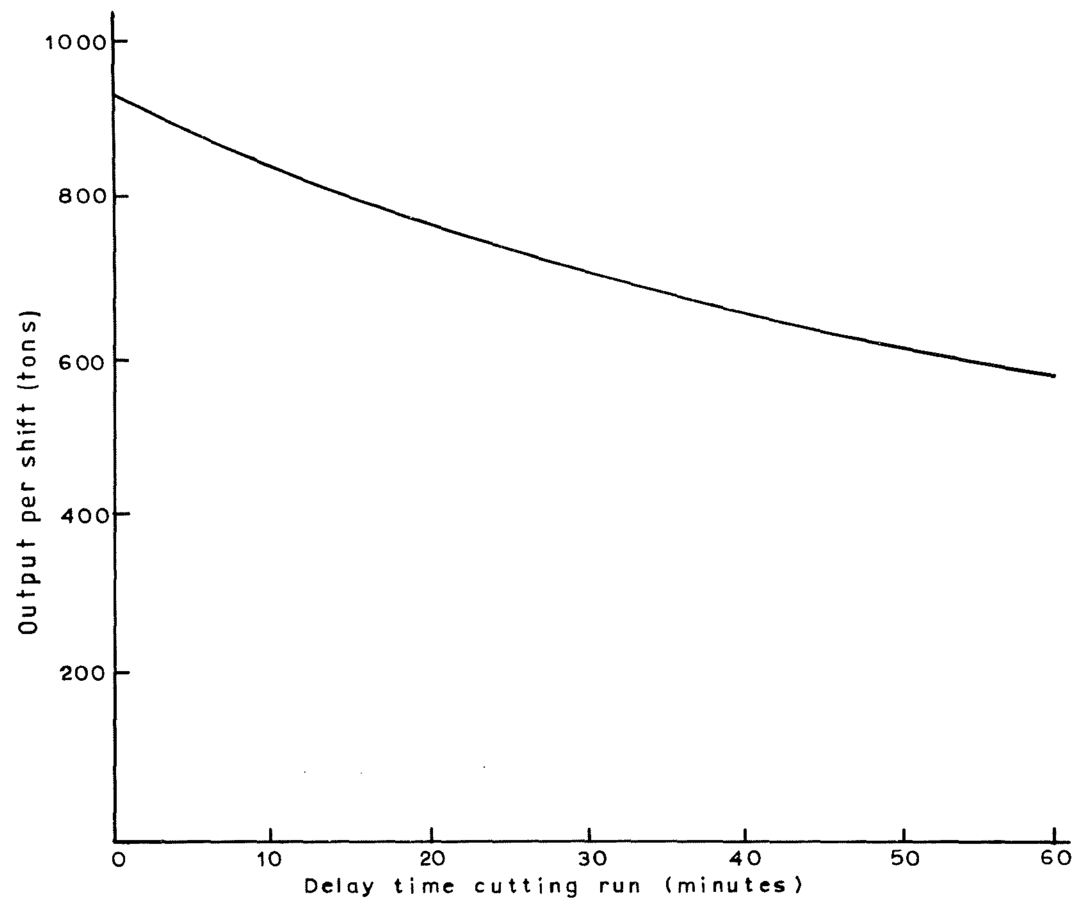
Wall length	960 ft
Headgate	25 ft
Shuffle distance	90 ft
Seam height	33 ins
Coal density	84·24 lb/ft^3
Haulage speed	11 ft/min
Delays per shear	30 min
Shift time	300 min

British Mining Consultants Ltd London

Engineer...... J H C	Date...... June 80
Traced by...... J W	Drg No...... 710 / 34

PLATE 35

LONGWALL
EFFECT OF DELAYS DURING
CUTTING ON OUTPUT PER SHIFT



Wall length	960 ft
Headgate	25 ft
Shuffle distance	90 ft
Seam height	33 in
Web thickness	36 in
Coal density	84·24 lb/ft^3
Haulage speed	11 ft/min
Shift time	300 min

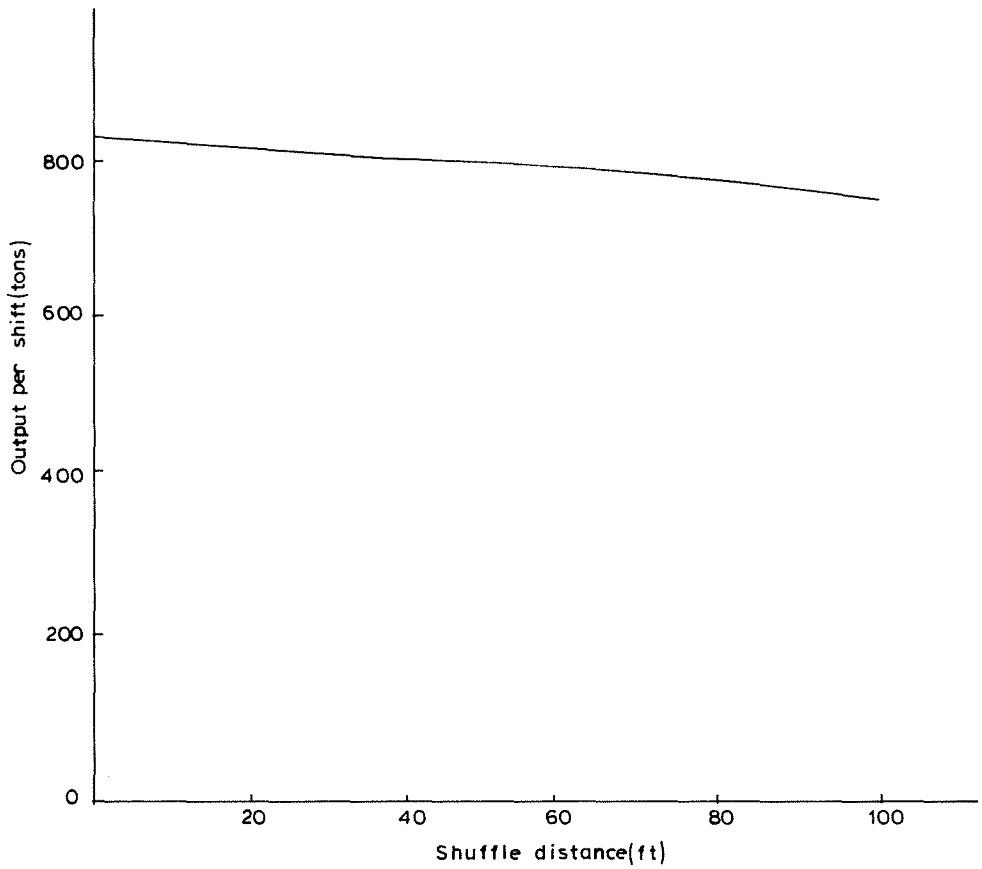
British Mining Consultants Ltd LONDON

Engineer...... J. H. C	Date...... June 80
Traced by...... J. W.	Drg No...... 710/35

PLATE36

LONGWALL

EFFECT OF SHUFFLE DISTANCE ON OUTPUT PER SHIFT



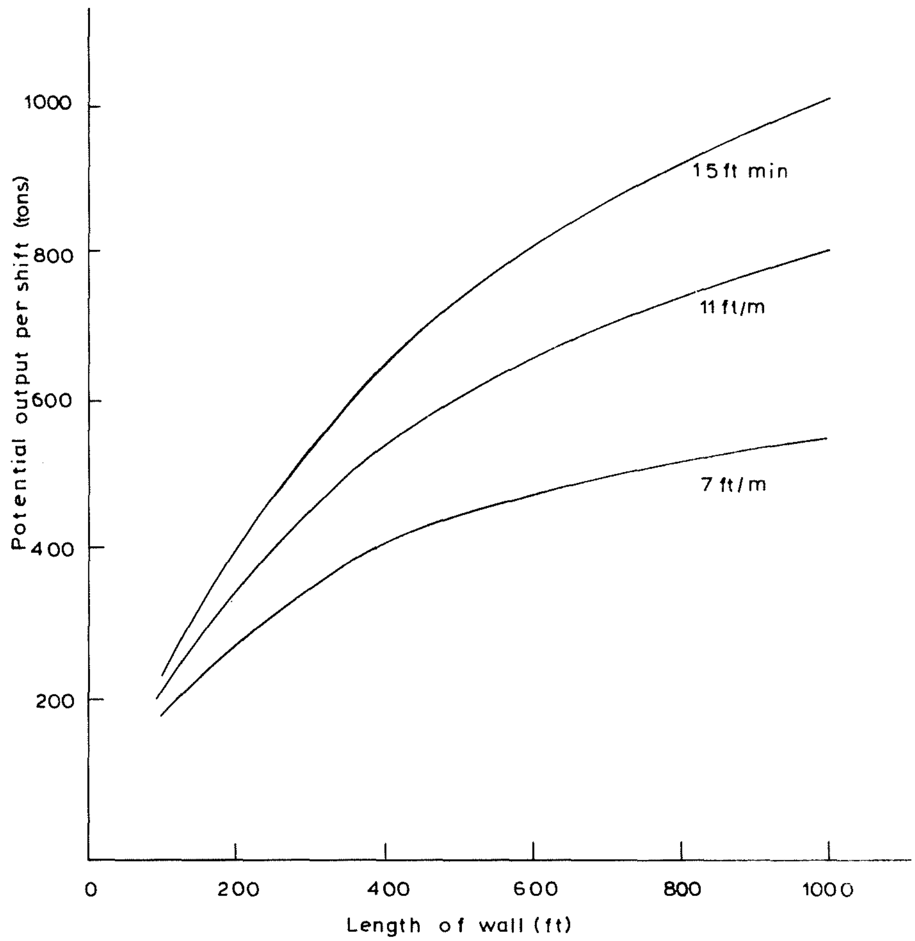
Wall length	960ft
Headgate	25ft
Seam height	33in
Web thickness	36in
Coal density	84.24 lb/ft^3
Haulage speed	11 ft / min
Delays per shear	30 min
Shift time	300min

British Mining Consultants Ltd London

Engineer.........J.H.C.	Date...........May 80
Traced by.......J.M.E	Drg No.........710/36

PLATE 37

LONGWALL
EFFECT OF FACE LENGTH ON OUTPUT



Headgate	25 ft
Shuffle distance	90 ft
Seam height	33"
Web thickness	36 in
Coal density	84.24 lb/ft^3
Haulage speed	7, 11, & 15 ft/min
Delays per shear	30 min
Shift time	300 min

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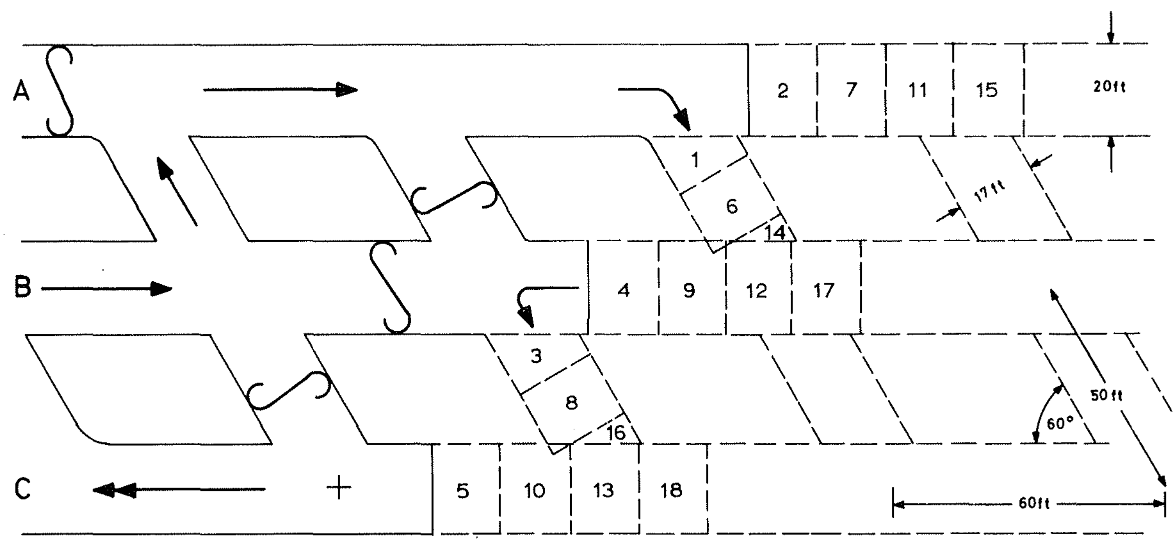
Engineer........... J.H.C.	Date........... July 80
Traced by.......... J.M.E	Drg No........ 710/ 37

PLATE 38

SHORTWALL DEVELOPMENT
CUTTING AND SUPPORT SEQUENCE



LEGEND

Neutral conveyor split, acts as headgate .... A
Intake .... B
Return, acts as tailgate on completion of development C
Cutting sequence .... 1,2 ... 18
Temporary brattice curtain ....

NOTE : Ventilation shown as when cuts 1-18 completed.

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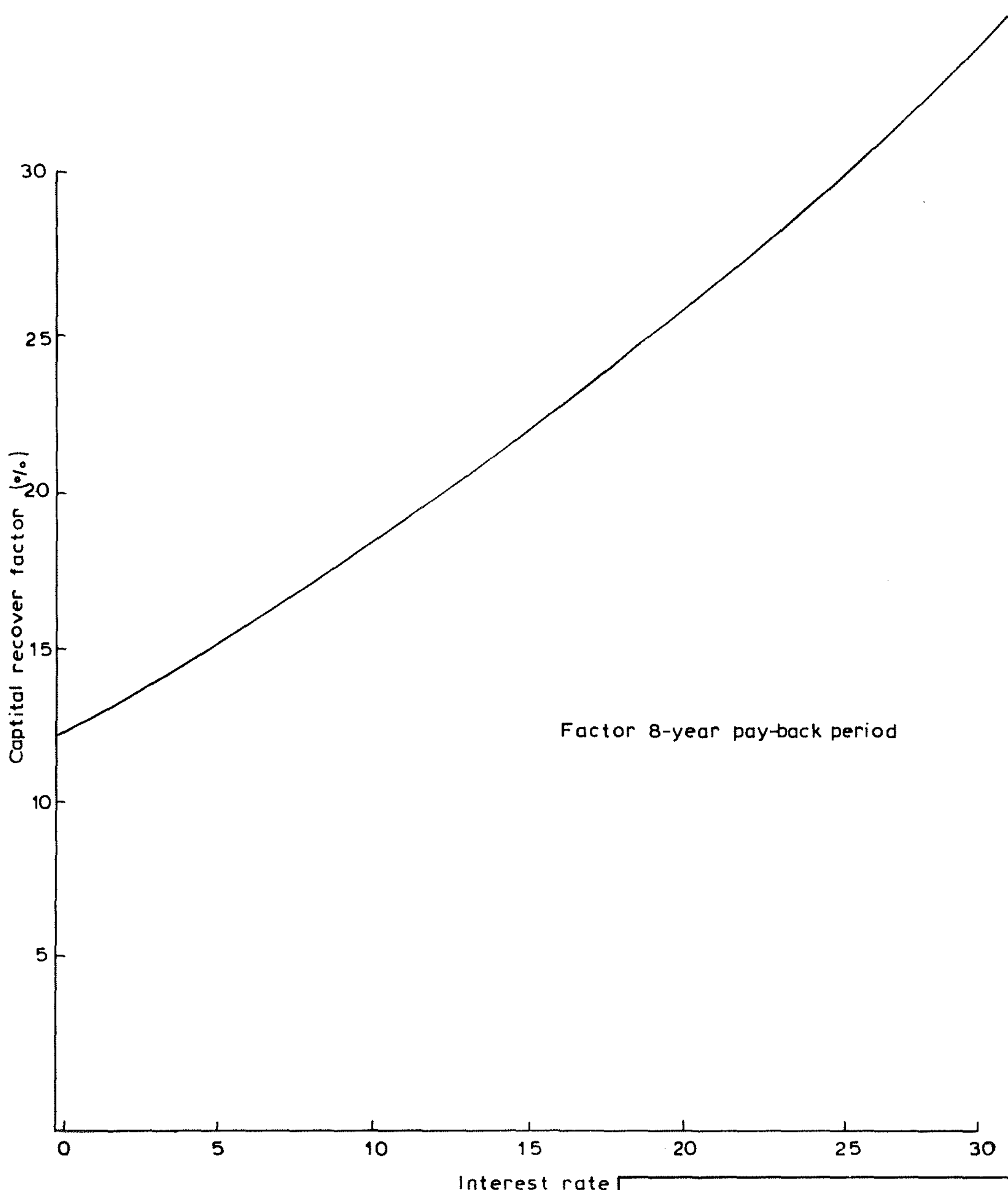
Engineer.........J.H.C.	Date..........June 80
Traced................J.S.	Drg. No........710/38

PLATE 39

CAPITAL RECOVERY FACTOR
EFFECT OF VARIATION OF INTEREST RATE
8-YEAR PERIOD



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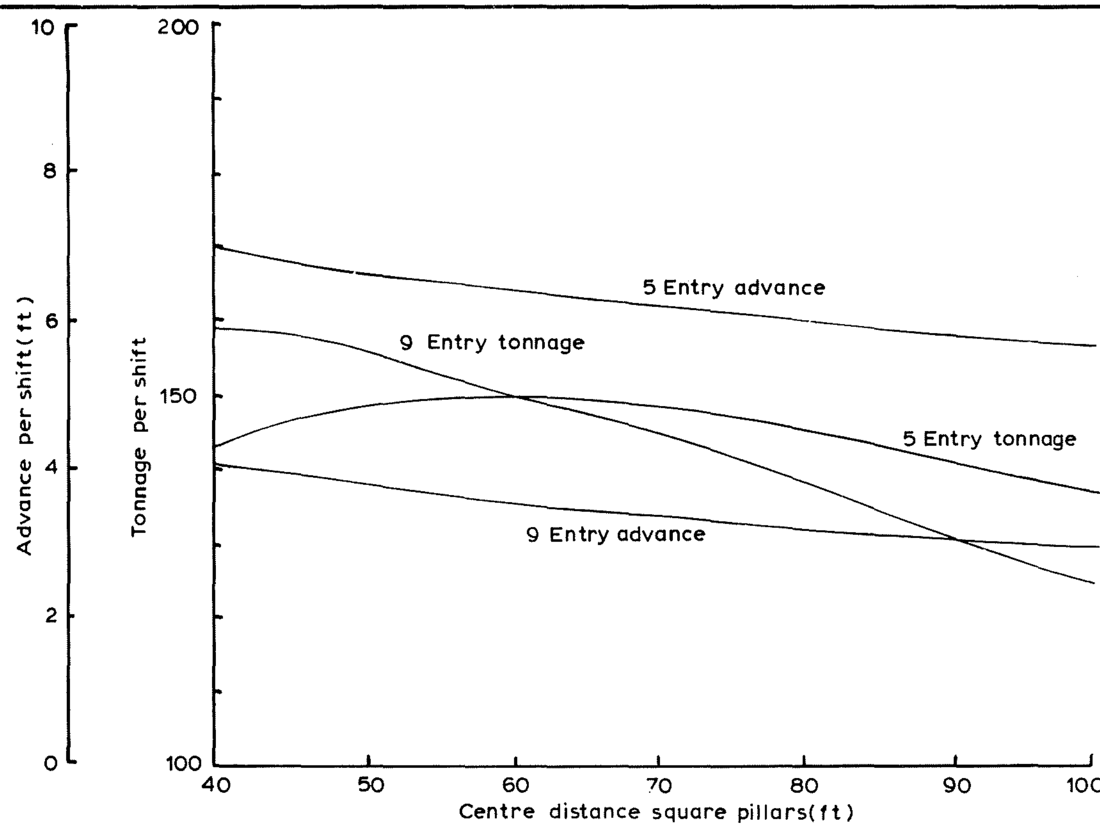
Engineer........ J. H.C. Date........May 80
Traced by....... J. M.E. Drg No.....710/39

PLATE 40

ROOM AND PILLAR
SHUTTLE CAR HAULAGE CONTINUOUS MINER WITH INTEGRAL BOLTER
EFFECT OF VARIATION IN CENTRE DISTANCE



Continuous miner average capacity	2 tons / minute
Continuous miner tram speed	30 ft / minute
Support distance	5 ft
Time delay to bolt	15 minutes
Shuttle car average speed	250 ft / minute
Shuttle car factor	3 tons / car
Car discharge time	1 minute
Centre distances	variable
Available shift time	300 minutes
Seam thickness	33 inches
Entry width	20 ft
Coal density	84·24 lb / ft^3

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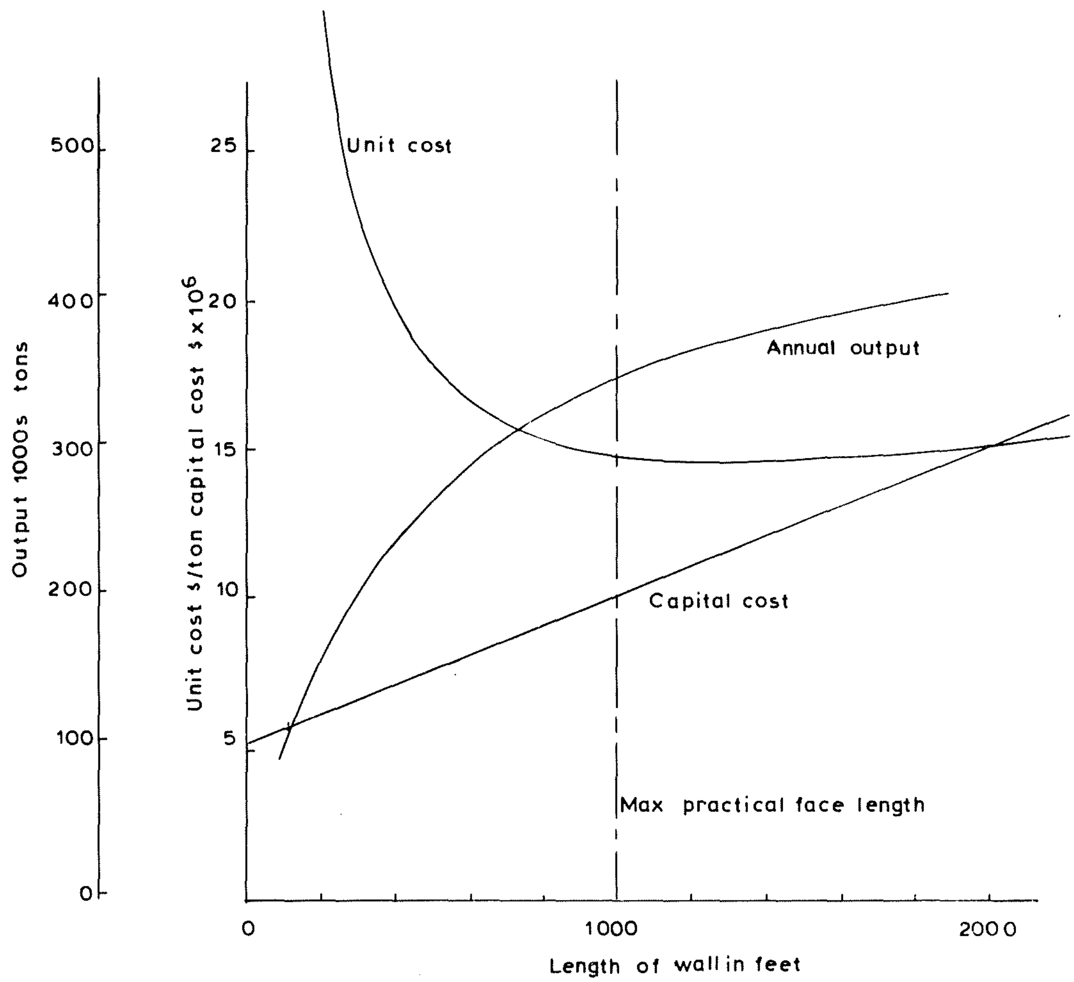
Engineer....... J.H.C.	Date......... June 80
Traced........ E.W.	Drg. No.....710/40

PLATE 41

LONGWALL FINANCIAL MODEL
EFFECT OF VARIATION IN WALL LENGTH



Note:
1000 ft probably represents the longest face that is feasible with present-day conveyor technology

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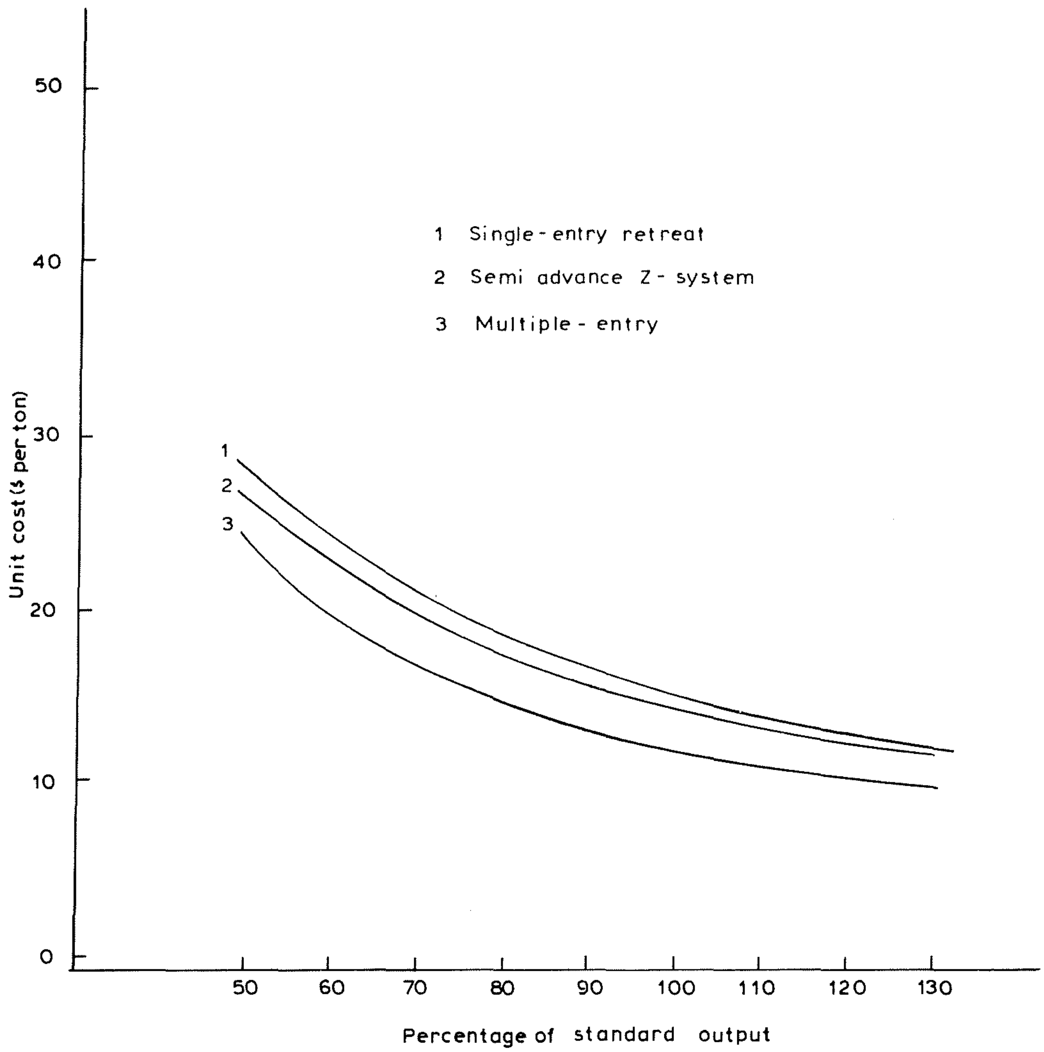
Engineer............J.H.C.	Date........June 80
Traced by..........J.M.E	Drg No.....710/ 41

PLATE 42

LONGWALL

EFFECT OF VARIATION OF OUTPUT ON UNIT COST



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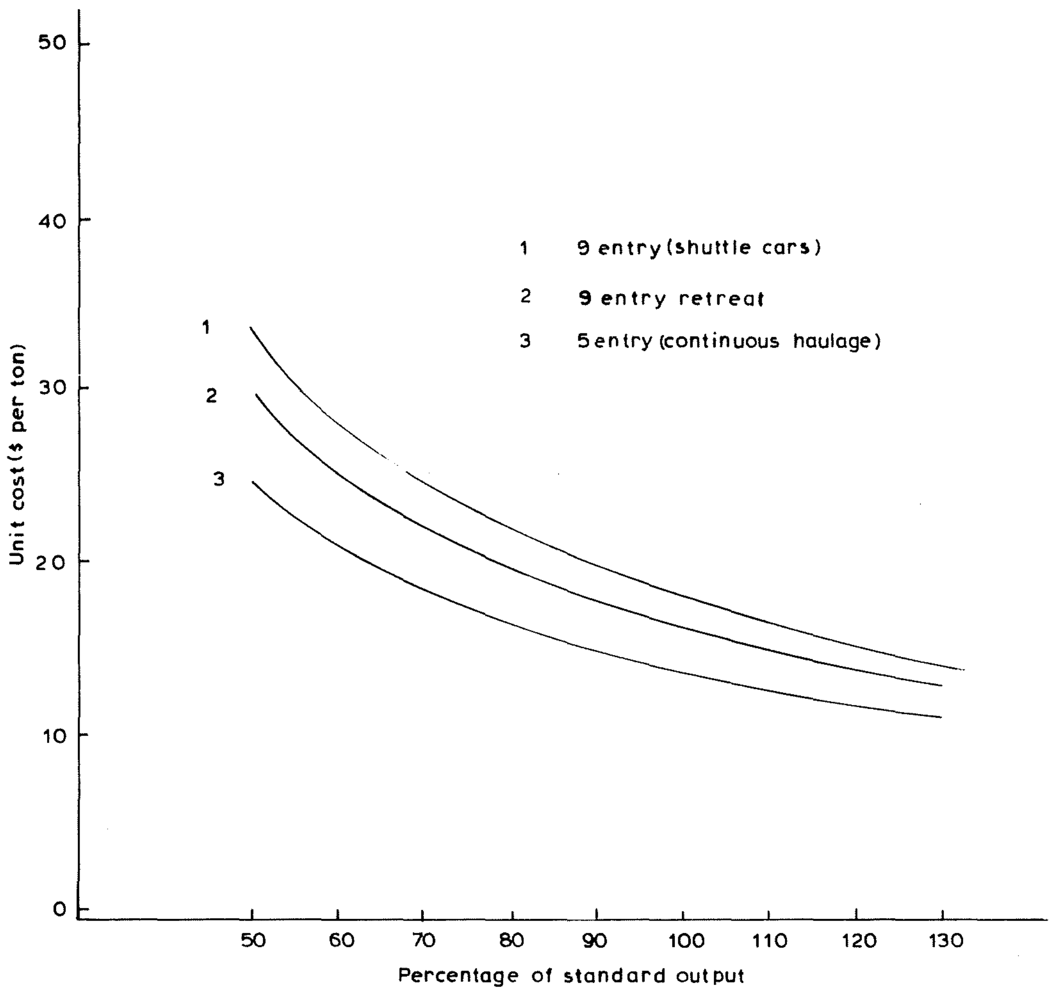
Engineer.........JHC	Date..........May 80
Traced by.....JME	Drg No......710/42

PLATE 43

ROOM AND PILLAR
EFFECT OF VARIATION OF OUTPUT ON UNIT COST

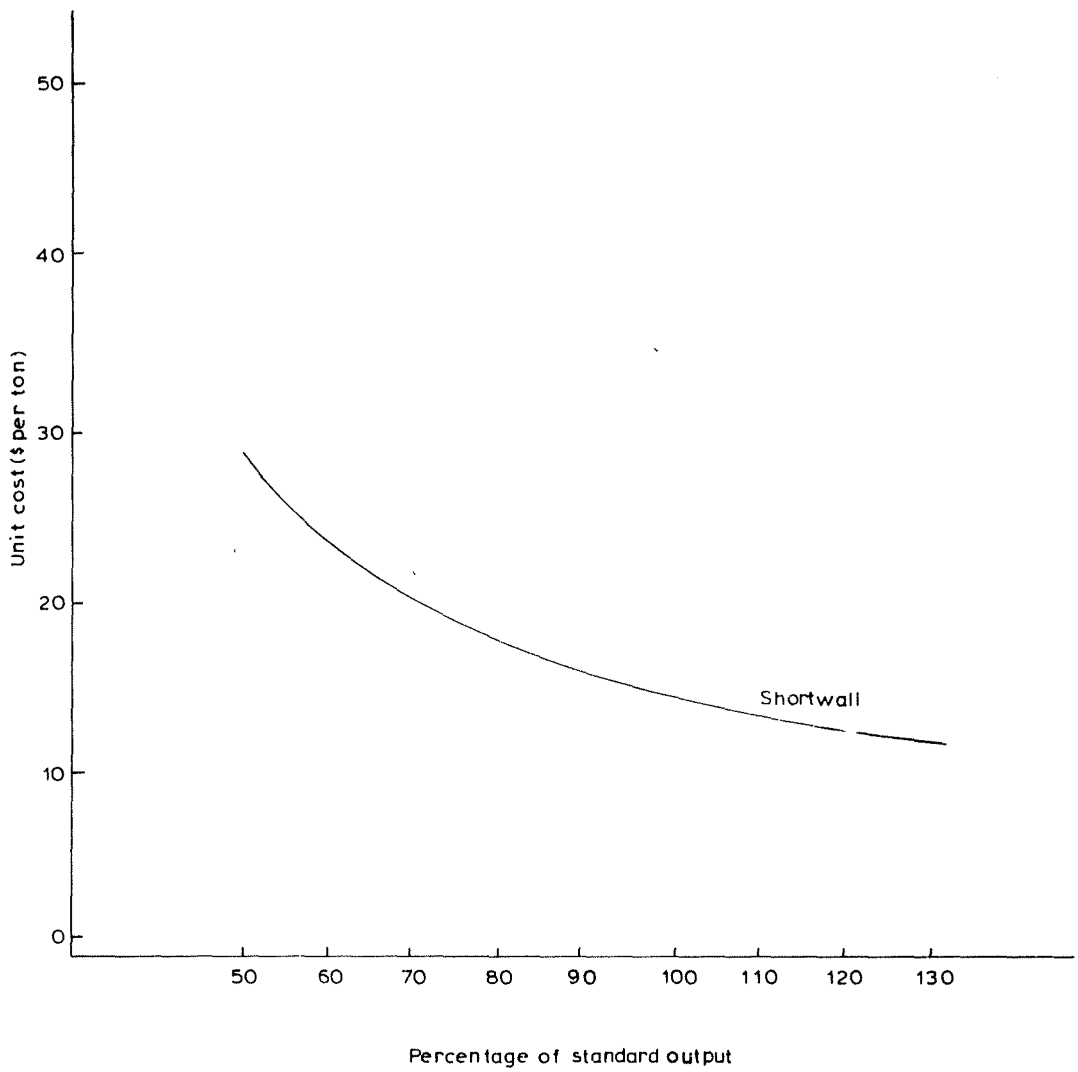


British Mining Consultants Ltd London	
Engineer........J.H.C Traced by........J.M.E.	Date.......June 80 Drg. No....710/43

PLATE 44

SHORTWALL

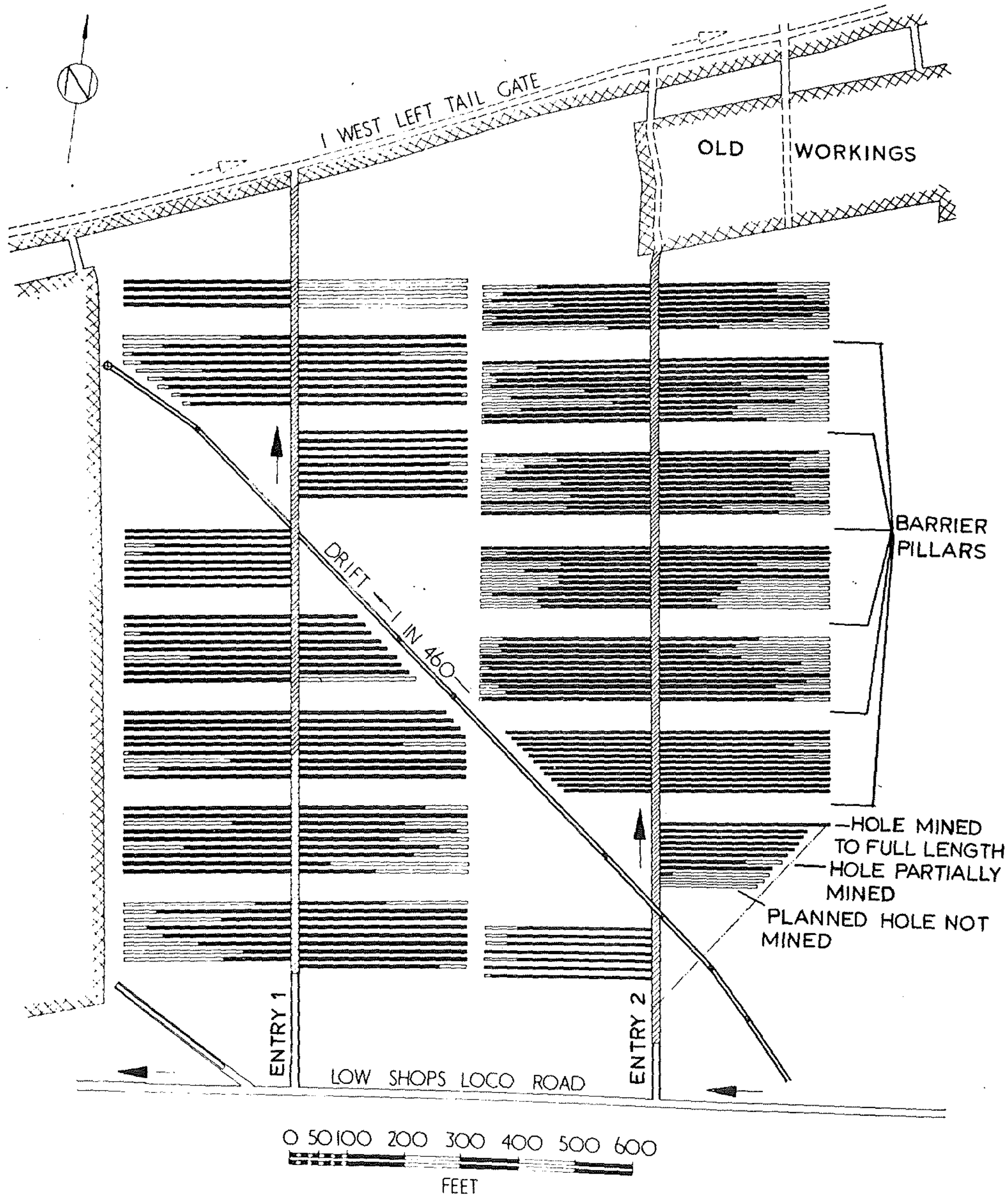
EFFECT OF VARIATION OF OUTPUT ON UNIT COST



British Mining Consultants Ltd London	
Engineer........J.H.C.	Date.......June 80
Traced by .....J ME	Drg No.....710/ 44

PLATE 45

COLLINS MINER
EXTRACTION LAYOUT



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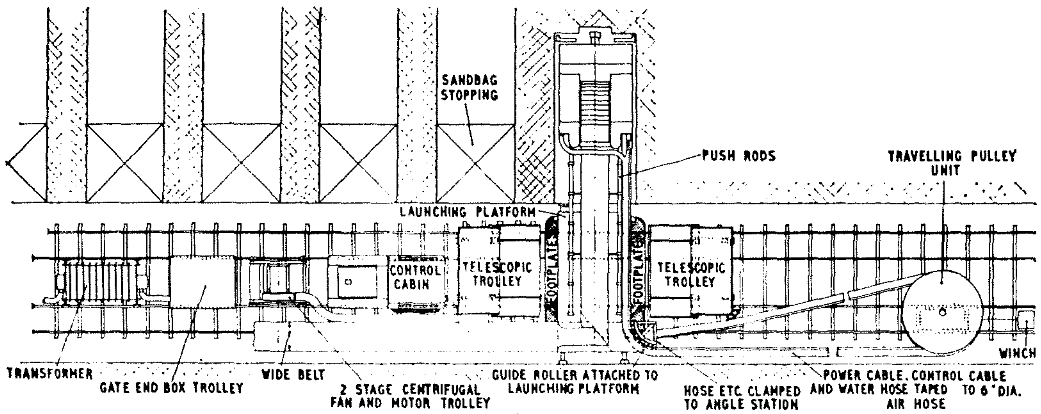
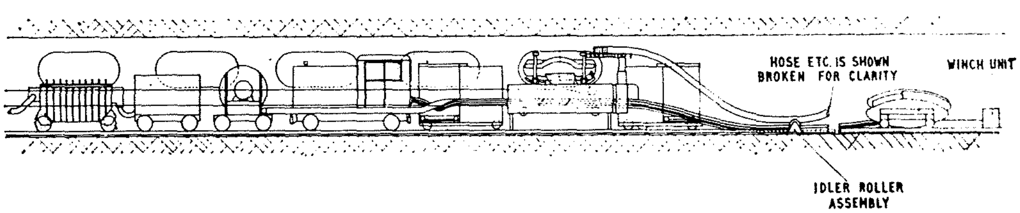
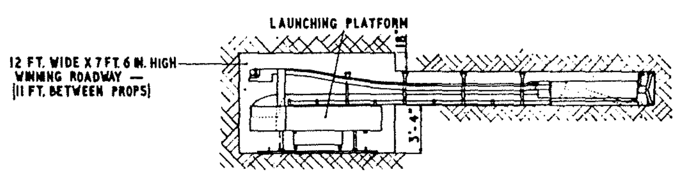
Engineer...... J.H.C.	Date........June 80
Traced....... E.W.	Drg. No.... 710/45

PLATE 46

COLLINS MINER
GENERAL ARRANGEMENT







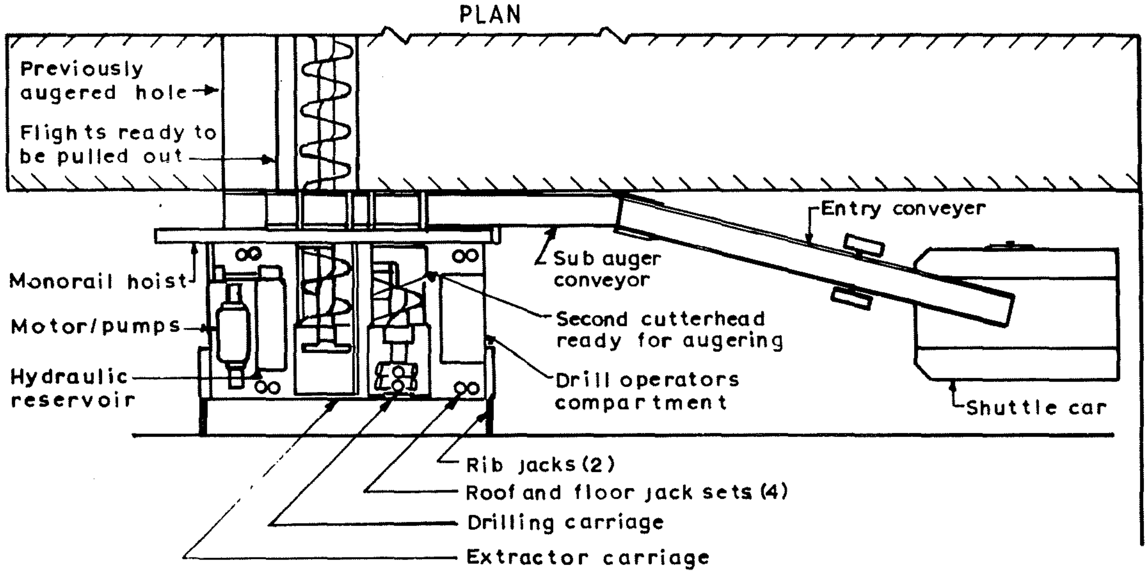
British Mining Consultants Ltd. London

Engineer...... J.H.C.	Date........ June 80
Traced....... E.W.	Drg. No.... 710/46

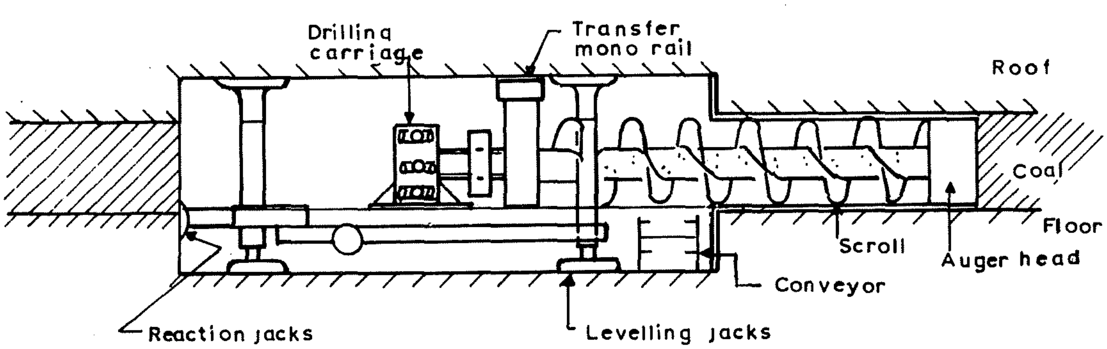
PLATE47

AUGER
GENERAL ARRANGEMENT
PLAN AND SECTION

PLAN



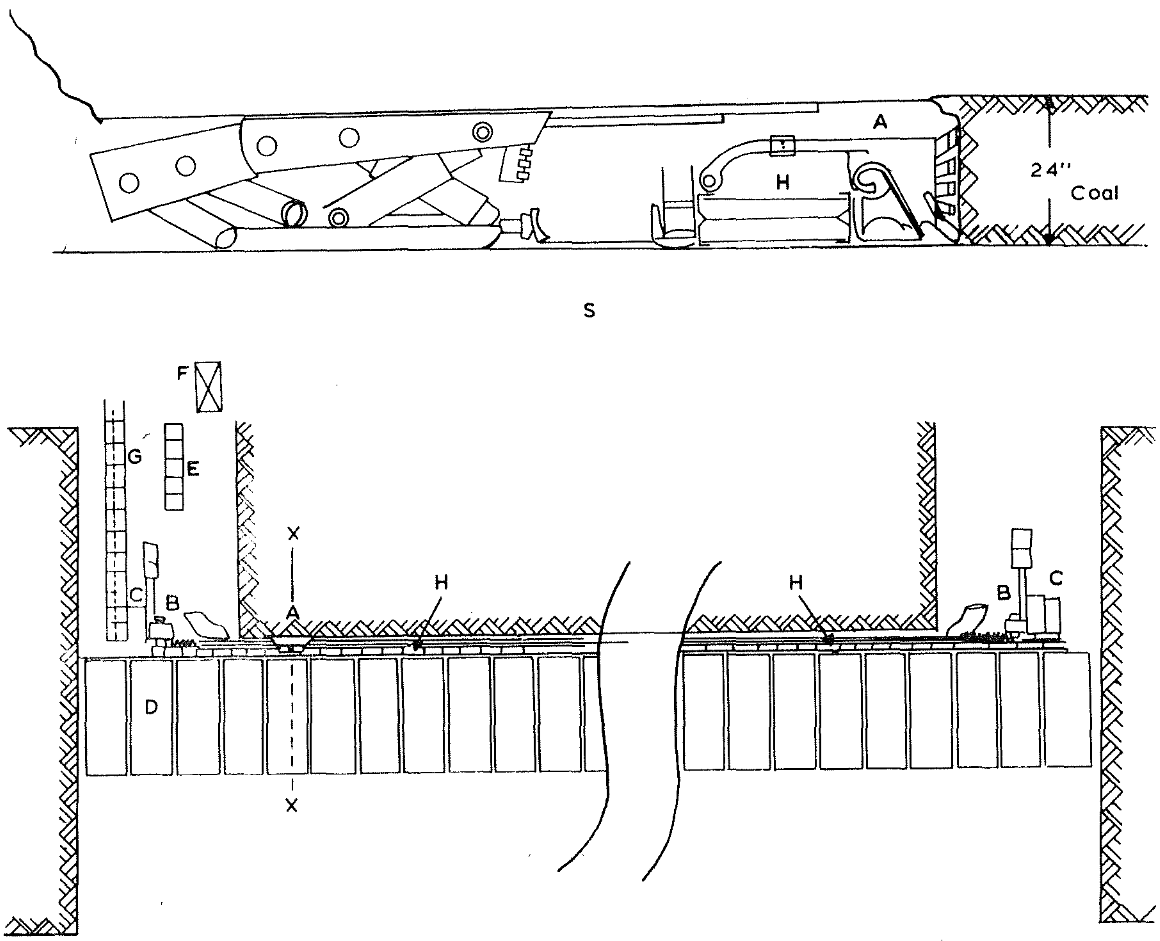
SECTION SHOWING REQUIREMENT FOR ADDITIONAL HEIGHT IN ORDER TO AUGER FULL SEAM THICKNESS



British Mining Consultants Ltd London	
Engineer...... J H C	Date....... June 80
Traced by....... J W	Drg No....... 710/47

PLATE 48

LONGWALL PLOUGH
SUGGESTED ARRANGEMENT USING POWERED SUPPORTS IN
A 20-IN TO 24-IN SEAM



LEGEND
A..................Plough
B..................Plough drives
C..................Conveyor drive
D..................Powered supports
E..................Switch gear
F..................Hydraulic pump
G..................Stage loader
H..................Face conveyor

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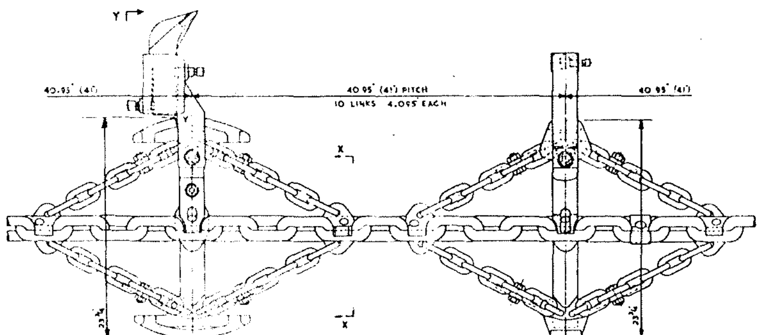
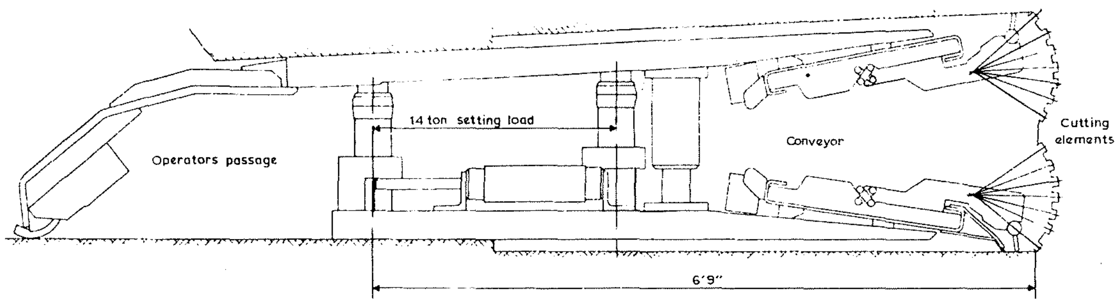
Engineer........ J.H.C.	Date.......June 80
Traced............J.M.E.	Drg No....710 / 48

PLATE 49

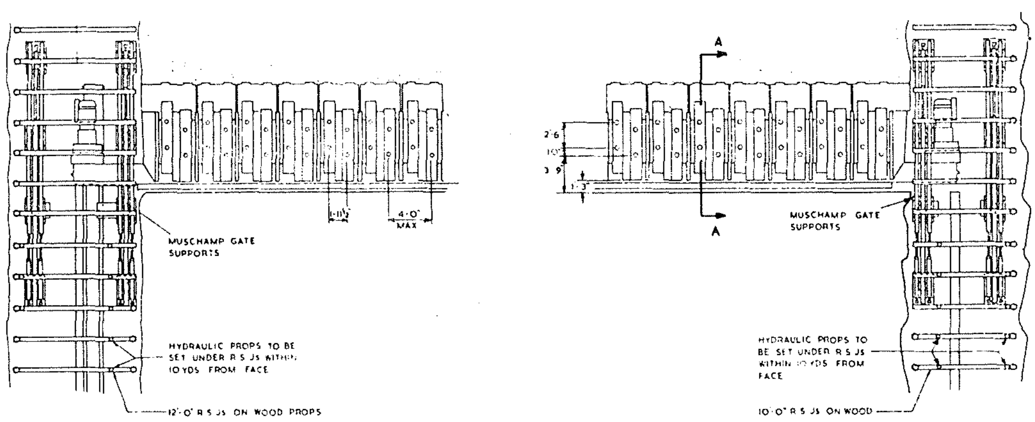
YARMAK MINER
GENERAL ARRANGEMENT



Cutting and conveyor elements



Cross section A-A



Face arrangement

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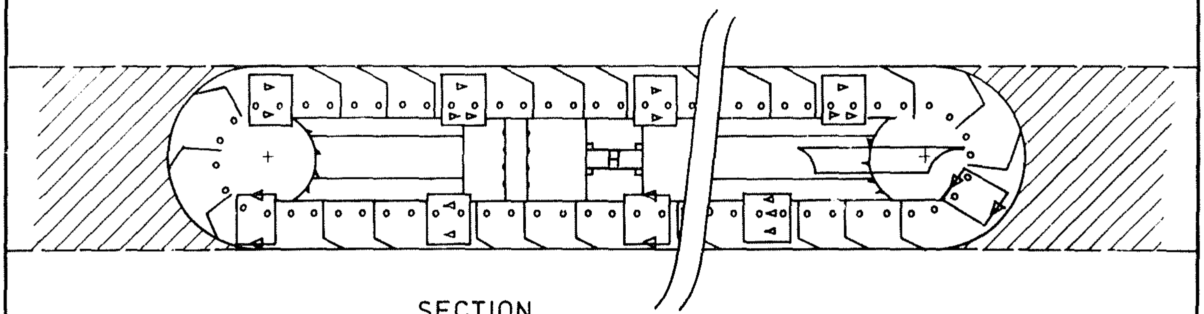
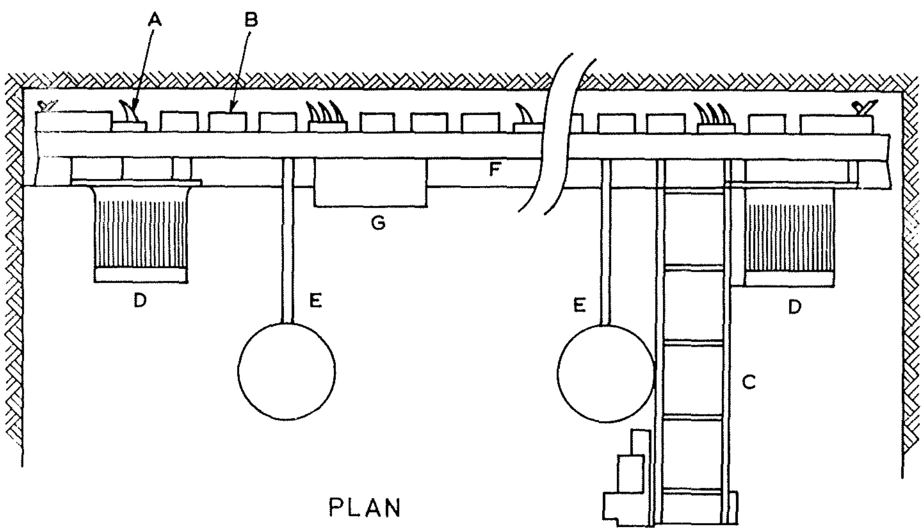
Engineer...... J.H.C.	Date........ June 80
Traced....... E.W.	Drg. No.... 710/49

PLATE 50

IN-SEAM MINER
GENERAL ARRANGEMENT



SECTION



PLAN

LEGEND

Cutters .... A
Conveyor scoops .... B
Discharge conveyor C
Power units .... D
(electric or hydraulic)
Thrust rams .... E
Main frame .... F
Control position .... G
Chain tension unit .... H

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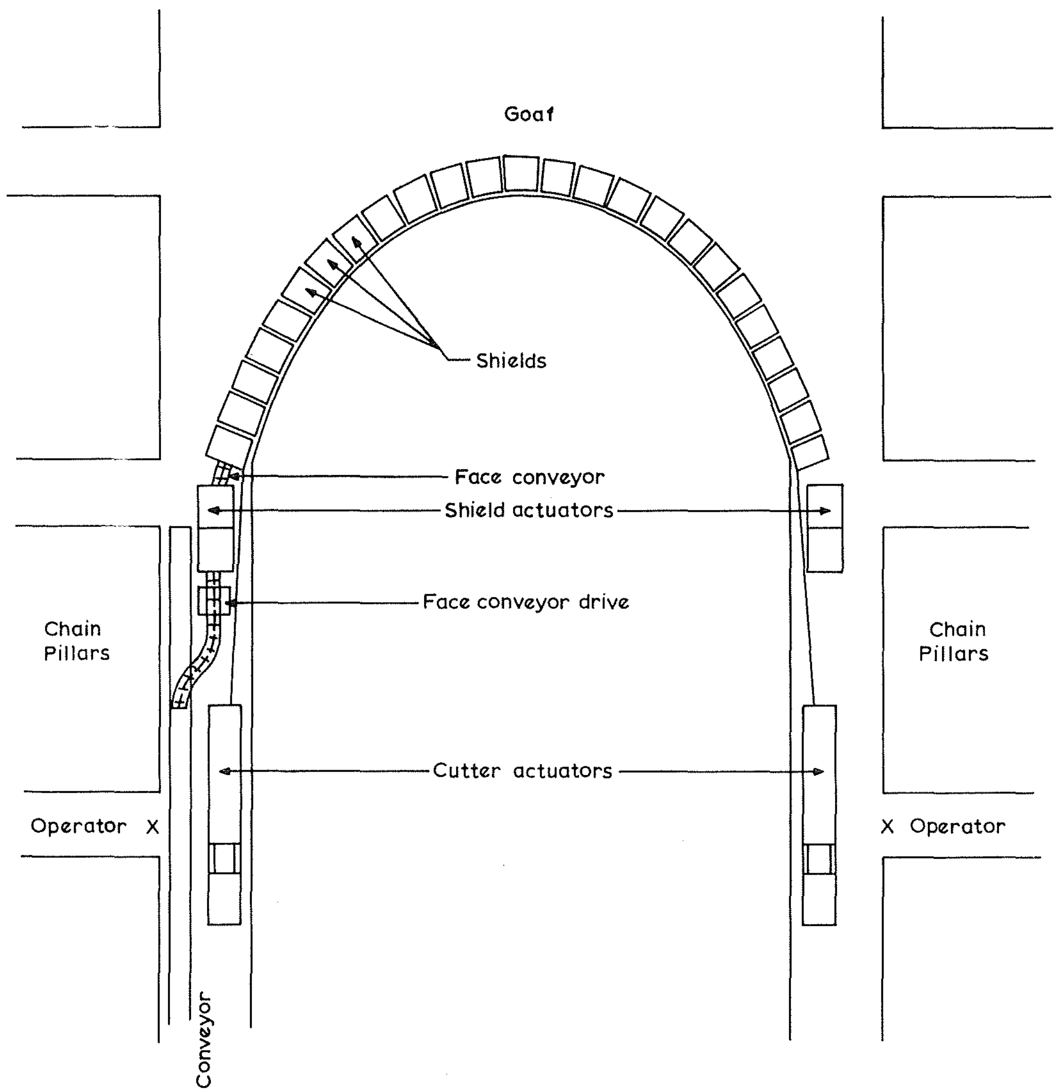
Engineer .... J.H.C.	Date .... June 80
Traced .... J. S.	Drg. No 710/50

PLATE 51

"MINI WALL"

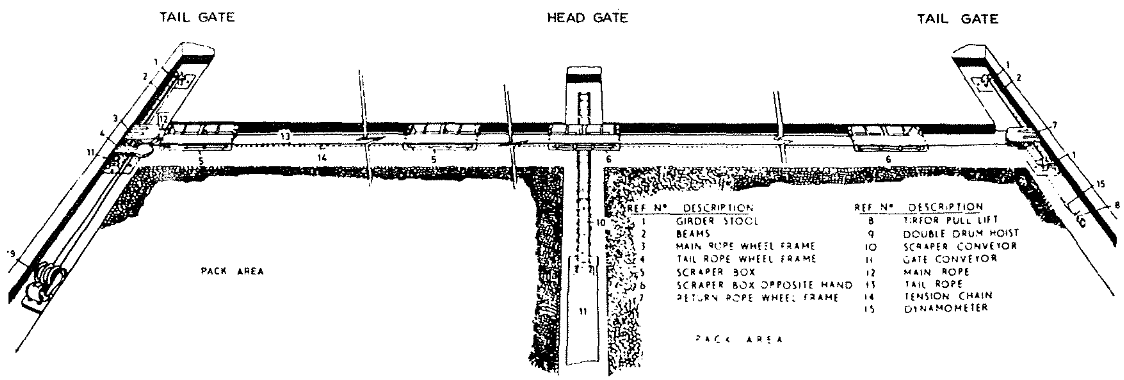
GENERAL SCHEMATIC LAYOUT



British Mining Consultants Ltd, London	
Engineer.........J.H.C.	Date.........June 80
Traced...............J.S.	Drg.No........710/51

PLATE 52

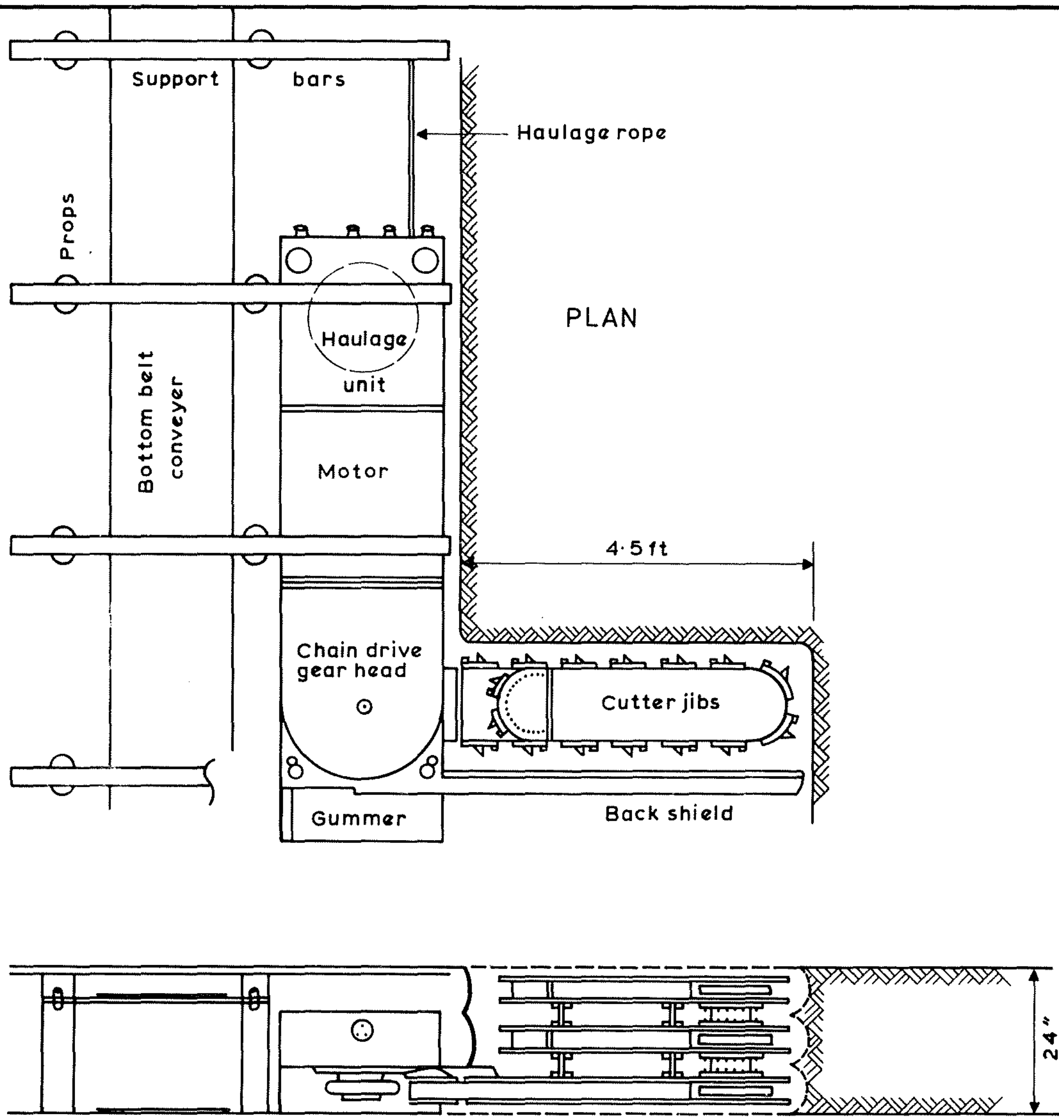
LAYOUT FOR CHAIN TENSION SCRAPER



British Mining Consultants Ltd. London	
Engineer...... J.H.C.	Date........ June 80
Traced....... E.W.	Drg. No.... 710/52

PLATE 53

MULTI-JIB CUTTER LOADER



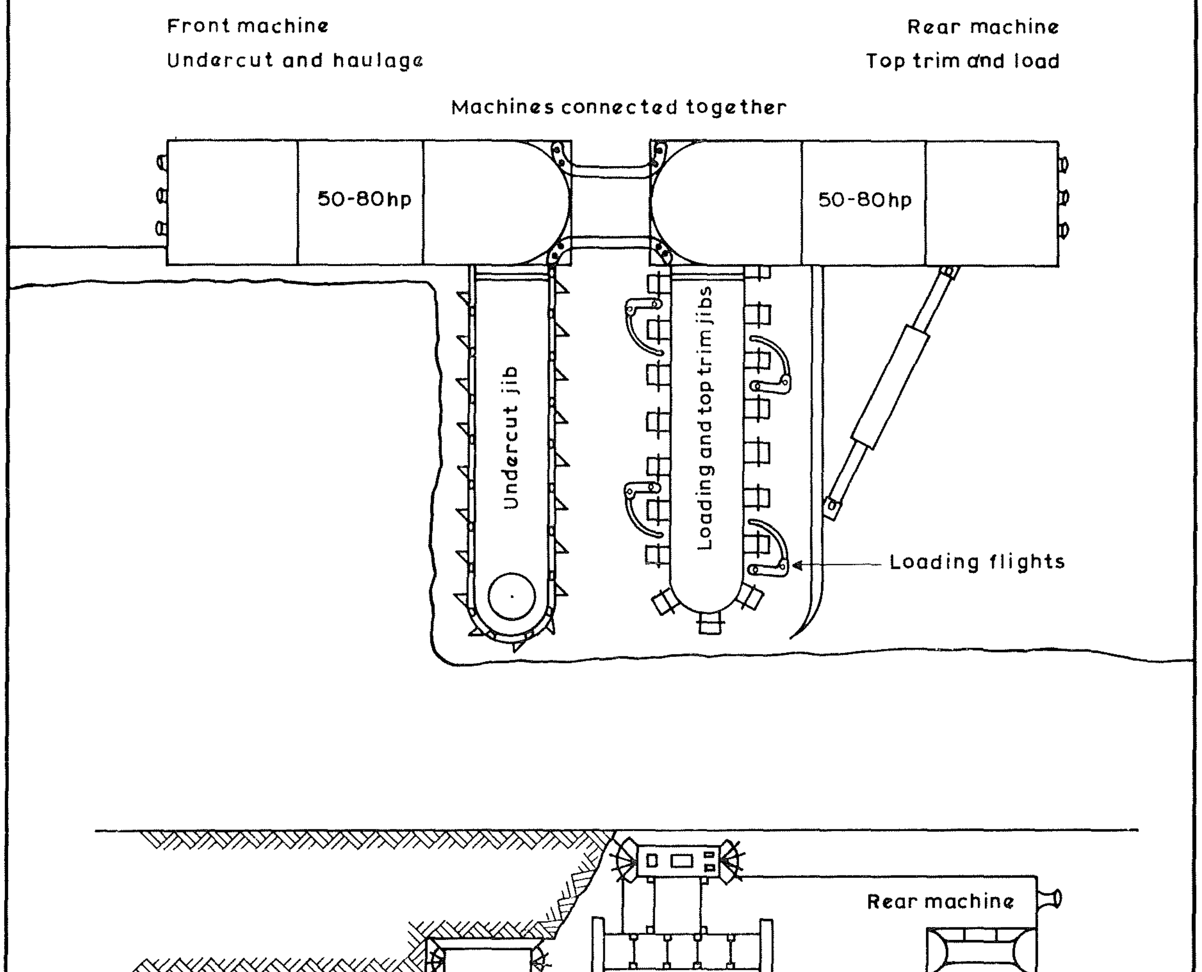
SECTION
(chains, shield and gummer removed)

British Mining Consultants Ltd London	
Engineer........ J.H.C.	Date .... June 80
Traced .... J.S.	Drg. No..... 710/ 53

PLATE 54

BACK-TO-BACK CUTTER LOADER

GENERAL ARRANGEMENT OF MACHINES



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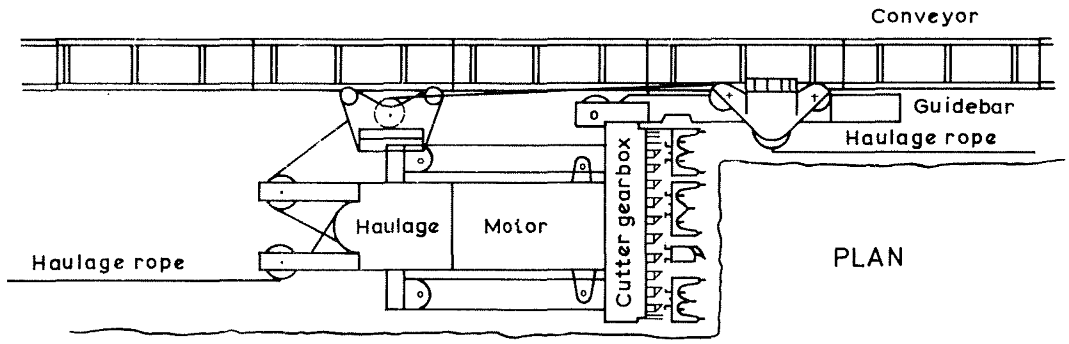
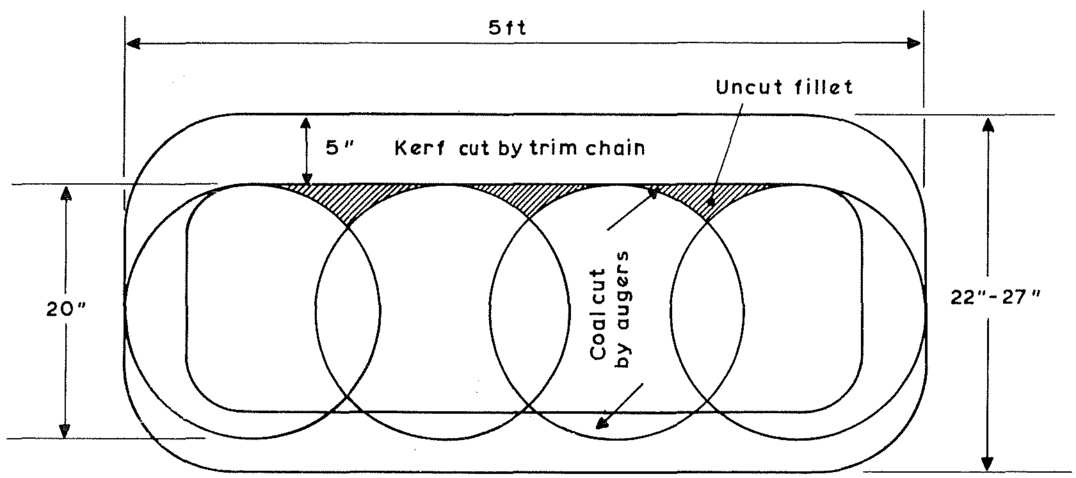
Engineer......... JHC	Date......... June 80
Traced .... J.S.	Drg. No......710/54

PLATE 55

MIDGET MINER

GENERAL ARRANGEMENT OF MINER, GUIDEBAR AND CONVEYOR





British Mining Consultants Ltd, London	
Engineer..........JHC	Date .... June 80
Traced .... J. S.	Drg. No...: 710/55