Papers Presented Before The

THIRD SYMPOSIUM on UNDERGROUND MINING

NCA/BCR Coal Conference and Expo IV

October 18-19-20, 1977

Kentucky Fair and Exposition Center

Louisville, Kentucky

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With Participation by the

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Zane Murphy, U.S. Bureau of Mines

PREFACE

The following pages contain the papers presented before the Third Symposium on Underground Mining during sessions on mine planning and development, health and safety, maintenance and improved efficiency, automation, longwall and shortwall mining, and illumination. These papers were selected by the Program Committee to detail the most current work being conducted by some of the leading scientists, researchers, and technical representatives of government and industry. The content of the papers has not been edited, and the views expressed are entirely those of the authors.

The procedure of preprinting obviously places a great burden on the authors, id we are indebted to them for adhering to the strict publication timetable that id to be established. In those few cases where the authors found this timetable impossible to meet, the full text of the papers will be available as handouts at the Symposium or can be obtained by contacting the author.

The objective of the NCA/BCR Coal Conference and Expo IV is to publicize, discuss, and disseminate new information on all aspects of coal--from mining to utilization. These proceedings therefore have not been copyrighted; however, it should be noted that many of the processes described herein have been patented.

In addition to the National Coal Association and Bituminous Coal Research, Inc., this Conference was presented in cooperation with the International Committee for Coal Research, Federal Power Commission, Kentucky Coal Association, Tennessee Valley Authority, Federal Energy Administration, U.S. Department of Agriculture, U.S. Department of the Interior, Appalachian Regional Commission, U.S. Environmental Protection Agency, Energy Research and Development Administration, Coal Industry Advisory Committee on Water Quality, Council for Surface Mining and Reclamation Research in Appalachia, and The Coal Association of Canada. We gratefully acknowledge the assistance of the members of these cooperating organizations who served on the Program Committee and helped arrange the details of the Symposium.

The Third Symposium on Underground Mining is an integral part of the 1977 NCA/BCR Coal Conference and Expo IV. Other Symposiums included in this Conference are the Third Symposium on Coal Management Techniques, the Fourth Symposium on Coal Utilization, the Third Symposium on Coal Preparation, the Seventh Symposium on Coal Mine Drainage Research, and the Fifth Symposium on Surface Mining and Reclamation. Copies of the proceedings of each of the Symposiums can be purchased from National Coal Association, The Coal Building, 1130 Seventeenth Street, N.W., Washington, D.C. 20036. Instructions for ordering these publications, and the proceedings of the technical Symposiums conducted in the past, can be found on the inside back cover.

NCA/BCR

PROGRAM

THIRD SYMPOSIUM

on

UNDERGROUND MINING

October 18, 1977

SESSION TITLE: MINE PLANNING AND DEVELOPMENT
Papers 1 to 5

SESSION TITLE: HEALTH AND SAFETY
Papers 6 to 10

October 19, 1977

SESSION TITLE: MAINTENANCE AND IMPROVED EFFICIENCY
Papers 11 to 14

SESSION TITLE: AUTOMATION
Papers 15 to 19

October 20, 1977

SESSION TITLE: LONGWALL/SHORTWALL
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* Paper not available for publication

DEVELOPMENT OF AN IN-SEAM MINER

Kenneth E. Hay, Civil Engineer
Spokane Mining Research Center
Bureau of Mines
Spokane, Washington

and

Jack B. McNicol, Director
Mining Operations
Semet-Solvay Division of Allied Chemical Corporation
Beckley, West Virginia

INTRODUCTION

In mid-1968, the Control Engineering Establishment of the National Coal Board (NCB) in England first proposed the development of a machine aimed particularly at the drivage of low-height headings and stables in the thinner seams (3 to 4 feet). This machine orginally called a "heading stablehole machine" proved both reliable and efficient. As a result, the machine found wide acceptance in longwall face development work and to a lesser extent in the drivage of retreat in-seam roadways. With these applications came a new name--"in-seam heading machine." Subsequent development work expanded the machines capabilities; the applications grew tremendously; and the machine was given yet another name--"in-seam miner."

Because of several features, the in-seam miner (ISM) exhibits promise for U.S. mining applications as well. The configuration allows installation of roof supports within 3 feet of the face, making it compatible with continuous forward advancement. The machine employs simple, straightforward design and robust components providing high reliability and good maintainability. Low pick speeds and constant pick engagement are used, resulting in low dust and noise emissions, and giving the unit a rock cutting capability superior to drum-type miners. This machine can cut a wide entry (32 feet is common, over 60 is possible in theory) with a flat roof and floor making it compatible with "single entry" techniques such as divider walls.

Background

Although conceived at the NCB's Central Engineering Establishment, the development was conducted by the Mining Research and Development Establishment (MRDE). MRDE designed, manufactured and tested prototypes. Following successful surface and underground trials, the NCB offered the machine for explotation to mining machinery firms. Dosco Overseas Engineering Ltd. became suppliers to the NCB and have co-operated with MRDE during the continuing development. To date over 120 of these machines are in service.

Although the British have many of these machines in service, few are applied to situations where the excavation potential can be evaluated, and those that are, are relatively low horsepower hydraulic machines. The In-Seam Miner in use today employs two 30-hp Staffa motors to drive the cutterhead. The relatively low horsepower chosen was primarily the result of the undemanding initial application in terms of excavation rate and the size limitation on the hydraulic power pack. As new applications were developed, it was recognized that higher horsepower units could be utilized. The result was the development of several prototype electric-powered in-seam miners.

The Bureau obtained one of the higher horsepower electric prototypes to evaluate the potential of this design approach, and depending on the potential exhibited, to determine what changes would be required to optimize performance under U.S. mining conditions.

Initial equipment trials were conducted in 1975 at Kaiser Steel's Sunnyside mine. Although instantaneous excavation rates of 15 feet per hour were realized, some problems were encountered with the machine. As a result, it was decided to make a number of design improvements prior to subjecting the unit to more extensive testing. In a combined effort, the National Coal Board and Dosco worked out the design details and built a new unit for the Bureau of Mines. This machine is presently in the U.S. undergoing underground trials. It is one of only two machines in existance powered by two 65-hp electric motors.

Machine Description

The ISM is based on a simple but unique arrangement of cutting picks mounted on pick plates (fig. 1). These pick plates are attached to and moved by a strap-link chain. Loading buckets are similarly attached to the same chain and transport the cut coal to a discharge conveyor via a chute mounted on the cutter jib (fig. 2). The discharge conveyor can be mounted on either side of the cutting head. The cutterhead is moved forward by hydraulic cylinders pushing against two staker chocks set between the roof and floor (fig. 3).

The equipment can be dismantled, transported, and reassembled in 5 to 10 shifts depending on crew experience and underground conditions. If needed, the machine can be broken down into components of approximately 1-ton maximum weight.

Present Configuration

As mentioned, the ISM presently undergoing testing has two 65-hp electric motors powering the cutter chain. All other machine components are hydraulic.

The cutting height is 56 inches and the width is 18 feet, 6 inches. The height range is 38 to 56 inches, the 52- to 56-inch change involving only a pick spacing adjustment. The width can be 18.5, 24.0, or 26.0 feet. Changing to any of these widths would require approximately three shifts. On the original machine, the control station was attached to the main frame. However, to allow for roof-support installation forward of the operator, the control station was removed and mounted on a skid. The skid is connected to the main frame by a chain which tows it approximately 10 to 12 feet behind the face. The ISM chain conveyor was extended from 27 feet to 57 feet to allow more room for face operations.

Other Configurations

Machines are available in the 54- to 70-inch height range; equipment trials are presently being conducted by the NCB on a unit with a 34-inch design capability, and under a Bureau of Mines Cooperative Agreement with the NCB, a machine with a 28-inch capability is being developed. As mentioned, the machine has been used successfully at widths up to 32 feet, and wider configurations are possible.

Instrumentation Plan

An instrumentation plan has been set up to aid in evaluating performance, trouble-shooting, and defining areas of improvements. Two recording units have been installed to record the data from the sensors. The sensors will measure pressures, flows temperature, and vibrations. Wear on the system, and environmental data such as noise and dust levels, and air velocities and flows will also be collected.

A series of curves will be produced indicating the optimum advance rate that produces the maximum efficiency, acceptable wear rates, and acceptable vibrations. Most likely these factors will not be optimized at the same advance rate so that a range of rates will be specified. The curves will be useful in predicting the machine performance at conditions other than those that were tested, in predicting the expected life and reliability of the machine, and in defining specific maintenance items and frequency of maintenance. Recommendations will be made on system or material modifications that will improve performance, safety characteristics, and environmental conditions. These results will be included in a final report along with a predicted economic impact of using the ISM in U.S. underground coal mines.

Test Site

Allied Chemical Corporation under contract to the Bureau of Mines will be testing the in-seam miner. The test site is at their Semet-Solvay Division, Harewood mine, Harewood, West Virginia. The mine is located approximately 30 miles east of Charleston, West Virginia, just off Highway 60. Approximately 419 individuals are employed underground at this mine. Both conventional and continuous room-and-pillar mining sections are in operation.

The test area is about 2-1/2 miles in from the main portal in an area called the "Turnpike" (fig. 4). Two parallel entries are being driven, one with the ISM and the other entry and crosscuts with a continuous miner. The continuous miner entry and crosscuts will be 20 feet wide, and the ISM entry will begin at a width of 18 feet, 6 inches, and go to 26 feet halfway through the entry. The test area and mining method being used was developed specifically for the ISM trials.

Equipment Trials

Before the in-seam miner and ancillary equipment were taken underground at the Harewood mine, it was completely assembled and operated on the surface. This allowed MESA, West Virginia Department of Mines, and UMW union representatives the opportunity to view the equipment and recommend changes or additions before operations began. Some additions were made at this time to improve the machine's mining potential and safety characteristics, and meet MESA requirements. The additions were:

1. Lights: Lighting was installed on the machine and around the face area to comply with MESA regulations and provide a safer work environment.

2. Canopies: Canopies were fabricated and installed to provide protection for the machine and roof-drill operators.

3. Continuous haulage system: To minimize downtime encountered in advancing the outby conveyor, a mobile tailpiece and belt take-up unit were installed.

4. Distribution box: A power box providing circuit breakers for the lights, conveyor tailpiece, geolograph, dust collector, and instrumentation box were added.

5. Instrumentation: Two recording units are provided to collect the data from the sensors.

6. Roof drills: Two roof drills were mounted on the machine to allow for bolting within 6 feet of the face.

7. Dust collector: A dust collector unit was assembled and mounted on a skid.

The ISM was then dismantled, transported underground, and reassembled. This took approximately 10 shifts. Final inspections were then made by MESA, West Virginia Department of Mines, and the UMW representatives.

Because of complications involved in obtaining an approval for the dust collector unit, the roof drills were removed and an alternate roof support plan was submitted for approval. The alternate plan is to install aluminum bars for temporary support on meter centers within 3 feet of the face, to a distance of approximately 50 feet. A roof-bolting machine would then be used to install roof bolts on 4-foot centers in increments of 25 feet, from 50 feet to within 25 feet of the face.

It is anticipated that support activities, primarily ground control, will prevent sustained periods of rapid advancement. However, the system as presently configured should allow an evaluation of the machines performance potential. The ISM will be constantly monitored throughout the trials and areas of improvement identified. Those improvements that can be incorporated during the course of the trials, i.e., changes consistent with the underground working conditions and test schedule, will be made and the resultant impact noted. The possible impact of other identified areas of improvement beyond the scope of these trials will be considered in weighing the performance potential.

If the ISM proves to be a good basic piece of equipment, design specifications will be developed for modifications to optimize its primary function and for support systems to make it truly compatible with U.S. mining conditions.

Applications

To realize the full potential of the ISM, it must be operated in a straight-ahead mode. If present mining law is adhered to, this requires that special support systems be devised, unless it is applied as a stablehole machine. This of course is true of continuous miners as well. But considering the mobility aspect, it is more important to efficient operation of the ISM.

This ISM does lend itself, however, to adaption to a straight-ahead operation. As noted, roof support can be installed immediately behind the cutterhead and a wide entry with flat roof and floor can be obtained. The latter is consistent with the placement of divider walls to, in effect, create two or more entries from one cut. Another approach would be to use two or more ISMs (which should be lower in unit cost than continuous miners) and drive crosscuts with a simple unit such as an auger.

A number of concepts can be developed to adapt the ISM to U.S. mining law, but the first order of business is to evaluate the basic machine.

Status

The ISM commenced cutting coal September 19, 1977. It is scheduled to drive 1,000 feet of entry. Allowing for in-process modification, scheduled downtime, and the usual unscheduled downtime experienced on a project of this nature, it is expected to complete the trials by early December.

CROSS—SECTION THRU JIB

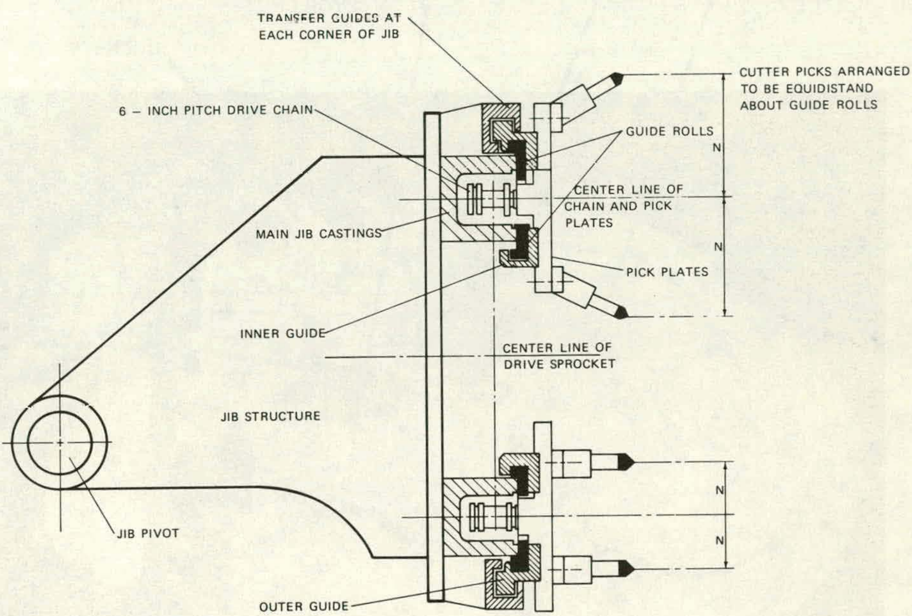


FIGURE 1.

IN—SEAM MINER CUTTING HEAD

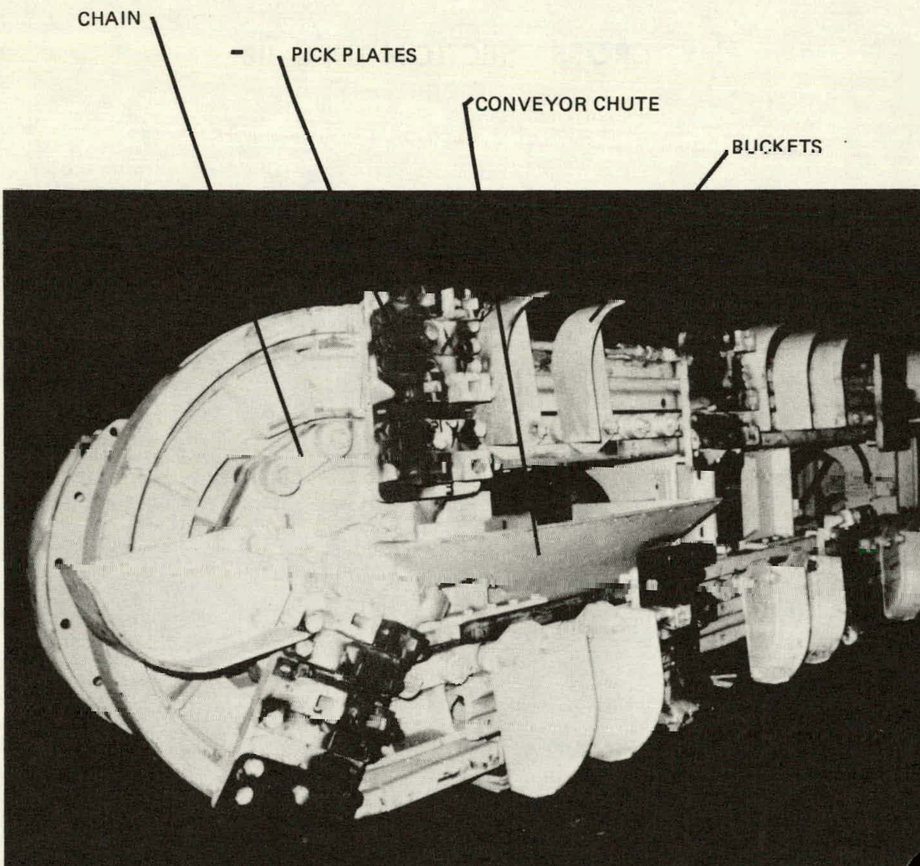
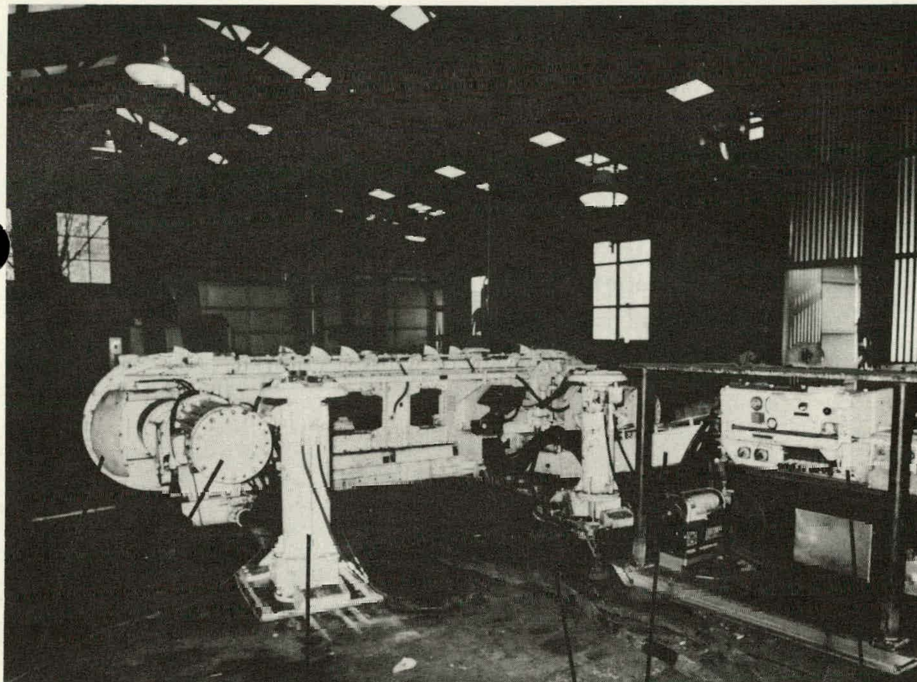


FIGURE 2.



IN—SEAM MINER

FIGURE 3.

PROPOSED MAP OF IN—SEAM MINER PROJECT

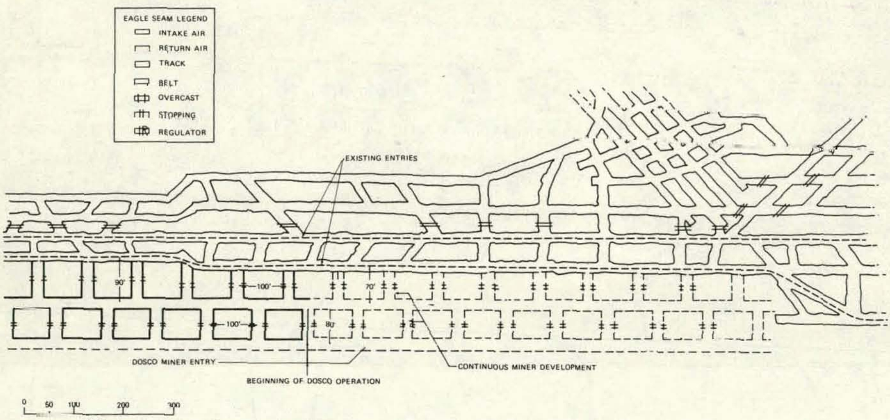


FIGURE 4.

RECENT DEVELOPMENTS IN UNDERGROUND MINING IN AUSTRALIA

J. Johnstone
Managing Director

Buchanan Borehole Collieries Pty. Limited
Singleton, N.S.W. Australia

Director - Australian Coal Industry Research Laboratories Ltd.
Elcom Collieries Pty. Ltd., Newcom Colliery Pty. Limited,
Huntley Colliery Pty. Limited,
Port Waratah Coal Services Ltd.

Recent increases in productivity in the Australian coal industry ave been achieved by open cut mining. The industry is conscious of the fact that as time passes the proportion of coal won by underground methods will increase and is seeking means to ensure that the necessary productive capacity will be maintained.

Although it is not possible to cover the whole subject in detail, the general approach by the industry to the problem is outlined. Important features of the approach are:

1. Conservation and recognition of the need to extract a higher proportion of the in situ coal resource.

2. Control of subsidence so that a higher proportion of coal under built-up areas, lakes and impounded waters can be extracted.

3. Accelerated prospecting to identify new coal resources.

4. Introduction of new forms of mechanisation such as longwall and shortwall and improvements in underground transport of men and materials.

5. Modification of existing mining methods to provide better roof support and strata control.

6. Increased research by Australian Coal Industry Research Laboratories Ltd. and others to identify and seek solutions to problems likely to be encountered.

COMPARATIVE STATISTICS - NEW SOUTH WALES AND QUEENSLAND

		YEAR ENDED 30TH JUNE		
		1956	1966	1976
A. Coal Production - million tonnes (raw)				
New South Wales	- Underground	13.9	24.4	32.2
	- Open Cut	.9	1.0	8.4
	Total	14.8	25.4	40.6
Queensland	- Underground	2.1	4.2	3.9
(estimated	- Open Cut	.6	1.4	28.5
tonnage 1956 & 1966)	Total	2.7	5.6	32.4

It can be seen that total production in Queensland increased by 480% and in N.S.W. by 60% between 1966 and 1976.

However, the comparison in total production won by open cut methods as opposed to underground in Queensland has been most spectacular in the last ten years.

In 1976, 20% of coal produced in N.S.W. was won by open cut methods and most of this was used for power generation. On the other hand, nearly all coal produced in Queensland is directed to the overseas coking coal market and that produced by open cut is 87.6% of the total State's production.

B. Output per Manshift

Significant improvements in O.M.S. in Australia's underground mines have occurred during the past twenty years.

	1956	1966	1976
New South Wales Underground	3.61	8.88	10.02
Queensland Underground	2.69	7.55	8.07

Most of the improvement in N.S.W. has been due to the rise in efficiency of underground mechanised methods. During this twenty year period, the percentage of total coal won by total mechanised means increased from 65.9% to 98.5%. It was during this period that traditional pillar extraction (bord and pillar) has been given away largely to the "Wongawilli", "Big Ben" or "Short Wall" methods. The O.M.S. improvements have also been achieved during a period when the normal length of shift reduced from eight hours bank to bank to seven hours bank to bank during the early 1970's.

C. O.M.S. - U.S.A. and Australia

International comparisons of labour productivity are difficult because of different bases used for measuring the two key factors used in the calculation. In the U.S.A. productivity is measured in short tons (2,000 lb) of clean coal per mineworker day. The quantity of coal used excludes rejects removed at consumer washeries as well as at coal industry washeries. The number of mineworkers excludes administrate and clerical staff. The number of mineworkers is multiplied by the number of days the mine worked during the year and used as the divisor in lieu of the number of manshifts worked.

On this basis, productivity for the United States bituminous coal industry was:

	1974	1975
Underground Mines (O.M.S.)	11.3	9.5

The Joint Coal Board has converted the Australian data as closely as possible to the United States basis and for 1975 gives the followi

New South Wales Underground - 1975	9.6
Queensland Underground - 1975	8.0
Australian Underground	9.35

RESOURCES OF HARD COAL

The Joint Coal Board of N.S.W. has carried out a recent assessment of coal reserves based upon an acceptable degree of geological assurance resulting upon diamond drilling and the knowledge of quality of these deposits. The criteria adopted in Australia are based on current economic conditions, known mining technology and the following:

a. Depth of cover not exceeding 600 metres

b. Minimum seam thickness of 1.5 metres

c. Maximum ash content of 30% (dry basis)

Depth of Cover (Metres)

	Less than 300m	300-600	Total
N.S.W.	13083	3071	16154 million tonnes
Queensland	16270	638	16908 million tonnes

It should be noted that in Queensland most exploration work has concentrated on the location of deposits which can be won by open cut methods and the proving of such area. Based upon seam thicknesses, the above coal reserves can be defined (million tonnes):

Thickness	1.5 - 2.99m	3.0 - 4.99m	Plus 5.0m
N.S.W.	12304	2515	1335
Queensland	63	5030	11815

Extractability and Workability

The technical limits to mining as practiced in Australia may be broadly summarised as:

1. Depth of Cover

a. Open Cut - N.S.W. - The limit currently accepted is a ratio of coal to overburden of 1 to 12. The deepest cut is about 100 metres although depths up to 300 metres are contemplated.

Queensland - The greatest depth at present being worked is 60 metres. The ratio of coal to overburden approximates 1 to 5.

b. Underground Mining - N.S.W. - 600 metres is generally regarded as a workable limit. As considerable coal reserves are available to this depth, the mining engineer is seeking improved recovery factors and to this end, longwall methods are being tried.

Queensland - The acceptable limits of bord and pillar mining is 450 metres. New mines developed to approach this cover are running into difficulties already and research is proceeding for improved techniques.

Coal Seam Thickness

New South Wales - Most collieries work in seams between two and three metres in thickness. Relative factors are the size of mining plant and the need to leave coal to provide additional roof support. A 9 metre seam is mined in the Cessnock area.

Queensland - Coal seams in excess of four (4) metres predominate in the extensive Bowen Basin. However, high productivity rates are not being achieved because of spontaneous combustion problems and high makes of methane. Local authorities regard 6 metres as a maximum safe-working thickness where depths are less than 450 metres.

3. Conditions of Surrounding Strata

New South Wales - The problems are the poor quality of some roof strata and sudden changes in the sequence of the strata. The solution adopted is generally to leave coal to the roof and roof bolt.

4. Geological Faulting

New South Wales - Major faulting is not common. The presence of faults makes mining a little difficult and only affects the planned layout of workings.

Queensland - In the Bowen Basin, two of the present five underground mines are troubled with faults. These coal measures are folded which range from tight to open gentle folds. They are accompanied by normal and thrust faults.

5. Spontaneous Combustion

This is a problem in the South Maitland area of N.S.W. and at mines in the Bowen Basin of Queensland.

Two unfortunate accidents in Queensland at Box Flat in 1972 and at Kianga in 1975 have focussed attention on this mining problem. This has resulted in the publication of a special safety handbook to set guide lines to handle incipient heatings.

Recoverable Hard Coal Reserves

Present mining practices in Australia when applied to the recoverable reserves indicate a substantial difference between N.S.W. and Queensland in the availability of open cut coal.

The Joint Coal Board assess that only seven per cent of the N.S.W. reserves are extractable by open cut compared to an estimated thirty-two per cent in Queensland.

CONSERVATION OF COAL - NEW SOUTH WALES

A. Revised Lease Conditions - In recent years, legislation for the granting of mining leases in N.S.W. has been updated in an effort to conserve our coal reserves. It is necessary to comply with certain conditions depending upon the method of mining and whether the underground operations are proposed below land surface or below tidal waters. These conditions are:

Bord or Pillar Mining

a. The percentage of coal to be left in pillars after first workin is specified depending upon depth from the surface. This range from 50% at 60 metres of cover to 85% at 600 metres. (No mines working at this latter depth at present).

b. The width of pillars is specified. This ranges from not less than 8 metres where the depth from surface does not exceed 60 metres to not less than 24 metres at 300 metres.

c. All bords, cut-throughs and headings must not exceed 5½ metres in width.

d. Where the seam exceeds five metres in thickness, the coal shall not be extracted to a greater height than four metres except with special consent.

Longwall Mining

Where it is proposed to work an area by a longwall method, special applications accompanied by plans and proposed mining equipment must be made to the Department. After consultation, these applications are submitted to the Minister for Mines for his approval and any special conditions that he may impose.

Open Cut Mining

Any proposal to mine coal by open cut mining operations requires special consent and conditions, including the lodgement of a bond or security for rehabilitation of the area after operations. Plans of the proposed operations are to be submitted and these plans indicate how the area is to be mined and the sequence of the operations. A second plan shows the approximate surface contours after rehabilitation and methods proposed to prevent contaminated water from entering creeks and water courses.

Besides these plans, details must be submitted on the manner of conserving the top soil and of the planting of grasses, plants and/or trees to minimise soil erosion.

Mining Under Tidal and Other Waters

Before mining operations commence under the ocean or under any river, lake or tidal waters, lease holders must notify the Department and submit a plan of the proposed system. Some of the requirements are:

a. All working places must be driven on surveyed lines.

b. All coal workings must be accurately surveyed and recorded. All fissures, joints, faults, dykes and anything encountered that may affect the mining operation, must be delineated on the Colliery plans as they occur - not as projected from other workings.

c. A system of advanced boreholes must be maintained ahead of development workings.

B. Surface Effects Due to Mining Operations - The study of mine subsidence has accelerated since 1965, in Australia when early surveys were commenced on the South Coast of N.S.W. by Australian Iron and Steel Pty. Ltd. The work in this area was undertaken to obtain information for mining under stored waters which form the Sydney water supply. In the Newcastle area, B.H.P. Collieries has carried out investigations and surveys since 1966. Here the surface effects became rather complex with multi-seam workings extending over many years combined with various methods of mining, with handmining in past years to full mechanisation in recent times.

In the Vales Point-Munmorah field south of Newcastle, detailed studies commenced in 1969, with regard to surface subsidence which occurs after partial pillar extraction using various mining percentages. These studies proved that extremely limited movement took place when 70% extraction of a panel was attained at depths of 150 - 200 metres. The coal seam in this area ranges from 2.2 to 3.0 metres thick and is overlain by massive conglomerates and shales.

As a result of the above, mining engineers agreed that the hitherto 40% extraction under tidal waters and inland lakes, should be increased to at least 60%. Such a recognition of this experimental work, immediately gave rise to an increase of 50% in recoverable coal reserves under tidal or non-tidal lakes.

C. Recent Mining Under Tidal Waters - The first practical application of percentage mining under water commenced at Newvale No.2 Colliery in April 1976. This mine has extensive areas beneath the tidal and non-tidal waters of the Tuggerah Lake system - midway between Sydney and Newcastle. Bord and pillar method of mining is adopted and square pillars of coal 24 metres in size are left to support the strong conglomerate roof. There is about 110 metres of strata to rockhead, which has been proved by diamond drilling from pontoons on the lake. The floor of the lake consists of silt in various thicknesses to the solid rock.

In the first panel attempted, fifty-six of the one hundred and thirty three pillars were taken successfully and were completed in Mid-November 1976. These were extracted in alternate rows after leaving some pillars for protection against a 5 metre fault along one side. Under the firm conglomerate roof only roof-bolts and props were used. Where the roof was weak, sawn cross-timbers were set at one metre centres.

Some water broke through the roof when the first small fall occurred; but this was characteristic of all pillar extraction at this mine. After this initial difficulty, the inflow gradually ceased to just a nuisance level. The employees in this panel have experience in pillar extraction at the mine and were not apprehensive of the Management's proposals. As a matter of fact, they have been urging for improved recovery for many years.

The second panel was opened up in November 1976 and has been formed of pillars designed specifically for partial extraction. This panel has been won out and the first pillar extracted on 6th May 1977. This section of pillars was completed by 10th June and the unit moved to form an additional adjacent section. These two sections and the sequence of extraction are shown on accompanying drawings, figure 1 and figure 2.

PROSPECTING OF NEW COAL RESOURCES

The coal fields of Australia extend over many distinct areas from Victoria to Queensland - an overall distance of about 3000 kilometres. Scout drilling has proved these various fields but the in situ reserves are considered as "inferred" until geological assurance of recoverable coal can be determined.

Drilling programmes have been concentrated in New South Wales over three main areas:

a. The deep coal measures South-West of Sydney where new mines are being sunk and will mine coal to 500 metres of cover.

b. West of Newcastle in the Singleton-Muswellbrook area where new mines are to be opened within the next five years.

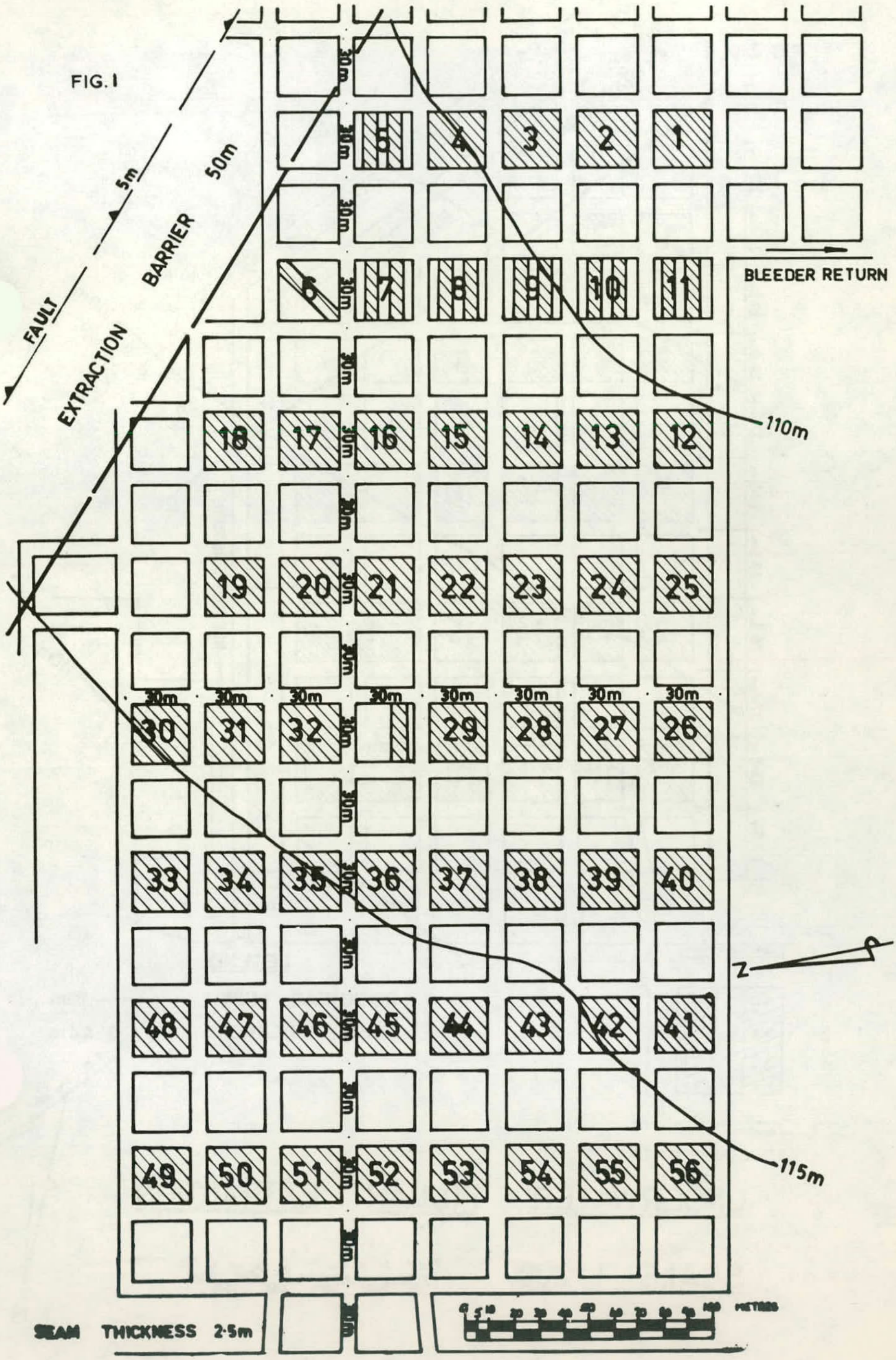


FIG. 1
FAULT
5m
EXTRACTION
BARRIER
50m
BLEEDER RETURN
110m
115m
30m
1
2
3
4
5
6
7
8
9
10
11
12
13
14
15
16
17
18
19
20
21
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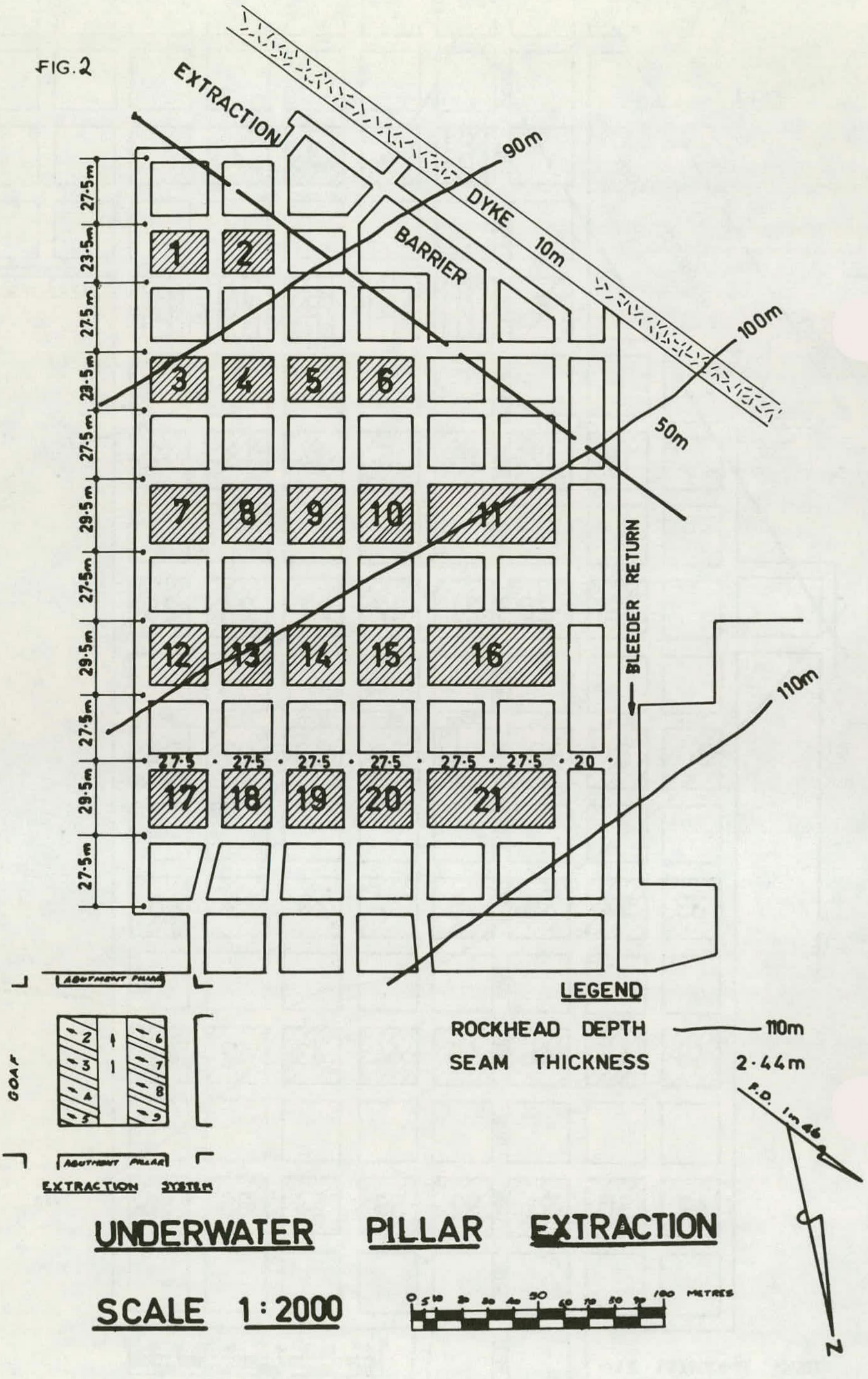
52
53
54
55
56
N
0 5 10 20 30 40 50 60 70 80 90 100 METRES
SEAM THICKNESS 2·5m

FIG. 2



UNDERWATER PILLAR EXTRACTION

SCALE 1 : 2000

c. At Boggabri - some 350 km. North West of Sydney where a new coal field was discovered recently.

In Queensland attention is now being given to the prospect of underground mining possibilities as well as open cut. From Blackwater immediately West of Rockhampton to Goonyella, 250 km. Northwards, the Bowen Basin is being proven by extensive drilling programmes. New names, like Norwich Park, Dysart and Saraji, are becomming known to mining engineers. Coal resources in this field alone have an indicated value of a further 10,000 million tonnes. These are now being proved by close drilling.

In the Singleton area of N.S.W. last year some 6000 metres of ımond drilling to depths of 300 metres, was carried out by mining npanies proposing to open mine in this district. As most of this --eld will comprise multi-seam operations, including open cut and underground workings, considerable drilling has been carried out at shallow depths to define outcrops, coal qualities and zones of oxidised coal.

Private mining companies entered drilling programmes in recent years to a major degree. Fifteen years ago all prospect drilling was carried out by Government instrumentalities - either the Federal Government through the Joint Coal Board, or State Government by the Electricity Commission or Department of Mines. Nowadays private enterprise wants to gain first-hand information by its own geologists and mining engineers in order to confirm heavy capital expenditures.

New concepts are being devised and employed to improve the level of knowledge obtained from drilling programmes. New procedures introduced by ACIRL have enabled more reliable estimates to be made from bore core samples re yields of clean coal and qualities which can be expected after mining and preparation of the raw coal. Advanced statistical methods will be used to establish the range of variation of the coal properties within the areas of interest to so provide information for coal preparation plants.

NEW FORMS OF FACE MECHANISATION

The traditional form of bord and pillar panel development is now giving way to more efficient extraction techniques, particularly in N.S.W. These techniques feature a minimum of first working development into a panel of coal. Pillars are then formed and extracted when "green" on the retreat. This keeps the face workings constantly in a stress-relieved zone. This is particularly effective on the Illawarra field South of Sydney where cover ranges from 300 to 400 metres. Besides attaining a greater percentage of extraction, these techniques reduce floor heave, improve roof conditions and safety while giving consistent productions.

Mechanised Shortwall Mining - In some areas particularly in the Newcastle field of N.S.W., older mines have sought forms of mining that are suitable for introduction in areas of limited extent and in some cases bounded by small faults and dykes. Seam heights of about 2½ metres below sandstone and shale strata were difficult to support in normal bord and pillar mining. In such cases, a form of shortwalling has proved most effective in mine safety and higher productivity.

This method can have many variations in depth of panel and application. The technique is to form a rectangular panel of coal by driving a pair of headings on either side of a block of coal so as to leave 60-80 metres of solid coal between them. A place is then driven to connect both pairs of headings at the inbye end of the block of coal. Ventilation is carried in one set of entries, across the back and returns out the other set of entries.

Coal is then sliced across the block of coal utilising a continuous miner and one shuttle car. The working place is kept narrow while self-advancing hydraulic supports are placed along the goaf side. The steel canopies of the hydraulic supports provide protection for the operator, who always works on the goaf side of the machine.

The distance across the block of coal should be fixed by the anticipated production of a shift. Several mines are now working three shifts daily and attaining shift averages of in excess of 500 tons. The technique greatly saves pit timber an requires a minimum manning. Good organisation in moving the hydraulic supports, alignment of working places and easily retractible conveyors are essential pre-requisites in this form of mining.

B. Mechanised Longwall Mining - Depths of coal from the surface of the Illawarra field and resultant difficulties of roof weight, and ventilation problems gave rise to the introduction of longwall mining with self-advancing supports about twelve years ago. The results were disappointing until recently, due mainly to the inability of the supports to handle the massive sandstone overlying strata, which are so different to conditions met in the more plastic strata of Europe.

Subsequent units installed by A.I.S. Collieries and Bellambi Coal Co. have had much stronger roof support systems and improved hydraulics. Outputs at these mines from longwall units is now more consistent and mining engineers are becoming more confident with this system. The new mines now being developed will depend largely on the longwall method of mining.

C. Old Ben and Wongawilli Systems - These two extraction systems have proved very successful during the past twenty years. Several modifications to the basic concept are used. The United States supplied the "Old Ben" technique which has been varied to suit the mining in the Vales Point-Munmorah area South of Newcastle. The roof strata in this field comprises massive conglomerates difficult in its initial "break". Under such conditions this form of mining enables an area to be progressively worked in adjoining panels with continuous roof falls to relieve the pressures.

The Wongawilli system, on the other hand, is practiced extensively in the South of N.S.W. and is used at depths of up to 500 metres. It incorporates the essentials of the "Old Ben" system in its strict planning and safety protection of men and machines. The main variation however, is that the extraction progress is kept within the stress-relieved zone between the main abutment pressure and the goaf.

It is interesting to note that in the year ending 30th June 1976 nearly 60% of all pillar extraction in N.S.W. was won by Old Ben or Wongawilli systems.

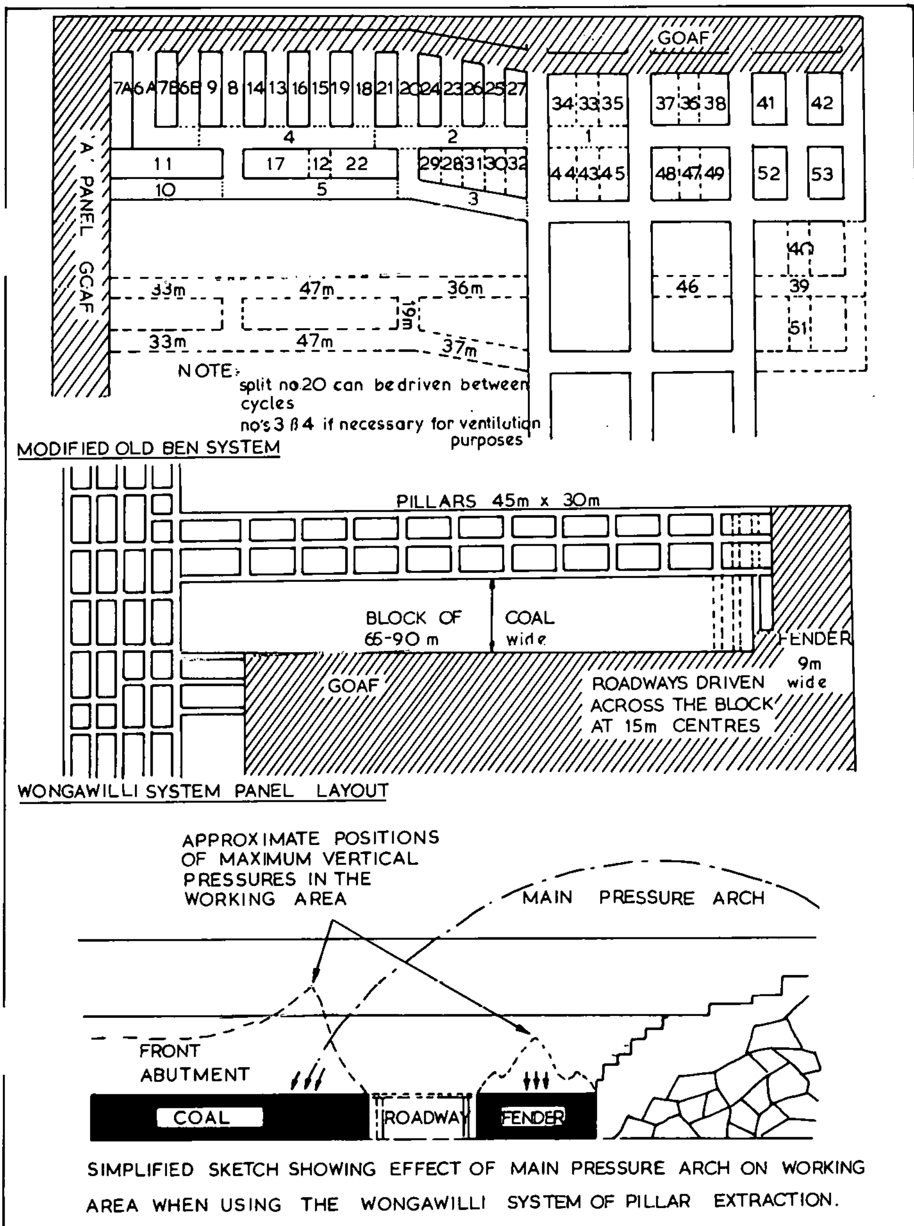

GOAF
'A' PANEL GOAF
7A 6A 7B 6B 9 8 14 13 16 15 19 18 21 20 24 23 26 25 27
4
2
1
34 33 35
37 36 38
41
42
11
17 12 22
29 28 31 30 32
44 43 45
48 47 49
52
53
10
5
3
40
46
39
51
33m
47m
36m
19m
33m
47m
37m
NOTE:
split no.20 can be driven between cycles
no's 3 & 4 if necessary for ventilation purposes
MODIFIED OLD BEN SYSTEM
PILLARS 45m x 30m
BLOCK OF 65-90 m
COAL wide
FENDER
9m wide
GOAF
ROADWAYS DRIVEN ACROSS THE BLOCK AT 15m CENTRES
WONGAWILLI SYSTEM PANEL LAYOUT
APPROXIMATE POSITIONS OF MAXIMUM VERTICAL PRESSURES IN THE WORKING AREA
MAIN PRESSURE ARCH
FRONT ABUTMENT
COAL
ROADWAY
FENDER
SIMPLIFIED SKETCH SHOWING EFFECT OF MAIN PRESSURE ARCH ON WORKING AREA WHEN USING THE WONGAWILLI SYSTEM OF PILLAR EXTRACTION.

FIG.3

THICK SEAM MINING - AUSTRALIA

In New South Wales the only thick coal seams worked have been in the South Maitland field where the Greta seam varies from seven to ten metres in thickness. Underground mining in the field has been carried out for the past seventy years. Mechanisation was introduced about forty years ago but to date no satisfactory system of mining has been evolved to obtain a satisfactory recovery factor during mining at an economic cost. Most of the old mines are now closed and thick seam mining is only proceeding at three collieries.

The high sulphur content in the coal forming the top plies of the seam prevents first working at the top to so gain benefit of the strong roof. The hard brittle nature of the coal gives rise to bursting failure under excessive loading. The virgin coal in the remaining areas is over 350 metres deep and difficult mining condit... are expected.

In spite of this, some progress is being made by the use of road-heading machines in development of a small panel system to counter spontaneous combustion. Unfortunately any success in pneumatic stowage, from experiments some years ago, has had to be scrapped because of excessive costs. Present mining applies continuous miners, shuttle cars and roof bolting units. The introduction of chemical anchors with bolting has been most successful.

Queensland on the other hand, has had some success in thick seam mining in areas away from the Bowen Basin. Near Ipswich at Westfalen Colliery up to seven metres of coal has been tackled using multiple face cutting, with a Marietta Borer and also a Joy continuous miner. Such operations comprised driving initially at roof level, supporting this roof and making a second or third pass below. It should be realised however that in this mine no pillar extraction took place because of future housing development.

At Collinsville experimentation on practical mining methods is proceeding. At these mines up to eight metres is being tackled in pillar extraction by a two-pass method although percentage recovery is only about 60% within the panel.

All mines are looking with interest at model testing by Australian Coal Industry Research Laboratories Ltd. in the Bellambi laboratories. It is believed that a multiple-slicing method leaving a septum of coal will ultimately present a safe method of pillar extraction and an optimum recovery factor.

Interest was created in September 1976 at a symposium on thick seam mining by underground methods, sponsored by the Australasian Institute of Mining and Metallurgy. At this gathering papers were presented by overseas authorities from France, Germany, Japan, Pol and United States. These engineers injected new thoughts and conf ence to those of Australia attending the seminar.

UNDERGROUND TRANSPORT

The reduction of the normal shift time from eight to seven hours about six years ago caused changes in thinking from two shifts to multiple shift production. This was countered industrially by substantial Union claims for higher wages. Production costs have been closely investigated by the industry and two areas of these costs have had serious attention paid to them.

Mining plant repairs and maintenance charges have shown some reduction. Change-out units are kept on hand for ready replacement, conveyor roller design has lengthened previous roller life, routine machine overhauls and heavier continuous miners have had a positive effect on costs.

Perhaps the most positive step has been a review of existing haulage systems for men and materials. When the time exceeds half an hour to take men to the working face, serious investigations must take place. All mines have made improvements, either by purchase of new rolling stock, new vertical shafts sunk near the face or the overlapping of the working shifts - a form of four-shifts per day. Diesel haulage underground at large mines in some cases is proving ·h in maintenance and serious thought is being given to alternate .ts.

As a great proportion of coal haulage is carried out on belt conveyors, belt designs are being critically looked at to obtain longer life. Solid woven carcassed with well-impregnated P.V.C. is extending average face belt life quite considerably indeed. Conveyor widths are becoming wider in the interests of spillage and improved coal handling. New mines proposed to work at depths up to 500 metres are now being developed around automatic bulk winders. Mines intended to work in more than one seam are to have fast haulage in one seam with short stone-drifts near the face for access to second seam. In one sentence, haulage of men and materials has become an important design factor in mining.

METHODS OF MINING THICK COAL SEAMS

One of the greatest challenges in the underground mining of a thick coal seam is to attain a maximum recovery at an acceptable cost and provide a safe and comfortable environment. Although satisfactory methods have been developed overseas under certain situations, it is recognised that in Australia there are conditions peculiar to this country which require general assessment and individual evaluation.

The thick seams of the Bowen Basin in Central Queensland are those receiving immediate consideration. Mining techniques being currently looked at and present opinions on such systems may be summarised:

Longwall Slicing and Caving - This system was developed in the United Kingdom and Europe to work thin coal seams, and a common system is long wall advancing. In N.S.W. where longwall has been introduced, it has been a retreating system and great difficulty has been experienced in maintaining approved dust standards. The friable coal of the Bowen Basin is expected to create a more difficult problem than that experienced in N.S.W.

The mining of thick seams by longwall could require a multiple operation leaving a septum of coal to separate the operation. The dust make under such a system could be excessive and uncontrollable. Considerable research is still required to see if a longwall seam in thick coal seams is capable of working satisfactorily in various Australian locations. If longwall proved satisfactory, the application of remote control and automation should be undertaken.

Shortwall Slicing and Caving - In some N.S.W. mines a shortwall method of pillar extraction is employed using continuous miners, shuttle cars and self-advancing hydraulic supports.

Under such a system the span between the chock and the face is much greater than in longwall and thus enables high productivity in a system familiar to us. If such a system were to be applied to thick coal seams, a careful assessment of roof support at the goaf edge becomes necessary. As for longwall, if a septum system between slices were to be adopted, the adequacy of dust control during chock movement would have to be evaluated.

RESEARCH BY AUSTRALIAN COAL INDUSTRY RESEARCH LABORATORIES LIMITED

This research organisation was incorporated in 1965 and took over the existing facility of A.C.A.R. Ltd.,which had served the coal industry since 1955. It has five laboratories in New South Wales and Queensland, and provides (1) Commercial Analytical Service, (2) Non Confidential Research, and (3) Confidential Research.

It is financed from charges for services rendered and from Government and private industry funds.

One of ACIRL's recent publications sets out in detail some of the problems currently facing the industry in Australia. These are:

a. High gas emission
b. Outbursts
c. Roof control
d. Thick seam mining
e. Prediction of mining conditions in advance of mining.
f. Spontaneous combustion.
g. Low percentage recovery.

To these ends, ACIRL has recommended that the mining industry should combine with the Australian Government to provide sufficient funds to expedite an expanded research programme.

The priority areas that require immediate attention and a concentration of effort include:

1. Improvement in percentage recovery by variation in mining methods.

2. Development and proving in practice new mining systems to extract thick seams.

3. Improvements in mine safety by -

   (a) Pre-mining degasification of coal and associated strata.
   (b) Prediction prevention and/or control of outbursts.
   (c) Prevention and/or control of spontaneous combustion.
   (d) Development of new methods of roadway supports.
   (e) Identification and measurement of rock characteristics relevant to mine design and extraction procedures.

ACIRL propose to engage a further seventeen professional engineers and assistants to carry out the expanded programme. It is estimated that cost during the first year will be $2.2 million.

As indicated in the foregoing sections ACIRL has carried out several model studies of methods of mining that can be applied to thick coal seams. Practical application of these new techniques is now awaited from the Collinsville area of the Bowen Basin where an area of coal 300 metres by 300 metres has been set aside for field trials.

DESIGN OF OPENINGS IN UNDERGROUND COAL MINES

Mechanised methods in Australia have been determined in nearly all cases to suit the available mining plant. Most of the production is won by continuous miners and rectangular entries are made in the winning out of the new areas. The trend in recent times is to increase the overall dimensions of the miner and to overcome some weaknesses in the machine by increasing the weight. To allow sufficient room around the machine for timbering and ventilation, is not unusual for the heading to be driven up to 6¼ metres in th.

The face haulage of coal has been improved over the years by the introduction of high capacity shuttle cars. These, too, have become heavier and wider than their counterparts of yesterday. The carrying capacity has risen to 12 tonnes in an effort to reduce car changing times behind the miner. Bord and pillar work is the normal method of layout - or a modification of it - and results in the roadway intersections being wider than desirable.

As mines become deeper and physical conditions deteriorate, it is most necessary to reconsider roadway shapes in order to improve roof support systems. The industry in Australia through sponsorship of ACIRL has examined the effects of variations in the size and shape of underground roadways. Some general conclusions have already been determined and certain mines are adopting new methods in heading development.

Some conclusions may be worth noting:

1. Roadways of near circular shape supported by radially placed roof bolts are stable in poor physical conditions.

2. The stability of rectangular roadways in typical N.S.W. geological environment decreases markedly as width increases.

3. Intersections should be at right angles and in preference roadway junctions used in lieu of intersections.

4. The use of chemical anchored roof bolts with maximum cartridge length for full anchorage.

The above valuable work by ACIRL has demonstrated that modifications to the traditional size and shape can result in more stable openings. The introduction of road-heading machines of the Alphine or Dosco type is hoped to supply a practical solution. Such nines can be operated in places 4½ metres wide cut an arched roof le the coal can be removed by narrow shuttle cars or some form extensible conveyor. Although the capacity of such machines is considerably less than continuous miners, the narrower width could result in less roof support and reduced maintenance costs during the subsequent years.

SPONTANEOUS COMBUSTION

At some mines in N.S.W. where spontaneous heating is a problem, the mining objective adopted is to attain complete extraction at maximum speed. Panel layout and method of ventilation are the essentials in a suitable technique. The system permitting a quick method of "sealing off" must be adopted and all officials and employees should understand how "seals" are to be erected in an emergency.

A mining method now favoured is for three or four headings driven to the boundary of a panel, say 400 metres deep and 200 metres wide. When the boundary of the panel is reached, an additional pillar is formed on each side of the three or four entries and a line of pillar extraction commenced on the retreat.

Such a system permits the erection of a minimum number of preparatory "seals" and ensures that adequate face ventilation is available, while minimising the amount of ventilation entering the goaf area. This ensures that oxidisation of any coal remaining in the goaf cannot take place. Also in the event of serious heating, minimum amount of coal is lost if the section is sealed.

Good work has been done in Queensland, when the Department of Mines, Coal Owners Association and the Colliery Employees Union jointly sponsored a handbook dealing with spontaneous combustion in underground coal mines. This handbook is distributed to all employees free of charge. It details how heatings occur, their detection, precautionary measures to be taken and the final sealing where there is a risk of explosion.

Consideration is being given in Australia to continuous monitoring of mine air. Where heatings may be a problem, such installations can be justified by early detection of an incipient heating. Appropriate action can be immediately taken and a high level of mine safety achieved.

In conclusion one must recognise that so far as Australia is concerned, very considerable reserves of coal are available for exploitation. It is apparent however, that as open cut type reserves become depleted and more operations become necessary in underground type mining, significant cost increases must be faced. It is also apparent that the complexities of underground mining at depths in excess of existing underground operations will present considerable challenges to all concerned. It is important therefore, that areas of research with regard to such problems be both expanded and intensified in financial application.

I believe it is also of importance for Australian mining technicians both in the fields of operations and research, to become associated with what is being done in other parts of the world so far as research into these matters is concerned. The need for energy in all its forms is important to us all, therefore close co-operation between the mining communities of the world becomes essential if we are to satisfactorily solve the many mining problems confronting us.

HELPING FINANCE NEW MINES WITH REVENUES FROM COALBED DEGASIFICATION

Maurice Deul
Research Supervisor, Methane Control and Ventilation
Pittsburgh Mining and Safety Research Center

Milford L. Skow
Staff Engineer
Office of Assistant Director--Mining
Washington, D. C.

Ann G. Kim
Chemist, Methane Control and Ventilation
Pittsburgh Mining and Safety Research Center

Bureau of Mines
Department of the Interior

Introduction

Even a cursory examination of this country's fuel requirements makes it clear that the demand for coal will increase dramatically in the future. The predicted extent of this demand depends on the bias and/or optimism of the forecaster, but according to most estimates, coal consumption in the United States may be expected to double in the next 10 years and triple or quadruple by the year 2000 (17,20).

In the past, the growth of the coal industry could be characterized as a boom or bust cycle strongly influenced by political and economic events (fig. 1). Over the past 15 years the coal industry has grown at an annual rate of 3 pct; in 1976 coal production reached an all-time high and from 1973-76 the mine mouth price of coal more than doubled (1, 15-18). In spite of the overall growth of the industry, production from underground mines has remained static or declined in the last 10 years (fig. 2). At present, about half of United States coal is produced from surface mines, but it is unlikely that this trend will continue over the long term. Not only can political and socio-economic factors be expected to put additional constraints on the growth of surface mining, but less than one-third of United States coal reserves can be mined by surface methods (1, 19). There are indications that the outlook for underground mining is already improving. In 1974, the number of underground mines increased (fig. 3), and underground mines account for 80 pct of the new capacity planned for the Appalachian region to 1985 (4, 18).

An immediate problem which should be faced prior to mine development is the probability that new coal mines will experience severe methane emissions. As readily accessible, relatively shallow reserves are depleted, new mines must be opened in deeper parts of the coalfields (fig. 4). Theoretically deeper bituminous coals contain more gas than shallow coals (fig. 5), and field studies have shown that this is usually the case (10, 12). If only traditional dilution methods are used to handle methane emission in new deeper mines, the ventilation systems will be expensive, in terms of capital invested and in operating costs; and it is also probable that productivity will be adversely affected by excessive methane emission at the face.

In addition to the technical problems in the development of new underground bituminous coal mines, the need to obtain large amounts of capital from external sources will also be a problem. When coal mining was the domain of the small independent producer, capital needed for mine expansion was often obtained from a local bank or from internal cash flow (21).

If annual coal production is to reach 1.5 billion tons by the turn of the century, 35 million tons of new capacity must be added each year. It has been estimated that 600 to 800 new large capacity mines will be needed to meet the increased demand for coal (2). In 1974, capital requirements for this large-scale expansion were estimated at $8 billion to $22 billion. Capital required for deep mines was estimated at $25 to $36 per ton of annual capacity (20-21). Depending on size, total capital investment in an underground bituminous coal mine will be between $32.5 million and $123 million (table 1) (8-9). Since coal mining is considered a high risk industry, it does not readily attract capital, particularly in times of economic and regulatory uncertainty. In order to compete effectively in public money markets the coal industry, particularly the independent producer, must be able to lower economic risk, reach full productivity on schedule, sustain increased earnings, and substantially improve the rate of return on invested capital (14).

TABLE 1. - Capital required for new mine development, $ million

Capacity, ... MMtpy	1	2	3	5
Initial capital	20.8	36.9	54.8	84.0
Deferred capital	11.7	18.6	26.2	39.0
Total capital	32.5	55.5	81.0	123.0
Capital/ton of annual capacity $	32.0	28.0	27.0	25.0

A concept developed, tested, and evaluated by the Methane Control and Ventilation group of the Bureau of Mines, Pittsburgh Mining and Safety Research Center can both reduce methane emission problems and help mine financing. Field experience has demonstrated that gas production from coalbeds through mine shafts for 3 to 5 years before mining commences is technically feasible and economically attractive. In less than 4 years, almost 2 billion cubic feet of methane has been drained through two mine shafts to the Pittsburgh coalbed in northern West Virginia (fig. 6) (6-7). Gas production from each shaft averaged over 600,000 cfd and the gas was sold as high Btu gas. As workings approached to within 500 feet of one of the shafts, methane emission at the face declined to 30 pct of what would normally be expected from virgin coal in this mine (fig. 7). It is evident that both commercial recovery of the gas and decreased methane emission at the face contribute to improved earnings.

Technology of Methane Drainage

When the organic debris of Carboniferous forests was converted to coal, methane and other gaseous hydrocarbons were also produced. Although a large percentage of this gas migrated away from the coalbed during geologic time, all coalbeds contain methane. The amount of methane in the coal may vary from 0.01 to more than 600 cu ft/ton. Theoretically the amount of gas in coal increases with increased rank, and with depth, as in the Mary Lee coalbed where gas content triples with a twofold increase in depth (fig. 8). Although the total methane emission from a mine depends on many factors including production rate, gob area, rib extent, and age, mines in gassier parts of coalbeds generally encounter more severe methane emission problems.

The gas in coal, although usually referred to as methane, is actually a mixture of gases. Methane is the major component, constituting between 80 and 99 pct of the gas, but higher molecular weight hydrocarbons, carbon dioxide, hydrogen, oxygen, nitrogen, and helium are also present. Coalbed gas does not contain carbon monoxide, sulfur compounds, or oxides of nitrogen. The heating value of gas drained directly from coalbeds is usually more than 900 Btu/cu ft. Since the composition and heating value of the gas in coal is very similar to natural gas (table 2), gas drained from coalbeds is interchangeable with natural gas. It can be added directly to commercial gas pipelines, used as a chemical feedstock, converted to LNG, used as boiler fuel, or in gas turbines to generate electricity. The gas in coal is a high quality, high Btu (11).

TABLE 2. - Composition and heating value of coalbed gas and natural gas, pct

Source	CH_4	CH[1]	H_2	Inerts[2]	O_2	Btu/scf
Pocahontas No. 3	96.87	1.40	0.01	2.09	0.17	1,059
Pittsburgh	90.75	0.29	-	8.84	.20	973
Kittanning	97.32	.01	-	2.44	.24	1,039
Lower Hartshorne	99.22	.01	-	0.66	.10	1,058
Mary Lee	96.05	.01	-	3.45	.15	1,024
Natural gas[3]	94.40	4.90	-	.40	-	1,068

1/ Other hydrocarbons.
2/ N_2, CO_2, and He.
3/ Moore, B. J., R. D. Miller, and R. D. Shrewsbury. Analyses of Natural Gases of the United States. USBM IC 8302, 1966, 144 pp.

If the average gas content of coal is estimated at 200 cu ft/ton, minable coals in the coterminous United States contain more than 300 trillion cubic feet of gas and all U.S. coal resources could contain over 750 trillion cubic feet of gas (table 3). Specifically, the Mary Lee coal group in the Black Warrior Basin of Alabama contains more than 1 trillion cubic feet; the Beckley coalbed in Raleigh County, West Virginia contains .1 trillon cubic feet of gas; and the Pittsburgh coalbed in southwestern Pennsylvania contains more than 0.5 trillion cubic feet of gas. To put these numbers in perspective, in 1974, proved reserves of natural gas in the United States totaled 237 trillion feet (18). According to a National Academy of Sciences study (13), routine drainage and utilization of coalbed gas is the most technically advanced and readily implemented means of supplementing conventional gas production and meeting short-term energy needs.

TABLE 3. - Estimated U.S. coal and coalbed gas resources

	Coal, billion tons[1]	Coalbed gas, tcf
Coal reserves		
Surface mining	137	-
Underground mining	297	59
Identified coal resources		
0-3,000 feet	1,297	259
Hypothetical coal resources		
0-3,000 feet	1,849	370
3,000-6,000 feet	388	78
Total	3,968	766

1/ Averitt, P. Coal Resources of the United States, January 1, 1974. U.S. Geol. Survey Bull. 1412, 1975, p. 1, 33.

Although there are several methods of draining gas from coalbeds (table 4) at present, draining gas through horizontal holes drilled from the bottom of shafts is the most economically attractive. In this procedure, an air shaft is sunk 3 to 5 years before it is needed for ventilation and long horizontal holes are drilled radially into the coalbed from the shaft bottom. The drainage holes are connected through a water trap to a receiver tank, and the gas is piped to the surface through a pipe located behind the shaft liner. The kind of surface installation needed depends on the end use of the gas (3, 7). Gas production rate from horizontal boreholes depends upon the aggregate length of the holes. In the Pittsburgh coalbed, the drainage rate averages 125 cu ft of gas per day per linear foot of drainage hole. In other words, five drainage holes averaging 1,200 feet in length, an aggregate length of 6,000 feet, produce 750 Mcf of gas per day.

TABLE 4. - Methane drainage methods

1. Horizontal boreholes from shaft bottoms.
2. Vertical boreholes from the surface.
3. Vertical boreholes into gob areas.
4. Directional slant hole from the surface.
5. Horizontal boreholes from underground mine workings.

Economics of Methane Drainage

The economic analysis of the cost and benefits of coalbed degasification presented here is relatively simple. It does not take into account depreciation, depletion allowances, tax benefits, accounting methods, and other factors which vary for each coal producer. In this paper capital investment, cost, and income for coalbed degasification are estimated for a hypothetical degasification program in which three ventilation shafts are sunk 3 years before coal mining begins and used to drain methane from the coal. One shaft drains gas for 3 years, one for 4 years, and one drains gas for 5 years. At the base of each shaft, five 2,000-foot-horizontal drainage holes are drilled to give an aggregate drainage length of 30,000 feet. The only constraint on the size of the shaft is that the area within the coalbed at the shaft bottom be large enough to accommodate drainage equipment and drilling operations. Since drilling and drainage operations have been accomplished in a 14-foot-diameter area (154 sq ft), most shafts require no significant modification to be used for gas drainage. The cost of sinking the shafts is estimated at $4 million. Since the shafts are needed for subsequent mine operation, the capital invested in their construction need not be charged against methane drainage.

A major cost incurred from degasification is the interest on capital invested for early shaft construction. If it is assumed that the annual interest rate is 8 pct and that the three shafts cost $4 million, the annual interest is $320,000. Over the 5-year period total interest is $1.3 million. But if it is assumed that the annual rate of inflation is 5 pct, the cost of constructing the shafts when needed would be $4.8 million. Therefore, the interest for early shaft construction chargeable to degasification is actually less than $0.5 million. Even this amount is justified considering that mine development costs have been rising faster than the average inflation rate.

Early construction of all three shafts has other advantages. Sinking shaft early and simultaneously can realize savings in site preparation, as well as in engineering costs; and costly delays due to equipment shortages or lack of qualified personnel can be avoided. When shafts are sunk in advance, interest charges add approximately $1.3 million to their initial cost; this amount can be recovered from revenue generated by gas drainage. It can also be justified in terms of rapidly rising costs and the fact that shafts are available when needed, avoiding production delays.

Other cost items related to methane drainage from coal include drilling horizontal holes in coal, underground equipment, surface installation, maintenance and servicing. Although actual costs depend on the individual situation, the following ballpark figures are considered reasonable. The cost of drilling the horizontal drainage holes is approximately $25,000 per shaft. Capital expenditure for underground equipment is approximately $250,000. Since the salvage value of this equipment depends on its condition and length of use, in this instance, it will be wholly depreciated over the drainage period. The cost of the surface installation depends on the ultimate means of disposing of the gas, but it will probably be $200,000 or less per shaft, again, no salvage value is calculated. Annual maintenance is estimated at $2,500 per well. Thus, total expenditure for draining gas from the three shafts, including equipment, drilling, and maintenance, would be about $1.5 million. Expenditures for gas drainage and ditional interest on invested capital increase capital investment in a new mine 2 pct or less. The costs of draining gas from coal (table 5), are small when mpared to other costs of putting a coal mine into operation.

TABLE 5. - Costs related to methane drainage through shafts, $ million

	In advance	As needed
Capital	4.0	4.0
	Interest @ 8%/yr	Inflation @ 5%/yr
1st year	0.32	0.20
2d year	.32	.20
3d year	.32	.20
4th year	.22	.14
5th year	.11	.06
Total	1.29	.80
Shaft cost	5.29	4.80
Cost of horizontal drainage holes	.83	-
Cost of surface installation	.60	-
Maintenance @ $2,500/yr per shaft	.03	-
Total	1.46	-
Total cost in advance	6.75	-
Shaft cost as needed	-4.80	-
Cost of degasification	1.95	-

Income from degasification depends both on gas production rate and the selling price of the gas. At the rate of 125 cfd/ft, an installation with an aggregate drainage length of 30,000 feet would produce 3.75 million cubic feet of gas per day; annual production from three shafts would be 1.3 billion cubic feet. Assuming that one shaft was used for 3 years, one for 4 years, and one for 5 years, total gas production would be over 5 billion cubic feet.

At the present time the wellhead price of new natural gas is $1.42/Mcf plus $0.01 annual escalation. However, the Federal Power Commission does not intend to gulate the price of gas drained from coal, and also proposes to modify its regutions to encourage the sale of methane from coal to interstate pipeline compaes (5). Under these circumstances, $2/Mcf is considered a reasonable price for gas from coal, although higher spot-sale prices will prevail in a "free" market. Gas drained at the rate of 3.75 MMcfd, sold for $2/Mcf, would generate revenues of $7,500 per day. At this rate, the costs directly chargeable to degasification are recovered in less than 1 year. At $2/Mcf, the gas drained and recovered from the three shafts would generate $2.6 million in gross annual revenue; total revenues would be more than $10 million during the 5-year drainage period (table 6). Income from degasification is obtained during the planning and early development stages, when coal mining itself provides little or no income.

TABLE 6. - Gross income from methane drainage through shafts

Shaft number	1	2	3	Total
Drainage rate .... MMcfd	1.23	1.25	1.25	3.75
Annual production .... MMcf	440	440	440	1,320
Annual income @ $2 per Mcf, $ million:				
1st year	.88	.88	.88	2.64
2d year	.88	.88	.88	2.64
3d year	.88	.88	.88	2.64
4th year	-	.88	.88	1.76
5th year	-	-	.88	.88
Total	2.64	3.52	4.40	10.56

Although this example is based on a 5-year drainage period, there is no technical reason for limiting coalbed degasification to such a brief period. Experience has shown that methane production rates from coalbeds have averaged approximately 600 Mcfd levels for over 4 years (fig. 9). It is feasible to envision the use of air shafts as gas producing installations for longer periods of time. Since all costs are concentrated in the initial period, coalbed degasification continues to generate revenue without additional investment.

Exactly how much money will be earned from coalbed degasification actually depends on variables such as accounting method, tax treatment, etc., which are beyond the scope of this paper. To simply indicate the possibilities, the net income from coalbed degasification, before taxes, was calculated for four cases (table 7), in which costs chargeable to degasification and the effect of inflation were varied. The cost of sinking the three shafts is $4 million. During the 5-year period they are used for drainage, the total interest on invested capital at 8 pct per year is $1.3 million. The costs of horizontal boreholes, surface installation, and annual maintenance is $1.5 million. Inflation at 5 pct per year adds $0.8 million to the original cost of the shafts. Income from the sale of the gas is $10.5 million.

TABLE 7. - Net income from methane drainage through shafts, $ million

Case	A	B	C	D
Capital invested in shafts.	0	0	4.0	4.0
Interest on invested capital	1.3	1.3	1.3	1.3
Effect on inflation	-0.8	0	-0.8	0
Drainage costs	1.5	1.5	1.5	1.5
Cost of methane drainage	2.0	2.8	6.0	6.8
Income (gas @ $2/Mcf)	10.5	10.5	10.5	10.5
Net income	8.5	7.7	4.5	3.7
Pre-tax, average annual rate of return ...... pct	85	55	15	10

In case A, the cost of the shafts is not charged against degasification, and a credit is allowed for inflation. This is considered reasonable since the shafts will be needed for mining and inflation will raise their cost by at least 5 pct per year. In this case the average annual rate of return on invested capital before taxes is 85 pct.

In case B, costs are similar to case A, except that no credit is allowed for inflation. Assuming that the effect of inflation is negligible reduces the average annual rate of return to 55 pct. If the capital invested in sinking the shafts is also charged against drainage income, the average annual rate of return is 15 or 10 pct (cases C and D) depending on whether or not the effect of inflation is included. Considering that a 10 pct annual rate of return on invested capital is average for the largest corporations, the rate of return on invested capital for coalbed degasification is excellent.

Effect of Methane Drainage on Coal Production

Capital investment for degasification adds very little to the total cost of a mine; it produces income from the sale of the gas and economic benefits extend to subsequent coal mining (fig. 10). In normal mine development, methane emission increases as headings are driven into virgin coal. Research by the Bureau of Mines shown that when gas is drained prior to mining, methane emission at the face is uced by 70 pct, and emission from the rib is correspondingly reduced.

To justify the large investment required, a mine must be capable of producing coal at a sustained rate, and it must reach full production as rapidly as possible. Development times for underground mines are usually estimated to be between 1 and 3 years. However, a random sample of productivity versus mine age shows that most young mines, those in operation between 3 and 10 years, are operating at substantially less than their planned capacity (fig. 11). During the development period, the income from degasification and enhanced productivity realized from mining a less gassy coal seam may be critical in determining the economic viability of a coal mine. For instance, labor and operating costs during development are estimated at between $5 and $15 million. If these costs can be covered by revenue from methane drainage, the need for short-term borrowing at this stage is reduced. Draining methane from coal prior to mining generates income before coal is mined and can reduce both development time and cost.

Although benefits in subsequent mining are difficult to quantify, these may have a substantial effect of profitability. Increased productivity in mining a less gassy seam might alone be sufficient justification for draining gas in advance of mining. Likewise, lower ventilation costs, in terms of less power and auxiliary equipment, and fewer man-hours for ventilation, can enhance the coal mines profitability. Certainly the reduced hazard of ignitions and explosions would be a significant factor in the overall ability of the coal mining industry to attract investment capital.

Conclusion

Implementation of a national energy policy which calls for increased dependence on coal will require vastly expanded coal mining capacity. It is highly probable that new mines will be more expensive, requiring public financing. To attract outside investment, the coal mining industry will have to lower risk, increase productivity, and sustain a high production rate. Many factors will determine whether this will be accomplished, but it will be particularly difficult if problems related to the gassy nature of deep coalbeds are ignored until they become crises. It can be reasonably expected that new, deep coal mines will experience very high methane emission rates, and dependence on ventilation alone to trol methane emission will be ineffective and will result in lower productivity. eau of Mines experience with degasification of very gassy coalbeds, especially Pittsburgh coalbed, has shown that coalbed degasification prior to mining will ameliorate gas problems during mining and will also generate additional revenue. In the example given here, for an additional investment of $2 million for degasification, more than $10 million was realized from the sale of drained gas. In other words, commercial gas production from coal through mine shafts makes monetary sense; it is technologically feasible, economically attractive, and will improve the profitability and reduce the hazards of coal mining. Degasification in advance of mining reduces the risk, both financial and physical, involved in mining the coal.

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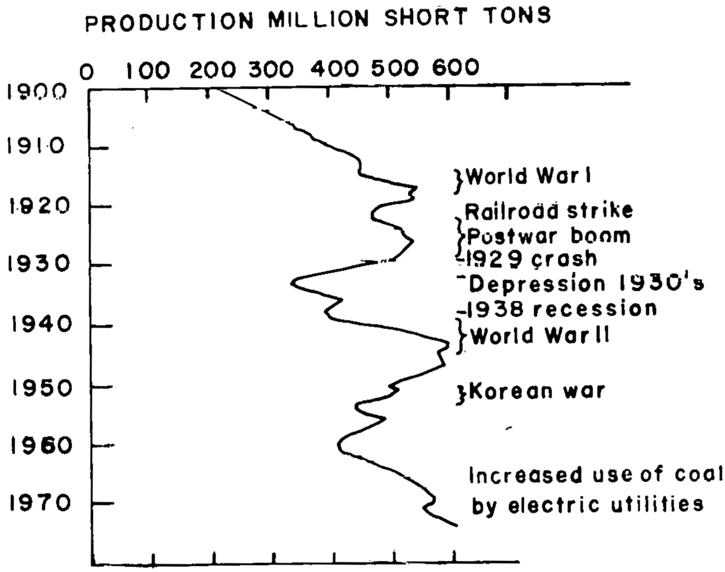


FIGURE 1. - U.S. bituminous coal production, 1900-74.

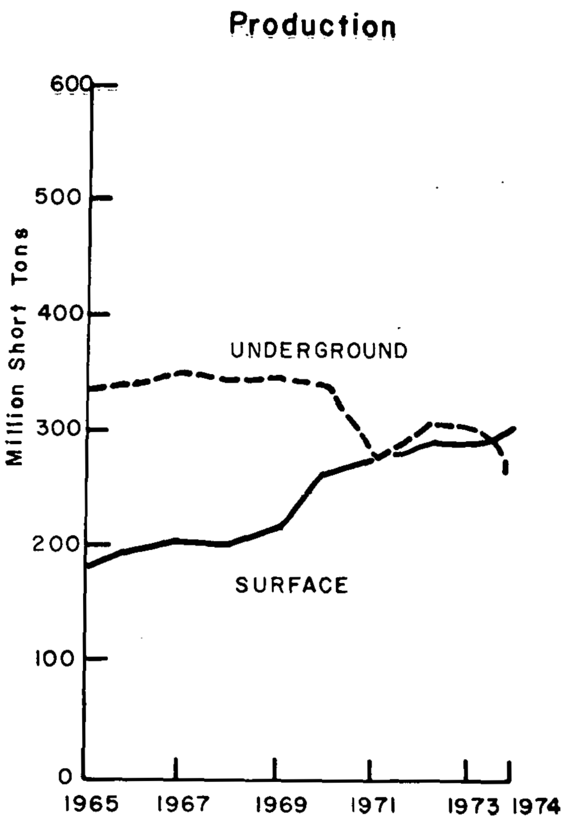


FIGURE 2. - Production from surface and underground bituminous coal mines, 1965-74.

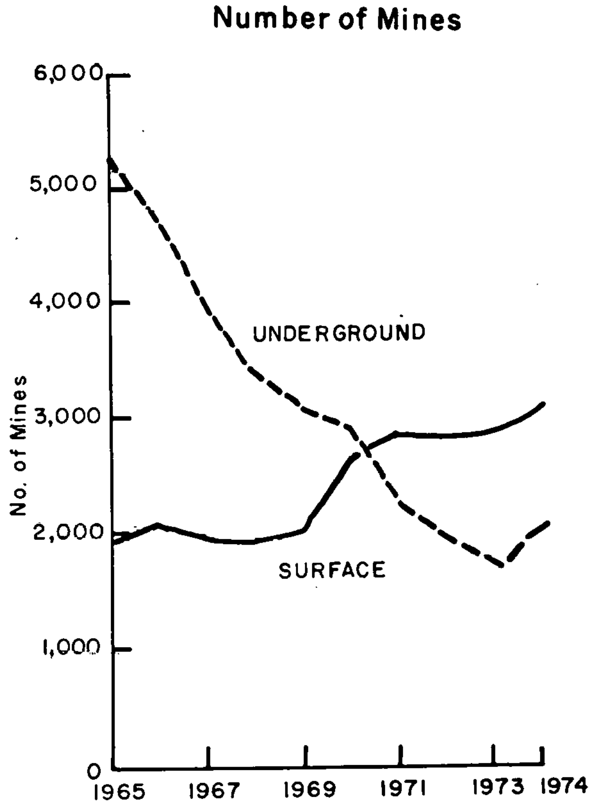


FIGURE 3. - Number of U.S. coal mines, 1965-74.

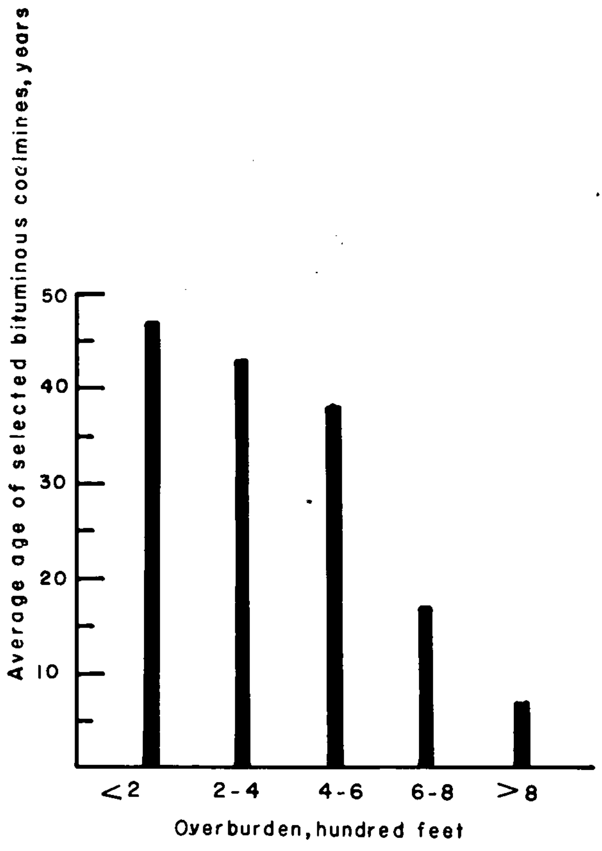


FIGURE 4. - Age of coal mines versus depth in selected coalfields.

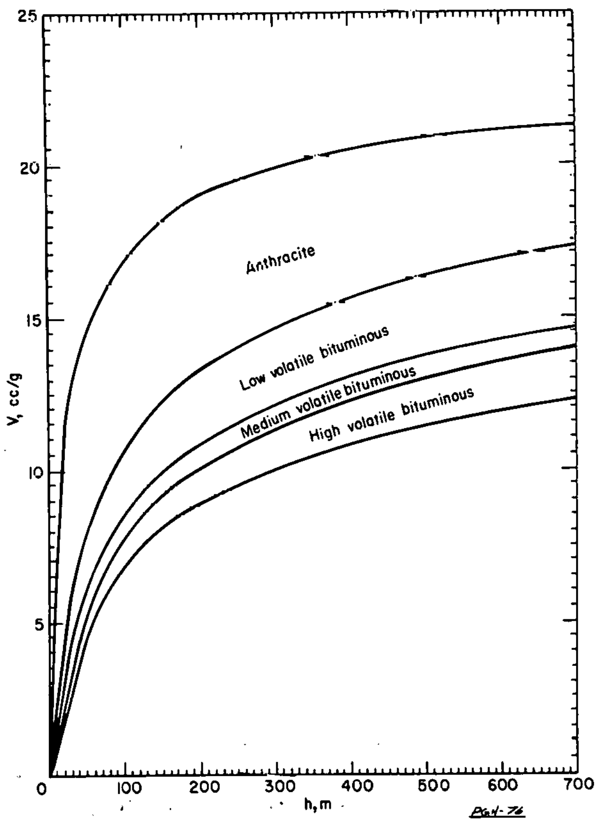


FIGURE 5. - Variation in gas content of coal with rank and depth.

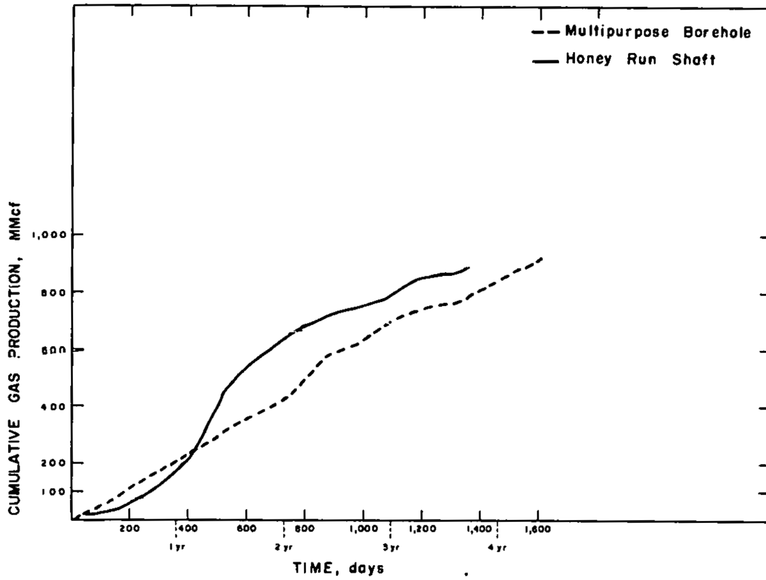


FIGURE 6. - Cumulative gas production from Multipurpose Borehole and Honey Run Shaft, Pittsburgh Coalbed.

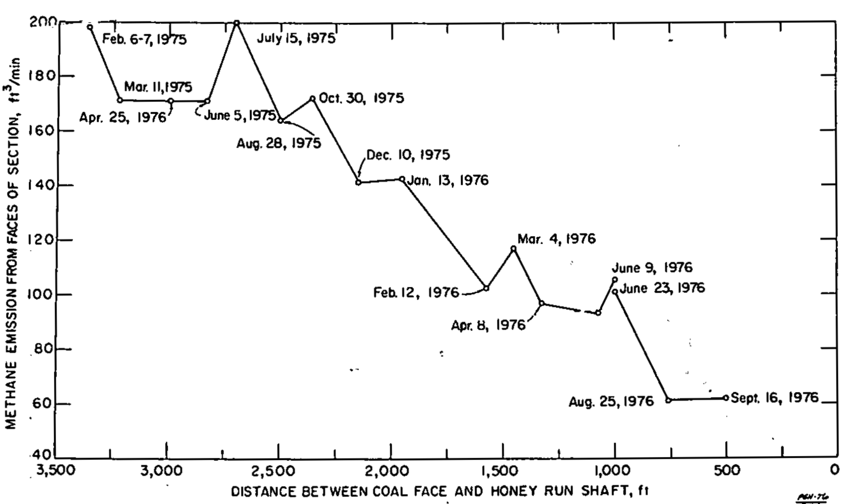


FIGURE 7. - Reduction in face emission as Honey Run Shaft is approached by mining.

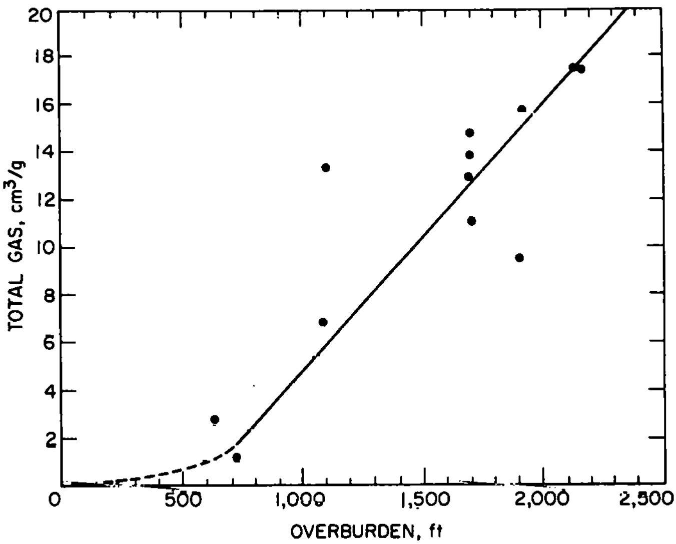


FIGURE 8. - Gas content of Mary Lee Coalbed, variation with depth.

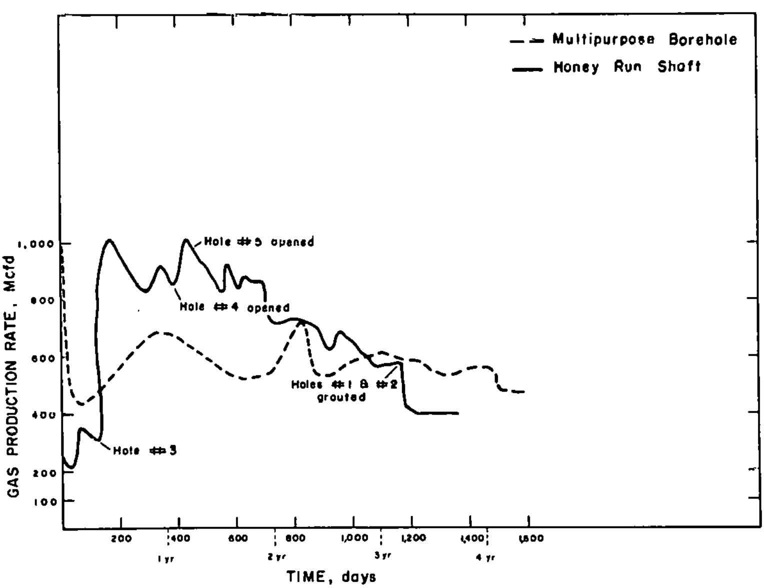


FIGURE 9. - Gas production rate, Multipurpose Borehole and Honey Run Shaft.

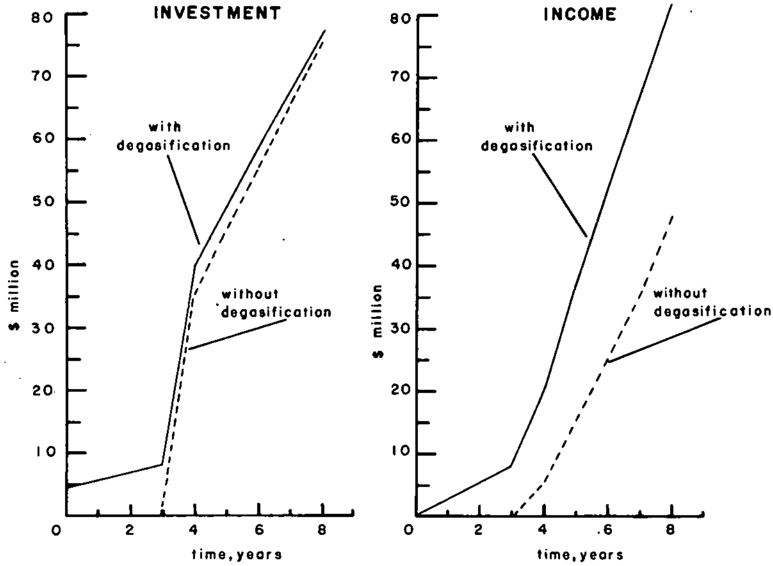


FIGURE 10. - Capital investment and income, with and without degasification.

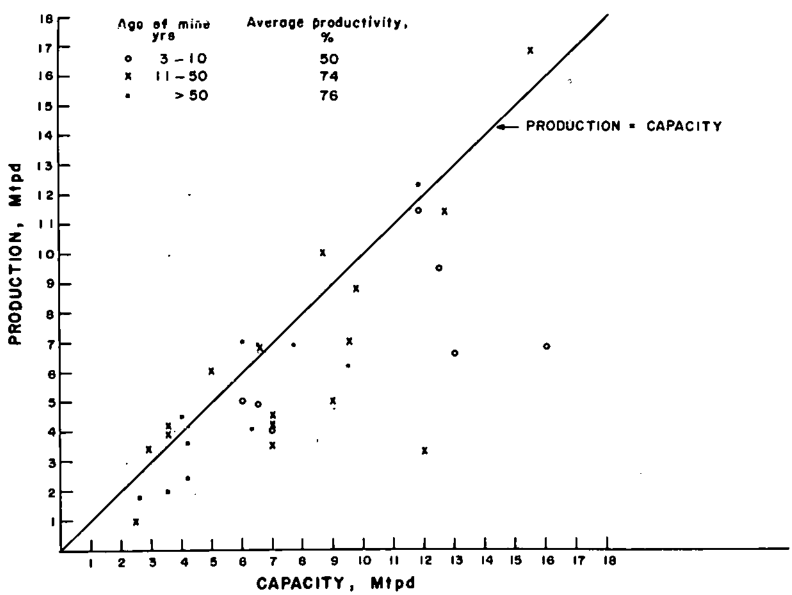


FIGURE 11. - Productivity of mine as of function of capacity and age.

DEVELOPMENT OF BLIND SHAFT BORING SYSTEM

by

James E. Friant, Program Manager, and
Paul F. Sands, General Engineer

Robbins Company, Seattle, Washington
Spokane Mining Research Center, Spokane, Washington

ABSTRACT

The Bureau of Mines is adapting technology existing in tunnel boring industry to mechanize vertical shaft sinking with a Blind Shaft Borer (BSB). The tunnel boring manufacturers have made significant advancements in cutting rock, supporting openings, and mobilizing and demobilizing equipment, all of which is applicable to a shaft sinking machine.

The Blind Shaft Boring system has been designed to bore and concrete line vertical coal-mine shafts up to 1,800 feet deep at rates up to 48 feet per day. The system will be demonstrated at a coal mine site in Alabama starting early in 1978. Final detail design has been completed and the machine is partly fabricated.

The Robbins Company is the prime contractor for the Bureau. Subcontractors to The Robbins Company have designed and will fabricate supporting equipment and will sink the demonstration shaft.

The BSB incorporates basic concepts developed for the tunnel boring industry. The major difference, however, requires that the cuttings be collected at the face and transported to the top of the machine where they are transferred to a skip-hoisting system. A mechanical muck pickup system has been tested in a full-scale facility, and a pickup system now exists which is compatible with machine production goals.

The machine will weigh about 350 tons and require about 1,000 horsepower. The cutterhead will be electrically powered, and the control system, e.g. grippers, guide shoes, torque reactors, and cutterhead advance cylinders will be powered hydraulically. The shaft will be ventilated with up to 50,000 cfm of air. Six operating personnel will be required in the shaft.

INTRODUCTION

The Blind Shaft Borer (BSB) project is part of the Bureau of Mines Advancing Coal Mine Technology program. This energy program is subdivided into 12 subprograms, each addressing a major technological area. One of these is the High Speed Coal Mine Development Subprogram, under which the Blind Shaft Borer is funded. This subprogram is committed, in part, to provide faster, safer, and cheaper methods of sinking large-diameter coal shafts to aid in meeting our expected energy goals. Shaft sinking using present methods and equipment is probably the most time consuming, costly, and hazardous operation associated with the opening of a new mine or in providing additional access to an expanding mine. The basic "drill, blast, and muck" cyclic approach has improved only marginally over the last hundred years with the development of delay detonators, drill jumbos, and mechanical mucking machines. This shaft sinking method is still labor sensitive, and provides such a poor working environment that miners willing to work in the shafts are becoming difficult to find.

The goals of the Blind Shaft Borer are to design, fabricate, and demonstrate a machine that will bore a 24-foot-diameter shaft from the surface to 1,800 feet. However, to test the machine, a commercial shaft will be constructed. Therefore, the shaft site must be cleared and cored, the system must be mobilized and demobilized, the surface facilities provided, and the shaft lined. This required the design and fabrication of a total system, and one that is in accordance with MESA requirements.

The construction of tunnels was a similar slow cyclic operation until the advent of the Tunnel Boring Machine (TBM), or as they are popularly called moles. Hard rock TBMs came into their own in the 1960's. Today they are virtually standard practice in tunnels longer than 1 mile. These mechanical mammoths progress at a rate frequently over 100 feet per day. Segmented or cast lining systems are integrated into the machine shield, and exposure of men to unsupported ground can be virtually eliminated. In short, both the efficiency and the working environment have been dramatically improved by the TBM.

By applying this technology to shaft sinking, similar benefits might be attained. Conventional sinking of shafts over 20 feet in diameter seldom progresses more than 30 feet per week. The Blind Shaft Borer system is designed to bore and line at 36 feet per day, with a possibility of attaining 50 feet per day. The shaft construction costs are estimated to be similar to that of the conventional sinking costs. However, there are cost benefits that give the borer system a distinct cost advantage. Refinements in its design and operation should occur through its use that will further reduce its operating costs.

Project Goals

Nine specific requirements were established in the contract:

1. The BSB system should excavate and line a finished 22-foot-diameter shaft having a minimum wall thickness of 1 foot.
2. The machine should be capable of an instantaneous advance rate of 3 to 5 feet per hour, and an average advance of 10 to 15 feet per shift.
3. The machine should be able to operate at least 35 to 55 percent of the time as with tunnel boring machines.
4. The machine should be constructed to permit in-the-hole assembly and disassembly. Replacement of cutters and other wear parts should not place personnel in hazardous locations.
5. The machine and support systems should operate with a six-person downhole limit.
6. The lining system shall reduce unsupported ground over the machine to the minimum, and shall progress continuously with the machine.
7. A water-control method shall be devised. This may include pregrouting, water pumps, the ability to drill and inject grouts from the machine, and water rings.
8. The system must conform to all applicable MESA regulations.
9. Mobilization and demobilization time shall be kept to the minimum.

Project Plan

Project goals will be accomplished in four phases. The project schedule is shown in figure 1.

Phase I

To aid in establishing design criteria for the system, current and past shaft sinking technology was researched. A preliminary design of both the machine and its support systems was completed. The operating mode of the system was established to obtain comparative performance and cost estimates compared to conventional shaft sinking.

Because the major development effort of the BSB evolved around removal of muck from the face, a full-scale test fixture was built to establish the feasibility of at lease one removal system by the end of Phase I.

Every significant coal district in the United States was considered in selecting a favorable site for the first field test of the machine and its support systems. The more favorable sites were visited, and a specific mine shaft in Alabama was finally selected.

A team was chosen to design and use the BSB system. This team and its major functions is shown in figure 2.

Phase I was started and completed on schedule.

Phase II

The project is currently in the final months of Phase II, which is the detailed design, fabrication and assembly of the blind shaft borer. After completing assembly in the shop, the boring machine will be tested, disassembled, and shipped to the shaft site.

Originally scheduled for completion by the summer of 1977, the project was delayed when testing indicated that the primary muck pickup scheme was only marginal in capacity. The test fixture was completely redesigned and set up with an alternate face profile and pickup system. The revised system doubled the pickup capacity of the machine.

In addition to the BSB detailed design, all systems engineering was completed in Phase II. The support systems were designed, and the techniques for their use with the BSB were established. Operational cycles to produce the required daily advance were established. Methods of grouting, water control and poor ground handling were developed. Planning for machine control, guidance, instrumentation and data collection also were completed during this phase of the project. Manuals for mobilization, operation, safety, and demobilization are being prepared.

Finally, contacts with regulating agencies were made. All the contractual and technical planning requirements for construction of the shaft were completed.

Phase III-Shaft Sinking

At the date of this paper, Phase III of the overall borer project has just begun. The machine is scheduled to arrive at the shaft site in mid-March 1978, and the shaft should be complete before December 1978.

All the support systems will be procured and the site preparations made. Construction of the shaft will proceed in accordance with prepared specifications and the MESA approved plan. During the sinking, data will be continuously collected on the performance of the machine and significant support systems such as lining, ventilation, and water control. Special tests will determine the capacities of the subsystems, and detailed costs of all functions will be recorded. After the boring machine reaches the bottom of the seam, it will be demobilized and the shaft bottom and headings will be completed in a conventional manner.

Phase IV-Project Completion

All data obtained from the demonstration and tests will be analyzed to evaluate the system performance and determine what modifications would be required prior to another demonstration.

BACKGROUND STUDIES

Technology Survey

Before embarking on preliminary machine design, past efforts were surveyed to assure that all available technology was considered. The results showed that there no blind shafts sunk with a downhole boring machine in a single pass. Most blind sinking attempts in rock, at 120-inches diameter or greater, used a drill with circulating mud as the means of removing the cuttings. The largest American single pass drill was 12 feet. A summary of large drilled shafts in shown in table 1. Little information obtained from the literature was directly applicable to the design problems; the best source of information was the tunnel borer industry.

Comparison of Schedule and Cost

Estimated System Performance and Costs

To estimate the benefit of the blind shaft boring concept, schedule and cost estimates were made for a conventional shaft and a bored shaft. This study compared the estimated costs of two shafts, each with a 22-foot finished diameter, concrete lined with a 12-inch minimum thickness, and 1,000 feet in depth. One shaft was assumed to be sunk using current drill and blast techniques, sequentially sunk with the blind shaft boring machine and its support systems. Both shafts were assumed to require a 50-foot-deep collar, similar construction of the shaft bottom, and a heading of about 30 feet in two directions from the shaft.

Schedule Comparison - Figure 3 shows a typical schedule for a shaft conventionally sunk by drilling and blasting. It reflects average performance and crew size. Approximately 11 weeks are required for mobilization including construction of the shaft collar. Due to the potential hazard of blasting damage to the stage, the initial 150 feet of shaft sinking below the collar progresses slowly. Normal difficulties in training and coordinating the crew also contribute to this reduced speed. Average performance assumed in sinking the remaining 800 feet of shaft is 40 feet per week. Six weeks were allowed for completing the shaft bottom The estimate also assumes that additional work in the shaft, such as grouting, water control, bad ground, etc., may add an additional 6 weeks to the overall schedule. Finally, 3 weeks were allowed for demobilizing the system, and the total project was estimated at 53 weeks.

Figure 4 shows the construction schedule for the blind shaft borer system. The 36 weeks estimated for completion of the bored shaft is considerably less than the over 1 year commonly experienced in conventional shaft sinking. As in the case of conventional shaft sinking, 11 weeks are required for site construction, mobilization and collar construction. At 125 feet of advance per week, shaft boring takes only 8 weeks. Removal of the boring machine, completing the station, a 6-week contingency allowance, and 3 weeks for site demobilization, complete the effort. As shown, the actual shaft boring would become a relatively small portion of the overall schedule time. Further reductions in the schedule time will probably occur with future improvements in the mobilization and demobilization time.

Figure 5 shows the benefit of the boring machine's faster sinking rate when three shafts are required, such as when opening a new mine. The main advantage is that only one boring machine is required. The other supporting activities can overlap and repetitive tasks performed simultaneously to speed the overall operation. The time required to sink three shafts is 73 weeks--a savings of about one-third over boring three shafts separately.

Relative Costs--Table 2 presents a comparison of estimated costs for a conventionally sunk shaft and a bored shaft. Both shafts are assumed to be 22 feet in diameter and 1,000 feet deep. This estimate does not include costs for severe water problems or bad ground. The estimates are a compilation of information received from several sources and do not represent a particular shaft or contractor. The total shown is somewhat higher than those currently found in areas where ground conditions are favorable, but is lower than found in more difficult regions. The analysis indicates that under these conditions, sinking with the BSB system cost about the same as when sinking conventionally.

The cost comparison changes, however, with a change in depth. As the shafts become deeper, the boring machine system would have an increasing advantage. At depths shallower than 1,000 feet, conventional sinking would have the advantage. This break-even depth, however, is expected to decrease in the future. This prediction is based on expected increases in labor and material costs, and on improvements in the efficiency of the boring system.

The cost estimates do not take into account the benefits of a shorter investment period before the shaft becomes productive and from possible earlier production of coal, or better working conditions and greater safety, all of which favor blind-shaft boring.

Selection of Demonstration Site

The following criteria were used to select a satisfactory site for demonstrating the BSB. These were classified as either "critical" or "favorable."

Critical--Site characteristics considered mandatory to successful completio of the demonstration.
Favorable--Site characteristics which would contribute to efficiency or convenience of operation, and would tend to minimize variables.

The criteria defined as critical were:
(a) A shaft depth of 1,000 feet or greater.
(b) A coal mining company interested in cooperating in the project.
(c) An existing mine development schedule compatible with the proposed program.

Favorable characteristics established were:
(a) Reasonably competent rock and consistent strata.
(b) An accessible site, preferably in a mild climate.
(c) Modest water inflow with proven control methods.
(d) No major environmental (spoil or water disposal) problems.

The criteria were then weighted, with the successful development of the machine as the foremost goal.

Major coal fields defined in the Keystone Industry manual were considered. All but a few localities were eliminated because of failure to meet the requirements for shaft depth. Other published statistics, such as underground capacity projections, mine size, topography, and geology were considered.

Mining companies and shaft contractors were contacted. Alabama was chosen because of the depth of shafts, relatively consistent and competent ground, and minimal ground water.

Preliminary Machine Design Trade Study

From a rock penetration standpoint, the standard, full-face, hard-rock tunnel boring machine has done a satisfactory job in coal-measure rock. In applying this technology to a vertical sinking situation, however, problems such as muck removal, cutter changing, and general access to the face had to be addressed. In vertical sinking, gravity and the invert scoops for picking up the muck cannot be used as they are in tunnel borers. Also, when cutters must be inspected or changed, the head of the tunnel borer is simply pulled back from the face, and personnel can to the front of the head with little hazard. In the vertical situation, however, the head would have to be securely blocked, and handling the 200-pound cutter assemblies from a prone position would be difficult.

Several alternate configurations were studied, and the results of these studies are discussed below.

Plunging-Head Borer--A plunging head design with a one-half shaft diameter head was studied. This configuration is shown in figure 6. The operating sequence includes plunging the cutter 3 feet down, relaxing the lower gripper shield and rotating the head 30°, reactivating the gripper and repeating the plunging of the cutter.

The design had two advantages: there would be good access to the face for cutter changing, probe drilling, and hand mucking; and the mechanical muck pickup system is simple. The design was abandoned because its complex cyclic operation would provide an unacceptably low penetration rate.

A Twin Head Borer--A twin head concept utilizing rotating twin discs mounted on a shaft at the machine centerline was studied (fig. 7). The twin cutterheads advance along a double-lead helix, cutting at the rock face with the leading edges. This advance motion is produced by the synchronious action of the main drive cylinders. The advantages would be access to the face and a simple muck removal system. Disadvantages include frequent stopping and starting of the cutterheads, and a complex hydraulic distribution system.

Full-Face Machines--Full-face designs such as are used in tunnel borers were also studied. Figure 8 shows a design where the muck is picked up by vertical bucket elevators from the low point in the shaft at a radius of 5 feet. The muck is dumped into a large storage hopper and transferred to a centrally located skip. This design was rejected because the large-diameter main bearing and seal did not adapt themselves to demobilization of the machine from the hole.

Figure 9 shows the full-face, center pickup concept. After a formal weighted trade study, this design appeared to be the most favorable, and was therefore selected for further development. The disadvantages of poor access to the face, and the difficulty in changing cutters was resolved by using a spoke-like construction for the cutterhead and using cutters which mounted through the head and transported to the skips using two bucket elevators located in the center of the machine.

MACHINE DESIGN

Muck Pickup Tests

The major departure from proven TBM technology is the muck pickup system. A full-size test fixture was constructed, and different ideas were tested.

A minimum capacity-goal of 40 cfm of muck was established. However, a goal of 80 cfm was informally set because if 80 cfm of muck could be removed, the system would be more reliable. Table 3 shows the relationship between mucking rate and capability of the machine to advance. At a rate of 40 cfm, the boring machine could advance about 18 feet in one shift.

Figure 10 shows the full-scale muck pickup test fixture. The muck used was tunnel borer spoil similar in physical shape to that which would be created by the boring machine. Figure 11 shows that the system was limited by the ability to move the cuttings into the bucket elevators. By increasing bowl speed (cutterhead rotation) to 4 rpm, 30 cfm of muck was the maximum obtained. As shown in figure 12, to achieve a 40 cfm minimum pickup, a head speed of about 5 rpm was projected which is faster than desired to optimize machine life and minimize cutter-bearing wear. Also, at an instantaneous penetration rate of 3.2 feet per hour (40 cfm), penetration would be 0.13 inches per revolution at 4 rpm. Based on experience with tunnel boring machines, this is insufficient penetration to optimize cutter wear and the machine's ability to break rock.

Therefore, the mechanical pickup design was revised. The new system is shown in figure 13. The cutterhead profile was changed and two flight conveyors were added. The bucket elevators and main structural design of the machine was retained. The revised system has a capacity of more than 70 cfm when handling either dry material or when the head is partially submerged in water.

Alternate Pickup Studies

While the testing program progressed, a secondary study was made of vacuum, hydraulic, and pneumatic muck-handling systems. The study covered hoisting methods as well as primary pickup methods in various combinations relative to both current requirements (40 to 50 cfm) and future requirements (about 100 cfm). The following combinations were investigated:

Primary Pickup	Hoisting
Mechanical	Mechanical
Mechanical	Hydraulic
Mechanical	Pneumatic
Hydraulic	Hydraulic
Vacuum	Mechanical
Vacuum	Pneumatic

Highlights of the study follow.

Two small vacuum-type pickup systems were borrowed from distributors and tested. While they picked up most dry tunnel borer spoil well, noise, downhole horsepower requirements, wet-muck handling, equipment size, and the large TBM chips were all considered problems requiring study beyond the scope of this project.

Pneumatic hoisting was examined as an alternative to the skips. Several existing systems were considered, but none have been designed to lift material of this size and specific gravity vertically at the current capacity requirements (40 cfm). Also, wear, reliability, and the effectiveness of noise control have not been evaluated. Pneumatics may be a feasible approach, but a development program was required to prove the extrapolation from current technology. This is being done under a separate contract.

A hydraulic hoisting system using a feeder to introduce solids into a pipe carrying clear water was studied. A new feeder that would feed 200 tph required designing and testing, and this was considered beyond the scope of the project.

Hydraulic pickup and hoisting using slurry pumps did not appear practical because the size of impeller opening for passing large particle sizes, uses prohibitively large volumes of water. The percent solids would be low and clarifying the excess water on the surface would be costly. If the muck were crushed, the size of the impeller and the volume of water would be reduced, but a crusher at the face appeared impractical. However, a 100-cfm muck-capacity machine would be more compatible with a slurry pumping system. A preliminary design of such a system is shown in figure 14.

As shown on figure 15, either hydraulic or mechanical primary pickup schemes may be possible in conjunction with a slurry pumping system. The major problems are cost and environmental considerations at the surface.

The results of this study, as summarized on table 4, indicated that the most reliable and rapid approach was to continue with the development of the mechanica system.

Final Boring Machine Configuration

Figures 16 and 17 show the current machine design and depict its principal functioning components.

Cutterhead--Figure 16 shows those components of the machine which are attached to the cutterhead structure and rotate with it. The cutterhead is fitted with 56 standard 13-inch-diameter cutters arranged in an approximate eight-spoke pattern. The outside diameter of the cutters is 24 feet, 5 inches. The face profile dips to a low point at a radius of about 6 feet. Cuttings are moved to this low point by a series of plowlike scrapers mounted on the cutterhead. Scrapers may be inspected or replaced from behind or from within the spokes of the cutterhead.

Primary muck pickup is by two single chain-modified Dosco^R flight conveyors operating in a horizontal plane. These units which are attached to the cutterhead and rotate with it, wipe through the muck piled in the low point of the face, and move it up a ramp towards the center of the machine. The conveyors then load dual bucket elevators which are also attached and rotate with the cutterhead.

Chain mounted buckets lift the muck approximately 47 feet vertically and dump into the annular collection hopper. The bucket elevators are hydraulically driven through the top sprockets, and with the current bucket spacing, have a capacity of lmost 100 cfm. At the top of the machine, the muck is dumped into a collecting arousel where blades extending from the elevator housing plow the cuttings to an pening. The muck drops through the opening into one of the skip loading pockets. Both carousel openings and the bottom of the pocket are controlled by hydraulic gates integrated into an overall interlock system. Distribution of the muck to the correct pocket and skip are controlled automatically.

Power to the cutterhead is provided by six permissible 125-hp electric motors (950 VAC) driving through a two-speed gear box. Manually shifted drive options at 1.91 and 3.16 are available. At the slower speed, the head has a maximum torque capacity of 2.05×10^6 ft/lb.

The cutterhead is constructed with two bolt-on outriggers which can be removed prior to lifting the cutterhead from the completed shaft. This reduces the overall size and weight to permit hoisting through the decks of the galloway stage suspended above the machine.

The 100-gpm Moyno pumps also rotate with the head. They are continuously available for dewatering even while the head is rotating. Their output will pass either through a cyclone or dump directly into a settling tank. Clear water pumps will send the excess water to surface.

<u>Cutterhead Support</u>--The nonrotating machine components are highlighted in figure 17. Most of the nonrotating components move forward with, and bear the face. The grippers, their structural ring, and the rear shield are the only stationary components while the machine is operating.

The cutterhead support is the principal load-bearing structure of the machine. It provides both torque reaction from the rotating cutterhead to the gripper ring and reacts the thrust required for penetration. In addition, this same structure absorbs the thrust of the stabilizer shoes on the shaft wall, providing directional stability and steerage for the machine. Finally, the cutterhead support houses the nonrotating inner race of the 86-inch I.D., 98-inch O.D. Torrington double rowed, tapered roller bearing. Thrust capability of the machine, 1.43×10^6 lbs at 2,000 psig, is provided by nine hydraulic cylinders acting in groups of three, each at 120° to the other. The six electric drive units, as well as five additional electric hydraulic power units (total 230 hp), are mounted on the cutterhead support eck. These latter units provide the power for all hydraulic cylinders, conveyors, uck elevators, and water pumps. To provide power to the conveyor and elevator ydraulic motors, the high pressure output is passed through a swivel at the top of the machine.

<u>Column and Deck Structure</u>--The center column of the machine is a 30-foot-long cylinder, 60 inches in diameter, supported at the bottom by the cutterhead support and extending to the top of the machine where the hydraulic swivel is located. The cylinder provides the mounting structure for the muck-transfer deck, and three additional decks from which the machine is controlled and the shaft-lining forms are handled. The muck measuring pocket, chutes and stationary portions of the muck handling carousel are also supported by the center column.

The transfer deck is the lower terminous for the muck skips. Mounted underneath the deck is the sheaves for the skip-guide ropes. The other three decks are supported both from the center column and from the transfer deck. These decks are spaced to allow a compatible boring and lining cycle using an 18-foot increment. The second deck will house a control panel for the machine operator, and the third deck houses an observation panel for monitoring skip loading.

Shield--Temporary ground support is provided from immediately behind the cutterhead to 2.5 feet above the transfer deck by a full shield as shown in figure 20. Windows in the shield permit operation of the cutterhead stabilizer shoes and grippers.

Machine Operation

Machine advance is cyclic, having a thrusting stroke of 30 inches followed by a regripping cycle. The cycle is initiated by releasing the gripping pads and moving the gripper ring and tail shield forward 30 inches. A new set is taken against the shaft wall. Then the head is turned and cutting commences at a rate controlled by the quantity of hydraulic fluid metered into the thrust cylinders. The forward stabilizer shoes are fitted against the wall but not gripped. Gradual steering corrections can be made by a deliberate unbalance in the force on these shoes. When the machine is extended to its maximum 30-inch stroke, the grippers are again released, moved forward, and reset. Theoretical utilization of the machine is over 90 percent, but in actual practice, overall utilization of the machine probably will be in the range of 50 percent. Overall utilization is defined as the actual time the cutters are turning against the face divided by 24 hours per day.

Guidance

Steering of the machine will be manually controlled in response to two vertical laser beams set at 180° from each other. The laser guns will be permanently anchored in the shaft collar and beamed through 5-inch pipes imbedded into the shaft lining. By blowing air down the pipes, the beams will pass through clean, turbulent air. Inaccuracies due to light bending or diffusion will be minimized. Photodiode array-type targets will be used having a resolution of a few millimeters. The system will feature an on-board calculator that, using head geometric constants and inputs from level indicators, will calculate the precise location of the center of the gage-cutting circle. In addition, it will have the capability of predicting the machine's position in several increments of machine length ahead, up to 12 feet. The principal display will be a four-quandrant, digital readout. The operator will keep offset readings at zero, or at the minimum number possible, by gradual machine corrections. Other readouts will indicate tilt in two directions, and rotation and deviation of the beam separation from a nominal value.

SYSTEMS DESIGN

BSB shaft sinking depends upon efficient backup systems and their design is dependent on the design of the machine and goals of the project. As soon as the machine design was underway, data such as component size, advance rate, spoil volume, and percent utilization affected the design of the support systems. For example, the large onsite crane necessary to mobilize the machine could be used to lift large preassembled sections of the headframe. This simplified the design of the headframe and allowed it to be assembled over the shaft more quickly.

Surface Facilities

Figure 18 shows the arrangement of the site facilities. It is typical of a modern, conventional sinking site where multideck work stages and multiple skips are used, except the shops and the provisions for down power and services are more elaborate.

The headframe was especially designed for this project by Lake Shore Inc. It was designed to withstand the operating loads of the work platform and man hoist in addition to the breaking load of one skip-hoist rope. Its design has some unique features so its structural support members were completely stress analyzed.

A gully a few hundred feet away will be used for spoil disposal. Spoil banks will be graded, drained, and a vegetative cover will be planted. Run-off water will be held in a settling pond and be treated according to the Alabama Water Improvement Commission regulations before it would be released from the pond.

Electric power will be provided by a 46,000-volt power line to a main transformer station. A 7,200 volt, 4,000 kva and a 4,160 volt, 1,750 kva power feeder line will be extended on poles to the shaft site.

Visitors will be encouraged to observe the demonstration. These visits will be coordinated to minimize interference with construction activities and to ensure personnel safety.

Machine Mobilization

As illustrated in figure 19, a conventional collar will be constructed to a 40-foot depth. The base of this collar must be strong enough to withstand the gripping forces of the machine. Some cracking of the collar is allowed however, because it will have a 25-foot inside diameter, but will be lined to 22 feet when the galloway and lining forms are installed.

Figure 20 shows machine installation. Major subassemblies of the machine will be built on the surface and lowered by crane into the shaft collar. When assembly is complete the machine will progress with the muck being disposed by crane and bucket as shown in figure 21. When the machine has progressed to about 95 feet, the operation will be halted while the headframe and galloway are installed. Because of available crane capacity, the stage and headframe can be partially assembled on surface and the final setup should proceed rapidly. This is illustrated in figure 22. At this point standard operations can begin.

Systems Operating Cycle

The machine and galloway progress in a cyclic fashion. The machine bores ahead 18 feet and concreting operations follow in an 18-foot pour.

The three-deck galloway will be suspended from a double-drum winch by two wire ropes in double purchase. It is located from 6 to 24 feet above the boring machine to permit placement of the shaft concrete. An 18-foot collapsible form will be used, which is similar in concept to a conventional hanging form.

The operational cycle is illustrated in figure 23. After an 18-foot advance, the machine will have reached the maximum separation distance from the stationary galloway stage. The stage houses a 7,200/950-volt transformer that will be deenergized from surface. The galloway stage can then be lowered about 12 feet to the most convenient position from which to extend shaft pipe lines. During the pipe extension operations, all other shaft activities will be at a standstill. The flexible, telescoping arrangement used with the exhaust-ventilation system will minimize interruption and permit the most rapid installation of a new length of duct to reestablish machine ventialtion. Having extended all service lines, the galloway will be lowered another 6 feet to its concreting position. At this point, following a satisfactory methane check, the 7,200/950-volt transformer will be reenergized from surface. This will reestablish the boring machine cutterhead power, auxiliary power, and the pumps. Because the lighting and instrumentation system will be powered by separate circuits, lighting on the machine and the machine-mounted methane detectors will be continuously operable.

The curb ring, attahced to winches on the galloway stage, will be removed from the bottom of the previous concrete pour and lowered 18 feet. It will be supported on eleven 1-1/8-inch-diameter hanging rods and centered by using the two machine guidance laser beams. The 18-foot concrete form will then be lowered using galloway-mounted winches and connected to the curb ring. Next, the concrete will be supplied through two, 6-inch-diameter pipelines to a galloway-mounted dash pot, to a distribution box, and then through pouring doors in the form. From this point, shaft-boring operations can proceed concurrently with concreting.

Operational Cycles

The operational cycles which will be employed during the first demonstration are shown in figure 24. Cycle 1 will be used during about the first half of shaft construction. The boring machine is available for two shifts during which the 18-foot advance is made. The third shift is reserved for services extension and lining. This represents an overall utilization of the machine of only 23 percent which is typical for tunnel borers during the initial startup of equipment and training of personnel.

Successful use of Cycle 2 is the goal. To do this, the boring machine must progress 18 feet in an 8-hour shift. This, as shown in previous tables, is compatible with the 3.2 ft/hr instantaneous boring rate of the machine and the 40 cfm of cuttings removal from the face. The services and lining shift is reduced to 6 hours and overlaps boring activity. This cycle will produce 36 feet per day. Cycle 2 represents an overall machine utilization of 46 percent. This is frequently attained in tunnel boring projects.

The third cycle represents the maximum feasible advance with the system. Three boring cycles of 7 hours each are accomplished in a 27-hour period. This represents 54 feet of advance in 27 hours and a 62.5 percent overall utilization of the machine. At times this rate is accomplished by tunnel borers, but seldom averaged. These rates, however may not be attained during the first demonstration.

Skip Design

The proposed skip cycle is compatible with the machine instantaneous advance rate and the proposed operational cycles. The cuttings will be removed with a balanced, two-skip hoisting system, each skip having a capacity of 100 cubic feet (5 tons). Because distance from the surface to the borer continually increases as the borer progresses, a single-clutched, double-drum mine hoist will permit the length of the rope to be adjusted. The system is designed to accommodate just over 30 inches of machine travel (shaft advance) between declutching operations. This matches the 30-inch stroke length of the thrust cylinders on the boring machine. The rope length will be adjusted when both skips are in a terminal position, one at the headframe and the other, which is attached to the declutched drum, resting on the transfer platform of the boring machine. the timing will coincide with regripping the boring machine. Skip cycles are shown in table 6. At 1,200 feet, with the galloway and the machine at the point of maximum separation, one cycle consumes 128 seconds. This permits muck removal at a rate of 40.5 cfm.

The empty skip will pass through the work platform and collar doors at creep speed and then accelerate to 1,500 fpm. It will pause 20 feet above the work platform, and the hoist operator will indicate its presence. The downhole skip attendant in turn will signal to bring the skip through to the loading position. When it leaves for the surface, it also proceeds at creep speed until it clears the galloway. Passing the skip through the galloway consumes 90 seconds of the 148-second cycle time.

Loading the skips is controlled by the skip attendant and an interlock system that directs the muck to the proper storage pocket and prevents loading of the same skip twice. The skip dumps automatically from the bottom by using Jeto-type dump scrolls mounted on the headframe.

Shaft Completion

When the machine reaches the bottom of the coal seam, it will be dismantled in the hole and lifted through the galloway stage as illustrated in figure 25. The two skip-guide ropes, each in double purchase, will be filled with a lifting harness for removal of the larger machine components. The heaviest item will be the cutterhead and bearing assembly which weigh about 60,000 pounds. As illustrated in figure 26, the galloway will then be used as a work platform for completing the shaft station and headings.

An alternative demobilization technique would be to bore some distance beyond the seam before removing the machine. This would provide a sump into which the muck from the shaft bottom could be dumped. In this way, the galloway could be sassembled and removed from the shaft ahead of the boring machine. This alter-tive method will not be tried on the first shaft.

Ventilation

Figure 27 illustrates the method of ventilating the shaft. There will be an intake and an exhaust fan on the surface, one on each side of the shaft. The intake and exhaust ducts will be 36 inches and 30 inches in diameter, respectively.

Up to 30,000 cfm of intake air will be supplied to the bottom of the galloway and up to 20,000 cfm will be exhausted from the face through the center column of the machine. In this manner, fresh air will flow down over the boring machine and operator, sweep the face and be drawn into the exhaust duct at the face. The excess of intake air over that exhausted by the exhaust fan, provides 10,000 cfm of fresh air that passes over the galloway stage as it moves up the shaft to the surface. When the boring machine is in operation, the exhaust-ventilation system functions both as a methane-dilution and dust-collection system. The exhaust sweeps all the main dust-producing areas including the elevator dump chute and the skip loading station.

Automatic methane monitors will be provided, both underneath the machine deck (lowest deck) and in the exhaust ventilation duct. A standard one percent warning and a two percent automatic-shutdown will be used. In addition, a third methane monitor will be in the shaft on the galloway and will monitor the lighting system. In this way, downhole lighting can remain active unless the shaft methane level reaches one percent.

Shaft-Water Handling

The first step of water control for the BSB is a thorough site investigation and a grouting program. The goal of grouting from the surface is to minimize additional grouting from within the shaft during sinking. Grouting also will stabilize incompetent ground. Full-depth probe drilling will verify the success of the program. Provisions to control excessive water inflows from within the shaft during sinking are:

a. Above about 400 feet, backwall grouting will be used.

b. Below 400 feet, panning, a collection ring, and a pump station in the aft will be constructed.

c. Provisions for downhole drilling and grouting have been designed into the machine.

To cope with possible water inflow, the boring machine will be equipped with two, hydraulically-powered Moyno pumps mounted in the cutterhead, each with the capacity of 100 gpm. An additional 200-gpm-capacity pump can be mounted on the lowest boring machine deck for emergency situations. This pump will deliver directly to surface.

CONCLUDING STATEMENT

As of this month, the design is complete and the machine is in the advanced stage of assembly. Existing technology has been used whenever possible, but where new concepts were required, design decisions were verified through testing. Although the machine is in fabrication, it will continue to be refined wherever possible.

The shaft-sinking activities will be monitored and evaluated continually to identify and resolve potential problems. Data from this demonstration will be used to improve the reliability and reduce shaft-sinking costs.

The BSB has many potential benefits:

For coal owners, reduced costs through more rapid mine development and less time between planning and production.

For miners, more desirable and safer working conditions.

For shaft contractors, an improved system that more easily meets MESA standards.

Finally, if a significant increase in coal production is needed, the Borer would provide a faster method for sinking the many blind shafts required.

The Bureau and The Robbins Company extend their invitation to visit the site in Alabama, and observe the demonstration.

ACKNOWLEDGMENTS

We thank the subcontractors; Cementation Company of America, U.S. Steel, Battelle Northwest, the Paul Weir Company, and Lake Shore Inc. We also thank the many coal company operators, shaft contractors, and equipment manufacturers that have contributed to the design. We believe that the various members of the team provide an expertise uniquely applicable to a successful project.

FIGURE 1. -

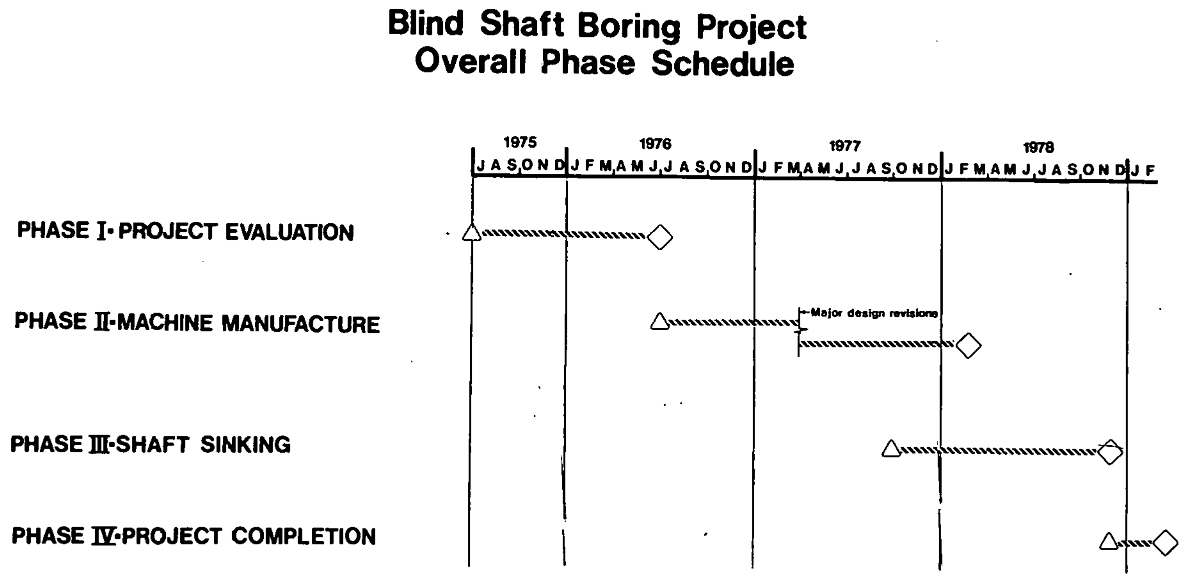

Blind Shaft Boring Project
Overall Phase Schedule
1975
1976
1977
1978
J A S O N D J F M A M J J A S O N D J F M A M J J A S O N D J F M A M J J A S O N D J F
PHASE I · PROJECT EVALUATION
PHASE II · MACHINE MANUFACTURE
Major design revisions
PHASE III · SHAFT SINKING
PHASE IV · PROJECT COMPLETION

FIGURE 2. -

BLIND SHAFT BORER
PROJECT TEAM

The Robbins Company

- Program Management
- Machine Design & Fabrication
- Machine Operation
- Field Support & Training

Cementation Company of America

- Collar & Shaft Design
- Construction
- Management
- Galloway Stage
- Forms & Concreting
- Grouting

Battelle Northwest

- Data Aquisition & Reduction
- Instrumentation
- Guidance
- Analysis & Forecast

Lake Shore Inc.

- Head Frame
- Skips
- Ropes, Sheaves, Cages
- Hoists

U.S. Steel

- Provide Site & Funding
- MESA Requirements
- Field Support

The Paul Weir Company

- Site Requirements
- Economic Studies
- Water Control

G A I Incorporated

- Geological Studies

FIGURE 3. -

CONSTRUCTION SCHEDULE - CONVENTIONAL SHAFT
1000' FT. DEPTH

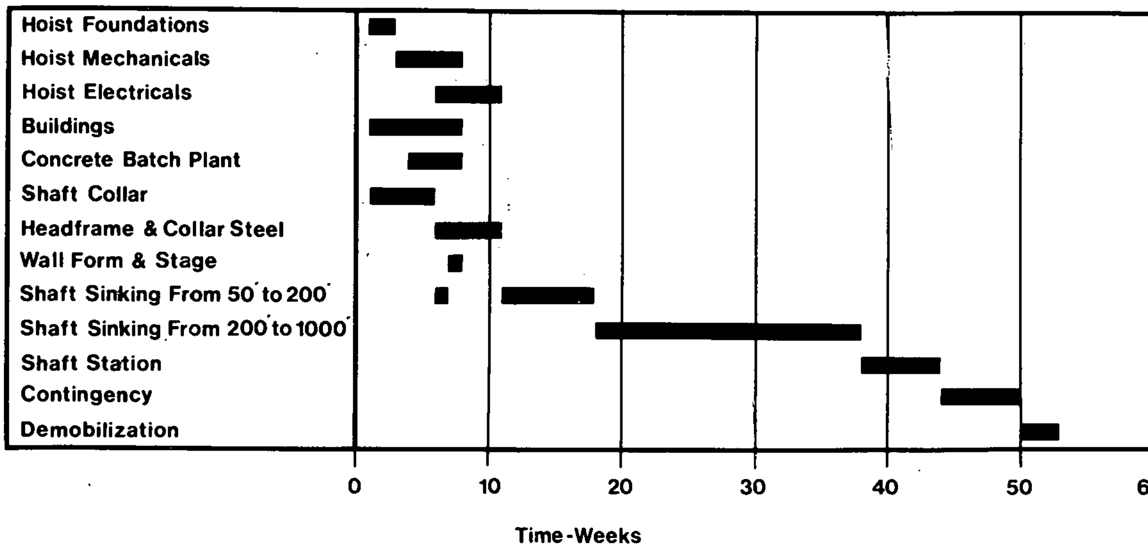


FIGURE 4. -

CONSTRUCTION SCHEDULE - BORED SHAFT
1000 FT. DEPTH

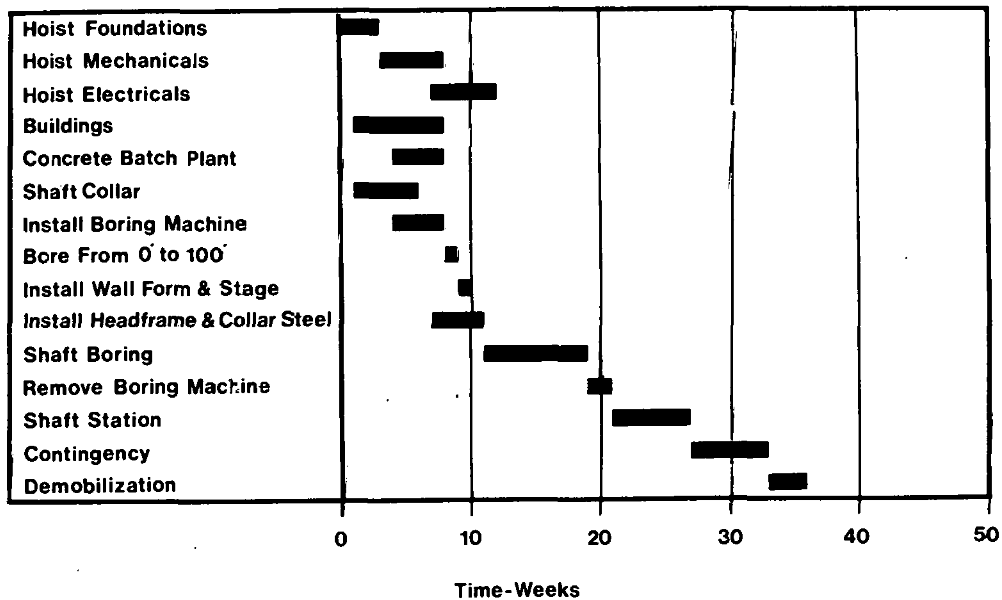


FIGURE 5. -

CONSTRUCTION SCHEDULE
THREE SHAFT SERIES - BLIND SHAFT BORING MACHINE

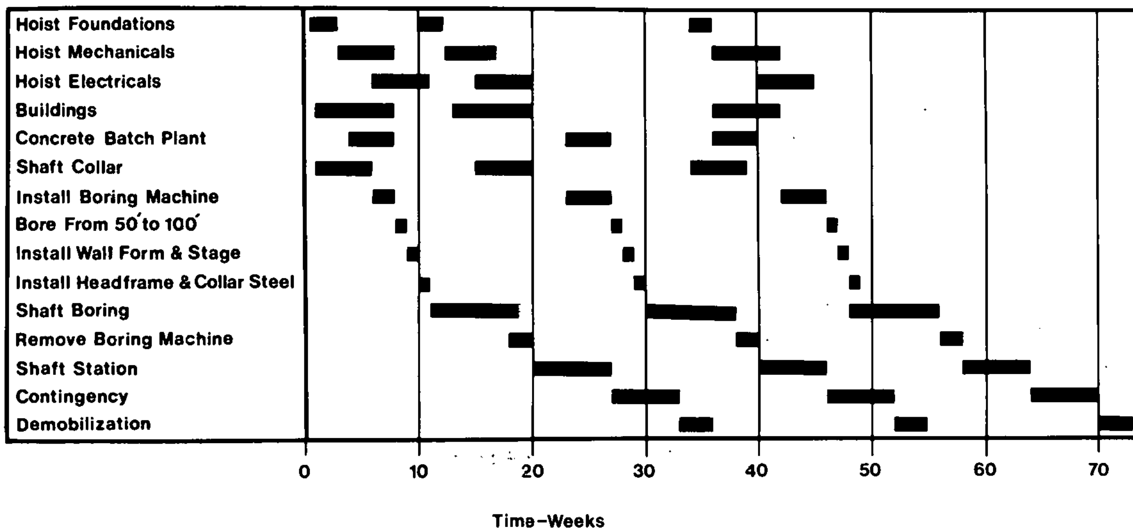


FIGURE 6. -

BLIND SHAFT BORER PLUNGING HEAD

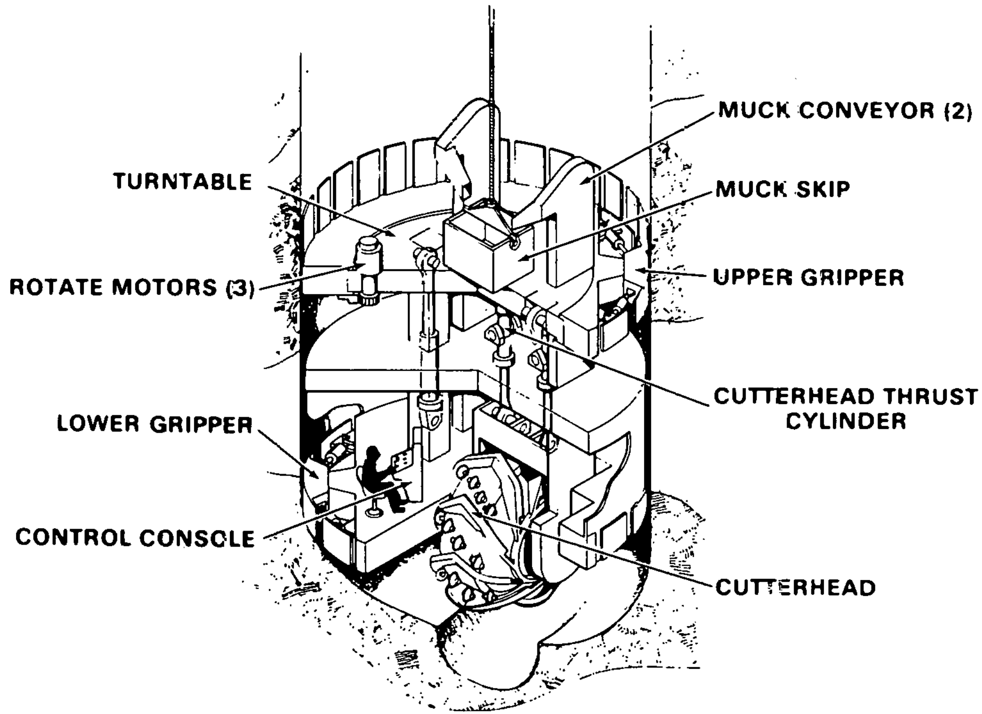


FIGURE 7. -

BLIND SHAFT BORER
TWIN HEAD

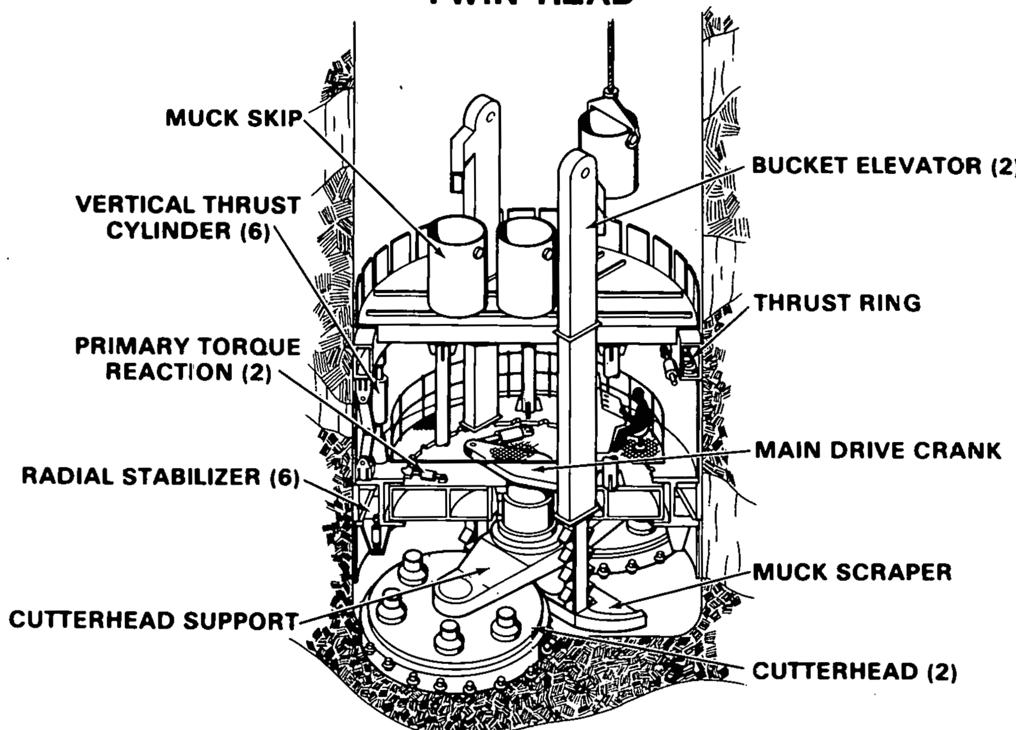


FIGURE 8. -

BLIND SHAFT BORER
FULL FACE

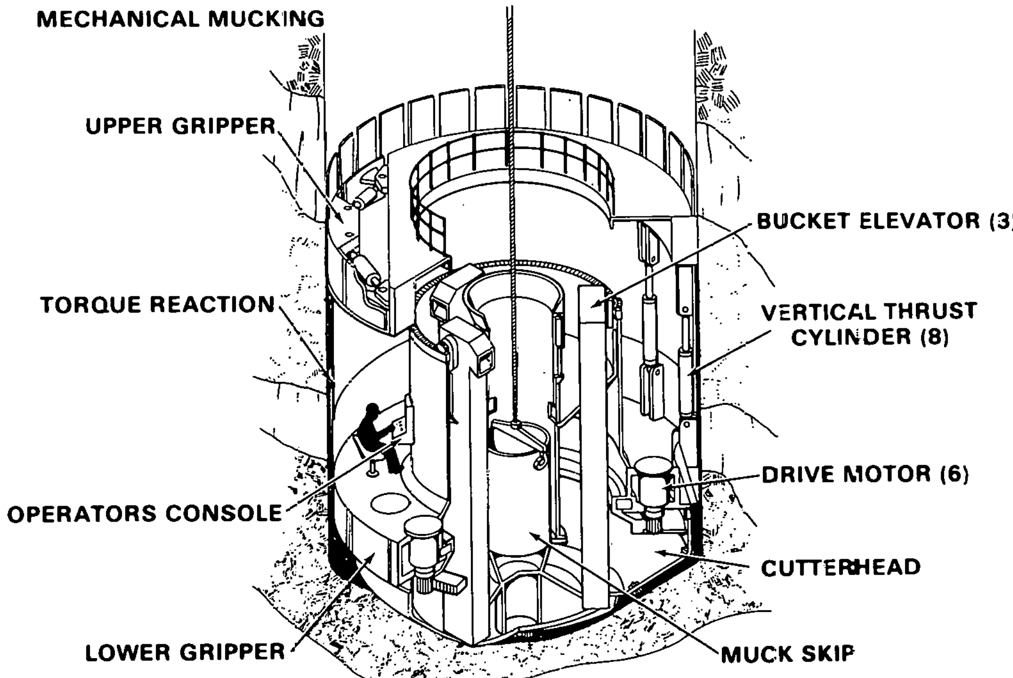
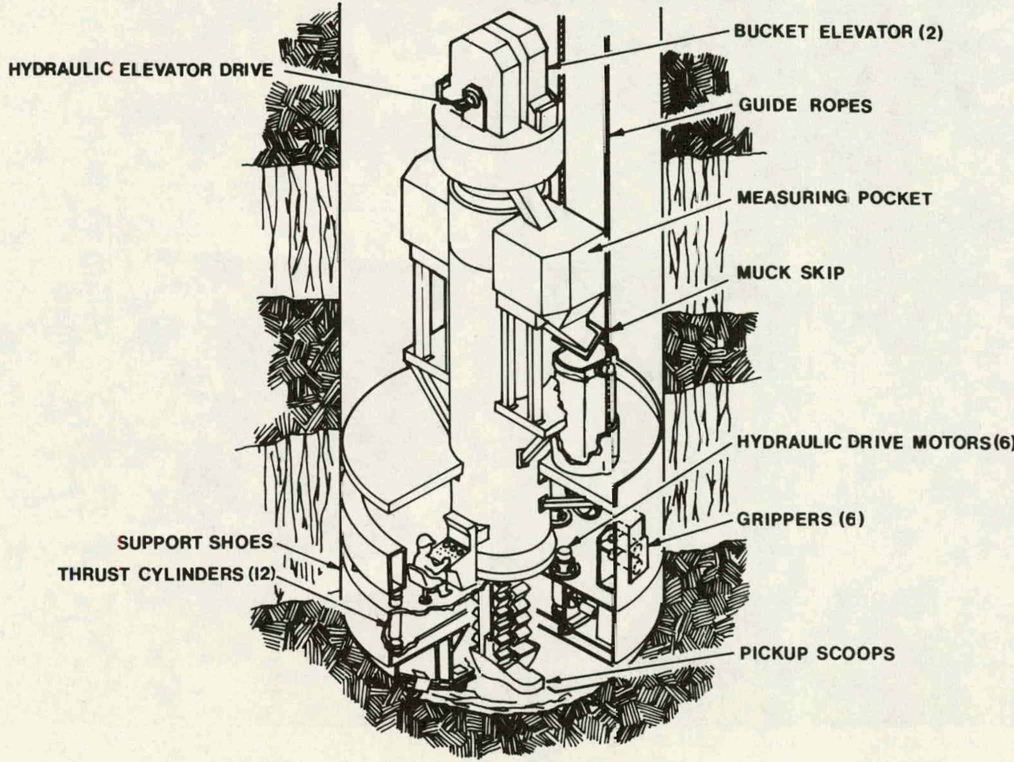


FIGURE 9. -

BLIND SHAFT BORER
FULL FACE MECHANICAL MUCK PICKUP



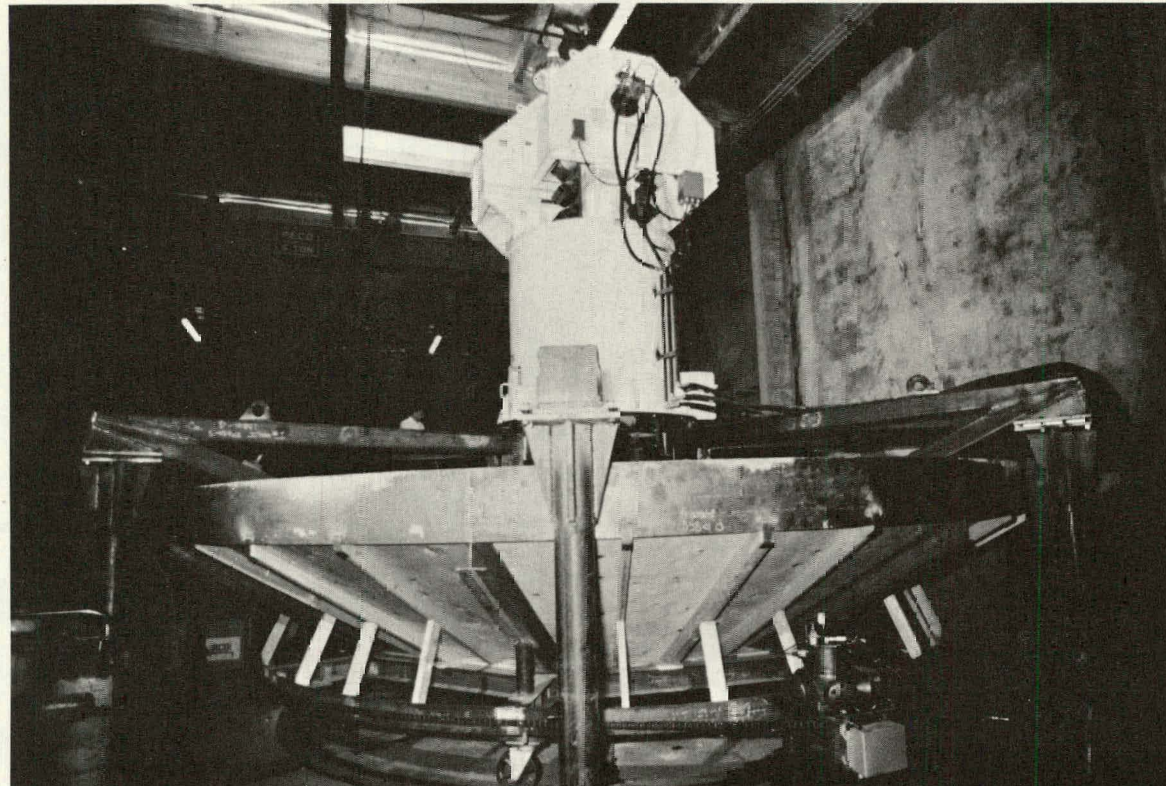


FIGURE 10. - Full-Scale Muck-Pickup Test Fixture

FIGURE 11. -

BLIND SHAFT BORER TEXT FIXTURE

DIRECT PICKUP SYSTEM PERFORMANCE

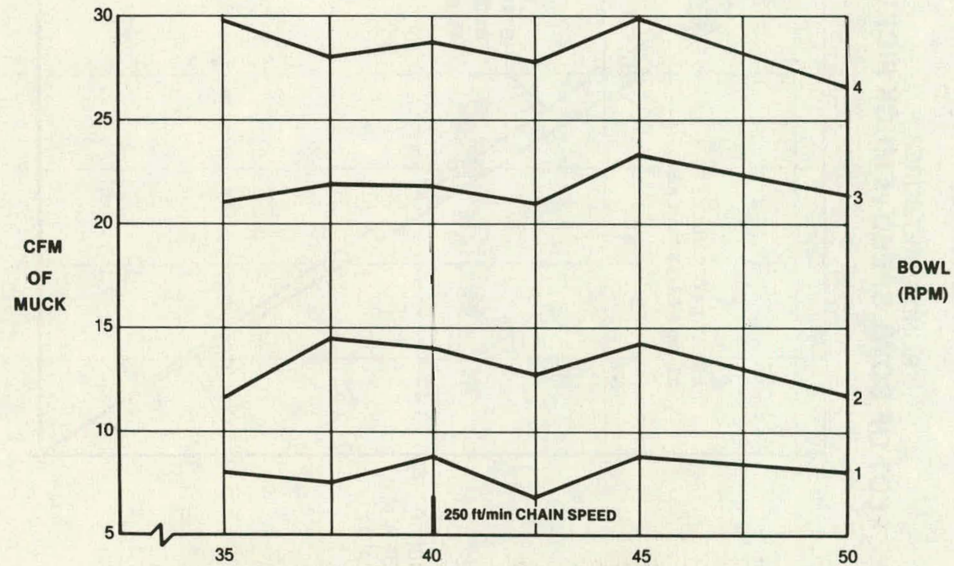


FIGURE 12. -

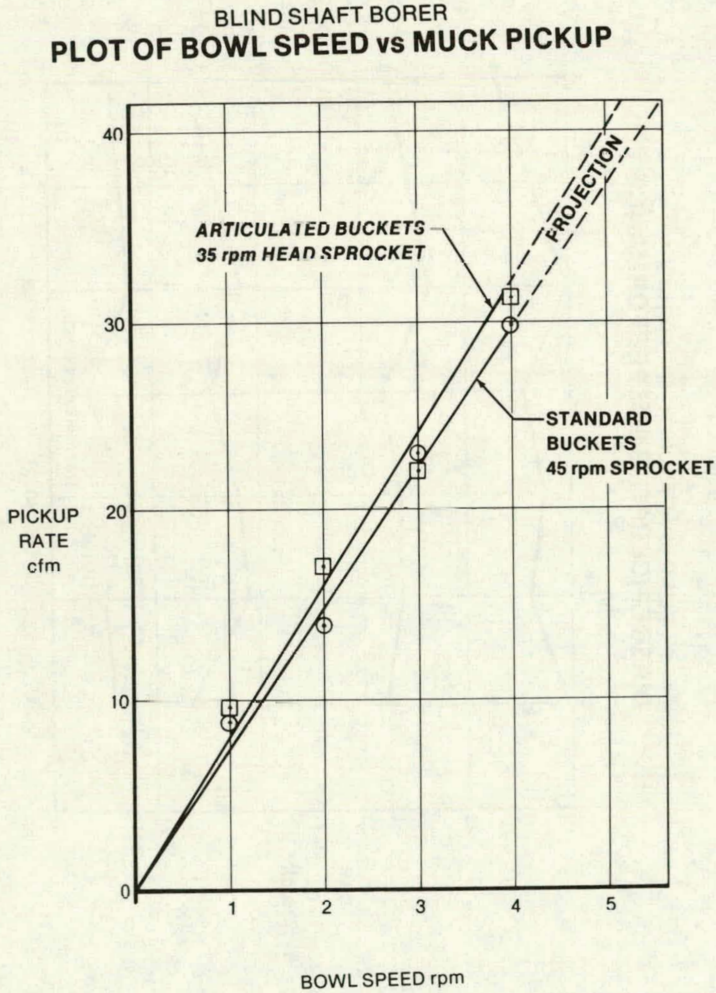

BLIND SHAFT BORER
PLOT OF BOWL SPEED vs MUCK PICKUP
ARTICULATED BUCKETS
35 rpm HEAD SPROCKET
PROJECTION
STANDARD
BUCKETS
45 rpm SPROCKET
PICKUP
RATE
cfm
0
10
20
30
40
1
2
3
4
5
BOWL SPEED rpm

FIGURE 13. -

BLIND SHAFT BORER

REVISED FACE CONTOUR AND PICKUP SYSTEM

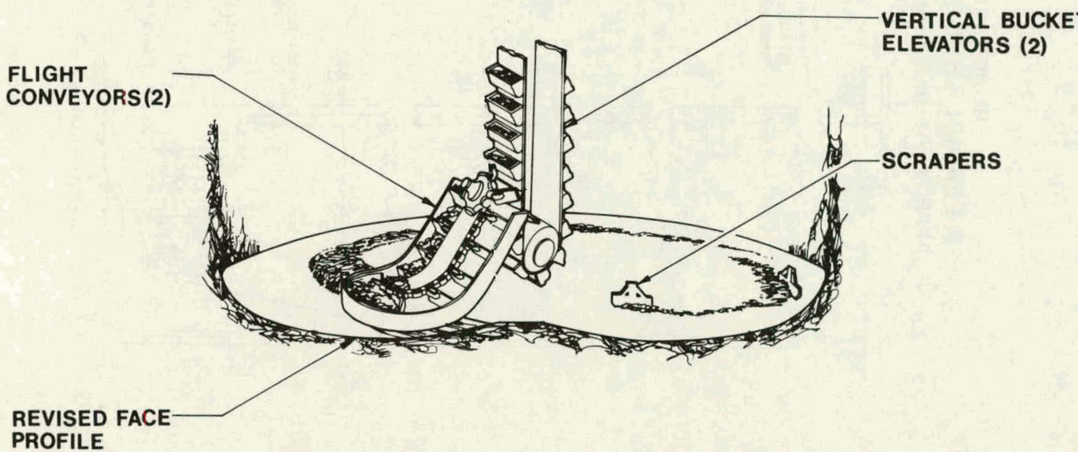


FIGURE 14. -

BLIND SHAFT BORER

A FEASIBLE HYDRAULIC CONCEPT

APPROX. 104 cfm PICKUP CAPACITY (300 ton/hr)

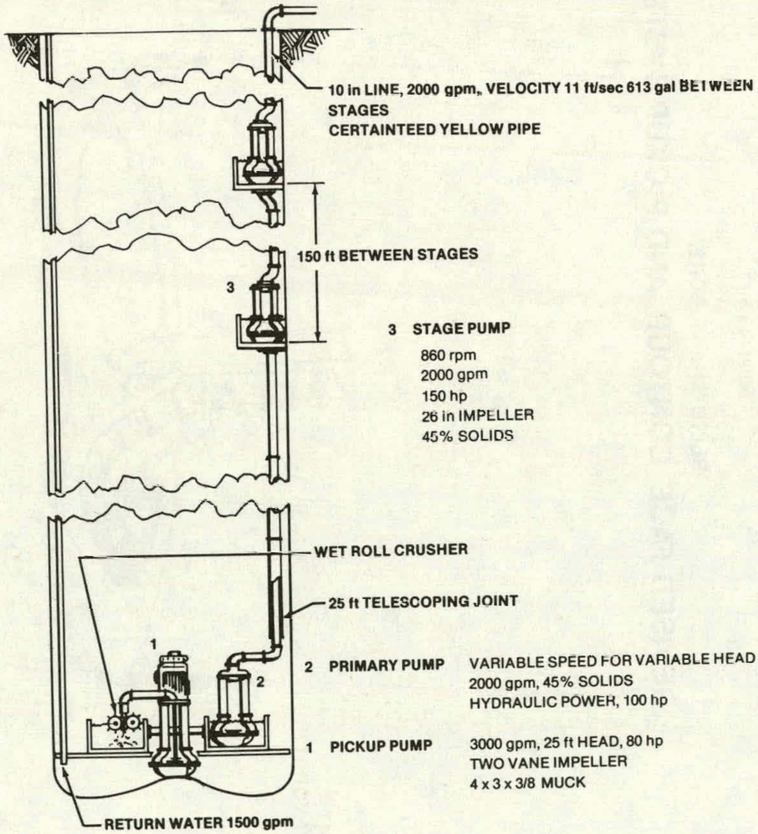


FIGURE 15. -

BLIND SHAFT BORER
PRIMARY PICKUP CONCEPTS

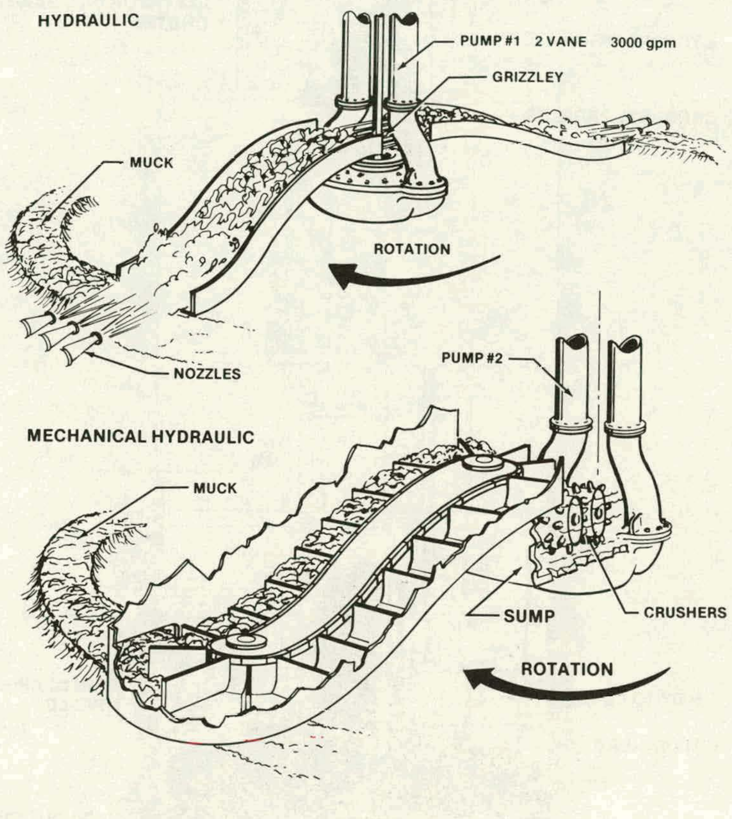


FIGURE 16. -

BLIND SHAFT BORER
ROTATING COMPONENTS

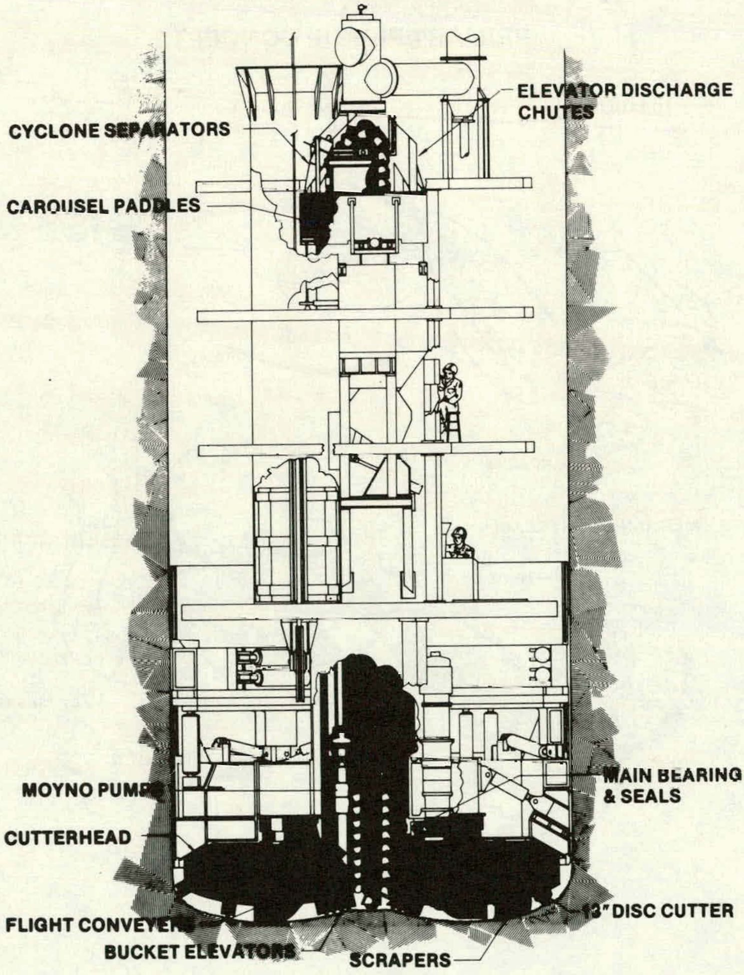


FIGURE 17. -

BLIND SHAFT BORER

NON-ROTATING COMPONENTS

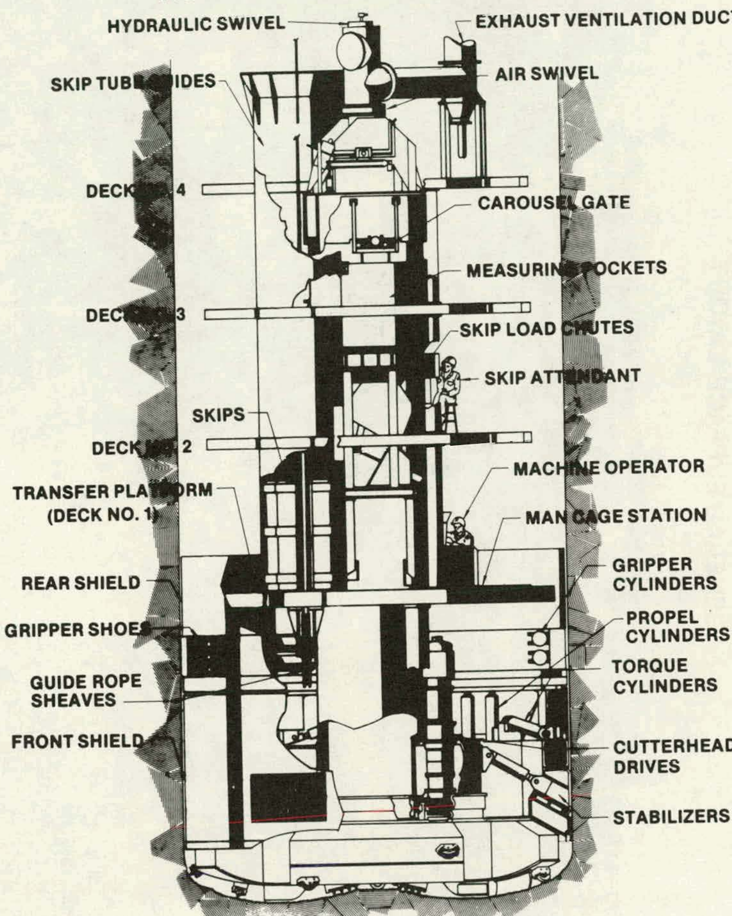


FIGURE 18. -

GENERAL SURFACE LAYOUT

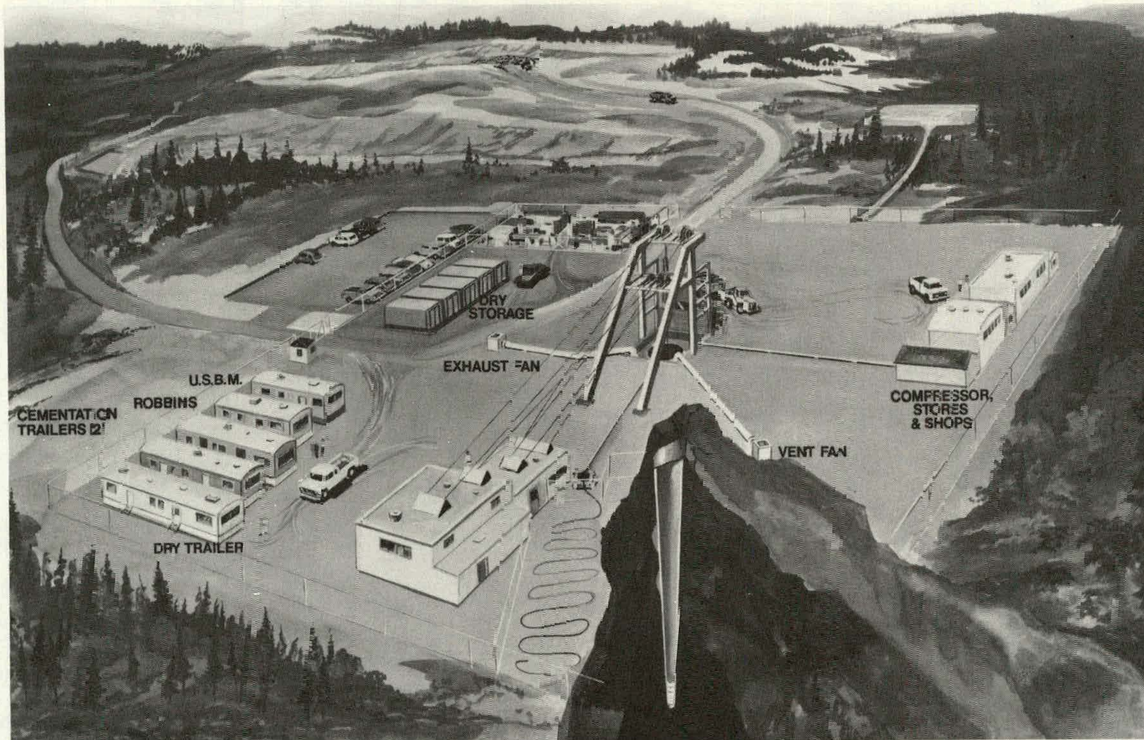


FIGURE 19. -

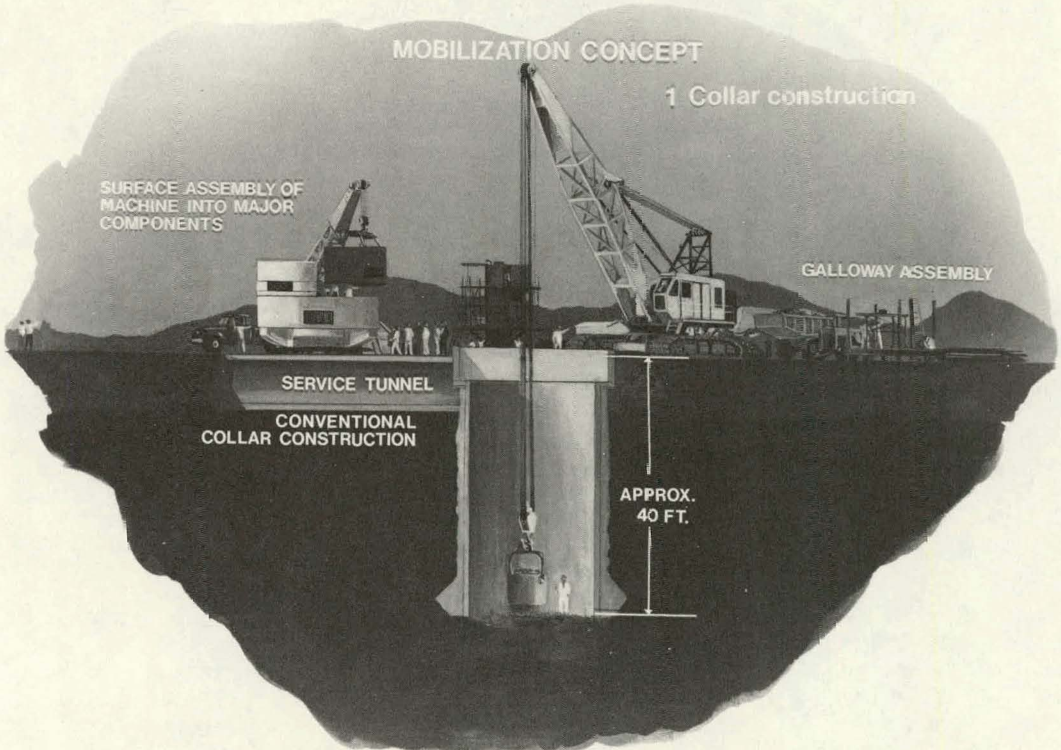

MOBILIZATION CONCEPT
1 Collar construction
SURFACE ASSEMBLY OF MACHINE INTO MAJOR COMPONENTS
GALLOWAY ASSEMBLY
SERVICE TUNNEL
CONVENTIONAL COLLAR CONSTRUCTION
APPROX. 40 FT.

FIGURE 20. -

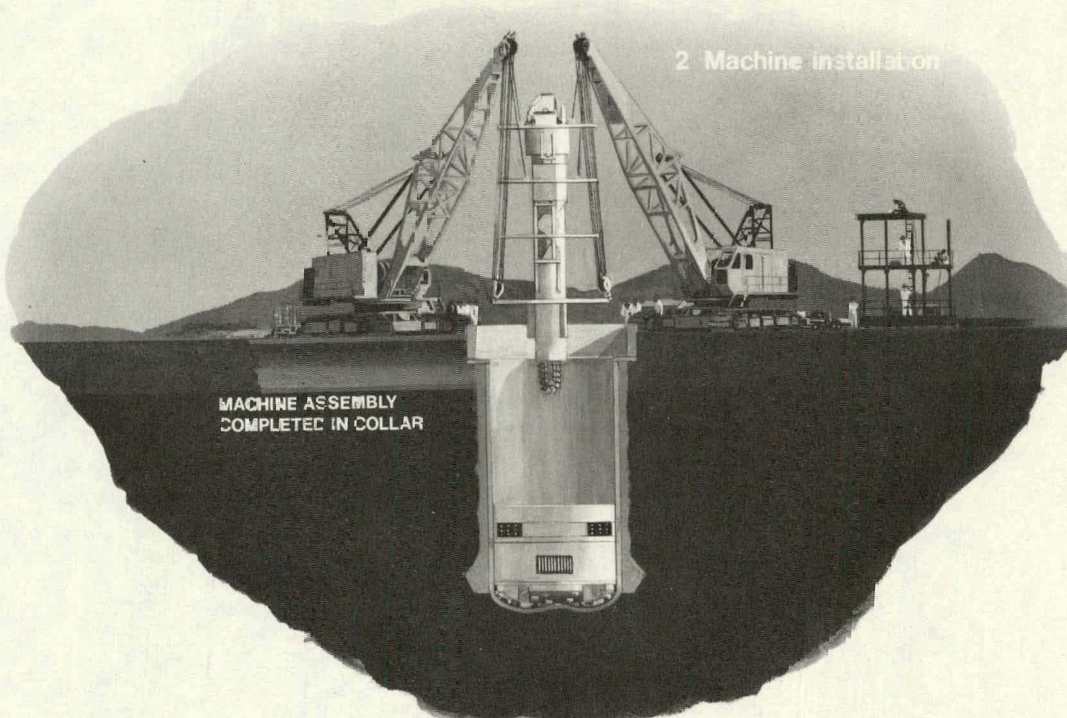

2 Machine installation
MACHINE ASSEMBLY
COMPLETED IN COLLAR

FIGURE 21. -

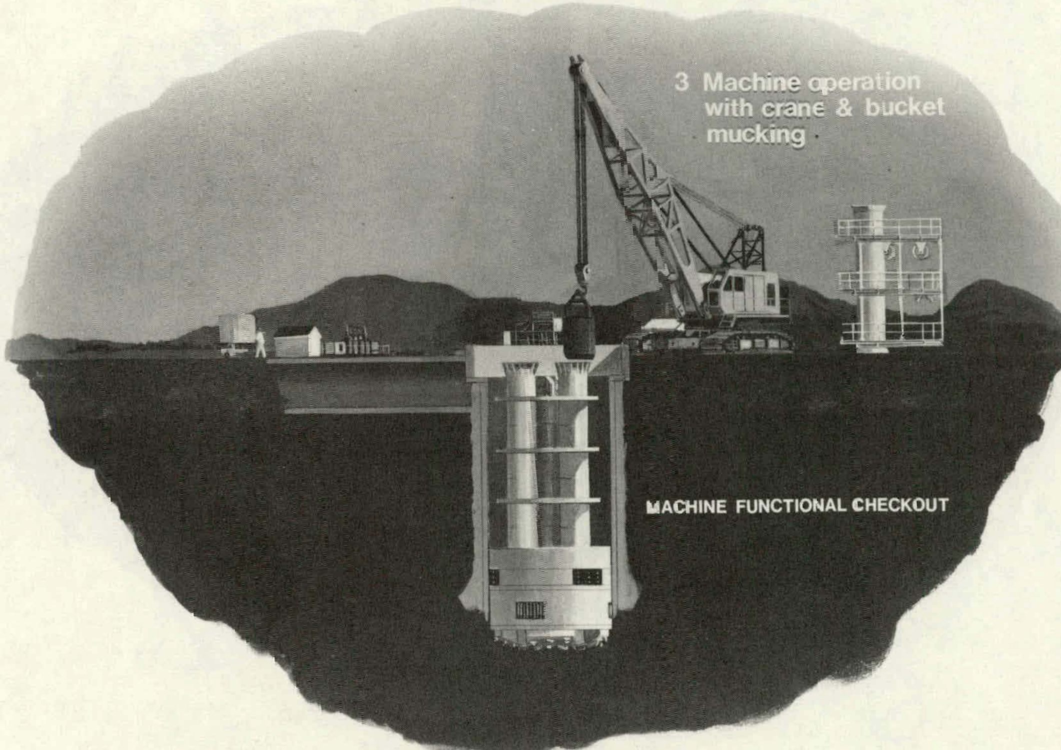

3 Machine operation
with crane & bucket
mucking
MACHINE FUNCTIONAL CHECKOUT

FIGURE 22. -

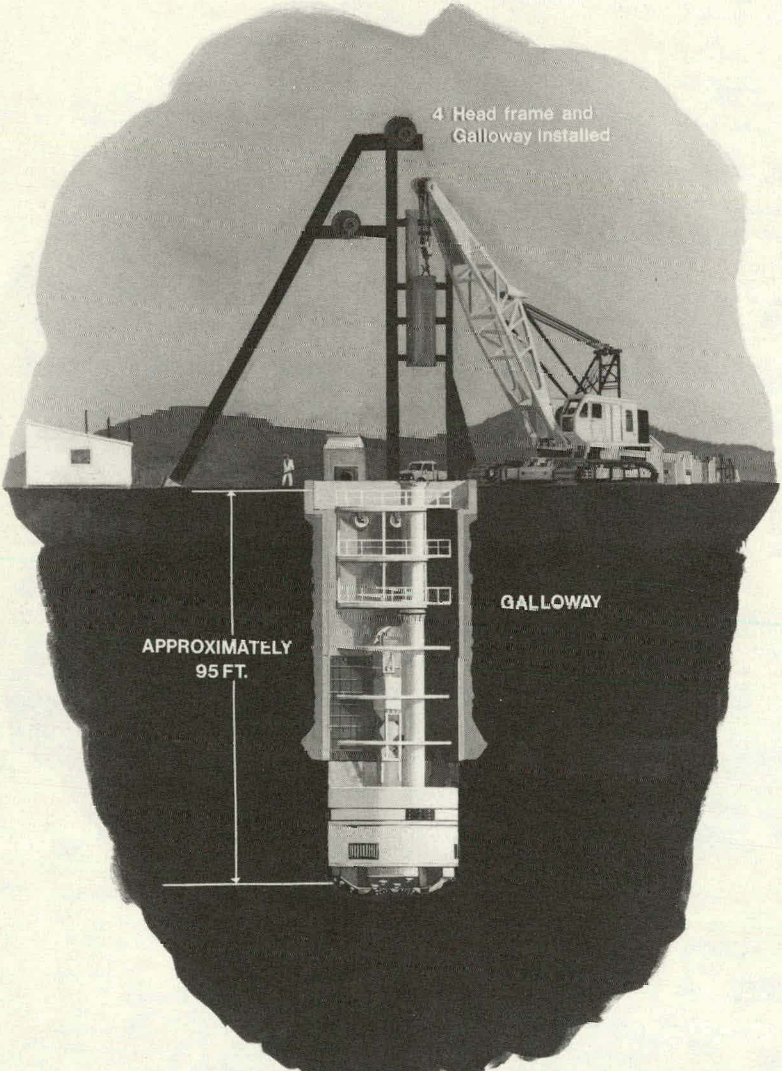

4 Head frame and
Galloway installed
GALLOWAY
APPROXIMATELY
95 FT.

FIGURE 23. -

STANDARD OPERATING MODE

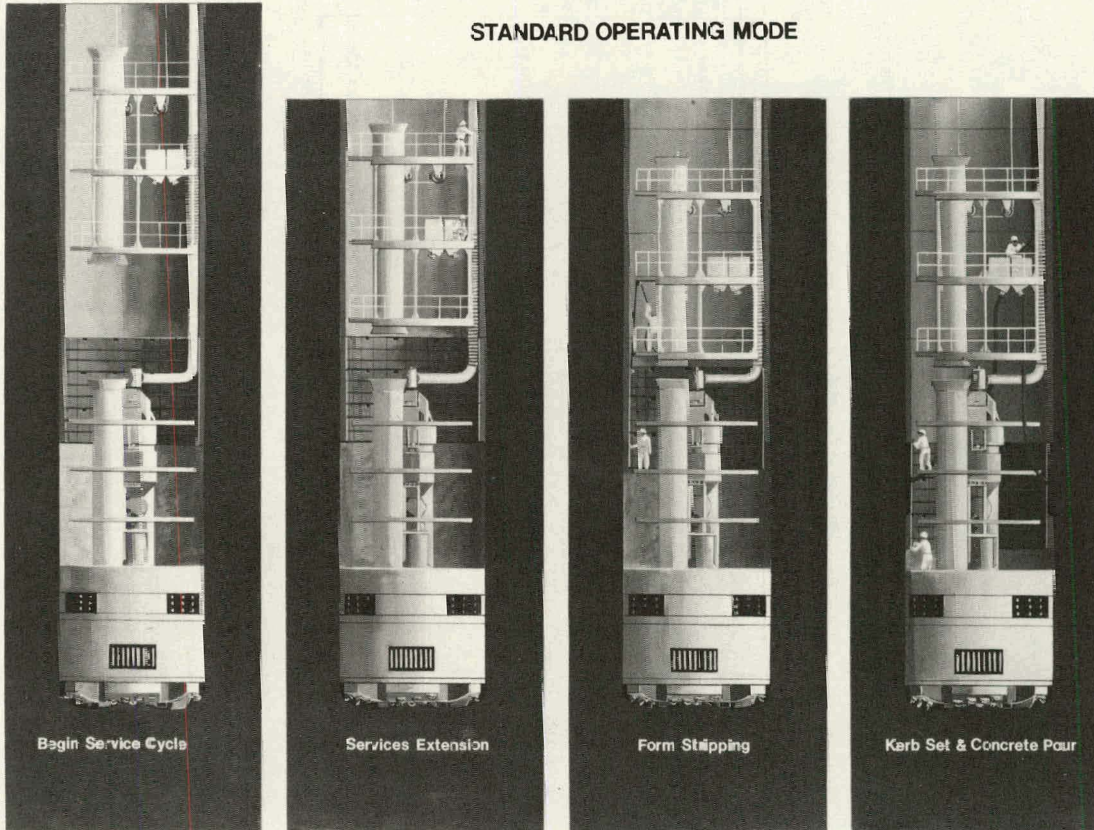
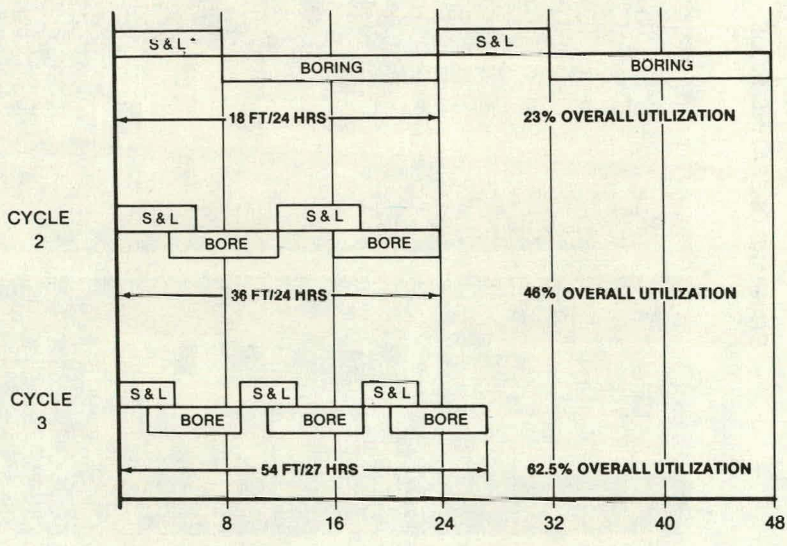


FIGURE 24. -

BLIND SHAFT BORER
PROPOSED BORING CYCLES
40 CFM BASIS



*SERVICE EXTENSION & LINING

FIGURE 25. -

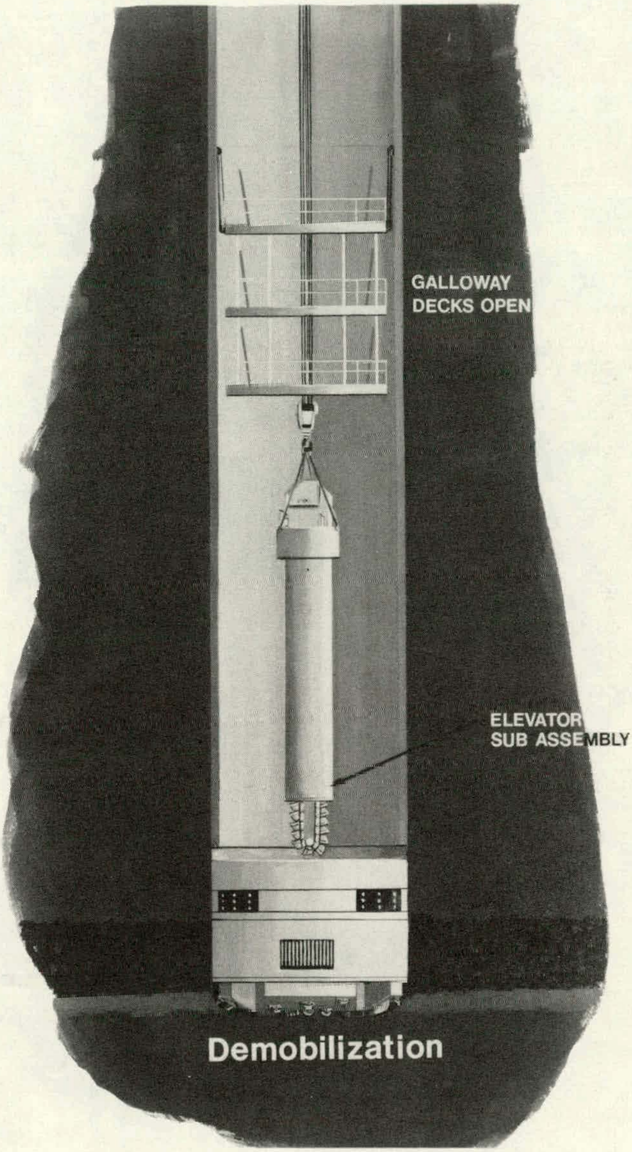

GALLOWAY
DECKS OPEN
ELEVATOR
SUB ASSEMBLY
Demobilization

FIGURE 26. -

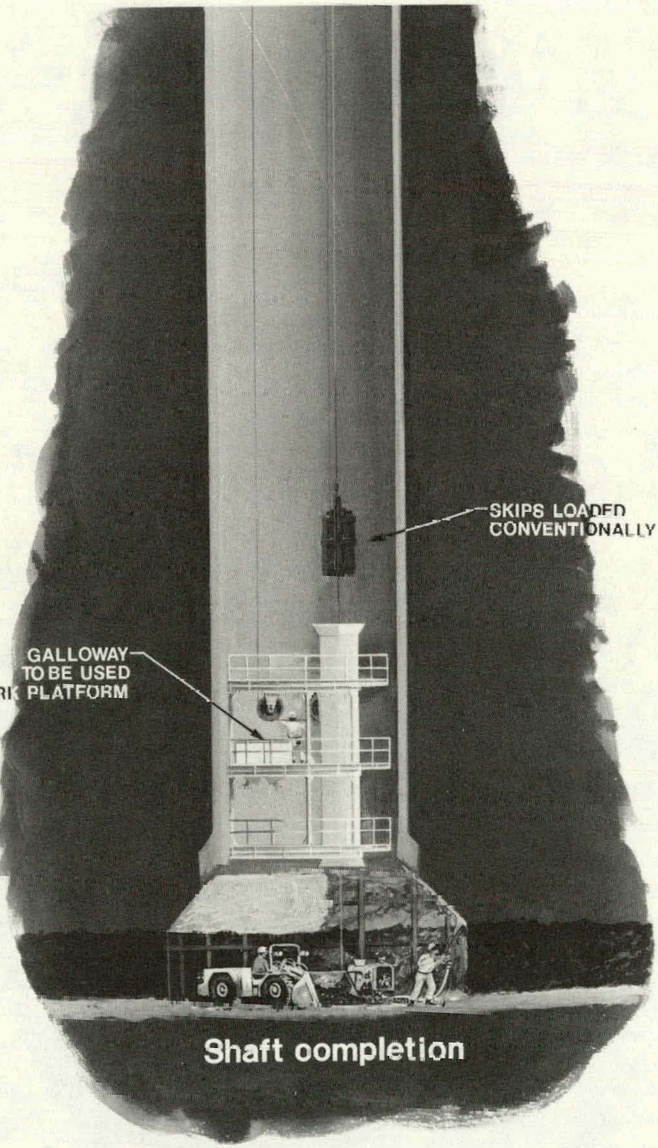

SKIPS LOADED
CONVENTIONALLY
GALLOWAY
TO BE USED
AS WORK PLATFORM
Shaft completion

FIGURE 27. -

BLIND SHAFT BORER
VENTILATION AND DUST CONTROL

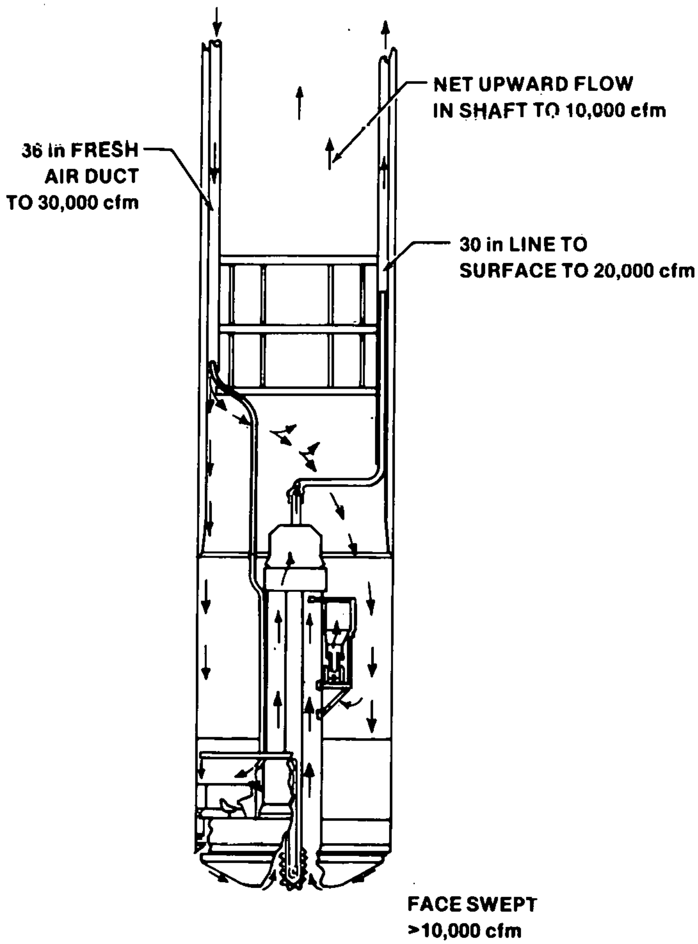


TABLE 1

A SAMPLE OF LARGE DIAMETER, MECHANICALLY CONSTRUCTED MINE SHAFTS

LOCATION	OWNER	CONTRACTOR	[1]Drill Method	[2]For-mations	Dia. Inches	Depth Ft	Time Months	Remarks
1954-1959								
Holland	Dutch State		BRM	Mr, L	301	1,680	43	1 hole
Holland	Dutch State		BRM	Mr, L	301	1,660	40	1 hole
Germany			BRM		301	1,660		2 holes
1958								
Russia					283	2,674		4 passes
1963								
Carlsbad, N.M.	Kermac Potash	Kerr-McGee	BDM	Sd, Sh	124 102	935 1,650	6	2 passes each sec.
Cote Blanch, LA.	Carey Salt	No. American Drlg.	BRM	Sd, Sh	120 90	570 1,400	2/3	
1964								
Jefferson Isle, LA	Diamond Crystal Salt	No. American Drlg.	BRM	Sd, Sh	130 90	242 881	2	
1965								
Bonanza, Utah	American Gilsonite	Teton Expln. Drlg.	BRM		132	450		1 Shaft
Waltonville, Illinois	Freeman Coal	Dravo	BRM	Sd, Sh	126	730	11	
Coalwood, W.V.	Olga Coal	Zeni-McKinney-Williams	BRM	Sd, Sh	144	970		
Russia					252			
1969								
White Pine, MI.	White Pine Copper	Dravo	DHM		144	1,591		3 Passes, Pilot Hole
1970								
Ambrosia Lake, N.M.	Kermac Nuclear	Kerr-McGee	BRA	Sd, Sh	198	784		2 Passes

1. BD = Blind Direct Circulation; BR = Blind Reverse Circulation; DHM = Downhole Machine; A = Air; M = Mud

2. SD = Sandstone; L = Limestone; Sh = Shale; Mr = Marl

Table 2
Cost Comparison, July 1976
1000 Foot Shaft, Concrete Lined
Production Usage

	CONVENTIONAL SHAFT	BORED SHAFT
ELAPSED TIME	53	36
MAN WEEKS	2269	1535
LABOR	$ 1,360,000	$ 921,000
STORES & SITE RUNNING	242,000	166,000
FREIGHT	110,000	160,000
PLANT FACILITIES - RENTS & DEPRECIATION	216,000	179,000
INSTALLED EQUIPMENT, AMORTIZED	75,000	285,000
CONCRETE, REBAR & FORMWORK	198,000	154,000
POWDER & DETONATORS	100,000	7,000
DRILL SUPPLIES	26,000	3,000
SERVICE LINES	40,000	51,000
BORER SPARES		100,000
BORER WRITE OFF (6 SHAFTS)		400,000
SUB TOTALS	$ 2,367,000	$ 2,426,000
OVERHEAD	296,000	253,000
MARKUP	200,000	171,000
TOTAL	$ 2,863,000	$ 2,850,000

TABLE 3

BLIND SHAFT BORER

MUCKING CAPABILITY vs ADVANCE RATE

24 ft. 5 in. DIAMETER BSB

MUCKING RATE	INSTANTANEOUS PENETRATION	PENETRATION AT 80% SHIFT UTILIZATION	FT. OF PROGRESS IN ONE SHIFT (7 HRS.)
30 cfm	2.4 ft/hr	1.92 ft/hr	13.4 ft
35	2.8	2.24	15.7
37	3.0	2.37	16.6
40	3.2	2.56	17.9
45	3.6	2.88	20.2
50	4.0	3.20	22.4
60	4.8	3.84	26.9
70	5.6	4.48	31.4
80	6.2	4.96	34.7
100	8.0	6.40	44.8

ASSUMPTIONS

GROUND EXPANSION FACTOR 1.6

BROKEN MUCK DENSITY 100 lbs/ft^3

TABLE 4

BLIND SHAFT BORER

MUCK PICKUP SYSTEMS POTENTIAL

	CONTRACT UP TO 50 cim (150 t/hr)	FUTURE TO 100 cim (300 t/hr)
MECHANICAL/MECHANICAL	KNOWN TECHNOLOGY AVAILABLE PARTS SHORTEST SCHEDULE LOWEST DOWN HOLE POWER	LIFT CAPACITY LIMITED
MECHANICAL/PNEUMATIC	AVAILABLE SCIENCE DEVELOPMENT REQUIRED	AVAILABLE SCIENCE POWER REQUIREMENTS UNDEFINED SIGNIFICANT DEVELOPMENT COST STUDY REQUIRED
MECHANICAL/HYDRAULIC	NOT FEASIBLE	KNOWN TECHNOLOGY AVAILABLE PARTS SHORT SCHEDULE COST & ENVIRONMENTAL STUDY REQUIRED
HYDRAULIC/HYDRAULIC	NOT FEASIBLE	KNOWN TECHNOLOGY ON LIFTING AVAILABLE SCIENCE ON PRIMARY PICKUP DEVELOPMENT WET FACE EFFECTS
VACUUM/PNEUMATIC	AVAILABLE SCIENCE SPACE DOWN HOLE POWER NOISE, DUST	EXTENDED AVAILABLE SCIENCE COST, PHYSICAL SIZE, POWER
VACUUM/MECHANICAL	AVAILABLE SCIENCE SPACE DOWN HOLE POWER NOISE, DUST	NOT FEASIBLE

AN ALTERNATE MEANS OF TRANSPORTING COAL OR WASTE ROCK

M. R. Carstens and Donn W. Leva
Professor and Director, respectively

Georgia Institute of Technology, Atlanta, Georgia and
Tubexpress Systems, Inc., Houston, Texas, respectively

Introduction

Capsule pipelines are an alternate means of transporting coal or waste rock. Pneumatic capsule pipelines are transportation systems in which cargo is loaded i--- wheeled vehicles (capsules), Figure 1, with the capsules being pushed through a pipeline by flowing air. Small-diameter pneumatic capsule pipelines have been ii use for over a century for movement of documents. Recent interest in large-diameter pneumatic capsule pipelines has been stimulated by the development of the technology for laying large-diameter steel pipelines and by societal demands for safer and less conspicuous transportation systems. The large-diameter pneumatic capsule pipeline is best suited for hauling particulate solids, such as coal, at a high flow rate on a nearly continuous basis.

Two commercial systems of large-diameter pneumatic capsule pipelines are being marketed at the present time (1977)--one by the U.S.S.R. called "Transprogress" and one by a U.S. firm called "Tubexpress". Both systems use wheeled capsules coupled together into trains as cargo carriers.

Development of the American system was begun at Georgia Tech in 1968. In 1970 a Houston-based firm signed a license agreement with the Georgia Tech Research Institute to develop and to market the system. A subsidiary firm, Tubexpress Systems, Inc., was formed to carry out the licensee's obligations.

State of the Art (Tubexpress)

In the period 1970-1976, Tubexpress Systems, Inc. with the aid of Georgia Tech developed a reliable pneumatic capsule pipeline for hauling dense materials, such as coal. This pioneer system is being actively marketed.

The pioneer system is a loop with one loading station and one unloading station as shown in Figure 2. The pressurized portions of the system, blocks 4 and 9 in Figure 2, are the pipelines connecting the terminals, Figure 3. The unpressurized pipelines are sloped downward through the terminals, Figure 3, in order to utilize the force of gravity for movement of the trains forward to the next block. Movement of the trains from temporary storage, blocks 10 or 5 in Figures 2 and 3, is controlled by gates in block 1 or block 6 as shown in Figure 4 which, in turn, are controlled by photo-electric sensors, shown at the upper left and lower right in Figure 4, and a control system, Figure 5, which allows a train to roll forward when the unit process is complete and when the next block is open to receive a train. All of the unit processes in the pioneer system including loading, Figure 6, and unloading, Figure 7, are automatic being controlled by feedback. Hardwar components for the pioneer system have been fabricated, installed, and tested in the 1630-ft-long, 16-in-diameter test pipeline located close to Houston. This p matic capsule pipeline, shown in plan in Figure 8, which is pressurized for 1380 ft with a 250-ft-long terminal, Figure 3, differs from an operating system, Figure 2, only in that both the loader and unloader are located in the same terminal.

In parallel to the hardware-development program, a software-development program for the design of pneumatic capsule pipelines has been pursued with equal vigor. While development of standard designs of hardware components has been a major goal, selection of pipeline diameter, mass flow rate of air and pump selection are design questions which can answered only by analyzing each particular system. In order to be able to design, a mathematical model based upon Newton's second law, conservation of gas mass, and a polytropic process of gas expansion has been formulated which can be solved by numerical methods.

In order to illustrate part of the information about flow characteristics available to the designer from the solution of the mathematical model, Figures 9, 10, and 11 are presented. The illustrative example is for the Tubexpress Test Facility shown in plan in Figure 8. Values of physical quantities used in the solution are listed in TABLE 1.

Pressure at the pump during a cycle of 20 seconds is shown in Figure 9. The highest pressure occurs at one second after injection of a 700-lb train. The pressure force required to accelerate the train from an initial velocity of 10 feet per second to a velocity of 25 feet per second two seconds later. The mass flow rate ir is maintained at nearly a constant value by use of a constant-displacement er of the Roots type driven by an electric motor. The power fluctuations are ially absorbed by the rotational energy stored in the blower and motor and are partially absorbed by the electrical system supplying energy to the motor. In the numerical solution a constant pump discharge of 2.42 lbm/second, TABLE 1, is assumed.

Figure 10 is a graph of train speed as a function of displacement. Since the process is periodic with a train being injected every 20 seconds, the train speed, U, shown in Figure 10 would apply to all trains passing through the pipeline. Since the live-load to dead-load ratio of a train hauling coal would be at least two, the live load in each 700-lb train would be at least 467 lb. Assuming a live load of 467 lb in each train, the live-load hauling rate would be 42 tons per hour. The unsteady velocity during movement through the pipeline is the result of trains being injected, trains being exhausted, and trains rolling uphill and downhill. The three trains in the pipeline at one time can be visualized as a lightly damped spring-mass system with the gas pockets between trains acting as springs.

The effect on the pressure at the pump, Pn, Figure 9, of a train being pushed uphill is demonstrated in Figure 11. Figure 11 is a picture of the conditions in the pipeline 7 seconds after the injection of train 3 when train 3 is being pushed uphill. Movement of train 3 up the hill is being accomplished partly by a decrease in momentum of trains 3 and 2, and partly by the pressure force. The pressure drop across train 3, Figure 11, is greater than the pressure drop across train 2 and 1. At this time, pressure at the pump is increasing, Figure 9, which is accomplished by compressing air into the air pocket between train 3 and the pump. At about 8 seconds after injection, train 3 has been pushed to the top of the hill and the air in the pocket begins to expand as evidenced by the falling pressure at the pump, Figure 9. Decompression of the air pocket behind train 3 after train 3 has climbed the hill results in acceleration of train 3 as shown in Figure 10. In a similar manner, the pressure increase at the pump, Figure 9, during the time from 14 seconds to 20 seconds results from train 1 moving up the second hill.

Two of the input quantities listed in TABLE 1 pertaining to the capsule trains, C_D and μ, must be determined experimentally. The coefficient of drag, C_D, is determined in a special apparatus of the Tubexpress Test Facility. In this apparatus, a train is tethered in a straight horizontal pipe with instrumentation to ure the mass rate of flow of air passing the train resulting from the controlled sure drop across the stationary train. The coefficient of rolling resistance, μ, is determined from a recorded pressure signature as a train passes through 50-ft long test section located at the downstream end of a 593-ft straight horizontal portion of the test loop, that is, from x=173 ft to x=766 ft in Figure 10. When performing tests to determine rolling resistance only one train is in the pipeline from pump to vent in order to achieve a steady velocity, U, as the train passes through the test section rather than having the unsteady velocity shown in Figure 10 when multiple trains are in the line.

<u>Assessment</u>

At the present time, 1977, the large-diameter pneumatic capsule pipeline has been developed as an alternative to trucks, conveyer belts, and short-haul railroads for surface movement of particulate solids, such as coal or waste rock. The following statements about the pneumatic capsule are based on the assumption that the pipelines will be buried between terminals.

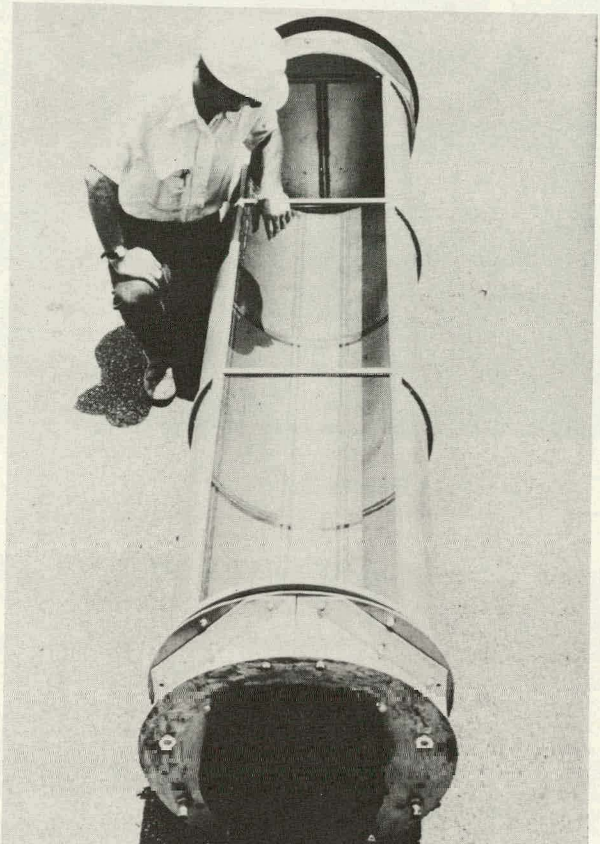
1. The pneumatic capsule pipeline is the safest inasmuch as transport is accomplished in an enclosed guideway, the pipeline.
2. The pneumatic capsule pipeline is environmentally superior. Between terminals the movement will be unheard, unseen, and unsmelled in the same way that movement of water, natural gas, and liquid petroleum products are hidden when being moved through buried pipelines.
3. The first cost of a pneumatic capsule pipeline is comparable to that of a covered conveyer belt. In 1975, a pneumatic capsule pipeline for moving waste rock about 3/4 mile was completely designed and priced. Two independent estimates of the c of a covered conveyer belt were also made for comparison. The capsule pipeline about 10 percent less in first cost than a covered conveyer belt by one estimate and about 30 percent less by the other.
4. Operating costs are believed to be lower that the alternatives. Since the capsule pipeline is automatic in operation, no operating personnel will be required. All of the maintenance is concentrated at the terminals. Coated buried pipelines can be expected to be maintenance-free for decades.
5. Energy requirements are comparable to that of the covered conveyer belt. However the pneumatic capsule pipeline has a greater fluctuation in power demand than a conveyer belt but less than a motor truck. In the systems investigated to date, the plan has been to rely on the electrical-energy supply system to accommodate for the power variations. Fluctuations will be relatively less in longer lines where more capsule trains are in transit because changes in linear momentum of the moving trains act to decrease the pressure fluctuations. In special cases, flywheels may be desirable for energy storage in order to release energy during perturbations such as occur during the initial acceleration of a train, Figure 9. The pneumatic capsule pipeline has a greater comparative advantage over trucks and railroads in hilly terrain because slopes up to 20 percent can be incorporated in the pipeline thereby reducing the route length and because most of the potential energy of a capsule train is recovered as the capsule train descends.

Future Development

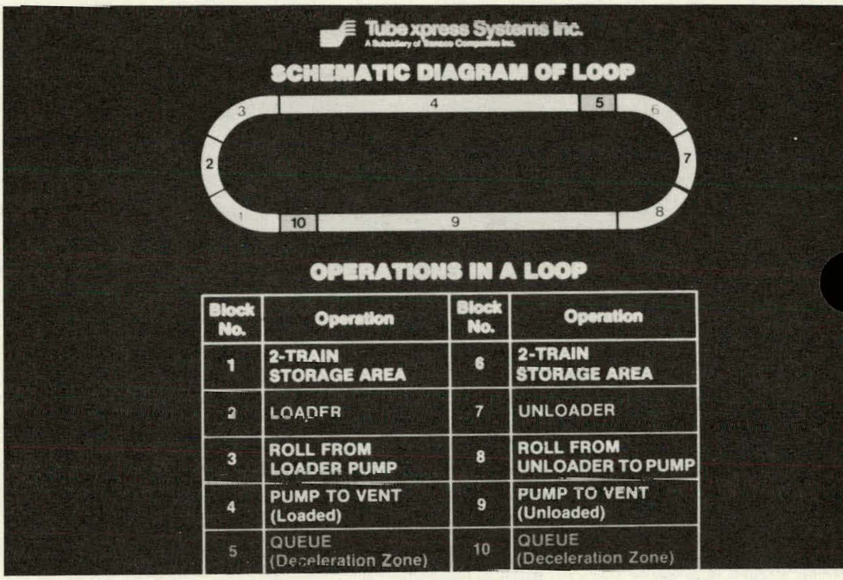
While development of improved capsules, switching devices, and so forth is being pursued, two very significant advances in capsule pipelines appear to be possible.

Doubling the energy-conversion efficiency of an already efficient pump appears to be possible as a result of the unique method of propulsion in a capsule pipeline. Analytical studies are in progress to determine the pipelines to which the improved pumping equipment could be applied. Successful implementation of the concept could result in a transportation system which is considerably more efficient than the alternatives.

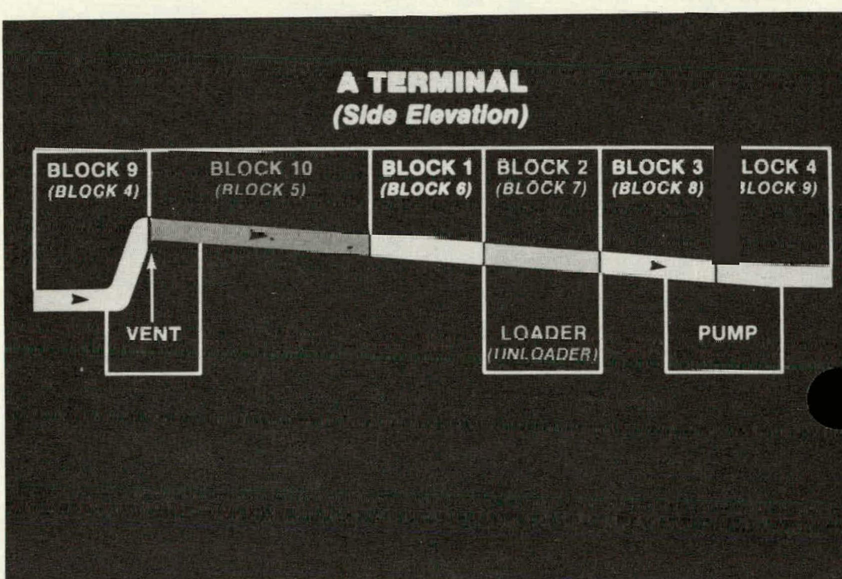
The second possibility is to utilize the pneumatic capsule pipeline to perform three functions in underground mining, that is, (a) to ventilate, (b) to provide an emergency exit, and (c) to haul ore. By design, the mass rate of flow of air i the mine could be greater through the empty capsule pipeline, block 9 in Figure than in the loaded pipeline, block 4 in Figure 2. The excess air would be discharged into the mine at the underground terminal to flow back through the mine and out through various surface openings. The two steel pipelines could be designed to resist damage from falling rock thereby providing ventilation and an emergency exit through a mine cave in. For underground transport, hardware components described previously would have to be developed for operation of the underground terminal where space is limited.



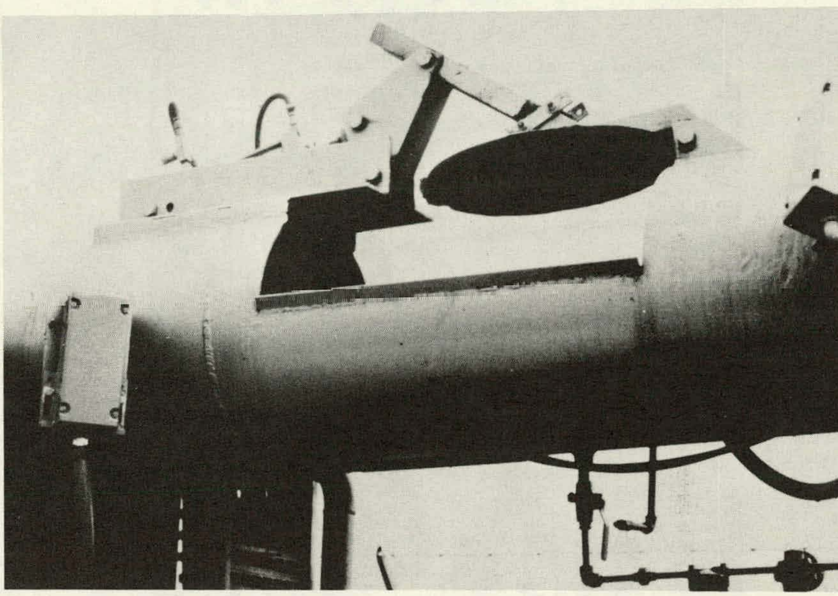
1. A capsule for use in a 30-in-diameter pipeline



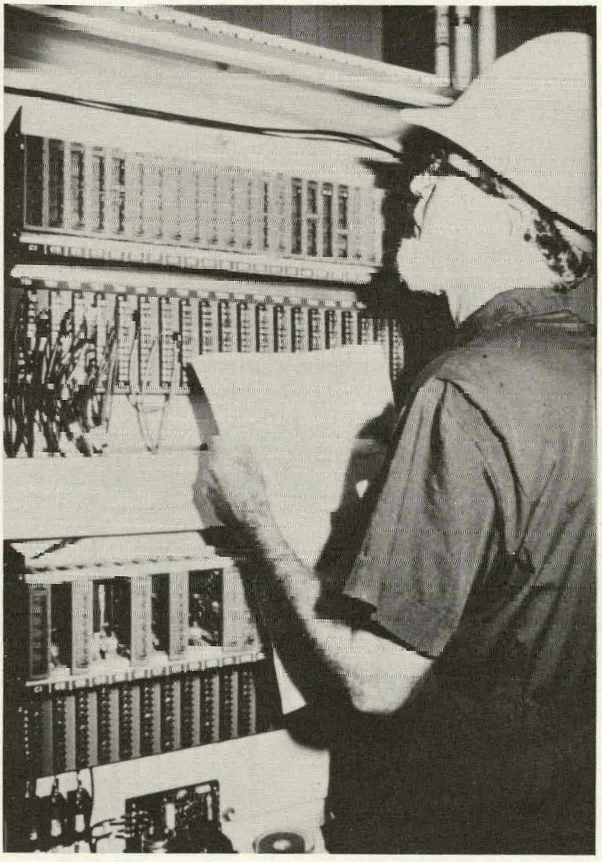
2. Schematic diagram of loop



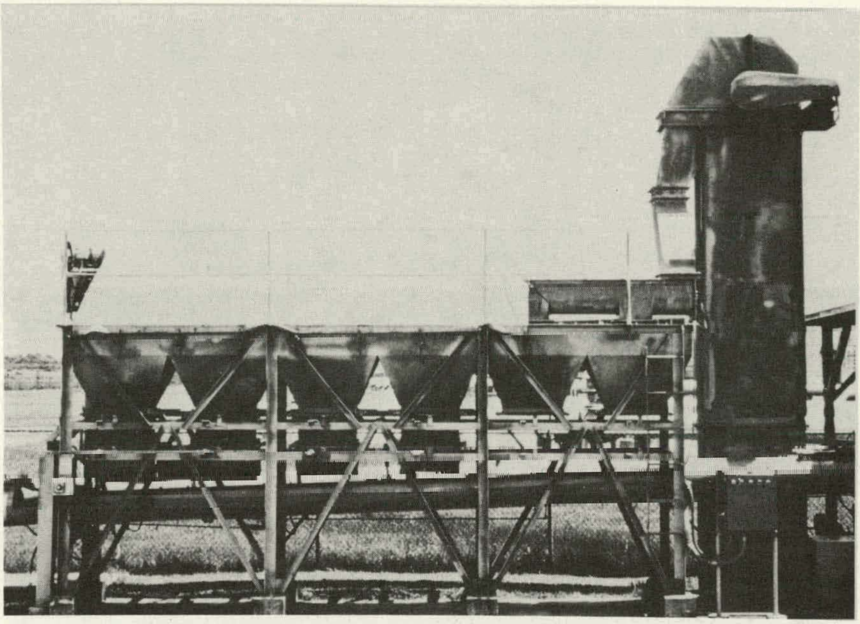
3. Schematic diagram of terminal



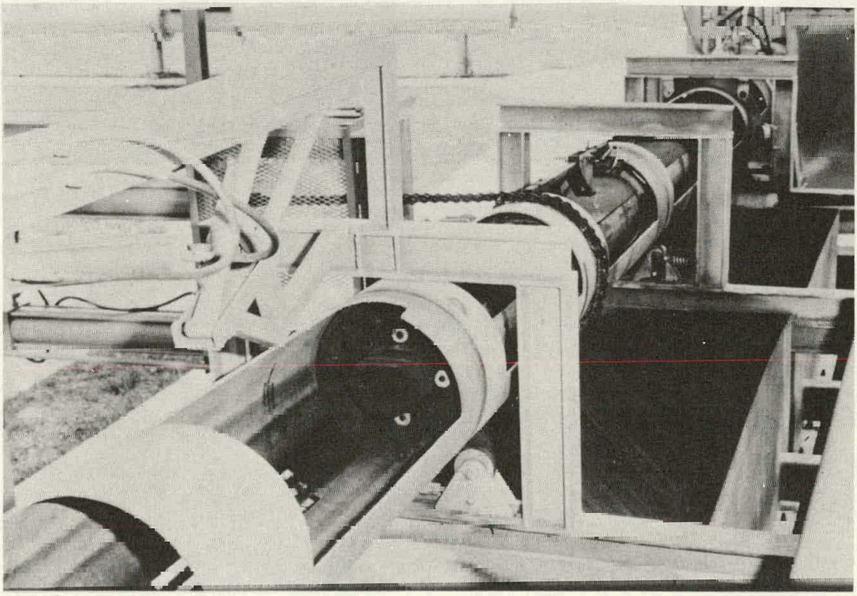
4. A gate and sensors



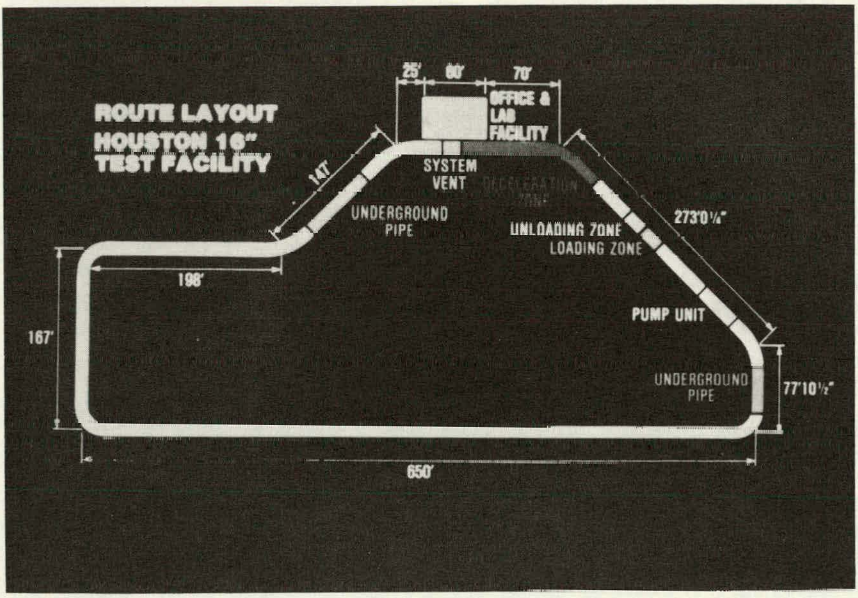
5. Electronic control system



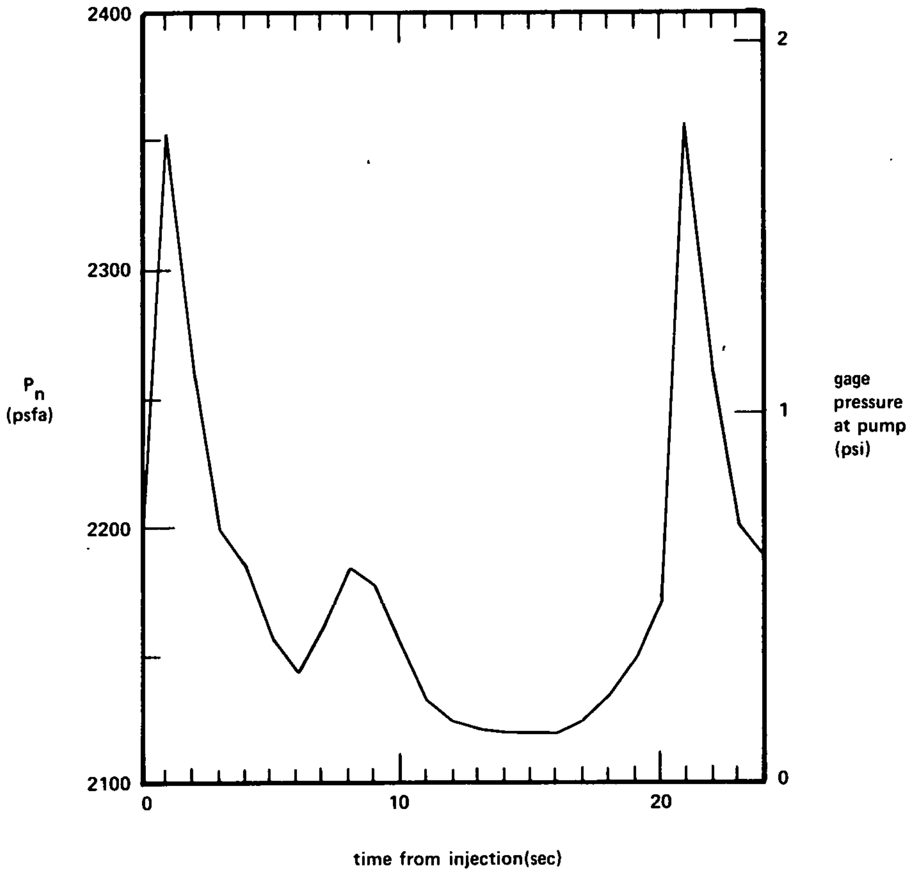
6. Loader



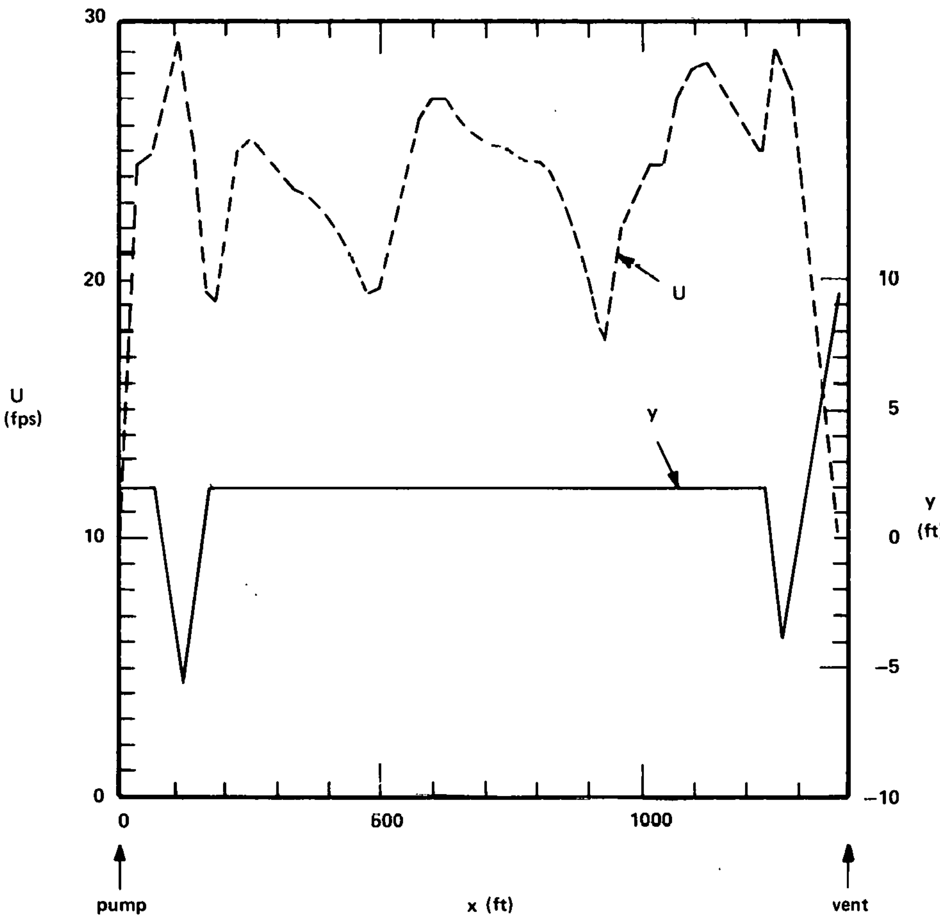
7. Unloader



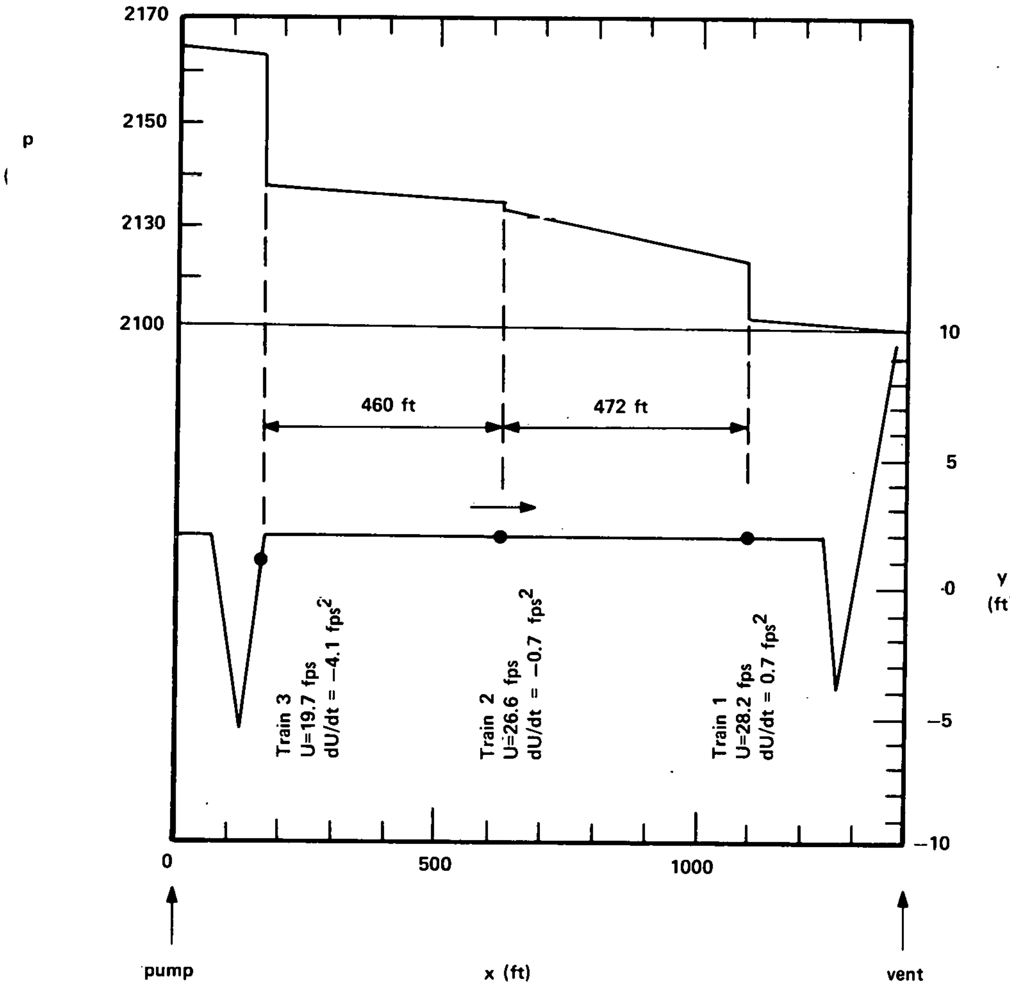
8. Plan of test facility



9. Pressure at the pump during a cycle



10. Profile of pipeline and train velocity during a trip



11. Pressure line in pipeline 7 seconds after injection of train 3

TABLE I

Physical Quantities (Illustrative Example)

Symbol	Definition	Units	Numerical Value
D	ID of pipeline	in	15.5
f	boundary-drag coefficient	-	0.017
x	axial coordinate	ft	x(pump)=0
			x(vent)=1379
y	vertical coordinate	ft	Figures 10 and 11
W	train weight	lb	700
C_D	coefficient of drag of a train	-	4000
μ	coefficient of rolling resistance	-	0.025
g	local acceleration of gravity	fps^2	32.13
G^{-1}	period of train release	sec	20
U (initial)	train velocity at x=0	fps	10
	pump discharge	lbm/sec	2.42
ρ_a	density of atmospheric air	$slugs/ft^3$	0.0023
p_a	atmospheric pressure	psfa	2100

NOISE CONTROL OF UNDERGROUND COAL MINING EQUIPMENT

Roy C. Bartholomae
Electrical Engineer

Dr. H. Kenneth Sacks
Supervisory Electrical Engineer

U.S. Department of the Interior
Bureau of Mines
Pittsburgh Mining and Safety Research Center
Industrial Hazards and Communications
Pittsburgh, Pa.

Abstract

Research has continued at the Bureau of Mines to develop and demonstrate mining machinery noise control technology as mandated by the Federal Coal Mine Health and Safety Act of 1969. The following discussion summarizes the present status of selected underground coal mine machinery noise research projects.

Introduction

The Federal Coal Mine Health and Safety Act of 1969 established mandatory noise exposure standards as shown in Table 1. The Bureau of Mines is actively developing noise reduction technology for various pieces of aboveground and underground mining equipment that presently are not in compliance. At the First and Second NCA/BCR Symposiums on Underground Mining, a review of Noise Control research being conducted at the Pittsburgh Mining and Safety Research Center was presented. This review included roof drills, diesel-powered equipment, continuous miners, and other equipment employing coal-carrying chain conveyors.

This paper presents the work presently being done in developing an inherently quiet stoper-type percussive drill, methods to predict machinery noise levels in underground mines, and mantrip noise control. The appendix lists noise publications resulting from Bureau projects and the Bureau's current programs.

Table 1.--Permissible noise exposures, per day

Duration, hours	Noise levels, dbA
8	90
6	92
4	95
3	97
2	100
1-1/2	102
1	105
3/4	107
1/2	110
1/4 or less	115

Inherently Quiet Stoper Drill

Extensive noise surveys have shown that the most serious noise source in underground coal mines is the percussive drill. Noise levels at the operator's position of 118 dbA are often reached during roof-bolting operations using pneumatic percussive equipment. Previous studies by the Bureau have led to practical means of "quieting" these machines to levels of 102 to 106 dbA via retrofit packages. The Bureau of Mines effort under contracts J0155099 (Development of Other Pneumatic Drills, Ivor Hawkes Associates) and J0177125 (Development of Six Prototype Production Stoper Drills, Ivor Hawkes Associates) is to develop an "inherently quiet" stoper drill for roof-bolting operations. The reason for developing a completely new stoper drill instead of retrofitting existing units with exterior acoustical treatment is to obtain a quieter, lighter, and less bulky stoper. This is being accomplished by analyzing a Joy L-47 percussive drill to identify major noise sources. Noisy components are then redesigned to make a quiet demonstration drill. A completely new quiet prototype percussive stoper drill that incorporates the noise-reducing features of the demonstration drill will then be developed and tested. At this time the quiet demonstration drill is completed and tested, and work has begun on the new quiet prototype productio stoper.

Under contract J0155099, a Joy L-47 percussive drill was analyzed to identify major noise sources. The noise from the percussive rock drills arises mainly from two sources: Air noise from the flow of high-pressure air, and impacts of the mechanical components. Air noise originates at the exhaust port and in the chuck region where air leaks between the drill rod and chuck. Air noise must be controlled by muffling before the benefits of reducing mechanically generated noise can be realized. In addition, the drill steel radiates high noise levels and must be quieted. Mechanical noise sources of the L-47 drill are summarized in figure 1.

In addition to these purely mechanical (impact) noise-generating sources, air noise-generating sources inside the machine, which couple to mechanical surfaces, must also be considered. Two sources can be identified:

1. High-frequency pressure oscillations associated with airflow through the valves and ports.

2. Dieseling in the return stroke chamber due to running "on cushion" or when drilling with inadequate thrust.

The vibrations (stress waves) generated at these sources propagate through the internal components and radiate noise from the rock, drill rod, and machine body. (The airleg can easily be isolated from these vibrations and generates very little noise.) The L-47 quiet drill is illustrated in figure 2 with a summary list of the modifications to the original L-47 drill.

Figure 3 shows the components of the L-57 quiet drill as well as the original L-47 drill. Figure 4 shows the modified drill in use. An explanation of the major changes follows:

Chuck-Related Modifications

To eliminate the rattle and at the same time permit easy rotation, the chuck was mounted in thrust bearings. The use of a roller-bearing-mounted chuck make the drill smoother running than the conventional L-47 and noticeably reduces th body vibrations.

The tendency to generate drill rod bending waves increases as the drill rod shank/chuck clearance increases. Therefore, this clearance was reduced and the chuck was manufactured from tungsten carbide to prevent galling. The new prototype stoper being developed will not have a tungsten carbide chuck because of cost considerations but will instead rely on reduced drill rod tolerances and a longer drill shank length to reduce drill misalinement.

In addition, the chuck was vibration-isolated from the machine body via a rubber collar and sealed in the front to eliminate noise generated by air venting from the front cavity.

Drill Body

The body was completely redesigned to produce a potentially lightweight, compact, and radially symmetrical construction that can be easily shrouded. The body consists of two aluminum tubes located axially by a steel buffer ring. The front half of the body contains the rotating chuck assembly described earlier, and the back half has a hardened steel liner for the piston and valve assemblies.

Grooves machined in the steel liner form the air passages when pressed into the aluminum inner body. These two inner body assemblies are preloaded together by two end plates connected by long steel bolts. The use of such bolts is universal in drill machines since they provide a springlike means of absorbing the impact energy of the piston when the drill is running "on cushion." The end plates are made of Incramute*,(1) which reduces noise radiation.

Exhaust Plenum and Muffler

An exhaust plenum chamber is formed around the complete machine body by a tube located on rubber gaskets between the end plates. The plenum chamber vibrates due to the discharging air and also to some extent due to the vibration of the end plates, so additional means are needed to damp its vibrations and/or shroud it. The final solution was to damp the plenum chamber tube and eliminate the outer shroud tube. (The plenum chamber tube for the new stoper now being developed will be made from a vibration-damping material consisting of a 1/4-inch aluminum-plastic-aluminum laminate.)(2)

From the plenum, the exhaust air was discharged through holes cut in the back head plate and into a Joy 150 muffler. The muffler consists of a stack of rubber baffle plates enclosed in an aluminum case.

Drill Rod Shroud Tube

There were two versions, each of which covers the drill rod with a tube that is supported only at each end. Other studies have shown that bonding a noise-absorbing coating to the entire length of the drill rod will substantially reduce drilling rates. Figure 5 shows the components of the "Integral Shroud Tube" and "Drill-Mounted Shroud Tube." The integral shroud tube encloses the drill steel by either a steel or plastic tube located between the collar and bit and mounted to the rod at its extremities by rubber isolating bushings. In this technique the shroud tube and rod are handled as one assembly, and the tube rotates with the drill rod. To remove the tube, the bit is detached and the tube and bushing are withdrawn over the rod.

The drill-mounted shroud tube encloses the drill steel by a tube fastened at one end to the drill front head and located on the rod by a rubber isolating bushing mounted immediately behind the bit. In this technique the drill rod rotates inside the tube during drilling. The tube can either be handled with the drill rod (by decoupling it from the drill body using a quick-release coupler) or be left mounted on the drill machine. In the latter case, the rod must be inserted into the shroud tube from the front. The shroud tube can be given an internal coating of plastic or rubber to damp the shroud tube vibrations. The internal coating should not contact the drill rod because this would reduce the drilling rate. Exterior coatings are not practical because they will wear against the sides of the drill hole.

* Use of brand names is for identification purposes only and does not imply endorsement by the Bureau of Mines.

An unmodified drill and a modified drill were demonstrated in a coal mine for Bureau of Mines (BOM) and Mining Enforcement and Safety Administration (MESA) personnel. Noise levels were reduced with the modified drill from around 114 dbA to around 95 dbA with a reduction of less than 10 percent in the drilling rate. Through numerous tests it was determined that the majority of the noise in the modified drill comes from the drill rod. Noise radiated from the drill body was determined to be about 89 dbA. Figure 6 shows quiet drill noise levels as a shrouded and exposed drill rod drills into a hole. As can be seen from figure 6 the drill rod shroud provides substantial noise reduction.

Because the shrouded drill rod is still the major source of noise, basic studies on drill rod noise emissions were undertaken.

Figure 7 shows an experiment in which a drill rod was hit at the end of a slight angle, and bending waves and sound pressure were recorded. As can be see the bending wave and sound pressure waveforms are very similar, showing that mos. of the noise is caused by bending waves. This is an interesting result because most of the energy is in the longitudinal waves, which are essential to the drilling process; bending waves, which result mainly from noncentralized-impact-worn chucks and bend rods, serve no useful purpose.

Another useful result of this study was the development of an equation to predict drill rod bending wave natural frequencies. Knowing the drill rod bending wave frequencies enables one to identify which components in the noise spectrum are being radiated from the drill rod and which are not.

As stated earlier, the development of the quiet prototype stoper drill has just begun under contract J0177125. The new drill will incorporate the noise reduction features of the demonstration drill and will include the following features:

1. A "valveless cycle" will be employed where the piston acts as the valve, and drill rod rotation is achieved by a separate air motor attached to the chuck housing. The valveless drills are much simpler than valved rifle bar machines (such as the L-47 where drill rod rotation is linked to the piston motion) and as such are easier to maintain and more trouble-free in operation. In such a cycle the air is ported to the downstroke and returnstroke chambers in exact relationship to the piston position. This is in contrast to conventional cycles in which the independent valve is activated by air pressure signals. Because of the exact relationship between piston position and airflow, valveless cycles can be precisely designed for high blow frequency and maximum efficiency. The cycle for the prototype quiet drills will use an expansion cycle and discharge the air into the plenum chamber at a lower pressure than would normally be possible with a conventional rifle bar-valve cycle. It is expected that reducing the exhaust pressure will produce a machine that is much more economical in the use of air and that will have a lower exhaust noise level to be muffled.

2. The external exhaust muffler will be eliminated, and an integral muffler will be designed into the exhaust plenum chamber.

3. The drill shroud tube will be "machine mounted," and improved shroud tube damping methods will be developed.

These quiet prototype stoper drills are scheduled for delivery in 1978, at which time inmine testing will begin. They will be the first "inherently quiet" percussive stoper drills manufactured. The technology developed for the quiet pneumatic percussive stoper will also be applicable for future "quiet" designs of other percussive drills such as jumbo-mounted drills and pavement breakers.

Effect of the Underground Environment and Loading Conditions on Equipment Noise Levels

In specifying acceptable noise levels for new mining equipment, the mine operator must provide the manufacturer with an aboveground sound level since in most cases the equipment must be tested on the surface at the manufacturer's site under no-load conditions. This requires that the mine operator must know and account for the increase in noise level at the operator's or bystander's position when the machine is placed in the underground environment. In addition, for those machines that interact with coal, i.e. continuous miners, loaders, etc., the mine operator must make allowances for the effect of load conditions on the noise level.

Because of the reverberant nature of the underground environment, the level existing underground will in most cases be equal to or greater than the level measured on the surface. This is illustrated in figure 8. The magnitude of the increase in level depends primarily on the absorption in the mine and the distance from the noise source. Thus, it is necessary to know the average absorption coefficient for a given environment as well as how sound propagates for the various underground configurations--room, tunnel, and room and pillar. Under contract H0346046 (Noise of Diesel-Powered Underground Equipment; Bolt Beranek and Newman), a methodology for predicting the increase in level using the technique of acoustic imaging was developed.

Correction factors were derived for the geometrical configurations of flat rooms and tunnels. Only a limited number of field tests were performed to verify this predictive scheme, and the testing was accomplished on diesel-powered equipment only, where the engine was always identified as the principal noise source and was easily characterized as the acoustic center which facilitated use of the corrosion factors.

In addition to using acoustic imaging techniques, for machines that cut or handle coal, it is necessary to estimate noise levels under loaded conditions in the mine from no-load noise levels measured on the surface.

The Bureau of Mines is presently extensively evaluating the acoustic imaging techniques under contract J0366030 (The Effects of Underground Environment and Loading Conditions on Noise Levels of Coal Mining Equipment, Donaldson Co.). Under this program noise levels of mining machinery are measured aboveground unloaded, and then noise levels of these same mining machines are measured underground unloaded and loaded. In addition, the mine dimensions, reverberant nature, and absorption nature are measured. This program involves obtaining noise data for approximately 22 machines, including continuous miners, cutting machines, coal drills, loading machines, shuttle cars, locomotives, and roof bolters. Noise data are presently being obtained on the aforementioned mining machinery. A preliminary analysis of the data that have been obtained indicates that the acoustic imaging technique works if the machinery major noise sources are assumed to be point noise sources.

The results of contract J0366030 will be available in 1978 and should provide a technique for the mine operator to predict machinery underground noise levels from aboveground measurements.

Mantrip Noise Control

Mantrip vehicles pose a unique problem because they are used by the majority of underground workers in traveling to and from their working sections. Therefore, the workers are exposed to the mantrip noise in addition to the noise at their work station. A recent study of sound pressure levels of mantrip vehicles showed noise levels from 80 to 102 dbA with an average of 94 dbA.

A continuous miner operator has more noise exposure than does a shuttle car operator, yet both use the mantrip vehicle for travel. If this vehicle is "noisy," it could cause the Noise Exposure Index (NEI) of the continuous miner operator to exceed unity, which would be in violation of the present noise standards, even though his occupational exposure is within prescribed limits. Thus, reducing mantrip noise below 90 dbA will help reduce worker exposure.

The Bureau of Mines, under contract H0166090 (Mantrip Noise Control, Bolt Beranek and Newman), is demonstrating noise control techniques for reduction of noise in mantrip vehicles to the minimum levels consistent with state of the art technology, with an eventual goal of 85 dbA or less, as measured at the approximate location of the passenger's ear. Under this program, noise control techniques are being demonstrated on a FMC-Galis 2190 portal bus. The FMC-Galis 2190 portal bus is shown in Figure 9. Table 2 shows noise levels measured under various conditions for the unmodified mantrip vehicle.

Table 2.--Unmodified FMC-Galis 2190 portal bus:
Measured noise levels (dbA) at various locations and operating conditions

	Passenger compartment		Driver compartment	
	15 mph	10 mph	15 mph	10 mph
Curve	93.5	92.5	88	88
Straight	90.5	80.0	87.5	82.5

The major noise contributors and the proposed quieting techniques are shown in table 3.

Table 3.--FMC-Galis 2190 portal bus:
Major noise sources and proposed quieting techniques

Major noise sources	Proposed quieting techniques
Wheel-rail structureborne noise..........	Resilient wheels.
Motor airborne noise.....................	Sound-absorbing motor enclosure.
Motor structureborne noise...............	Constrained layer damping.

As shown in figure 10, resilient wheels are basically composed of three sections; the inner and outer circular metal sections are separated by rubber blocks that provide the required vibration dampings. These wheels are commercially available(3) and are presently used on mass transit vehicles.

The demonstration retrofit noise control package delineated in table 3 is scheduled for installation and testing in the near future. Similar quieting techniques should be applicable to other rail-type mantrips.

Conclusions

With awareness that noise presents a major health problem to the coal miner in the form of occupational hearing loss, the Bureau's noise research projects are being directed at critical areas that have been previously identified. Progress is being made in applying available acoustic technology to mining machinery. The total cooperation of the mining industry is required for these efforts to be successful, and it should be emphasized that the industry-wide application of available noise control technology is still the responsibility of the mine operators and equipment manufacturers.

APPENDIX A

PUBLICATIONS RESULTING FROM BUREAU OF MINES NOISE PROJECTS

The following is a list of reports that are available in the area of Noise as a result of the Bureau of Mines contract research program. These reports may be obtained as follows:

Open File Reports (OFR) - An open file report is an unpublished Bureau of Mines report that has been made available as reference material. Open file reports may be inspected during working hours at the Bureau of Mines libraries in Pittsburgh, Pa., Denver, Colo., Spokane, Wash., Twin Cities, Minn., at the Energy Research and Development Administration library in Morgantown, W. Va., and at the Central Library, U.S. Department of the Interior, Washington, D.C.

National Technical Information Service Reports (NTIS) - Paper and/or microfiche copies may be purchased by writing to the National Technical Information Service, U.S. Department of Commerce, Springfield, Va. 22161.

Bureau contract number	Contractor	Report date	Report title	Report numbers
S0111345	Hydrospace Research Corp.	Jan. 72	Design of a Program of Acoustic Research Volume I Volume II	 OFR 2 (1)-72 OFR 2 (2)-72 Not in NTIS
G0122004	Penn State University	Nov. 72	Aspects of Noise Generation and Hearing Protection in Underground Coal Mines. 106 pp.	OFR 19-73 NTIS PB 219 087/4
H0220048	U.S. Steel Engineers and Consultants	Jan. 73	Muffler for Pneumatic Drill. 81 pp.	OFR 28-73 NTIS PB 220 372/7
H0133027	Bolt Beranek and Newman Inc.	May 74	Coal Cleaning Plant Noise and Its Control. 99 pp.	OFR 44-74 NTIS PB 235 852/AS
G0133026	Penn State University	Dec. 73	A Study of Roof Warning Signals and the Use of Personal Hearing Protection in Underground Coal Mines. 238 pp.	OFR 46-74 NTIS PB 235 867/AS
S0144091	Naval Ammunition Depot, Crane, Ind.	Dec. 74	Feasibility Study of Portable Calibration Instrumentation for Audio Dosimeters. 93 pp.	OFR 41-75 NTIS PB 242 577/AS

APPENDIX A - Continued

Bureau contract number	Contractor	Report date	Report title	Report numbers
H0220048	U.S. Steel Engineers and Consultants, Inc.	Jan. 75	Noise Control of Stoper Drills. 153 pp.	OFR 91-75 NTIS PB 246 381/AS
H0122054	Apt Bramer Conrad Assoc., Inc.	Apr. 75	Noise Abatement in Mining Machinery. 108 pp.	OFR 1-76 Not in NTIS
H0144079	Bolt Beranek and Newman Inc.	July 75	Noise Control in Surface Mining Facilities. 156 pp.	OFR 64-76 NTIS PB 253 257/AS
G0155032	Penn State University	May 76	Evaluation of Speech Processing Systems. 150 pp.	OFR 96-76 NTIS PB 255 787/AS
H0155142	Naval Weapons Suport Center	March 76	Characteristics of an Acoustic Coupler for Use in an Audio Dosimeter Calibrator. 37 pp.	OFR 113-76 NTIS PB 259 586/AS
H0155131	FMC Corporation	May 76	Evaluation of Wet Head Drilling Techniques. 69 pp.	OFR 44-77 NTIS PB 264 997/AS
H0144078	Foster-Miller Associates, Inc.	Oct. 76	Alternate Conveyor Design for Mine Machinery.	OFR 52-77 NTIS PB 265 151/AS
H0144079	Bolt Beranek and Newman Inc.	Dec. 76	Practical Reduction of Noise from Chutes and Screens in Coal Cleaning Plants. 73 pp.	OFR 59-77 NTIS PB 265 344/AS
H0346046	Bolt Beranek and Newman Inc.	Mar. 75	Noise of Diesel-Powered Underground Equipment.	NTIS PB 243 846/AS
H0133027	Bolt Beranek and Newman Inc.	May 74	Coal Cleaning Plant Noise and Its Control	OFR 44-74 NTIS PB 235 852/AS

The following is a f reports published by the Bureau of Mines as a result of the inhous e program. To obtain copies of these reports, apply to Publications Distribution Branch, Bureau of Mines, U.S. Department of the Interior, 4800 Forbes Avenue, Pittsburgh, Pa. 15213. Because of the limited editions, only one copy of any publication can be sent to the person applying.

Year	Title	Authors	Report No.
1975	NOISE CONTROL - Proceedings: Bureau of Mines Technology Transfer Seminar, Pittsburgh, Pa., January 22, 1975	Staff, Pittsburgh Mining and Safety Research Center, U.S. Bureau of Mines	IC-8686 NTIS PB 245 571/AS
1975	Noise Reduction of a Pneumatic Rock Drill	A. Visnapuu and J. W. Jensen	RI-8082
1974	Noise Abatement of Pneumatic Rock Drills	C. R. Summers and J. N. Murphy	RI-7998 NTIS PB 240 095/OWD
1974	Noise Dosimeter Performance	K. C. Stewart and T. Y. Yen	RI-7876
1973	Response Variations of a Microphone Worn on the Human Body	T. L. Muldoon	RI-7810
1971	Noise in Underground Coal Mines	J. A. Lamonica, R. L. Mundell, and T. L. Muldoon	RI-7550

APPENDIX B

BUREAU OF MINES CURRENT NOISE RESEARCH PROJECTS

Bureau contract no.	Contractor	Description
J0155099	Ivor Hawkes Associates	Development of Other Pneumatic Drills
J0177125	Ivor Hawkes Associates	Develop Six "Inherently Quiet" Prototype Production Stoper Drills
J0366030	Donaldson Co.	Study of the Effects of the Underground Environment and Loading Conditions on Coal Mining Equipment Noise Levels
H0166090	Bolt Beranek and Newman, Inc.	Mantrip Noise Control
H0166012	Donaldson Co.	Auger Miner Noise Control
H0166057	Bolt Beranek and Newman, Inc.	Noise of Surface Coal Mining Equipment
J0265021	Booz, Allen, and Hamilton	Engineering Noise Control Guidelines Handbook for the Coal Mining Industry
J0166122	VAST Corp.	Design of a Quiet Rock Drill Using Principles of the Leavell Model D Pavement Breaker--Feasibility Study
S0366041	Bendix Corp.	Fabrication of a Time Resolved Audio Dosimeter System
H0155133	Bolt Beranek and Newman, Inc.	Continuous Miner Noise Control
H0155155	Bolt Beranek and Newman, Inc.	Coal Preparation Plant Noise Control
H0262013	Bolt Beranek and Newman, Inc.	Noise Control of Underground Load-Haul-Dump Machines
H0366024	Bolt Beranek and Newman, Inc.	Noise Control of Jumbo-Mounted Drills
J0377014	Bolt Beranek and Newman, Inc.	Noise Control in Surface Noncoal Plants and Mills--Source Diagnosis and Abatement Techniques

APPENDIX B - Continued

Bureau contract no.	Contractor	Description
J0377002	Aerophysics	Noise Control in Surface Noncoal Plants and Mills--Noise Control in Secondary Crushers
J0377001	Industrial Acoustics Co.	Noise Control in Noncoal Plants and Mills--Noise Control for Secondary Crushers
J0366055	Donley, Miller, and Nowikas, Inc.	Noise Control in Noncoal Plants and Mills--Study of Noise Exposure in a Cross-Section of Taconite Plants
J0177049	Contract to be awarded in near future	Demonstration of Bulldozer Noise Control
J0177060	Contract to be awarded in near future	Investigation of Direct Airborne Noise Generated During Coal Cutting
J0177039	IIT Research Institute	Flammability Evaluation of Noise Control Products for Use in Underground Coal Mines

References

1. Incramute (AMPCOLOY 420), AMPCO Metals, Milwaukee, Wisconsin 53201.

2. Antiphon. Speciality Composites Corp., Newark, Deleware 19713.

3. Penn Machine Co., Porter Building, Pittsburgh, Pa. 15219.

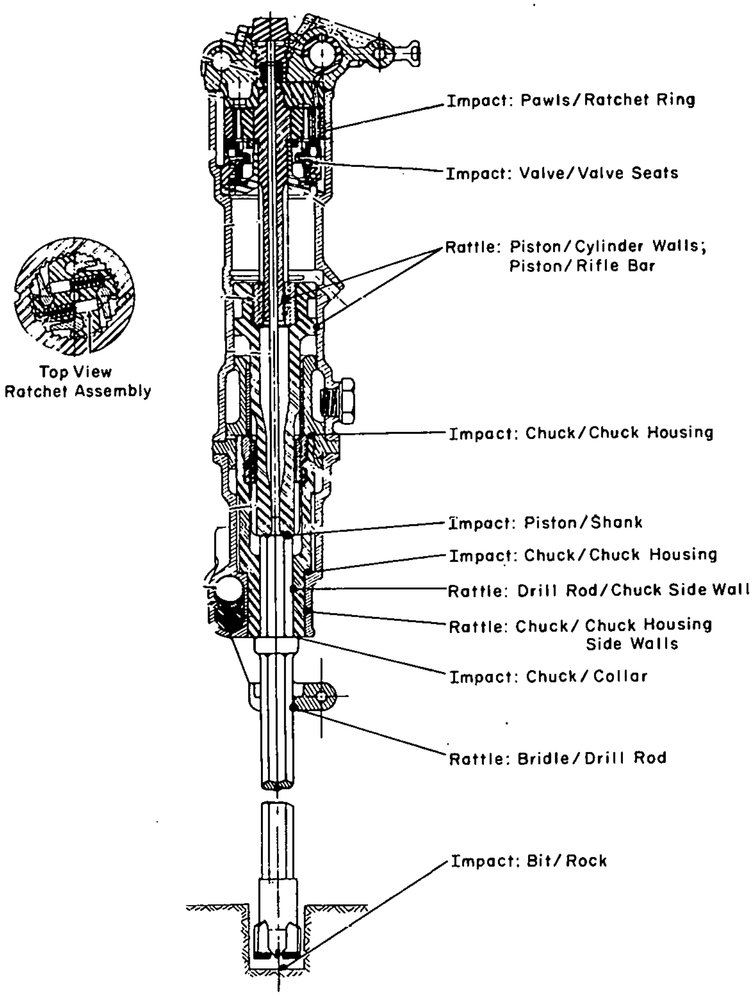


Figure 1. - Mechanical noise sources in the Joy L-47 drill (excluding exhaust and flushing air).

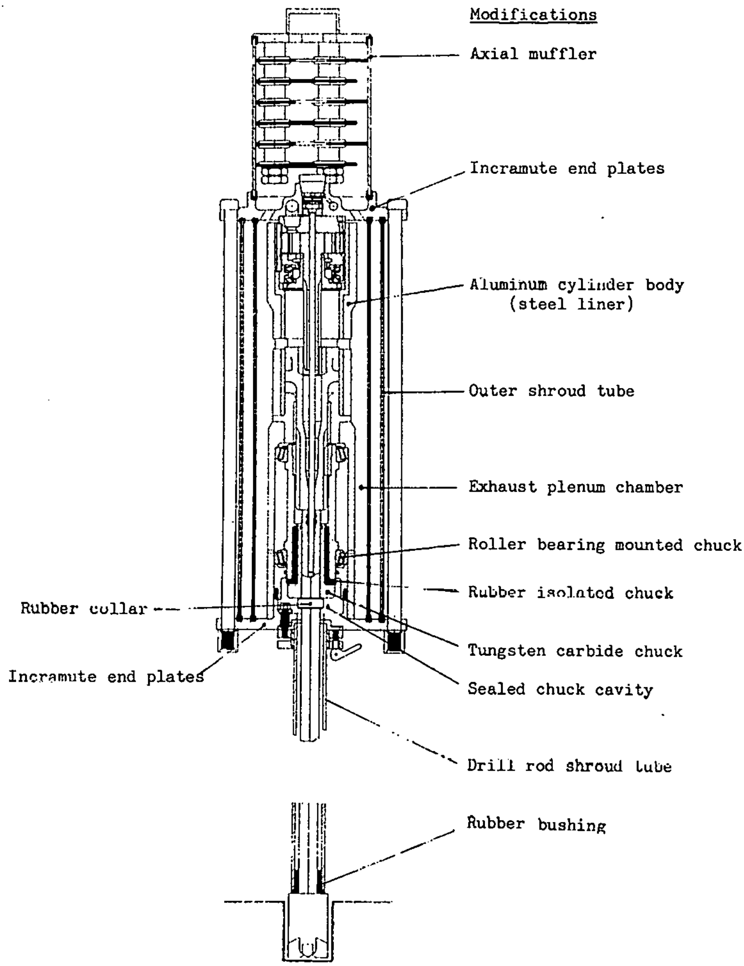


Figure 2. - Modified Joy L-47 drill and list of modifications to the original drill.

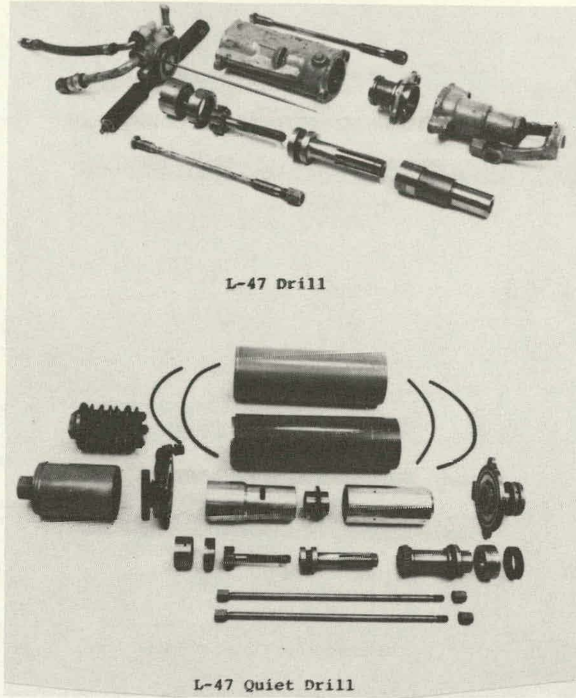


Figure 3. - Joy L-47 drill and quiet drill components.

Figure 4. - Quiet stoper drill in operation.

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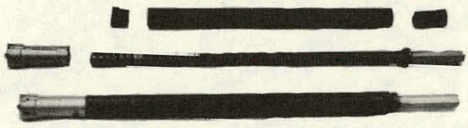
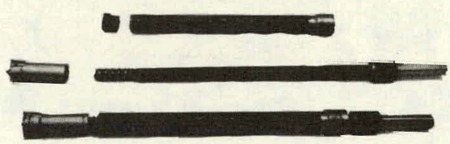


Integral shroud tube



Drill mounted shroud tube

Figure 5. - Drill rod shroud tube components.

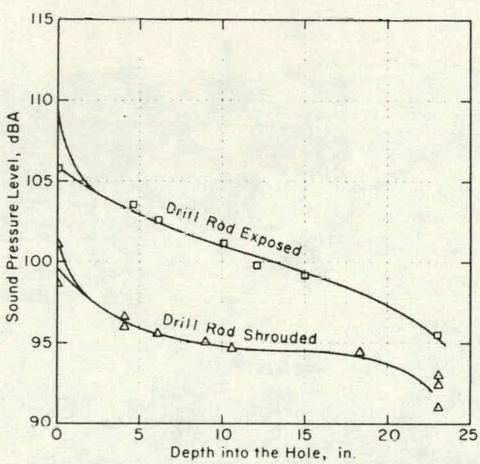


Figure 6. - Drill rod noise (1-inch diameter, 2-ft long rod). Machine body noise was approximately 89 dbA for these tests.

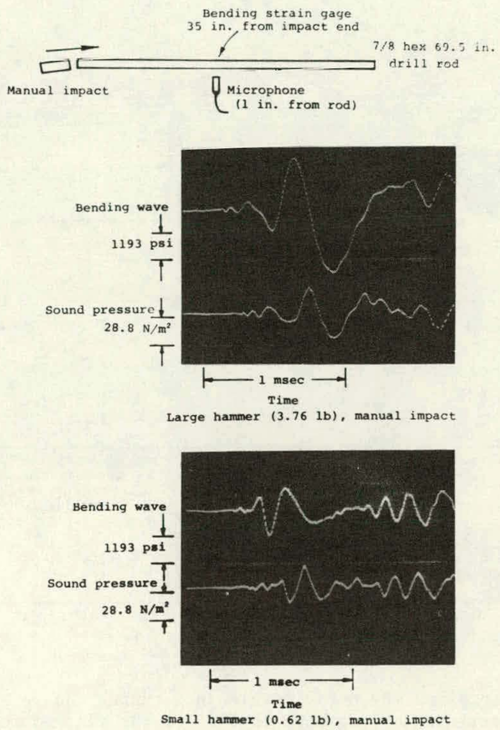


Figure 7. - Simulated drill rod bending waves and sound pressure levels.

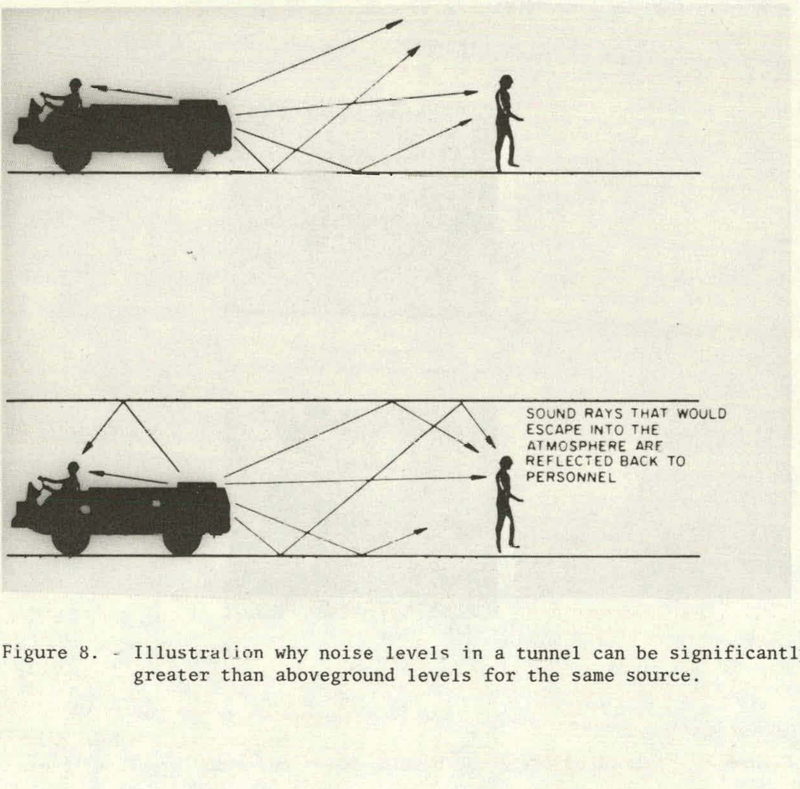


Figure 8. - Illustration why noise levels in a tunnel can be significantly greater than aboveground levels for the same source.

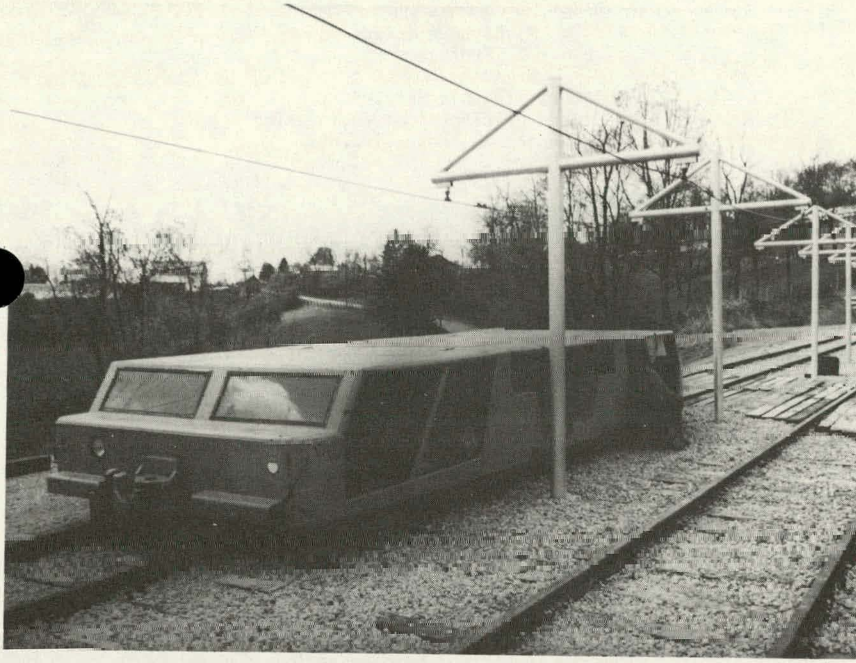


Figure 9. - FMC-Galis 2190 portal bus.

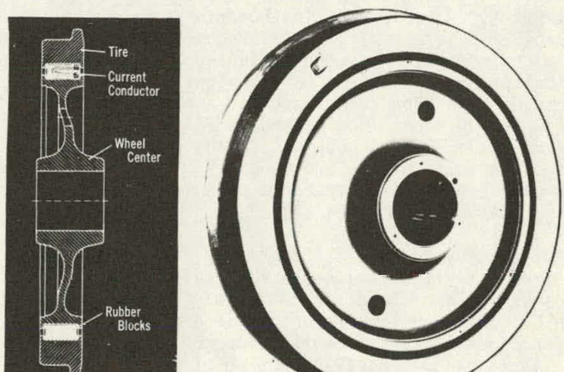


Figure 10. - Resilient wheel construction.

ELECTRONIC MONITORING AND CONTROL OF COAL MINE VENTILATION

M. Dayne Aldridge
Professor

West Virginia University
Department of Electrical Engineering
Morgantown, West Virginia

Introduction

One of the key factors in the safe operation of any coal mine is a reliable and adequate ventilation system. Although ventilation networks can be subjected detailed mathematical analysis and computer simulation, everyday operation of coa mine ventilation systems depends very little on such techniques but rather on experience and "cut-and-try" methods. While such methods have been adequate in the past, it is obvious that the ability to know the state of the total ventilation system at any given moment and to readjust flow distribution by remote control would be valuable from both safety and economic points of view.

Since 1970, the West Virginia University Department of Electrical Engineering has been developing, under the sponsorship of the U.S. Department of the Interior's Bureau of Mines, a system to remotely monitor critical mine ventilation parameters at key locations in mine airways. This work has led to the installation of a mine-wide monitoring system which has made experiments with remote ventilation control possible for the first time in the United States.

This paper describes the basic objectives and design criteria for mine ventilation monitoring and control. The WVU Mine-Wide Monitoring System installed in Eastern Associated Coal Corporation's Federal No. 2 mine is discussed followed by a description of an electronic remote controlled mine ventilation regulator designed for experimental use. Results of the testing program are presented.

Monitoring Mine Ventilation Networks

The need for monitoring conditions in coal mines for safety purposes has been recognized for many years. A variety of instruments are used in coal mines today to insure the presence of a safe environment. However, most devices used in United States coal mines operate only when activated by the miner. Continuous monitoring of critical parameters offers many obvious advantages over current practices. These advantages have been recognized for a number of years in the European mining countries [1]. The National Coal Board is researching sophisticated computer based monitoring [2,3]. Continuous monitoring has been under development for gold and uranium mines in South Africa and Canada [4,5]. In recent years the U.S. Bureau of Mines has conducted several research projects in electronic mine monitoring [6], including the work described here.

Since the ventilation system plays the primary role in the control of methane and dust in normal mine activities and smoke and noxious gases during fires and explosions, the flow of air and the gases contained therein are the most importa parameters that can be monitored for prevention of explosions and fires. Thus, work described herein concentrates on ventilation monitoring with the following three objectives: 1) to provide early detection of problems well before human injury occurs, 2) to provide information on situations that develop beyond the incipient stages to aid the safe evacuation of personnel and control and elimination of the problem, and 3) to aid mine personnel in the day-to-day operation and maintenance of the mine system in minimizing hazards and maximizing efficiency.

Following a study of the candidate parameters for measurement and the availability of suitable sensors, the variables chosen for the WVU program were air velocity, differential pressure between airways, methane, carbon monoxide, temperature and relative humidity. Details of this study are presented in previous reports [7,8,9].

In order to most effectively monitor each variable, it would be best to have sensors placed at each position where problems are most likely to occur. Obviously this is not practical. One solution is to place sensors at key positions so they can maintain surveillance over designated areas. In view of the way the air is supplied to each working section and fed back along nearby return airways, a natural location for the sensors is near the major splits of air. By properly choosing the splits to be monitored, the ventilation system can be effectively broken into zones or areas where all the air going into and out of each zone is under surveillance. Thus the occurrence of fire, abnormal methane liberation, ventilation disruption, ., can be detected and the zone in which it is located determined.

There are several advantages to the area monitoring method. By judiciously positioning the sensors, the monitored areas can be established so as to put an entire mine under surveillance with a reasonable number of sensors. The sensor installations would be relatively permanent thus minimizing the need for relocating and calibrating. Disadvantages are primarily two-fold: obviously these sensors cannot provide information on conditions at or near the working places, and there is an inherent time delay of minutes in the transportation of gases in the ventilation system. Because of these disadvantages, additional sensors may also be required near the working areas and at other positions in the mine to yield the desired results.

Study of the overall monitoring problem has led to the conclusion that sensors should, in general, be placed as follows to effect an acceptable surveillance of an entire mine:

1. Each active section:
   a. complete array of sensors in each airway near the last split serving each section
   b. sensors near working place - methane sensor on mining machine as a minimum requirement
2. Monitor at least methane and carbon monoxide at critical airway junctions, especially ventilation from "worked out" areas.
3. Measure pressure between airways along main routes with sensors placed at major splits, spaced as often as desired between splits, and near fans.
4. Place carbon monoxide or other acceptable fire/smoke sensors along trolley and belt haulageways with spacings as required to achieve a maximum response time of about five minutes. They should also be placed near electrical equipment such as rectifiers and pumps.

These conclusions are based on the observation that the sections are generally located at the extremities of the ventilation network where the air becomes contaminated and is channeled into the return airways. Since the majority of the workmen in a mine are located on or near the sections, it is important to know if fresh air is reaching them. Thus by locating a complete array of sensors in the airways serving each active section, conditions for each individual section can be moni- d. A minimum of measurements need to be made near the actual working place to rtain that adequate fresh air is actually reaching the workmen and not being ted into a return. For example, in a gassy mine, monitoring methane via the mining machine mounted instrument should be adequate to determine if sufficient ventilation is present.

Problems can develop at locations other than an active area. Since these need to be quickly detected and analyzed before endangering the miners, sensors must be placed at other locations in the ventilation system. In order to protect the safety of miners who enter return airways, sensors must be located at important junctions. At the very minimum, sensors for methane and carbon monoxide should be used at these locations. In order to analyze changes in the ventilation system, differential pressure should be measured throughout the main airway system. Also, the measurement of pressure at each fan will determine if fan performance is proper. Detection of fire can be effected by using carbon monoxide sensors which are placed along track haulage lines and beltways. The sensors can be spaced in accordance with the normal air velocity to achieve any maximum desired response time. Existing beltway fire sensor systems should be integrated into the overall monitoring system. Sensors should be placed so as to detect fire from unattended electrical equipment such as rectifiers and transformers which are ventilated into return air.

Data from all the sensors discussed above must be converted to engineering units, checked and processed to determine the status of conditions in the mine, and displayed as required by operating personnel. As originally conceived, the research reported herein began with the hypothesis that the most optimum arrangement was to locate all data processing and display equipment at one centralized location out the mine. Early studies indicated that one of the major mine management problem during normal and emergency situations is the lack of an up-to-the-minute knowle of conditions throughout the mine. Thus, it was concluded that the major thrust of the system design should be to accumulate as much useful data as possible at a central point and to process and display it in a useful and practical manner. Because of the number of sensors required to cover an entire mine, it is apparent that some form of digital data processing must be employed to scan the data automatically to detect unusual or abnormal conditions. Because of the cost of digital processing equipment only a few years ago, the only practical solution to this problem was the centralized processing configuration. If it would ever be desired to display data near sections or other underground locations with this configuration, the processed data must be transmitted back to the underground display point. However, the recent advent of the microcomputer has made decentralized processing practical and thus is requiring that mine monitoring system architecture be reconsidered. It should be emphasized that the authors are not suggesting that the data processing should be totally decentralized. On the contrary, the major value of mine-wide monitoring is the accumulation of all data at one point for display, analysis and storage. Any system architecture must supply adequate centralized processing for these functions.

The work described here utilizes a totally centralized processing and display arrangement. All conversions of voltages to engineering units, generation of alarms and display of data is done at the central control station. An effort has been made to simplify the calibration and other underground maintenance procedures as much as possible. While this tends to shift most of the responsibility for system update to the central control operator, it places the major requirement for training at this position. An effort has also been made to simplify the normal operation of the central computer thus minimizing the training to be required. For example, communication with the computer will be through a system of code words that are derived from the normal terms used at the mine.

Remote Control of Mine Ventilation

The main objective of remotely controlling mine ventilation systems is to permit changes in air flow distribution and quantity in a matter of minutes rather than over a period of several hours or days as now required. The value of such capability for mines with methane and dust control problems is obvious. In the case of explosion or fire, the ability to redistribute and control air flow wou be extremely valuable in aiding the safe evacuation of men and in controlling t... spread or propagation of fire. It should be recognized at the outset that remote control of mine ventilation must be coupled with a mine-wide monitoring capability. That is, the remote control capability is of little value unless its effect can be immediately measured throughout the mine.

Remote control can be effected in at least two different ways. One way is to control the blade pitch or speed of ventilation fans thus controlling the total quality of flow contributed to the system by that unit. This could be used on primary units to minimize power requirements for ventilation or on underground booster units to assure adequate flow is maintained in certain areas. Another approach is to remotely control the aperture on underground regulators and thus control the distribution and total quantity of flow in the system. It is this latter method that is under consideration in this report.

There has been very little experimentation with remote control ventilation. The only other work known to this author has been in Poland, the USSR and Belgium [10,11,12]. Extensive use of automatic ventilation control has been reported from the USSR [13]. It is believed that the efforts reported here are the first attempts of regulator control in room-and-pillar mining.

The WVU Experimental Monitoring System

In order to determine the feasibility of designing a monitoring system using approach discussed above, a two phase program to test the concept in an opera- coal mine was carried out. For the first phase, a system was designed and tests conducted to determine if the parameters could be accurately and reliably measured in a mine environment and to develop data processing techniques to yield a reliable alarm system. A secondary objective was to learn about the practical problems of operating and maintaining such a monitoring system.

Provisions were made to instrument two mining areas in the Eastern Associated Coal Corporation's Federal No. 2 mine with a set of sensors in four airways serving each section. Two telemetry stations served four sensor packages each and a special twisted pair telephone line carried frequency shift keyed signals to outside the mine. An additional telephone circuit provided communication to the central data processing and control center at West Virginia University about 40 kilometers from the mine. The equipment began operation during November 1972 and continually provided data for two years.

Due to the success of these experiments, phase two of the testing program was established. This involved the expansion of the experiment to cover the entire coal mine involving 13 different monitoring areas.

Figure 1 shows the Federal No. 2 mine outline and the approximate sensor locations. It should be noted that since the ventilation network is continually changing with the mining activity, minor alterations to the monitoring system are required on an infrequent but continuing basis. Equipment was prepared to monitor at least 13 operating areas. The operating areas include two longwall sections, one shortwall section and one experimental tunneling section. The remaining sections utilize continuous mining machines in room and pillar developments.

With the sensor layout shown in Figure 1, the following types of conditions can be detected if they occur anywhere in the entire coal mine with sufficient intensity:

1. If any parameter falls outside of acceptable bounds.
2. The occurrence and approximate locations of:
   a. major roof falls,
   b. short circuits in the air system,
   c. abnormal methane liberation,
   d. fire, and
   e. partial or complete fan failure.

A block diagram showing the system hardware configuration is presented in Figure 2. The telemetry system uses a dedicated audio telephone circuit which is shared by all units in the mine. Two frequency multiplexed channels are utilized where several units timeshare one frequency channel using frequency shift keyed modulation.

The sensors used in this system are listed in Table I.

TABLE I
TYPES OF SENSORS USED

Parameter	Sensor Type
Air Flow	Vortex/Ultrasonic
Differential Pressure	Diaphragm/Magnetic Balance
Methane	Catalytic Hotwire
Carbon Monoxide	Solid State
Temperature	Thermistor
Relative Humidity	Solid State

The last four sensors in the above list are enclosed in a steel housing shown on the left side in Figure 3. A diaphragm pump pulls an air sample through an (faced filter and is fed to the sensors through a small airway manifold in a sen: mounting block. Voltage regulating and amplifying circuitry is also enclosed i the housing. Electric power is fed to the package and the output signals are transmitted to nearby telemetry equipment by a cable containing 12 pairs of No. 19 solid conductors.

The air flow sensor is enclosed in a separate housing as shown on the right side of Figure 3. This device senses air movement by measuring the rate of vortex formation in the wake of a generating rod. An ultrasonic method is used for detecting the vortices and is unaffected by normal dust and water spray [14]. A typical installation of a sensor package and air flow sensor is shown in Figure 4.

The digital telemetry units have a capacity for 20 analog channels and four on-off command channels for each monitoring station. The telemetry equipment along with the sensor power supply, intrinsically safe interface circuits, and differential pressure transducers are housed in a steel enclosure mounted on skids as shown in the typical installation in Figure 5. The housing is positioned in fresh air and does not require additional protection from possible roof falls or damage during transportation.

A block diagram of the central data processing and control center is shown in Figure 2. All alarms and acknowledgments are simultaneously printed on one typewriter and stored on magnetic tape. The display panel shows the location of any alarm via a blinking red light on a mine map. The light blinks until the alarm is acknowledged then will continue on until the parameter falls back in acceptable limits. A green light is on when all parameters are within bounds. One red-green light is used for each mining section or return air station. Although the blinking red light indicates that at least one parameter is alarming in that area, the typewriter prints the location (station and airway), type of alarm (caution or danger), and a unique identification number for each parameter in the alarm. To acknowledge an alarm, the operator must use the identification number and his initials. A separate acknowledgment must be made for each alarm or an "acknowledge all" statement can be entered. All alarm and acknowledge data are printed on the system log typewriter.

Each sensor is queried once each minute, the data converted to engineering units and alarms levels checked. The data is stored in a disk memory and retai for one hour. Thus, at any given time all data for the immediate previous hour available for quick retrieval. At the end of each hour a statistical analysis summary of the previous hour's data is conducted and placed on magnetic tape. Once each day a summary of the hourly summaries is conducted, stored on an additional tape for permanent record and printed on the second typewriter. The daily summary tape contains the immediate past day of data at any given time. At any time desired by the operator he may call most any combination of data from the disk memory or hourly summary tape. This may be printed at high speed onto the CRT for rapid viewing or printed on the second typewriter for a permanent record. The output device is determined by the keyboard on which the command is typed.

Experimental Remote Control of Ventilation

An experimental remote controlled regulator has been constructed and tested. The regulators have been installed in the return airways which serve one working section of Federal No. 2 mine. A pictorial diagram of the experimental remote control ventilation regulator system which has been constructed is shown in Figure 6. The system consists of the remote control panel which is located at the monitoring data processing center, the underground control electronics which is located in a fresh airway with a monitoring telemetry station, and two regulators each of which are located in one of the return airways which serve one mining area. Signals are frequency division multiplexed over the same telephone circuits used for monitoring. A typical underground installation is shown in Figure 7.

The regulators used have a multiple vane, opposed action configuration as own in Figure 8. This design was chosen because it is not only readily available Trom the commercial heating and air conditioning industry, but also offers many advantages over other configurations considered. A man door must also be provided near the regulator as shown in Figure 7. A steel frame is used which mounts in a standard block stopping and allows easy disassembly and transport.

The regulator is actuated by an electric motor placed in an explosion proof enclosure and the angular position of the vanes is determined by an absolute digital position encoder attached to the shaft of one of the vanes. The position of the regulator vanes are constantly monitored and transmitted for display to the control panel as shown in Figure 9. The display consists of 16 red LED's in a circle. Although the encoder can resolve angles to less than one degree, the system is capable of commanding and reading out moves to only 16 different positions. A 16 position dial switch is located in the center of the display. To move the regulator to a new position, the following procedure is employed. The dial switch is rotated to the desired angular position. The command button is pushed and a command word is transmitted to the underground control which echoes the received command back to central control. The transmitted and echoed commands are compared and if a match occurs, the green ready light is activated. Otherwise a red error light is turned on for a few seconds indicating the command must be reinitiated by the operator. After the green ready light is activated, the operator has approximately 10 seconds to issue the final execution command. This is accomplished by simultaneously pushing a spring loaded toggle switch and pressing the execute button. As the motor rotates the vanes, the LED's light to indicate the actual position. The vanes should stop moving at a position within about one degree of the chosen position. If the execute command is not given within ten seconds of the activation of the green light, the system defaults briefly to an error condition and then falls into the normal position monitoring mode.

Experimental Results

The continuing operation of the experimental monitoring system is providing results and conclusions on a daily basis. This section presents a summary of several important results and conclusions derived therefrom.

Experiments with processing the outputs of the air velocity and differential essure transducers have shown a smoothing time of about one minute is required. ectronic averaging of the outputs with simple resistance-capacitance integrators with time constants of 40 to 60 seconds have proven adequate.

The most valuable parameters have been found to be air velocity, percent methane in air, and differential pressure. With sensor types listed in Table I, recalibration is required about once every two months to maintain acceptable accuracies (±10% for air velocity and differential pressure and ±0.1% methane).

Experience has shown that the most fundamental feature that a mine monitoring and control system must possess is versatility. This is necessary because of the wide variety of conditions that exist among different mines and the continually changing conditions at any one mine. Versatility must exist at various levels in the system including both hardware and software features.

Remote stations should be able to operate without supervision from a central computer if conditions require. There should be provisions for local readout of data in engineering units and of alarm status. Provisions should be made for battery operation of the station for periods of at least an hour in the event primary power is disrupted.

Many situations require that a readout and/or control function take place several hundred meters from a remote measuring station. For such cases it would be desirable to have a device which reads out data or executes control based on alarms from one particular remote station.

Provisions should be made for the system to use either a simple or sophisticated central control station. Small mines will not need the complexity and capability required in large mine situations.

Experiments with the remote ventilation control indicates its practicality and general usefulness. Efforts should be made to simplify the actuator design to be intrinsically safe. Such simplifications should make remote regulator control economical.

Concluding Remarks

The experiments reported here are proving the value and practicality of electronically monitoring and controlling ventilation in United States coal mines. The value for both safety and operating efficiency will be demonstrated as additional design improvements and experiments continue. While similar experiments have been conducted in other countries, the effort reported here utilizes the latest in electronic technology. Continued development will hopefully contribute to a safer and more acceptable environment for coal miners throughout the world.

Acknowledgments

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Disclaimer

The views and conclusions contained in this document are those of the author and should not be interpreted as necessarily representing the official policies of the Interior Department's Bureau of Mines or the United States Government.

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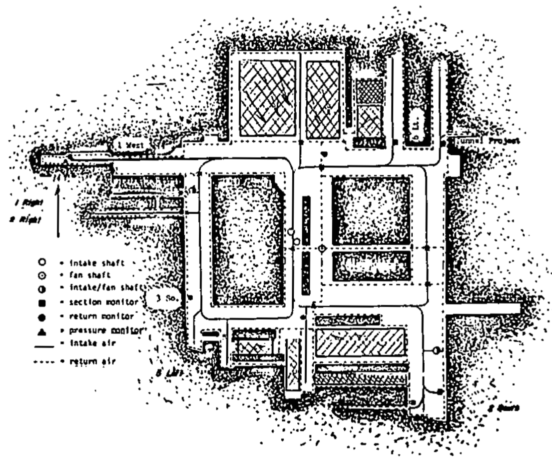

FIGURE 1: LOCATION OF SENSORS IN FEDERAL NO. 2 MINE

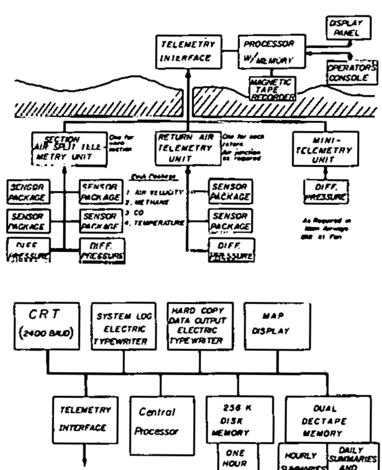

FIGURE 2: BLOCK DIAGRAM OF MONITORING SYSTEM

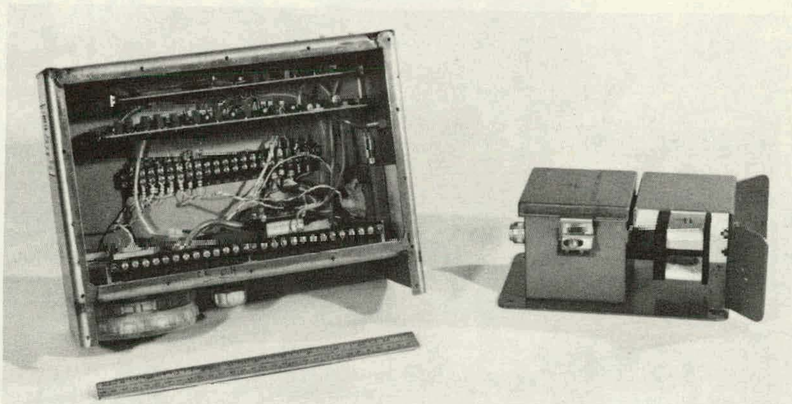

FIGURE 3: SENSOR PACKAGE AND AIR FLOW SENSOR

FIGURE 4: TYPICAL INSTALLATION OF SENSORS

FIGURE 5: TYPICAL INSTALLATION OF TELEMETRY STATION

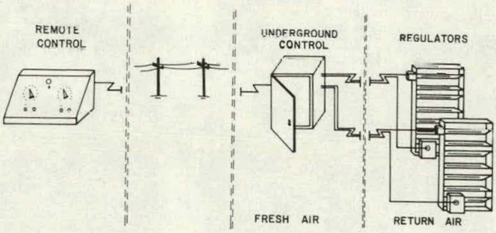

FIGURE 6: REMOTE CONTROL VENTILATION REGULATOR

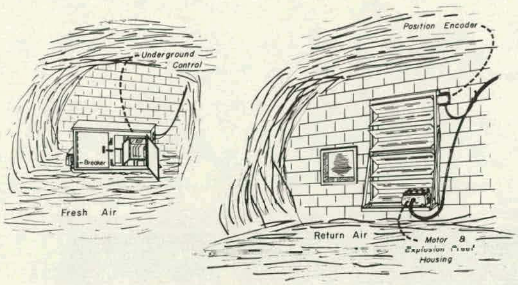

FIGURE 7: TYPICAL UNDERGROUND INSTALLATION OF REGULATOR

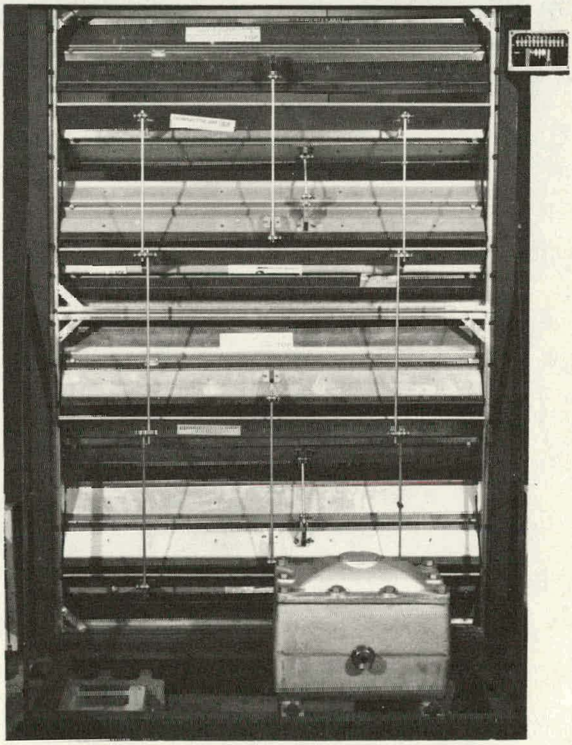

FIGURE 8: MULTIPLE VANED VENTILATION REGULATOR

FIGURE 9: CONTROL CONSOLE FOR VENTILATION REGULATOR

A SYSTEMATIC METHOD OF ESTIMATING THE COST OF INJURY-ACCIDENTS IN UNDERGROUND COAL MINING

Douglas C. Boyd
Operations Research Analyst

U.S. Bureau of Mines - Denver Mining Research Center
Denver, Colorado

and

Daniel G. DiCanio
Systems Engineer

FMC Corporation, ESD
Santa Clara, California

Introduction

Not until a dollar is invested is there a risk of making a return on that investment. Not until one person starts work in a mine is there a risk of an accident. If we do not invest or put that person in the mine, we are not confronted by any risks. This paper will address one of the factors which affects the risk of a favorable return on the dollar invested in coal mining, that of the costs involved with underground coal mining accidents.

The Bureau of Mines has pioneered accident cost investigation since the 1920's providing the first real evidence that advancement of industrial safety pays returns in dollars as well as in human life and well-being. The bulk of these early indagations dealt with only direct cost of insurance premiums and medical expenses which resulted from an industrial accident — only the costs which appeared as payouts on the company books.

In the 30's H. W. Henrich pioneered investigations which showed that industrial accidents impacted more than just the direct operating expenses of a company in higher insurance premiums. His work produced strong evidence that the uninsured costs of accidents far outweighed those covered by insurance. He developed and published a ratio of insured to uninsured costs of 1:4. It is used even today to estimate total costs of accidents despite his cautionary illusion to blanket applications.

In the late 40's and early 50's, R. H. Simonds developed what is now regarded as the principal method for estimating the total economic impact of industrial accidents by extending the definition of indirect or uninsured costs.

This method served as the basis for the logic development of the Accident Cost Indicator Model (ACIM) developed for the U.S. Bureau of Mines under Contract)225031 by the Engineered Systems Division of the FMC Corporation. Several depar-ıres from the Simonds' method were made, however, to satisfy certain requirements :quested by the Bureau of Mines and to accommodate some of the information collected in the study.

This paper will try to set forth the logic and methods used in the ACIM development, a brief discussion of the elements used in the estimated costs, observational foundations for the estimation of post-accident productivity losses, and a brief summary of the indicators obtained from an application of the Accident Cost Indicator Model to 1974 industry data.

At the outset it must be understood that an accident in this paper refers only to work-related accidents which cause an immediate traumatic injury or death to the accident victim. It may also be noted that the methodological principles and cost elements described may be applied to any industrial work environment, although the application here is to the underground bituminous coal mining industry.

Objectives

The primary objective in developing the ACIM was to estimate the total tangible cost of accidents in the underground bituminous coal mining industry in any given year, and to divide the total cost among the societal sectors affected, the work tasks involved at the time of the accident, and other characteristics of the work environment.

The tangible accident costs are measurable financial losses incurred as a consequence of the accident. Losses that can be measured with validity and those that would have been avoided had the accident not occurred both quality as tangib accident cost elements. Loss by direct payout (e.g., indemnity payments of wage compensation), indirect expenditures of resources (e.g., salaries paid during an accident investigation), or the non-realization of an expected income (e.g., unearned future wages) are all considered measurable losses to one of the societal sectors on which an industrial accident has an impact.

Avoidance losses are treated equally with direct payouts for two reasons. The first reason, besides that of the direct payouts of insurance premiums and disability compensation showing on a company financial statements as costs of doing business, is the loss in revenue from decreased productivity following a serious accident or fatality. This can be many times greater than direct payouts. This very real loss also appears in the statements, as lower gross income, and must therefore be considered in the losses to society. The second reason is it is a means of expressing the measurable hardship which a disabling or fatal accident thrusts upon a family in the form of reduced income for the victim's remaining or expected working life.

This last reason is pertinent because, in recent years, coal mining has become a relatively well-paid occupation in all the major coal mining states. and the average annual wages of the miners is at least twice the average wage of all other occupations. In these states, both state and private workers compensation funds are the major source of wage compensation for miners who are permanently disabled or killed in accidents. In all of these funds, the maximum wage compensation is limited by the average wage of the state. Thus, in many cases, a coal mining family struck by permanent disability or death from an accident is compensated for less than half the future earnings the family could have expected had the accident not occurred. Including the federal and union fund compensation paid to a permanently disabled miner or the surviving family, only a fraction of the income that would have been earned had the accident not occurred is received by the mining family.

Another requirement in developing the Accident Cost Indicator Model was to base the procedures for estimating accident costs on information in publically available sources so that the procedures could be updated periodically to reflect changes in the economic conditions. To meet this objective, a series of algorithms using published information were developed in the ACIM to compute elements of cost based on characteristics of the accident and ensuing injury. The philosophy used to develop the cost algorithms was to estimate resource units expended or lost as a result of an accident, and then to compute the cost of the resource by applying u costs which could be updated for each resource.

For example, Figure 1 shows the structure of the algorithm used to compute the fraction of income paid by workers' compensation funds to a totally disabled accident victim. The resources expended (days of disability; impairment of body function) are determined from data about the accident. Characteristics of the victim's family profile (e.g., married or not; number and ages of children) are generated from statistical data. The state in which the accident occurred is used to select the appropriate unit compensation cost schedules. The accident data, family profile statistics, and unit costs are processed by the algorithm to compute the annual cost, and the expected total cost over the expected remaining working life of the victim.

Similar logic is used in the other algorithms. That is, certain information in the accident data is combined with statistical data to estimate resources expended, and unit base costs are applied to the estimates to compute total cost.

In conjunction with the requirement for information from public sources of information, the Bureau of Mines required the computer data bases maintained by MESA's Health and Safety Analysis Center (HSAC) in Denver, Colorado be used as the source of all data about accidents, production and worktime. The two data bases which the ACIM use are the HSAC master files:

(1) Coal Accident and Injury File (CAIF) which contains coded information for all injuries reported in a calendar year in accordance with the 1969 Coal Mine Health and Safety Act.
(2) Address/Employment Master File (AEMF) which contains monthly production and employment for all mines which operated during a calendar year.

Use of these data bases insures that the source information used by the ACIM is the same as that used in MESA publications, and enables the ACIM to be run periodically with consistent data.

ACIM Cost Elements

The following elements include cost elements which are covered in part or entirely by occupational insurance. The costs computed for these elements are not, however, based on the cost of the insurance to a mining company, but on algorithms which compute, either directly or statistically, the cost of a specific injury based on the characteristics of the injured miner and his family. This approach was taken so that the cost indicators computed by ACIM could be used to compute relative economic impact by occupational and injury characteristics. Using an average insurance cost per injury does not permit this type of comparison.

1. Medical Treatment Cost includes the cost of medical resources required to treat the injury and repair the damage caused by the injury, the cost of in-patient hospital care associated with the treatment and repair procedures, and the cost of out-patient followup treatment. The approach selected for use in the model is a statistical one, but the basis of the distributions for medical costs is a set of injury and treatment scenarios developed by a physician. Injury scenarios were developed for each of 61 combinations of nature of injury and part of body injured, and a treatment scenario was developed for each injury scenario.

2. Compensation from Occupational Insurance includes wage compensation paid to a miner for periods of partial or total disability, benefits for loss of a body function as a result of the accident, and death benefits paid to the miner's surviving spouse and dependent children. The algorithms used to compute these benefits are based on state worker's compensation laws, and the benefits for a specific injury are computed using the schedule of benefits for the state in which the accident occurred.

. Compensation from Social Security Funds includes wage compensation for periods of total disability and death benefits to the widow and dependent children. The algorithms are based on the social security laws for disability and retirement benefits.

4. Compensation from Union Funds includes wage compensation for total disability, basic death benefits to the widow from the union retirement fund, and additional death benefits for accidental death. The algorithms are based on the current wage agreement between the United Mine workers of America and the Bituminous Coal Operators Association.

5. Administration and Overhead Expenses include the incremental costs to each of the above agencies for processing of claims for death and disability benefits.

6. Loss of Income to Mining Families includes the net expected loss of current and future income, after deduction of wage compensation and other benefits defined above, to the disabled miner or to the family survivors in the case of accidental death. The gross income is based upon the wage of the miner at the time of the accident as specified by the UMWA wage agreement in force at the time of the accident.

7. Cost of Fatal Accident Investigation includes the wages expended by MESA and state inspectors, representatives and employees of the mining company where the accident occurred, and union representatives. The algorithm is based on a se of frequency distributions developed with data obtained from MESA inspectors, and from a sample of 95 MESA reports of fatal accidents which occurred in 197 and 1974.

8. Losses in Coal Production as the result of an accident which causes a fatality or an amputation injury, including the loss due to the shutdown of the mine where a fatal accident occurs, and post-accident loss resulting from reduced efficiency on the section where the accident occurred. This algorithm is based on an analysis of the daily section reports at three mines which had experienced fatal or amputation injuries in 1974-1975. Results of this analysis are discussed later in the paper.

It should be noted that several cost elements are based upon the generation of a stochastic profile of the miner's family. This profile includes the life expectancies of the miner and his spouse, his marital status, and the characteristics of his wife and children. This profile is required for the computation of compensation benefits. Table 1 shows the family profile parameters generated for each CAIF injury accepted by the Accident Cost Indicator Model. The miner's age at the time of the accident is obtained from the CAIF data record. The remaining parameters are stochastically generated from statistical data contained in the model.

Each cost element is allocated to one of the three societal sectors shown in Table 2: mining company; mining family; or public agency. C_{med} and C_{stb} are costs to mining companies, as insurance premimums, even though actual payments of the costs may be administered by the state or by private insurance carriers. C_{ssb} is borne by the Social Security fund, which is entirely paid for by working persons and businesses in general.

Time Phasing of Accident Costs

Investigation brought out that most or all of the financial impact of certain cost elements occurred within a year of the occurrence of accidents which caused permanent disability or death. The value of lost coal production (amputation or fatal injury), the cost of investigating a fatal accident, and most or all of the cost of primary medical treatment are realized within a year of the accident. Certain lump sum benefits from indemnity funds can also be realized as first year costs, but are treated in the ACIM as incremental payouts.

Figure 2 illustrates the time phasing of the cost elements for a fatal injury accident. About 50 percent of the total cost of the accident is realized as a first year cost. The ACIM generates two cost indicators to emphasize the high fraction of first-year impact. One indicator is the total cost of the accident. The second indicator is the first-year cost (annual cost) defined as the sum of C_{med}, C_{ecp}, C_{inv}, the annual costs of compensation benefits, and the annual net loss of income.

Figure 3 and Table 3 show the estimates of total and annual cost for the fatal roof fall accident. For comparison, Figure 4 and Table 4 show the costs of an accident which resulted in permanent total disability. These two cost indicators are computed for each injury in each societal sector according to the allocations shown in Table 2.

Aggregation of Individual Accident Costs

The ACIM computes three indicators of accident cost for the total population of accident records extracted from the HSAC CAIF for each societal sector in Table 2 and for subsets of the population. Population subset S is defined by a combination of characteristics common to all accidents (e.g., all accidents involving roof bolters engaged in roof drilling when the accident occurred). The number of accidents in subset S is the same for each societal sector, but, as inferred in Table 2, the cost indicators are different for each sector.

The indicators are total cost, annual cost, and the annual cost per accident in subset S. The last indicator is a normalized measure of relative annual cost among different population subsets and/or different societal sectors.

To summarize, the three indicators of accident cost generated by the Accident Cost Indicator Model for an aggregated subset of accidents are
(1) Total present and future costs - the total expected economic impact of the accident subset;

(2) Present annual cost - the portion of total cost realized within the first year after the accident;

(3) Present annual cost per accident - the present annual cost divided by the number of accidents in the accident subset.

Post-Accident Production Loss in Fatal Roof Fall

The results from the analysis which underlies the development of the algorithm which estimates the value of post-accident production losses are a significant and startling indication of the long term disruption caused by a fatal accident. Many of the historical investigations alluded to at the beginning of this paper presented observational and anecdotal data to support the general hypothesis that a relationship exists between the severity of an industrial injury and the magnitude and duration of decreased productivity following the accident. In particular:

(1) a general relationship exists between injury severity (i.e., degree of damage to the injured) and the duration of work stoppage in the immediate area of the accident;

(2) major injuries, in which the injured person is severely damaged (e.g., amputation or disfigurement) and requires assistance by others to reach medical facilities, and fatal accidents cause a measurable reduction in productivity for some days after the accident.

When this hypothesis is viewed in terms of the work environment in an underground coal mine, one would expect to observe significant decreases in production following a major injury or fatal accident. Fifteen mines were visited to collect information from daily section reports to test this hypothesis by plotting daily production in sections where an accident occurred against the dates of the accident. In only two mines were the reports offered to support the project. One mine had experienced a roof fall fatality and both mines had experienced a number of lost-time injuries during the periods covered by the section reports.

Analysis of the section reports showed consistent losses in production for several days following the occurrence of (1) the fatal roof fall accident and (2) accidents which inflicted an immediate amputation injury. Inconclusive results were obtained for accidents which inflicted disabling injuries (non-amputation). Accidents which inflicted minor wounds (first-aid treatment or emergency room treatment with no ensuing disability) consistently caused no observable loss in post-accident production.

The daily section reports for three continuous mining sections at the mine where the fatal roof fall occurred were analyzed for a period of 39 workdays prior to the accident and 35 workdays following resumption of normal operations (after the fall was removed). These sections are described in Table 5.

Analysis of the reports was made to determine the pre- and post-accident means for productivity in terms of tons/operating minute and the number of miners in the face crew. The mean pre-accident values were computed, and linear regressions were made of the post-accident data. Table 6 shows the results of this analysis. Figur 5 shows a plot of productivity for the section-shift on which the accident occurred The post-accident trend in Figure 5 was observed on all other shifts in Sections A and B. The right-hand column of Table 6 is the number of working days at which the regression line of post-accident productivity intercepted the pre-accident mean. This table shows that a consistent loss of production was experienced on all shifts in the section where the accident occurred and in an adjacent section as well. Table 7 shows the average loss in productivity in Sections A and B and the normalized fractional loss in each section-shift.

It is important to note (1) that the productivities are expressed in tons/operating minute to eliminate non-productive time due to equipment malfunctions, place changes or other delays from productivity changes and (2) that crew sizes did not change significantly, which eliminates this factor as a possible reason for productivity changes.

When the magnitude and duration of the productivity losses shown in Table 6 was translated into tons of coal and lost sales, as shown in Table 8 for only Section A, then the economic impact becomes quite significant as a cost element. At a price of $15 per ton, this is a loss of $324,240 in gross sales within a calendar time of less than five months. When one observes that an adjacent working section also experienced significant losses and that four other working sections are located in the same area of the mine, the potential total loss in revenue and profit is staggering.

A regression analysis similar to that described above was performed for two injuries which resulted in amputations. Similar trends were observed and showed an average loss of 8 percent and a maximum loss of 15 percent in post-accident productivity over an average of 26 working days. This amounted to more than 1,000 tons of coal production lost.

These productivity losses from fatalities and dismemberment are direct reductions in gross profit because full wages and direct overhead costs are paid during the period of reduced efficiency. In a mine operating normally, these costs are relatively constant whether or not coal is produced.

<u>Selected Cost Indicators for 1974 Data</u>

The Accident Cost Indicator Model displays the cost indicators for each sector (mining companies, mining families, and public agencies) and for the total in a summary table for all injuries from the CAIF processed by ACIM, and in a series of tables for cross-sections and detailed summaries of the CAIF. In the cross-section tables are

mining activity at the time of accident by regular job title and
type of accident by degree of injury.

Included in the summary tables are

mining activities,
regular job title,
nature of injury,

underground mining method,
type of accident, and
degree of injury.

Each cost element record computed by ACIM contains a reference to the original record of the CAIF. This permits future capacity to display cost indicators for other accident parameters in the CAIF.

The ACIM was used to compute accident cost indicators for calendar year 1974 using copies of the Health and Safety Analysis Center's data bases.

A total of 9,286 accident records for underground bituminous coal mines were accepted by ACIM from the 1974 CAIF computations. The result of the program run was an estimated total of $56.9 million (1974 dollars) in current and future costs of these accidents. About 60% ($34.0 million) of this total was incurred in 1974 — the year in which the accidents occurred. The average annual cost of an accident was $3,700. About 41% ($23.6 million) of the total cost was incurred by mining companies as compensation payments, lost coal production, and investigative costs. Wage losses to the injured miner and his family accounted for 47% ($26.6 million) of the total. Compensation payments and investigative costs borne by public agencies accounted for the remaining 12% ($6.7 million).

A total of 89 fatalities in the 1974 CAIF accounted for almost 54% ($30.5 million) of the total cost; the average annual cost was $125,000 for a single fatality. A total of 5,325 disabling injuries (lost-time) accounted for 45% ($25.5 million) of the total cost, with an average annual cost of about $4,000 per injury. A total of 3,872 medical injuries (no lost-time) accounted for 2% ($0.95 million) of the total cost, with an average annual cost of about $250 per injury.

Mining task activities associated with roof support (roof scaling and bolting, jack and prop handling) were, as a group, the most costly, accounting for 22% ($12.4 million) of the total cost. Tramming/positioning of equipment accounted for 5% of the total. Handling supplies/materials, hauling coal (shuttle cars and tractors), and handling of wires and cables each accounted for 4% of the total cost.

Table 9 shows how the total cost of $30.5 million for the 89 fatal accidents was distributed among the societal sectors and types of accidents. As one would expect, roof falls accounted for the largest fraction (51%) of fatal accident costs. Machinery and haulage accidents each accounted for about 17% of this total. As we might also suspect, the distribution of total cost among the accident types is roughly the same as the distribution of the number of accidents within the type.

Mining companies and mining families bear nearly an equal economic impact from fatal accidents. Two points of significance should be noted here:

(1) the majority (80%) of the cost impact upon mining companies occurs from first-year (annual) costs through reduced production, medical treatment, and investigative cost;

(2) the economic impact of a fatal accident to a typical mining family is endured for many years after the accident at about the same annual level as the first year's cost.

The significance of the first point is that the majority of the first-year cost to mining companies is from reduced production efficiency, which is an uninsured and largely unrecoverable cost. On purely economic grounds, this point should provide ample reason for increasing company expenditures for accident prevention and safety improvement programs.

The second point is significant in that it dramatizes the compounding of personal and familial loss through continued economic hardship. Consider that the typical victim of a fatal underground mining accident is age 39, married with three dependent children less than 18, and earns about $9,000 per year (based on statistical data for 1973-1974 compiled for use in the ACIM). Statistics also tell us that the victim would have worked for about 26 more years. Since lost income comprises the total cost of $141,000 borne by mining families per fatality, the surviving family of the victim can expect a net loss of about $5,500 per year in potential income for as many years as the family remains nuclear. Therefore, it should be easy to vizualize and difficult to accept the economic hardship which compounds the shock, trauma, and disruption of the family unit struck by an accidental coal mine fatality.

In Conclusion

It is hoped that the cost indicators generated by the Accident Cost Indicator Model will serve to accelerate and expand the efforts of industry, government, and coal miners to work toward safer extraction of coal. Most projections of the nation's future energy requirements point toward coal as an increasingly important energy source. This means increased annual production and, with the continuance of present mining methods, increased exposure of coal miners to the hazards of underground coal mining. Perhaps the translation of the grim health and safety statistics of coal mine accidents into our society's prime yardstick — the dollar — will help us to achieve an energy future in which increasing utilization of our coal resources is achieved with decreasing waste of our human resources.

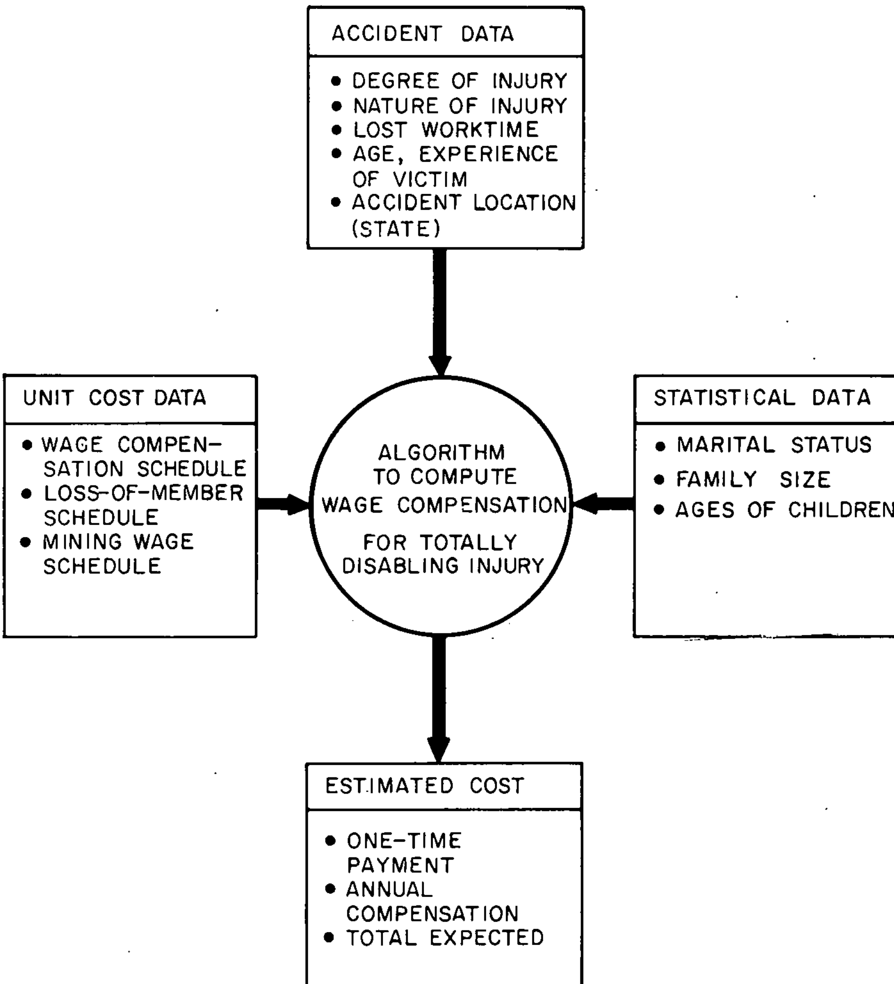

Figure 1 GENERAL STRUCTURE OF ACIM COST ALGORITHMS

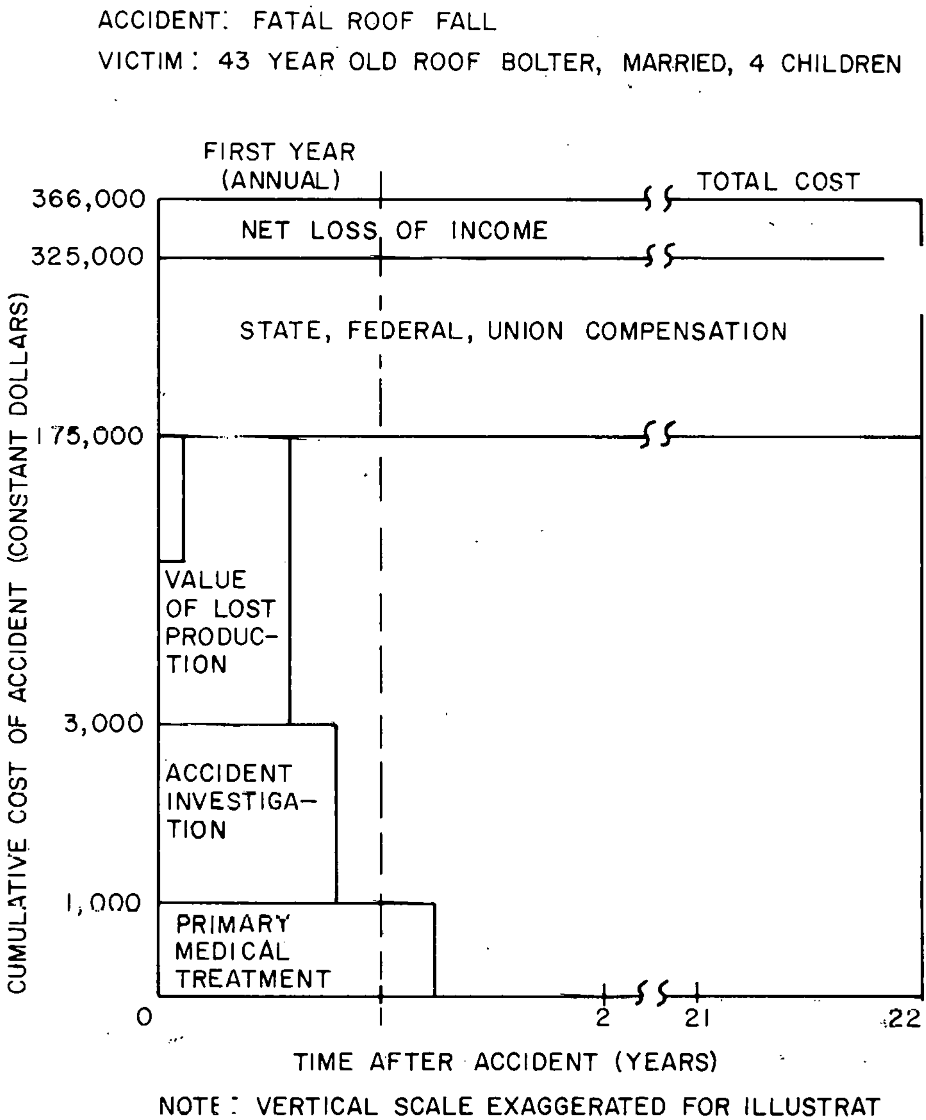

Figure 2 TIME PHASING OF FATAL ACCIDENT

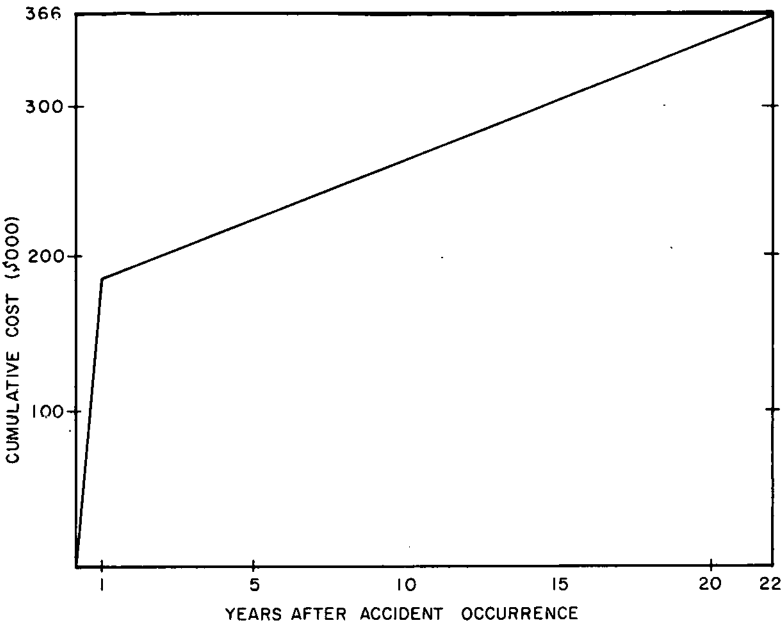

Figure 3 CUMULATIVE COST OF FATAL ACCIDENT

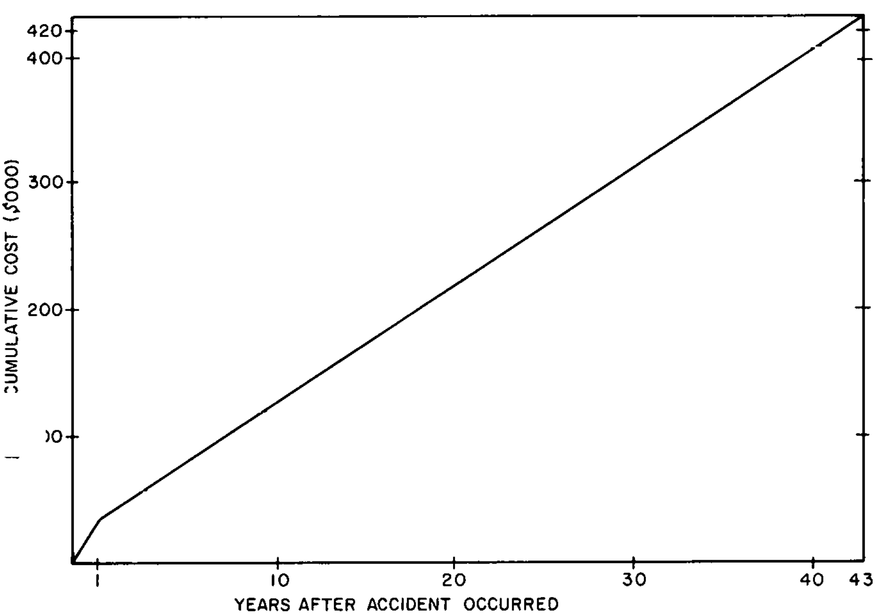

Figure 4 CUMULATIVE COST OF TOTAL DISABILITY INJURY

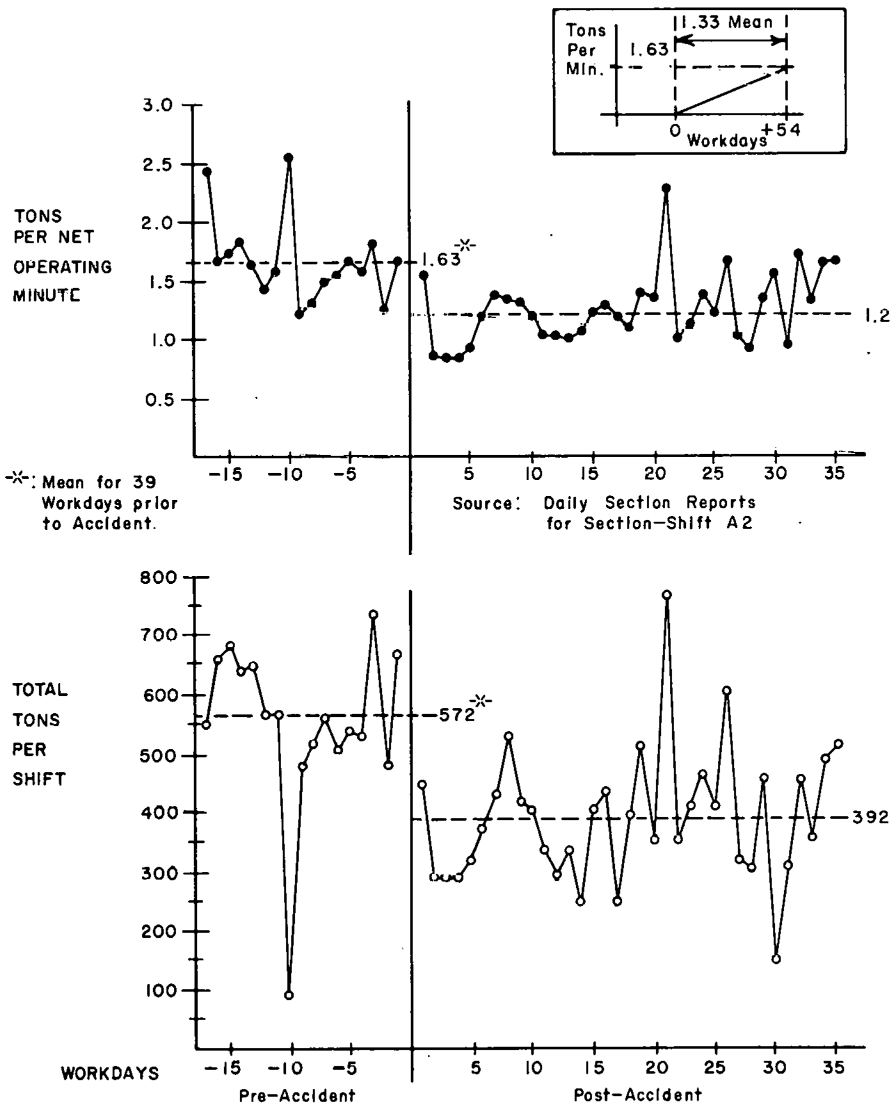

Figure 5 DAILY PRODUCTION AND OPERATING EFFICIENCY FOR FATAL ROOF FALL ACCIDENT — SECTION A, SHIFT 2

Table 1 STOCHASTIC FAMILY PROFILE PARAMETERS

PARAMETER	DEFINITION OF THE PARAMETER
S_m	MARITAL STATUS OF MINER (0 = UNMARRIED, 1 = MARRIED)
Y_w	AGE OF WIFE
N_c	NUMBER OF DEPENDENT CHILDREN LESS THAN 18 YEARS OF AGE IN MINER'S FAMILY
Y_{ci}	AGE OF A DEPENDENT CHILD
Y_{rw}	AGE AT WHICH THE WIDOW OF A MINER IS EXPECTED TO REMARRY
P_{65m}	PROBABILITY THAT A MALE OF AGE Y_m WILL SURVIVE TO 65 YEARS OF AGE
P_{65w}	PROBABILITY THAT A FEMALE OF AGE Y_w WILL SURVIVE TO 65 YEARS OF AGE
P_{rw}	PROBABILITY THAT A WIDOW OF AGE Y_w WILL REMARRY BY AGE Y_{rw}

Table 2 ALLOCATION OF ACCIDENT COST ELEMENTS

COST ELEMENT COMPUTED BY ACIM (C_x)		TOTAL COST ALLOCATED TO		
		COMPANY	FAMILY	PUBLIC AGENCY
MEDICAL TREATMENT	(C_{med})	✓		
LOSS IN COAL PRODUCTION	(C_{ecp})	✓		
FATAL ACCIDENT INVESTIGATION	(C_{inv})	✓		✓
TATE DISABILITY BENEFITS	(C_{stb})	✓		
FEDERAL DISABILITY BENEFITS	(C_{ssb})			✓
UNION DISABILITY BENEFITS	(C_{unb})	✓		
LOSS OF PERSONAL INCOME	(C_{pin})		✓	

Table 3 ELEMENTS OF FATAL ACCIDENT COST

ACCIDENT: ROOF FALL, FATAL
VICTIM: 43 YRS OLD, ROOF BOLTER, MARRIED, 4 CHILDREN, 25 YRS EXPR

COST ELEMENT	TOTAL COST (CONSTANT $)	TIME PERIOD (YEARS)	FIRST YR COST (ANNUAL COST)
PRIMARY MEDICAL TREATMENT	1,000	< 2	1,000
VALUE OF REDUCED PRODUCTION	172,000	< 1	172,000
INVESTIGATION	2,000	< 1	2,000
STATE COMPENSATION	28,900	22	1,315
FEDERAL COMPENSATION	92,200	22	4,190
UNION COMPENSATION	28,400	22	1,290
NET LOSS OF INCOME	41,100	22	1,870
TOTALS	365,600		183,665

Table 4 ELEMENTS OF TOTAL DISABILITY INJURY

INJURY: FOOT AMPUTATION
VICTIM: 22 YRS OLD, UTILITY MAN. MARRIED, 1 CHILD, 1 YR EXPERIENCE

COST ELEMENT	TOTAL COST (CONSTANT $)	TIME PERIOD (YEARS)	FIRST YR COST (ANNUAL COST)
PRIMARY MEDICAL TREATMENT	750	< 2	750
VALUE OF REDUCED PRODUCTION	8,500	< 1	8,500
STATE COMPENSATION	150,000	43	3,490
FEDERAL COMPENSATION	15,000	43	350
UNION COMPENSATION	39,000	43	910
NET LOSS OF INCOME	206,000	43	4,790
TOTALS	419,250		18,790

Table 5 LOCATION OF SECTION IN FATAL ROOF FALL ACCIDENT

SECTION	LOCATION IN MINE RELATIVE TO ROOF FALL AREA
A	ROOF FALL OCCURRED IN THIS SECTION
B	SAME PANEL IN ADJACENT WORKING PLACES
C	APPROXIMATELY 5 MILES FROM SECTION A

NOTES: 1. ALL CREWS WORKING IN SECTIONS A, B, AND C ENTERED THE MINE FROM THE SAME SHAFT

2. ACCIDENT OCCURRED ON SHIFT 2 (4:00 P.M. - 12:00 P.M.)

Table 6 PRE- AND POST-ACCIDENT PRODUCTIVITY FOR FATAL ROOF FALL

SECTION -SHIFT	PRE-ACCIDENT PRODUCTIVITY[1,2]	PRE-ACCIDENT CREW SIZE	POST-ACCIDENT PRODUCTIVITY[1,3]	POST-ACCIDENT CREW SIZE	DAYS TO REACH NORMAL[4]
A1	2.28	12.8	1.82 (20%)[5]	12.7 (1%)[5]	96
A2[6]	1.63	9.5	1.33 (19%)	9.1 (4%)	54
A3	2.01	10.5	1.65 (18%)	10.5 ( )	25
B1	2.85	12.5	2.41 (12%)	12.4 (1%)	111
B2	2.20	8.8	1.95 (11%)	8.8 ( )	54
C1	2.20	13.3	2.21 ( )	14.2 ( )	--
C2	1.82	11.1	1.82 ( )	11.3 ( )	--
C3	1.49	10.1	1.49 ( )	9.9 (2%)	--

NOTES:
1 TONS/OPERATING MINUTE
2 MEAN VALUE FOR 39 WORKDAYS PRIOR TO ACCIDENT
3 MEAN VALUE FOR POST-ACCIDENT PERIOD REQUIRED TO RETURN TO PRE-ACCIDENT MEAN
4 PERIOD CORRESPONDING TO POST-ACCIDENT MEAN PRODUCTIVITY
5 PERCENT LOSS FROM PRE-ACCIDENT MEAN
6 SECTION AND SHIFT OF THE ACCIDENT

Table 7 AVERAGE AND NORMALIZED POST-ACCIDENT PRODUCTION LOSSES

	LOSS IN POST-ACCIDENT PRODUCTIVITY		
SECTION-SHIFT	OBSERVED	DAYS	NORMALIZED[1]
A1	0.20	96	0.28
A2	0.19	54	0.15
A3	0.18	25	0.07
B1	0.12	111	0.20
B2	0.11	54	0.09
AVERAGE	0.15	68	0.15

NOTE: 1 NORMALIZED FRACTION = OBSERVED FRACTION x OBSERVED DAYS / 68

Table 8 POST-ACCIDENT PRODUCTION LOSS IN SECTION A

SECTION SHIFT	PRE-ACCIDENT MEAN TONS/SHIFT	% LOSS IN PRODUCTIVITY	DAYS	TONS OF COAL LOST
A1	680	20	96	13056
A2	572	19	54	5869
A3	598	18	25	2691

TOTAL LOSS IN POST-ACCIDENT PRODUCTION 21616

Table 9 COST IN[illegible]TORS FOR FATAL ACCIDENTS IN 1974

TYPE OF ACCIDENT	NUMBER	PERCENT OF TOTAL COST			COST/FATALITY ($000)	
		MINING COMPANIES	MINING FAMILIES	PUBLIC AGENCIES	TOTAL	FIRST YEAR
ROOF FALLS	45	39	44	17	346	116
MACHINERY	15	46	39	15	348	143
HAULAGE	15	42	41	17	324	123
ELECTRICITY	5	48	33	19	366	154
FALL OF FACE/RIB/SIDE	4	59	21	20	334	194
OTHER	5	27	44	29	332	69
ALL FATALITIES	89	42	41	17	343	125
TOTAL COST PER FATALITY		$143,000	$141,000	$59,000		
FIRST-YEAR COST PER FATALITY		$114,000	$6,000	$5,000		

The Use of Horizontal Roof Strain Indicators to Determine Changes in Top Conditions and the Effectiveness of Anchor Type Roof Bolts

Gerald B. Rupert, Chairman, Department of Geology & Geophysics
George B. Clark, Professor of Mining Engineering
Sunder Bajaj, Graduate Research Assistant

University of Missouri-Rolla
Rolla, Missouri
Colorado School of Mines
Golden, Colorado

Abstract

Pairs of four-foot resin and mechanical anchor roof bolts (without collars) were installed in the top along the intake side of an active coal mine. To each pair of the same type was affixed a horizontal roof strain indicator or HORSI and the instrument readings monitored for a period of approximately four months. These values when fitted to a linear regression model suggest that the HORSI's mounted on resin bolts more accurately monitor day to day changes in roof top conditions while those mounted on anchor types not only indicate trends but provide information concerning day to day movement of the bolts in the bolt holes. Thus by the subtraction of the two sets of values, changes in anchor effectiveness may be estimated. As a consequence, it is recommended that both types be used for HORSI installations when a bolting program dictates that anchor type bolts are to be utilized.

Introduction

There is a considerable, and justifiable concern within the coal industry as to the safe distance to which an operating surface mine may approach an underground mine before there is detectable damage found in the underground mine. For this reason, the University of Missouri-Rolla contracted with the Department of Interior's Bureau of Mines to investigate the criteria relating to the proximity of such operations. The effects of surface blasting are most likely to be detected in the structure of the underground mine, and as such the effects can be determined through monitoring of the behavior of the roof supports.

In the study which was carried out during this contract, two different sets of roof supports, resin anchored dowels and mechanically anchored roof bolts, were monitored for two purposes: firstly to determine if there had been any changes in the mine roof due to the blasting, and secondly should this prove to be the case to determine what changes in the mine support plan might be required to minimize or accomodate the effects measured.

Based on the results of the investigation carried out on the above cited contract data on the two bolting systems has been further analyzed and the conclusions are reported herein.

Test Site and Instrumentation

Thirteen monitoring stations were established along a 700 ft portion of the intake side of the Ferguson Mine in Nicholas County, West Virginia (Fig. 1). This punch mine is overlain by approximately 400 ft of Pennsylvania rock of Allegheny age. The coal is the Clarion and varies from 36 to 42 in. in thickness. The normal support program includes both mechanical anchor bolts, and timbers with headers are installed where needed. The mine lies adjacent to an operating strip mine which, during the course of the research program advanced towards the underground mine from a distance of 3/4 of a mile. In order to complete the data, University personnel included test firings at closer distances to the mine including the firing of several shots over the instrumented mine section.

Three-component seismometers were located on top, bottom, and ribs of the coal mine to record the motion of these surfaces during the course of the blast. Previous research (Stehlik, 1964) on the effects of blasting on anchors had utilized compression pads to monitor loads between the bolt plate and the top and these were incorporated into the research program to monitor bolt tension. Horizontal roof strain indicators (HORSI's) had been suggested by Panek in 1973 as being a means of comparing the behavior of the mechanical and the resin bolts and these were also, therefore, included in the instrumentation system.

Initially, only mechanically anchored bolts were installed in order to permit recovery of the pads after termination of the project. After installation of HORSI units, however, concern developed that the absence of collars or wedges and possible crushing of the rock by the bolt anchors could conceivably permit bolt wobble and slippage respectively and lead to erroneous measurements. Resin dowels were then installed within a foot distance from the mechanical bolts and additional HORSI's were affixed to these units; this gave the opportunity for the comparison of blasting damage on both types of support.

The total instrumentation used in the program is tabulated in Table I. During the course of the experimentation, particle velocity measurements made using the geophones indicate readings ranging from less than 0.01 in/sec to greater than 2.0 in/sec peak particle velocity, which latter figure is the level normally used to indicate the onset of damage to surface structures.

<u>Analysis</u>

Several correlation and analysis techniques have been utilized in the reduction of the data obtained in the program. In this paper, the results under discussion relate to the relative damage to the mine structure as indicated by the response of the anchoring systems.

Initially, a linear multiregression model was assumed of the form:

$$Y = B_0 + B_1X_1 + B_2X_2 + B_3X_3 \tag{1}$$

where Y is the dependent variable of horizontal roof strain or compression pad values and the independent variables X_1, X_2, and X_3 are temperature, humidity, and roof particle velocity, respectively. Computations were by means of the Statistical Analysis System (Barr, et al, 1976). Significant independent variables were determined by the forward selection procedure and Table II summarizes the resulting statistics. An R^2 value of 0.500 or greater is assumed to be statistically significant and indicative that the model is reasonable.

It is of interest to note that of the thirteen R^2 values greater than 0.500, nine are associated with mechanical anchors. Furthermore, the dominant coefficient of the compression pad responses is the constant term. Thus for small changes in load less than that detectable by the compression pads ($\pm$250 lb), it is suggested t mechanical anchor bolts are more sensitive to blasting than are the resin e. The data representative of each HORSI pair were then plotted for not only parative purposes but also as an aid for further analysis. These are as in Fig. 2 through 16.

It was assumed that resin anchored dowels would more accurately reflect changes in the rock mass of the mine top, whereas the anchor bolts would reflect transient movement such as that experienced by a seismic disturbance. The differences ΔT and ΔL between the transverse and longitudinal HORSI values were calculated, the resultants plotted, and the regression models assumed:

$$Y = B_0 + B_1X_1 + B_2X_1^2 + B_3X_2 + B_4X_3$$

For the preceeding, Y is the dependent variable ΔT or ΔL and X_1, X_2, and X_3 are top particle velocity, temperature, and humidity, respectively.

Of the stations for which data was analyzed, the R^2 values of only three imply mechanical bolt movement is a function of velocity (Table III). However, this represents a total of six measurements. Furthermore, the R^2 values are greater than 0.500 and the inclusions of temperature and humitity increases R^2 to greater than 0.700.

Discussion and Conclusions

Within the range of experiments covered in this investigation, the majority of the roof peak particle velocities lay below 1 in/sec and as a result, the change in the bolt and the roof conditions was anticipated to be small. In fac the size of the movement and change in the roof conditions was smaller than had been anticipated such that the changes in the readings of the instrumented bolts was of the order of accuracy of the dial measuring gage which was used to determine the changes. While this indicates that the roof, floor and sides of the mine were stable under the imposed loading, it did make a correlation of the effects of blasting on the support systems more difficult.

It is interesting to note that the best correlation can be obtained between the difference the bolt behaviors rather than the absolute movement of each system. This is because of the difference between the anchorage and response of the two systems. The resin system is attached along its length to the roof rock, and, as such, it will reflect the response of the rock to the blasting, conversely, the mechanically anchored bolt is attached only at the plate and the anchor to the roof and thus the response of the bolt will indicate not only the behavior of the roof but also any displacement of the anchor. The data and correlation therefore, suggest that the most accurate indicator of the effect of blasting on the mine support system is through the change in the anchoring characteristics of the mechanically anchored bolts, rather than in the response of the roof itself. This is substantiated by the presence of temperature and humidity as independent variables. The aforementioned is not unexpected as moisture laden air of varying temperature is free to circulate around mechanical anchor bolts whereby its flor is obstructued around untensioned bolts by virtue of their grouted nature. In conclusion, for purposes of monitoring the effects of blasting upon mechanical anchor bolts; resin bolts should also be installed, HORSI's affixed to a pair of each type and the associated measurements subtracted. The difference then reflects only bolt movement and does not include that of the surrounding rock mass.

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List of Figures

TABLE I. INSTRUMENTS AT EACH STATION

Station Number	Top	Geophone Bottom	Rib	Pad No.	2C	4C	TTH No.	TRH No.	LTH No.	LRH No.	WMS
1	X	X	X	-	X	X	3	4	5	6	X
1a	-	-	-	27	X	X	8	9	-	-	
2	-	-	-	16	X	X	10	11	12	13	X
3	X	X	X	51	X	X	14	15	-	-	
4	X	-	-	19	X	X	16	17	-	-	X
5a	-	-	-	-	-	-	18	19	20	21	
5	X	X	X	14	X	X	22	23	-	-	X
6	-	-	-	15	X	X	24	25	-	-	X
7	X	X	X	17	X	X	26	27	-	-	X
8	X	-	-	8	X	X	-	-	-	-	
9	X	X	-	50	X	X	28	29	-	-	
10	X	X	X	20	-	-	30	31	-	-	X
11					X	X	32	33	-	-	

2C -- Two foot roof bolt and floor hook for convergence determination.
4C -- Four foot roof bolt and floor hook for convergence determination.
TTH -- Transverse oriented HORSI mounted on torqued roof bolts.
TRH -- Transverse oriented HORSI mounted on resin roof bolts.
LTH -- Longitudinal oriented HORSI mounted on torqued roof bolts.
LRH -- Longitudinal oriented HORSI mounted on resin roof bolts.
X -- Present.
- --Absent or non-functional.

TABLE II. MULTIPLE LINEAR REGRESSION COEFFICIENTS

Station	Response	B_0	B_1	B_2	B_3	R^2 Percent
1	LRH	0.02963	0.00032	-	-	33.63
1	LTH	0.01906	0.00081		0.00184	70.53
2	LRH	0.04519	-	-	0.00280	27.48
2	LTH	0.04277	-	-	0.00585	16.12
1	TRH	0.00831	-	0.00021	0.00185	65.83
1	TTH	0.07851	0.00044	-	0.00165	79.72
1A	TRH	0.03844	-	0.00020	-	17.15
1A	TTH	0.04585	-	0.00021	0.00225	58.05
2	TRH	0.05836	0.00060	-	-	30.43
2	TTH	0.04947	-	-	0.00495	77.44
3	TRH	0.04593	-	-	0.00489	59.38
3	TTH	0.00795	-	0.00009	0.00192	82.87
4	TRH	0.00988	-	-	0.00995	69.95
4	TTH	0.01295	-	-	0.00226	42.56
5A	TRH	0.03640	-	-0.00021	-	24.28
5A	TTH	0.02691	0.00024	-	-	51.49
5	TRH	0.06927	0.00050	-		50.21
5	TTH	-	-	-	-	-
6	TRH	0.07184	-	-	-0.00166	12.13
6	TTH	0.03591	-0.00015	-	-	76.38
7	TRH	-	-	-	-	-
7	TTH	0.05617	0.00028	-	-	64.09
9	TRH	0.05193	0.00035	-	-	3.49
9	TTH	0.02423	-	0.00032	-	4.53
10	TRH	-0.01315	-0.00141	0.00057	-	16.20
10	TTH	-	-	-	-	-
11	TRH	0.01998	-	0.00014	-	28.23
11	TTH	0.05694	-	-	0.01466	76.30
1A	PAD	2.97736	0.02311	-	-	25.31
2	PAD	3.59422	0.03047	-	-	21.51
3	PAD	5.00395	0.03031	-	0.15851	78.62
4	PAD	4.02682	0.04774	-	-	46.70
5	PAD	-	-	-	-	-
6	PAD	5.74271	0.03334	-0.01222	-	81.00
7	PAD	5.25613	-	-0.00517	-	12.40
8	PAD	5.51615	0.01478	0.00555	-	58.88
9	PAD	-	-	-	-	-
10	PAD	3.97641	0.01542	-	-	15.14
11	PAD	3.65310	-	-	-1.64729	36.99

TABLE III. MULTIPLE REGRESSION COEFFICIENTS

Station	Response	B_0	B_1	B_2	B_3	B_4	R^2 Percent
1	DT	0.05845	-0.00243	-	-	-	0.8
5	DT	-0.06636	-0.04807	0.08213	-	-	0.71055
2	DL	-0.00504	0.040499	-0.02686	-	-	0.54197
1	DT	0.05845	-0.00243	-	-	-	0.85333
5	DT	-0.06599	-0.04608	0.079948	-0.001	-	0.72561
2	DL	-0.04046	0.049667	-0.035152	-	0.000502	0.725835

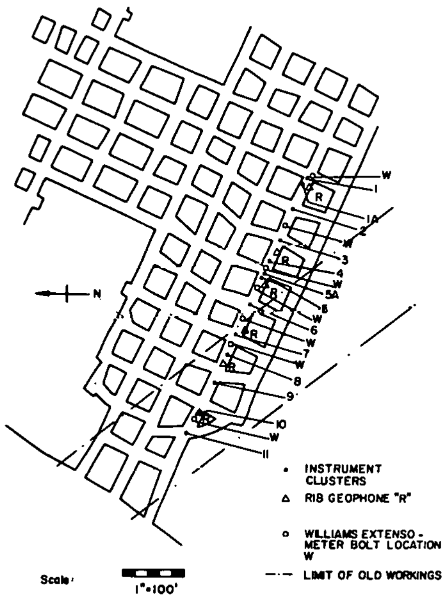


Figure 1

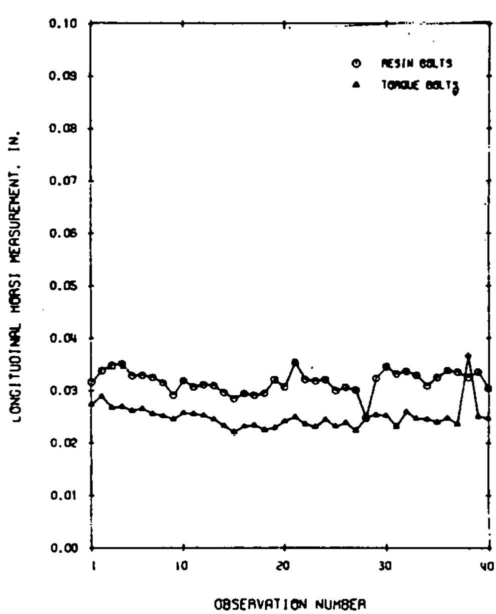


Figure 2

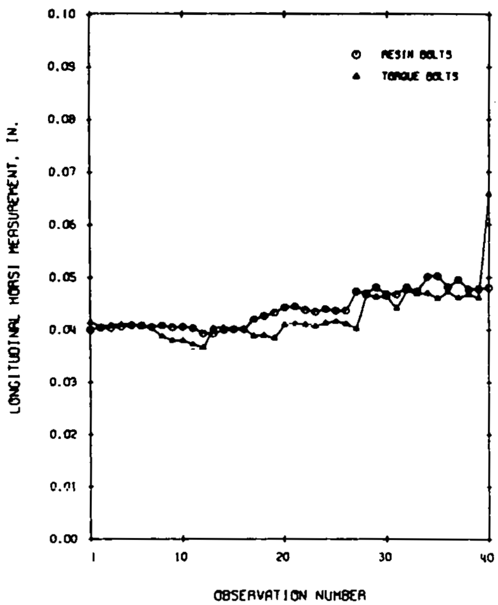
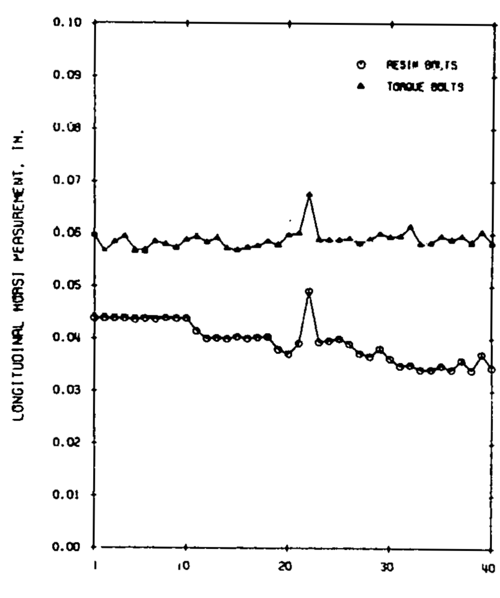


Figure 3



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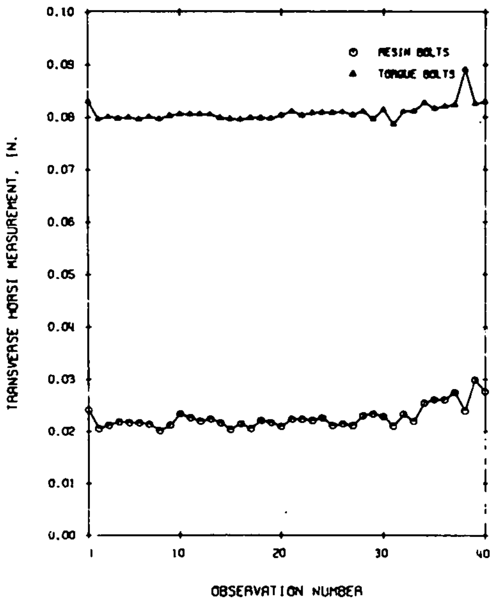


Figure 5

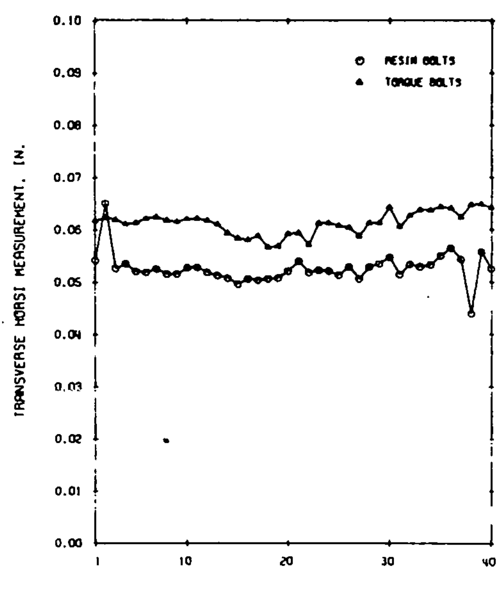


Figure 6

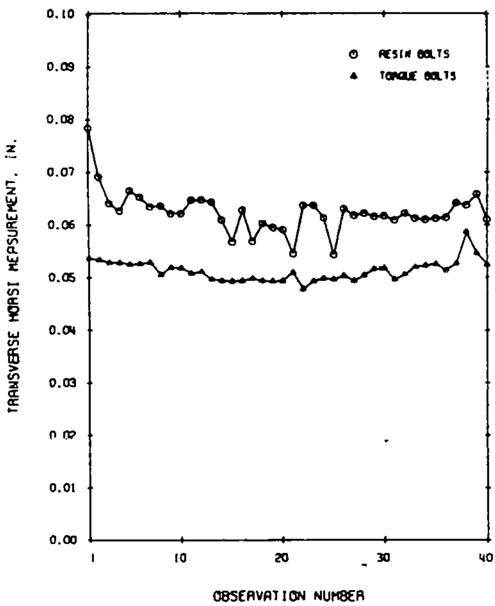
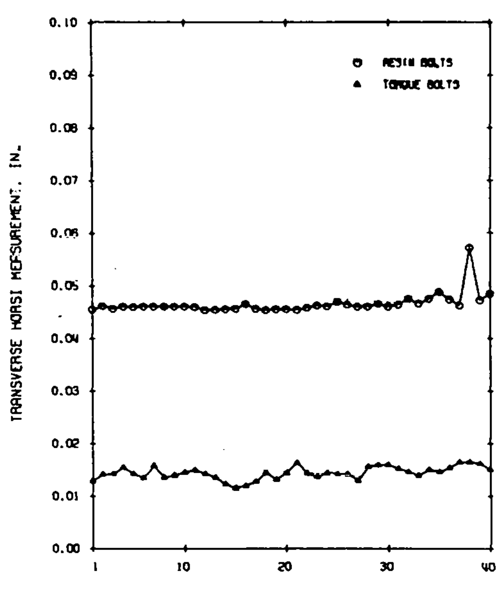


Figure 7



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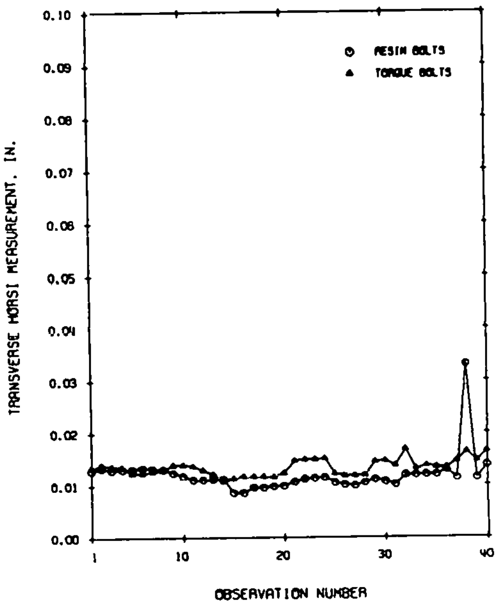
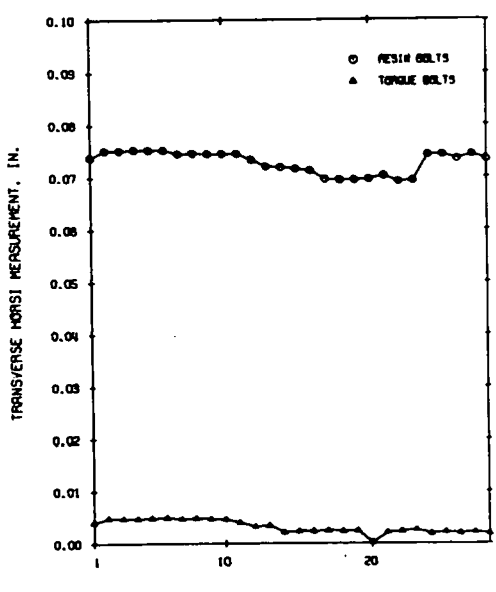


Figure 9

Figure 10



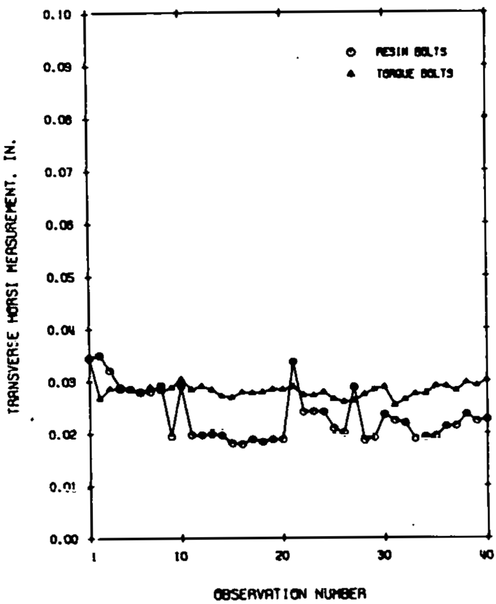


Figure 11

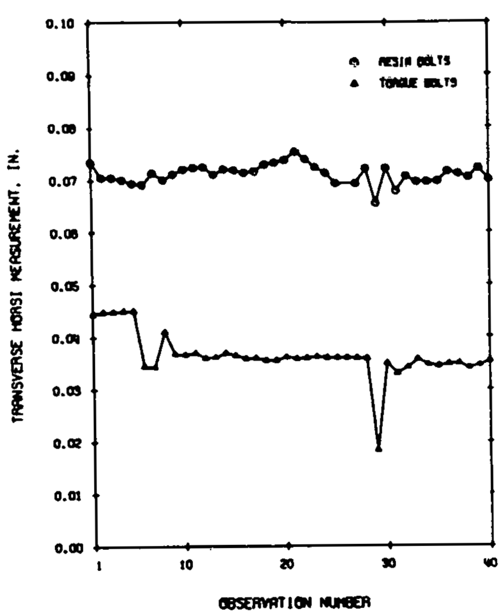


Figure 12

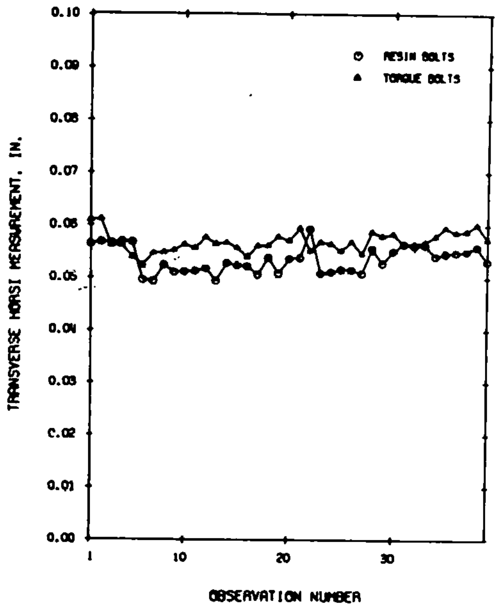


Figure 13

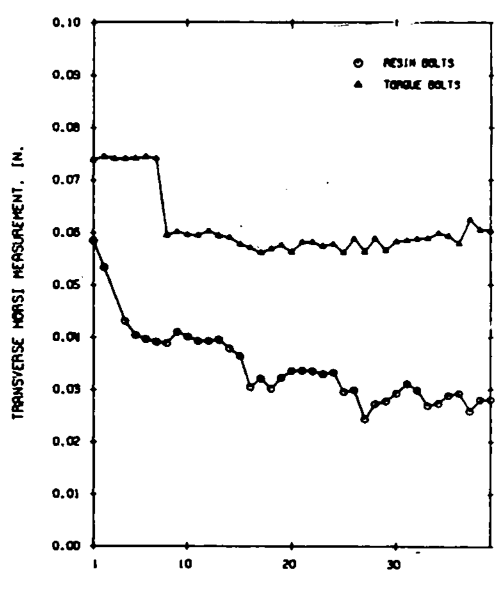


Figure 14

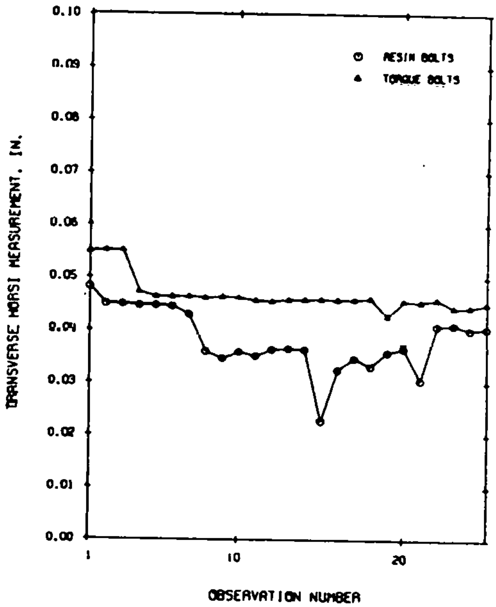


Figure 15

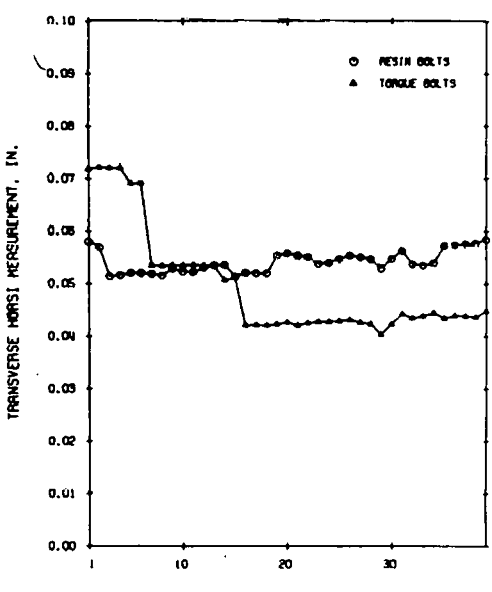


Figure 16

IMPACT OF DEEPER DEPTHS OF CUT ON COAL WINNING MACHINES AND RESPIRABLE DUST

This paper is unofficial and has not been completely reviewed by the Bureau of Mines.

Sigmund Black
Corporate Vice President of Engineering

Fairchild Incorporated
Beckley, West Virginia

Bradley V. Johnson
Physicist
and
Robert L. Schmidt
Underground Mining Research Supervisor

U.S. Bureau of Mines
Twin Cities, Minnesota

Dr. Bani Banerjee
Assistant Director of Research

Ingersoll Rand Research
Princeton, New Jersey

This paper presents (a) the final results of a Bureau of Mines sponsored test program on the effects of both deep cutting and bit speed on respirable dust and specific energy and (b) the impact these results will have on future coal mining machine designs. The preliminary results of these tests have been presented at a Bureau of Mines Technology Transfer Meeting held in September 1976.[1,2*]

BACKGROUND

Ingersoll-Rand Research, Inc. was granted Contract No. HO122039 to initiate the largest, full-scale underground test for measuring respirable dust ever undertaken. Using a modified drum type continuous mining machine (a Lee-Norse 455, "Hardhead") with adjustable cutting features and equipped with precision measuring devices, a controlled set of experiments, varying depth of cut and rotational speed, were conducted in the Pittsburgh Seam with the cooperation of EACC at their Joanne Mine in Rachel, West Virginia.

The HH456 test machine, shown in Figure 1, has a 44-1/2 inch bit tip to tip diameter, high pull-out torque capacity, semi-automatic controls and the following special features:

A. Matched sets of removable gear boxes permit changes in drum rotation speeds of 51, 18 and 9 RPM (600, 200 and 100 ft/min bit tip speed).

B. Sump and shear jacks for a positive control of drum horizontal (sump) and vertical (shear) velocities.

C. Extra long plumb bob style picks with contoured bit blocks to allow up to 5 inches maximum depth of cut per revolution.

D. A belt gathering head was used to keep up with potential high instantaneous sump production rates at 51 RPM and 4 inch depth of cut.

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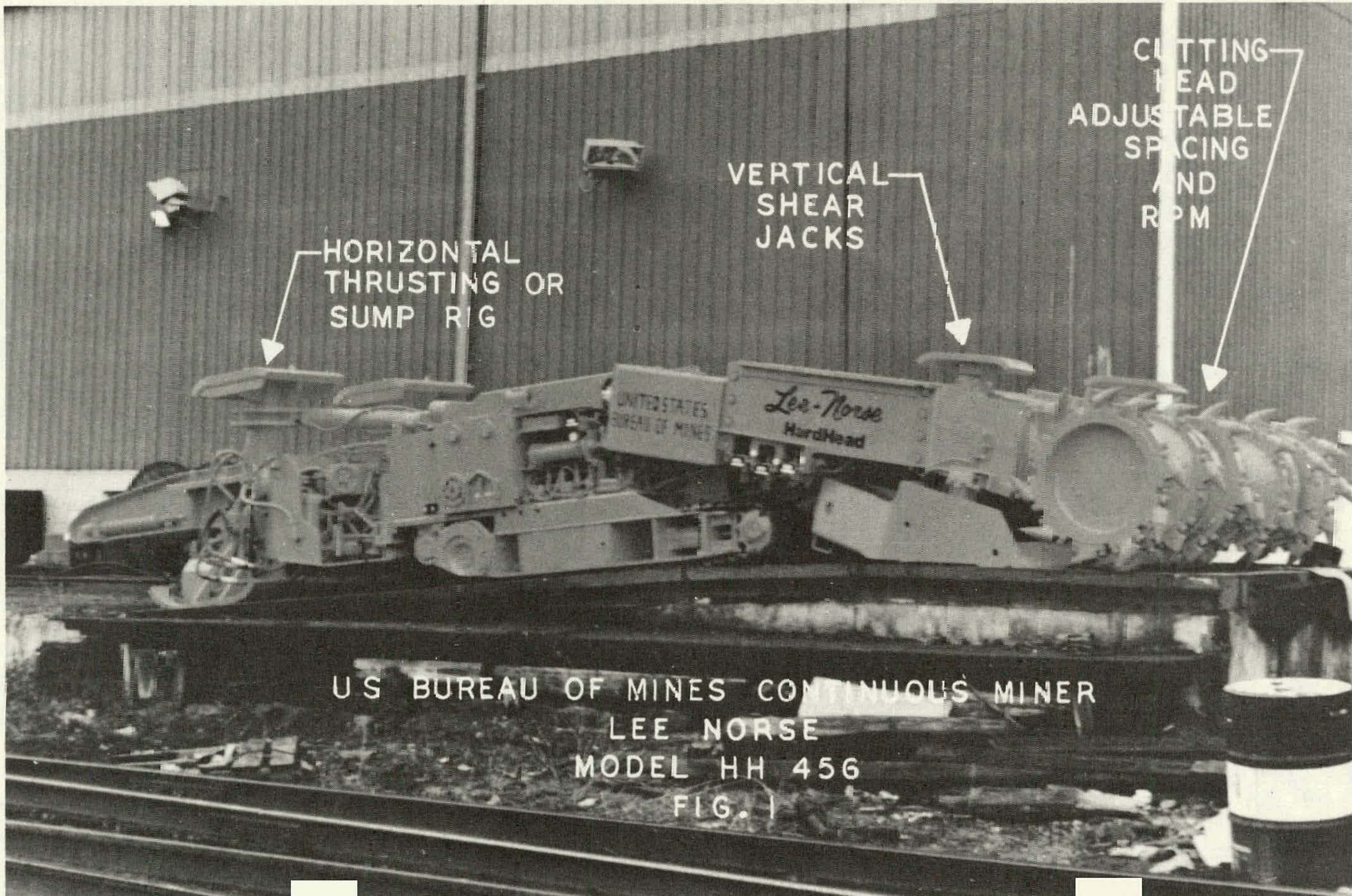
*All references listed at the end.



US BUREAU OF MINES CONTINUOUS MINER
LEE NORSE
MODEL HH 456
FIG. 1

Conventional underground coal mining machines operate at a cutter bit speed of 600 ft/min and shallow, 1-inch depth of cut. Thus, the test machine performance at 51 RPM and 1-inch bit penetration approximates the standard industry practice.

Figure 2 shows the position of the miner and the dust monitoring stations in an underground test set-up. The airborne dust generated was measured at S1 in front of the operator's cab, S2 close to the face, and S3 and S4 behind the brattice in the return air. Each test consisted of two automated sump and shear cycles covering a 3-foot advance and producing about 6-1/2 tons of coal. Air flow was maintained at 4000 CFM (±5%) and measured by a hand held anemometer.

The respirable dust at each station was collected by an air sampling pump[3] with a midget impinger in an alcohol solution to determine particle size distribution. The water sprays (boom sprays) facing the cutter were turned off during each test but were maintained at the gathering head and conveyor to minimize secondary dust generation.

Additional information recorded was the rotational speed, sump and shear velocity, power consumed and noise level.

The range of parameters tested was as follows:

Rotation speed	9, 18 and 51 RPM
Depth of Cut	1, 2 and 3.5 inches

The bit spacing was 4 inches and the bit attack angle was 45°, and each test combination was replicated 4 times.

RESULTS

Each of the Figures 3, 4 and 5 contain two types of data plots: the "Actual Averaged Data" and the "Statistical Results." The former represents the log mean averages* of the data gathered for each of the replications. The Statistical Results represent a regrouping of data after statistically testing for levels of significance by combining values that did not differ significantly and were the basis of numerical calculations for expressing final results. The purpose was to separate that data for which there were no detectable significant differences from that which shows valid significant statistical differences.

A. Airborne Respirable Dust**

The results presented in Figures 3 and 4 are for the air return (S3/S4) and operator's position (S1).

Respirable dust levels were reduced (Figure 3) by 63.3% (in the air return) by increasing the depth of cut from 1 inch to 2 inches at 51 RPM.*** It is interesting to note that at the standard 1-inch depth of cut, the lower rotational speeds had a slight effect on dust of 26.6%, but it was not significant at greater depths of cut. On the other hand, increasing the depth of cut from 2 to 3.5 inches did not affect the overall airborne respirable dust reduction in the air return.

---

* Expressed in antilogs for convenience.

** Relative quantities of dust are expressed in terms of milligrams per ton (mg/ton) of coal produced and are used to avoid problems of variations in section air flow and coal produced in different mines. The results could be equally expressed in terms of mg/M^3 for any known combinations of production and air flow.

*** Note that at 9 and 18 RPM the change was 53%.

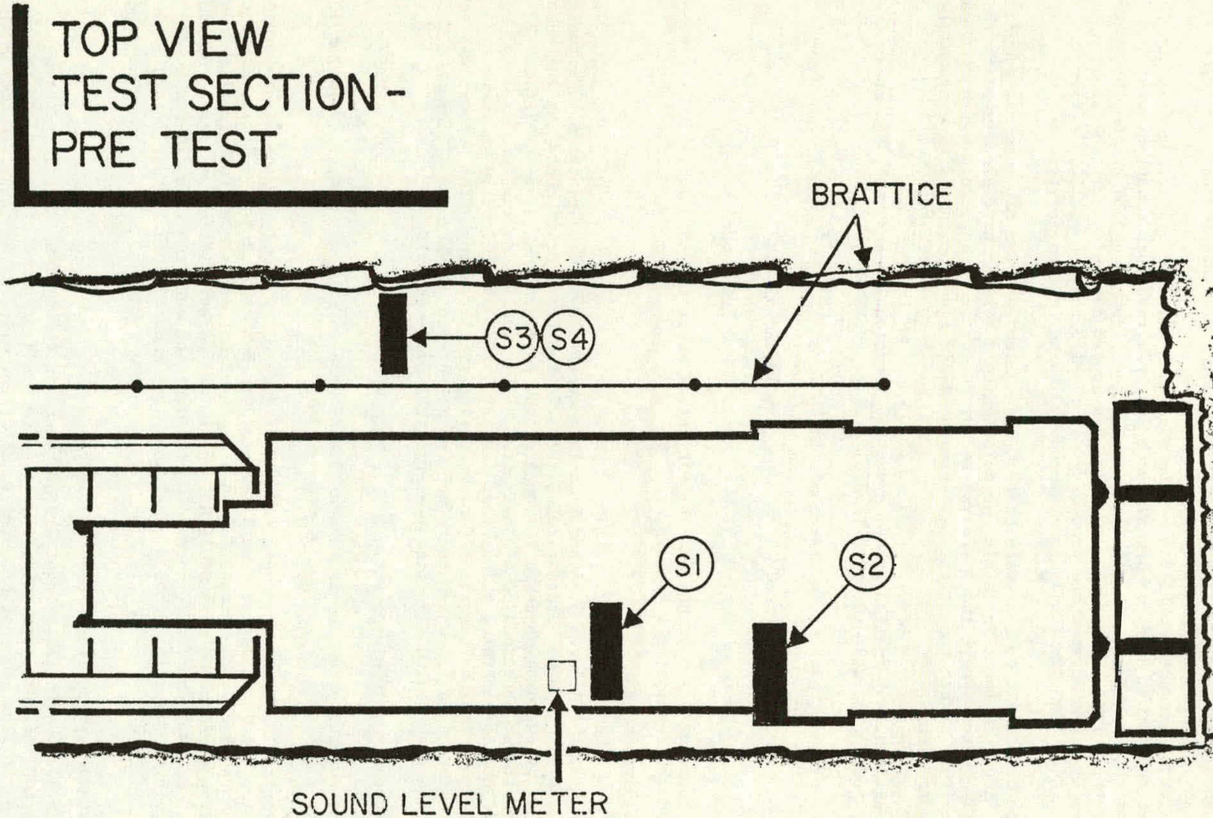

TOP VIEW
TEST SECTION -
PRE TEST
BRATTICE
S3
S4
S1
S2
SOUND LEVEL METER

FIGURE 2

DUST VS. S_M^*

ANALYSIS OF RESPIRABLE DUST DATA IN RETURN AIR (LOCATION 3-4)

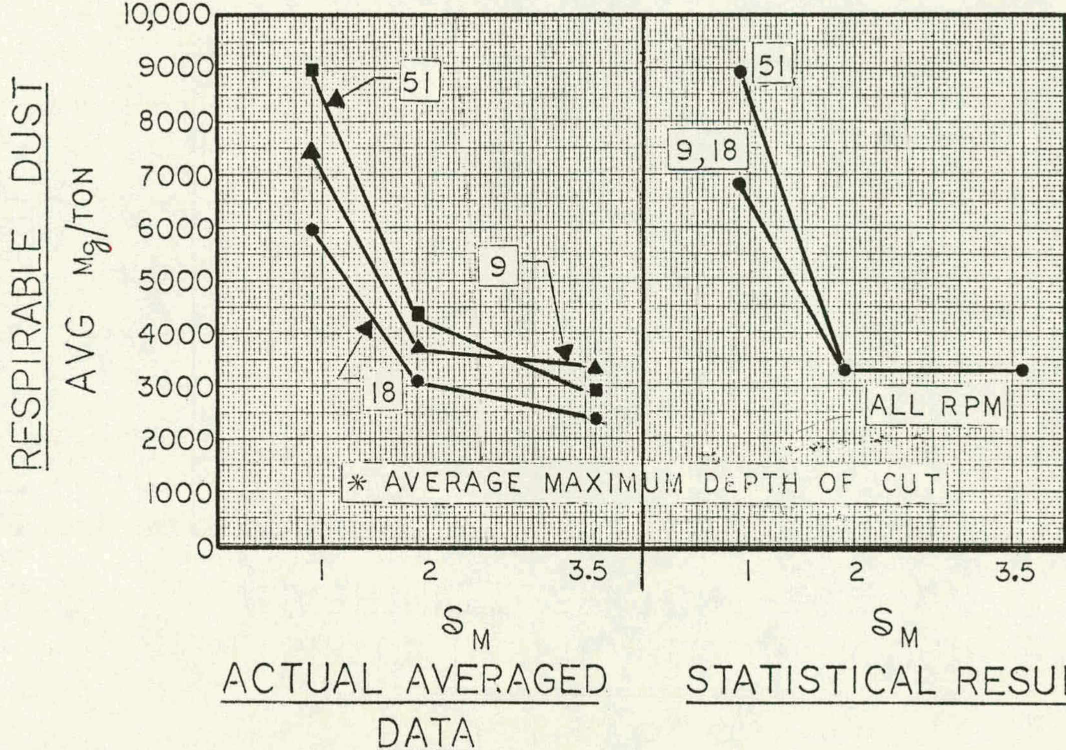


FIGURE 3

DUST VS. S_M^*

ANALYSIS OF RESPIRABLE DUST DATA
(MACHINE OPERATOR — LOC I)

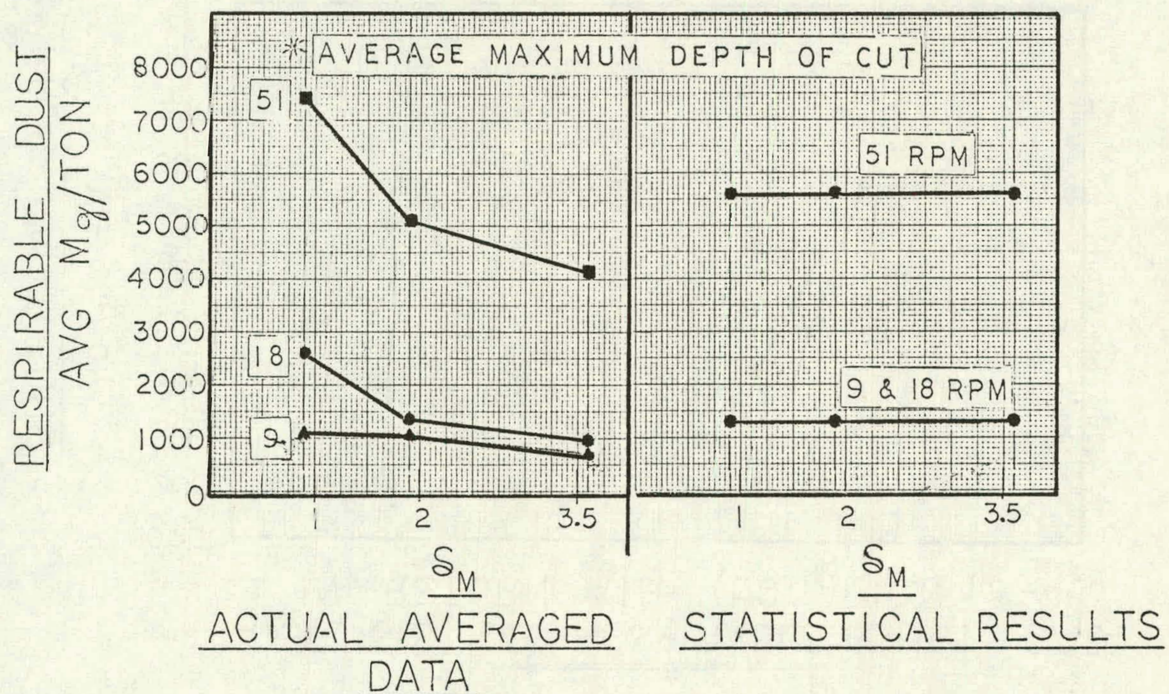


FIGURE 4

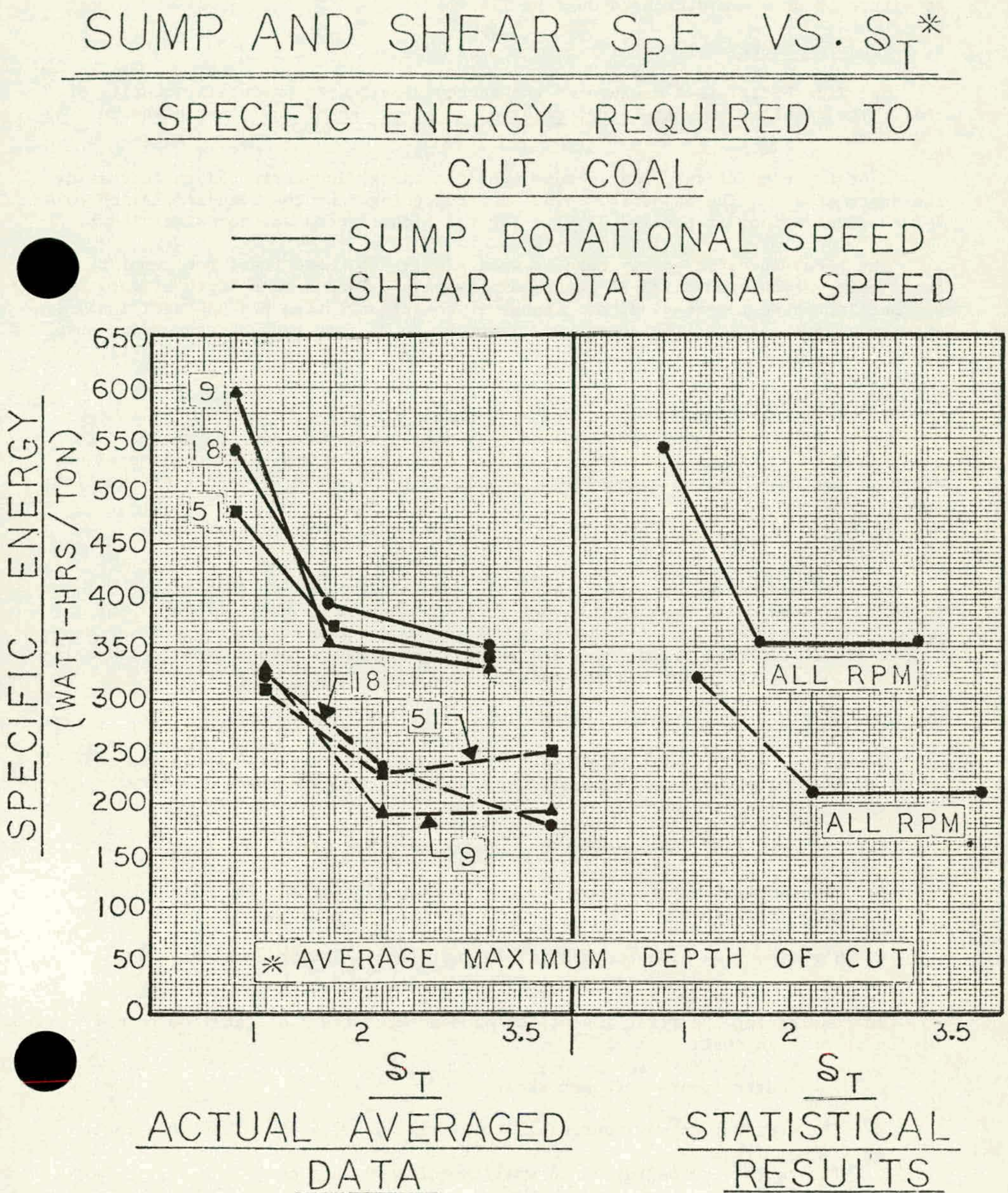

SUMP AND SHEAR SpE VS. ST*
SPECIFIC ENERGY REQUIRED TO CUT COAL
——— SUMP ROTATIONAL SPEED
- - - - SHEAR ROTATIONAL SPEED
SPECIFIC ENERGY (WATT-HRS / TON)
650
600
550
500
450
400
350
300
250
200
150
100
50
0
9
18
51
18
51
9
ALL RPM
ALL RPM
* AVERAGE MAXIMUM DEPTH OF CUT
1
2
3.5
1
2
3.5
ST
ST
ACTUAL AVERAGED DATA
STATISTICAL RESULTS

FIGURE 5

At the operator's station, S1 (Figure 4), the cutter head's rotational speed, not the depth of cut, was the highly significant variable. The change from 51 RPM to either 18 or 9 RPM decreased dust by 77%.*

B. Specific Energy

Specific Energy is a measure of the energy required to produce a quantity of coal. The results, in watt-hours, required to cut a ton of coal, are given in Figure 5.

A highly significant drop in sump specific energy is attributed to increasing the depth of cut. The major reduction occurred going from the standard 1-inch to a 2-inch penetration; it equaled 33.8%. The rotational speed was not significant.

The shear specific energy changes were similar to those found for sump; the total energy reduction of 33.9% occurred between a 1- and 3-inch depth of cut. Neither increased penetration, nor change in rotational speed had any additional effect. The shear specific energy was lower by 40.2% than each corresponding sump value.

C. Coal Sizing

From each test a 300- to 500-pound coal sample was screened and analyzed to determine the relationship between coal size distribution and rotational speed or depth of cut; however, no significant relationship was found at 4 inch bit spacing and 45° bit angle.

D. Noise

A-scale decibel readings for each sump and shear showed no significant relationship between noise level and either rotational speed or depth of cut. The shear noise level was 3 db-A lower in sound intensity than the sump and shear cutting cycles. Clearly, noise level and machine performance have a complicated relationship and no conclusion was reached.

JOANNE RESPIRABLE DUST RESULTS VERSUS OTHER SITUATIONS ARE PROMISING

Based on these dramatic results, it is recommended that mine operators consider adopting low rotational speed, deep cutting machines. There are three aspects to this conversion:

A. Projection of the Joanne Mine results to other mines.

B. The machine modifications needed for deep cutting and low rotational speed.

C. The effect on airborne respirable dust levels and production rates.

In considering the first aspect, the Joanne Mine tests differ from normal mining methods in that:

A. Face water sprays were not used.

B. The ventilation was controlled to within ±5% of 4,000 CFM to the face.

C. The tests were performed in small modules on butt coal.

D. The coal was a dry Pittsburgh seam.

---

* Note that experimental error at the operator's position was considerably larger than at air return.

Since the thrust of the program was to provide basic data on the relationship of airborne respirable dust generation to fracturing method and to prove the value of the deep cutting principle, the test conditions were deliberately made unusually severe. By dispensing with the water sprays, curtailing ventilation to a safe but marginal level and working a dry seam, the normal dust suppressive advantages associated with cutting were minimized.

According to water spray data collected by the U.S. Bureau of Mines,[4] a standard drum type continuous miner (operating at 600 ft/min at a 1-inch depth of cut) reduces airborne respirable dust by 20% to 60%, when using water sprays. This discussion assumes as a first order approximation that face water sprays are as effective on low-count dust-laden air as on high-count dust-laden air. A 30% reduction is considered typical of current practice[4] and is the base value used for comparison in this report. If deep cutting at low rotational speed is also used, an additional 63% reduction in the air return can be realized in the return. Similarly, the machine operator would experience an additional 77% reduction. These airborne respirable dust level reductions presume the comparison of mining conditions in which no changes occurred in production per shift or air flow to the face.

In general, most mines do not maintain ventilation practices as strict as those for the Joanne tests. Since airborne respirable dust concentration is dependent on airflow, it is difficult to make an accurate projection of these tests to other mining situations. This is particularly true at the operator's position which is very susceptible to the placement of air ducts (tubing, brattice, etc.). Regardless of the ventilation system used, less airborne respirable dust will be generated at the face. If the ventilation method used is consistent, this will result in a 63% improvement in the airborne dust level for the air return. Thus, similar percent reductions can most probably be achieved at other mines.

It is generally accepted that the airborne respirable dust generated is a function of the coal seam and direction of mining with respect to the cleats. This test program was limited to the one mine site and small test modules in what was believed to be a mining seam orientation that produced the most dust. In fact, the preliminary tests indicated that cutting face coal was 30% lower in airborne respirable dust generation than butt coal. The benefits of deep cutting should be applicable to any orientation of mining.

The test modules were sized to produce over six tons per test and closely approximate typical mine operations for a box cutting machine. The mechanics of coal fracturing and airborne respirable dust generation should not change whether a box cutting machine or full-face machine is used.

The projection of these results can logically be extended to other mining situations by considering possible principles of deep cutting.

A. Deep cutting produces less fines, hence lower airborne respirable dust.

B. Deep cutting reduces the total length of contact time between the tip of the picks and the coal compared to shallow cutting, since the airborne respirable dust may be generated by the crushing of the coal under the tip of the pick. Deep cutting reduces the length contact and hence the amount of airborne respirable dust.

There are pro and con viewpoints for both of these concepts; either or both may be correct. In any event, the operation of this principle should be applicable to other seams. The results may vary; and the percent reductions could either be higher or lower, but a significant reduction should be evident.

It is firmly believed that the low rotational speed, deep cutting machine will be significantly superior to existing continuous miners in all underground operations.

The Bureau of Mines recognized that the test conditions and controls exercised to obtain the above results were ideal and sponsored a full-scale production test of the comparison of high RPM - shallow depth of cut (51-1) to low RPM - deep depth of cut (18-3). These tests are in progress at the Joanne Mine using ventilation tubing. The preliminary results confirm the dramatic reductions in respirable dust using 18 RPM - 3-inch depth of cut without loss of production per shift. A separate report will be issued on these results at the conclusion of the work.

PRESENT DAY MACHINE MODIFICATIONS ARE UNLIKELY

The possibility of machine modification is of critical concern to the miner operator who is considering adopting deep cutting and low rotational speed. Unfortunately, there are numerous problems in retrofitting present machines.

A major problem is to provide sufficient thrust for sumping. Whereas the HH456 used an auxiliary thrusting rig, it is not practical for production use. The machine design solution may lie in:

A. A new auxiliary thrusting rig of practical design.

B. Utilizing the weight of the machine to provide the thrust. (The HH456 has proven itself capable of cutting the butt face in sump at up to 3-inch depth of cut without a sumping rig; this depth is more than sufficient to achieve the major deep cutting dust benefits.)

Another possible solution would be to use a different type of bit. One such bit is a chisel bit with a 15° positive rake angle. When tested[5] in chalk*, cutting this bit required less normal force for cutting than the plumb bob bits. Other work[6] on point attack bits cutting coal has also become available. With this information, a lower thrust might be similarly needed for coal cutting in sump, and lighter weight machines could possibly sump to deeper depths.

A second problem is the need for a new boom, cutter head and drive train. The deep cutting miner needs a boom with enough clearance for the longer bits and a high torque head. It is possible that many machines could use their same motor drives but with new gear transmissions, head booms and cutting heads to both provide and withstand the higher torque loads.

Any of the above are difficult and in all probability will offset the balance of the machine. Therefore, it is recommended that the mine operators consider replacement machines equipped for deep cutting rather than retrofitting. A few manufacturers have indicated an interest in providing such machines.

IMPACT ON PRODUCTION

Figure 6 shows multiple plots of short-term production rate in a full-box cut as a function of rotational speed and depth of cut. Horsepower requirements are superimposed on the data based on the results of the HH456 performance during this project.

In general, many current machines operate at point A:

1-inch depth of cut

51 RPM (600 ft/min bit speed)

300 average HP supplied to head

5.5 tons/min production

* Author equates chalk closely to coal.

YPOTHETICAL PRODUCTION AND AVERAGE H.P. ; HH456
ASSUMING TWO COMPLETE SUMP/SHEAR CY ;

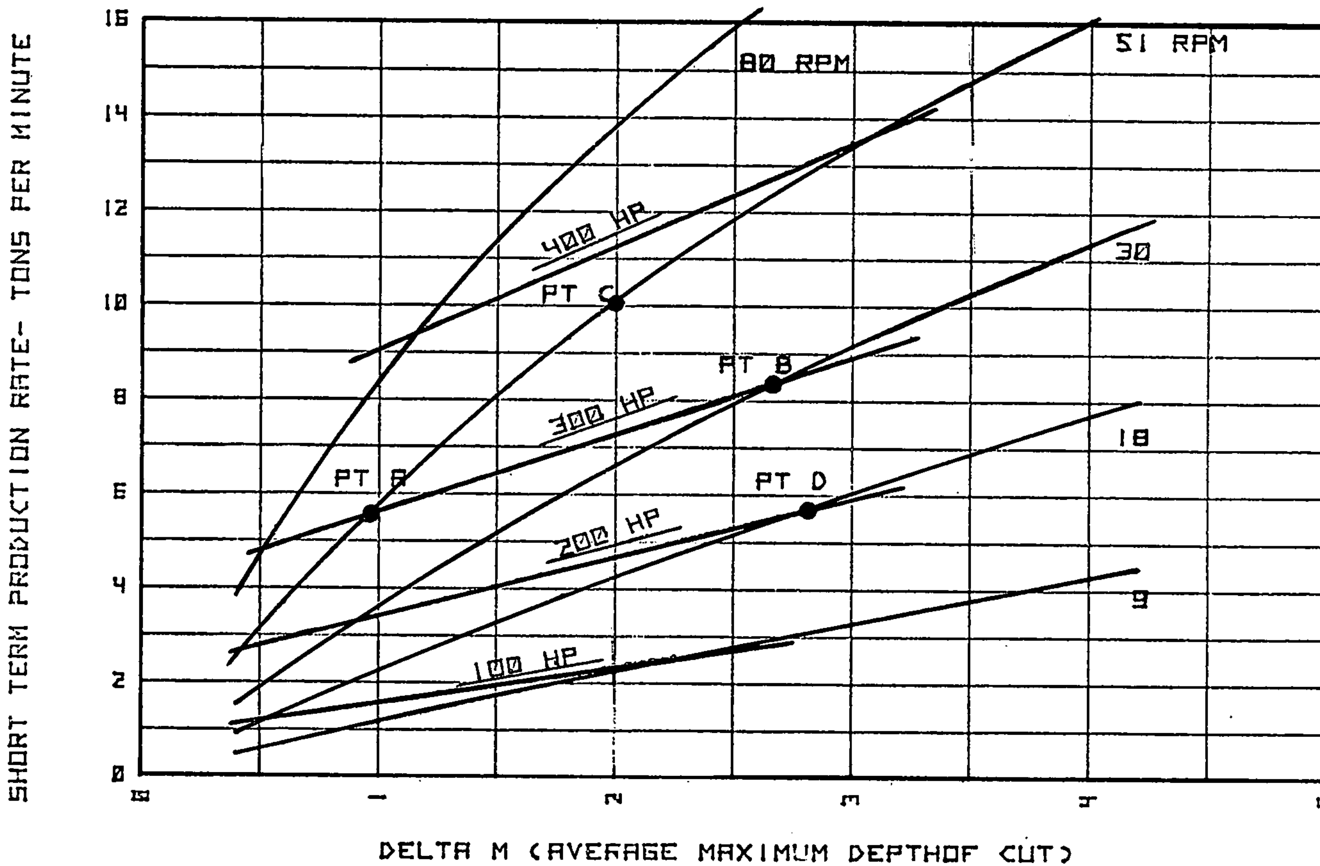


FIGURE 6

To achieve airborne respirable dust reduction in the air return, the replacement machine must have a deeper depth of cut of at least 2 inches or more.

If the rotational speed is maintained while trying to achieve a 2-inch depth of cut, (pt. C), then both the HP and production would increase dramatically. However, this would be an expensive approach. It is also not recommended because it does not include the low rotational speed dust reduction benefits to the operator. From a practical point of view, the continuous miner production rate is not the limiting shift production factor. A large majority of mines are limited in production by haulage or roof support.

The second alternative to operation at point A is to drop the head speed to 30 RPM and increase the depth of cut to slightly over 2-1/2 inches, (pt. B). Thi offers the advantage of a smaller gain in production, but still uses the same ave age 300 HP head motors. This increase in production rate from 5.5 tons/min to 8. tons/min at constant average horsepower is due to a reduction in specific energy resulting from increased depth of cut.

The following examples develop numerical values on dust reduction comparing operations at pts. A and B with water sprays. From the airborne respirable dust point of view, several possibilities are available depending on the mine section's ability to convert the 8.4 tons/min into some additional production.

A. Assume that the production rate per shift remains the same as at pt. A. Operating at pt. B would decrease the airborne respirable dust generated and consequently lower the dust level.

   1) At the machine operator's position, dust would be reduced due to lower rotational speed. The experimental results showed a 77% reduction when changing from 51 RPM to 18 RPM. At the 30 RPM of pt. B, the Multiple Regression analysis projects a 59% reduction in dust generation.* Thus, a 4.8 mg/M^3 dust level should drop to 2 mg/M^3.

   2) In the return air, the reduction would amount to 63% due to increasing the depth of cut from 1 to 2 inches.

B. On the other hand, assuming that the 53% increase in production rate is converted to a 33% increase in shift production, then the following results can be projected.

   1) The net airborne respirable dust reduction at the machine operator's location would be 45% rather than 59%.

   2) The net airborne respirable dust reduction in the air return would be 51% instead of 63%.

A third alternative is the operation at pt. D, which pinpoints an 18 RPM and 2.8-inch depth of cut. This achieves significant airborne respirable dust reductions while maintaining the same production rate and hence production per shift rate experienced at pt. A. Two conclusions can be drawn:

A. The airborne respirable dust level at the operator's station is reduced a net 77% due to operation at 18 instead of 51 RPM.

B. The airborne respirable dust level in the air return is still lowered 63%.

---

* Note that the MR equation predicts a 77% reduction between 51 and 18 RPM; therefore, a prediction of 59% between 51 and 30 RPM should be valid.

In addition to the dramatic dust drop, the other advantage to operation at the conditions set at pt. D is that a lower average horsepower is required for the same production as yielded at pt. A. Finally, it is believed that a machine requiring lower horsepower will inherently require less maintenance. This leads to more operating time and slightly more production at lower operating costs.

The production tests at the Joanne Mine are alternating on one-month cycle between pt. A and pt. B operation confirming the plot in Figure 5.

A different aspect of the deep cutting principle is its potential impact on future high production sections. For example, consider a conventional mining section presently operating at a 2.0 mg/M^3 dust level at the operator's position. With a new deep cutting, low rotational speed machine which generates 77% less dust at the operator's position, the shift production can be raised 4.3 times and still maintain the same 2.0 mg/M^3 reading.

As a final encouragement in support of the deep cutting principle, the quantification of the airborne respirable dust measurements illustrates the health benefits achievable if there is no increase in shift production. At the machine operator's location, a current dust level of 2 mg/M^3 can be reduced to 0.5 mg/M^3 by utilizing the deep cutting principle. It is assumed in the comparisons that both present day and deep cutting machines have water sprays operating.

DEEP CUTTING LOWER SPECIFIC ENERGY IMPACTS ON MACHINE DESIGN AND RESPIRABLE DUST MEASUREMENTS

The specific energy analysis of the Joanne test data is important for two reasons. First is its implications to future machine design of deep cutting machines. Second is the relationship developed between specific energy and dust which may be a useful research tool.

A. Impact of Deep Cutting

The impact of specific energy results are shown in Figure 6 and are indicated by the increase in production rates at a constant HP with increased depths of cut. As stated earlier, these curves are based on the gross specific energy for the HH456 and represent the true average cutting head horsepower. If total specific energy, which does not include machine losses, had been used, the graph would have projected overly optimistic results.

The pros and cons of shifting machine operations from current practice at pt. A (51 RPM at 1-inch depth of cut) to either B, C or D are discussed below in terms of specific energy.

A. Operations at C, (51 RPM at 2-inch depth of cut) would provide an 82% increase in the production rate at the expense of a 16% rise in average horsepower.

B. Operations at B (30 RPM at 3-inch depth of cut) would permit a 53% increase in production with no change in average horsepower.

C. Operations at D (18 RPM at 2.8-inch depth of cut), which maintains the same shift production rate as A, would allow for a 34% decrease in average machine head horsepower.

If the shift production is limited by conditions other than the mining rate, it is strongly recommended that mine operators accept option D. As stated earlier, this option has the advantage in that maintenance is generally less for lower horsepower machines, and lower maintenance directly leads to higher production and lower operating costs.

B. Implications of An All Shear Machine

The 40% reduction of shear specific energy as compared to sump specific energy has some important design implications for future deep cutting machines. The data used in Figure 6 are derived from an average sump-shear horsepower. If a hypothetical all shear machine could be created, the average horsepower would be about 40% lower. Equally important is the highly significant decrease in peak torque and hence horsepower.

Examination of the strip chart record in Figure 7 shows the peak instantaneous wattage for sump and shear. The 66% reduction in shear peak wattage means that there is a similar reduction in peak torque at the head. In turn, this results in lower cutting forces, less thrust required, lower head stresses and longer bit life.

As pointed out earlier, one of the major design problems for deep cutting machines is providing adequate thrust for the sumping portion of the cutting cycle. An all shear machine would eliminate this sump thrust factor. Moreover, since the thrust required for shearing should be around 66% lower than the sump, the shear design applications are possibly more amenable to practical solution.

A far reaching impact of an all shear machine is its potential for combining with deep cutting to achieve even greater reductions in airborne respirable dust. Bureau of Mines tests of a conventional continuous mining machine showed that 70% of the total dust came from the sump portion of the cycle. (Equal tonnage was produced in sump and shear.)[4]

As an indication of this, see the SRI dust level shown in Figure 7 for a continuous sump-shear cycle. Note the large peak of dust that occurs after each sump and how it tapers off during shear.

Based on the Bureau's 70% figure, an all shear cycle should generate 40% less dust than a 50% sump plus 50% shear cycle. Combining 40% all shear with 63% deep cutting and 30% water sprays, reductions should yield an overall 78% reduction in generated dust without decreases in shift production.

This 78% projection is an estimate assuming that the 63% deep cutting airborne respirable dust reduction is the same for both sump and shear. The Joanne data are for a complete test module and do not provide a convenient method for separating the effects. There is reason to believe that the deep cutting fracturing concept, which relates to larger pieces of coal producing less dust, might be more effective in shear than sump. This is based on both the fact that the cutting bits in shear are breaking to a free face and can produce larger coal sizes and that deep cutting in shear reduces bit tip contact. Thus, 53% is considered acceptable for shear.

These results should inspire the Bureau and possible deep cutting machine suppliers to look innovatively into the development of new machine designs with inherently low airborne respirable dust production.

Consider the very dramatic impact shown by the following numerical example projected for a 78% airborne respirable dust reduction for the air return. The dust readings would go from:

A. 9.0 mg/M^3 to 2 mg/M^3.

B. Or a 2 mg/M^3 level could be reduced to .4 mg/M^3.

C. Alternatively, a current section working at 2.0 mg/M^3 could raise production 4.5 times and still maintain the 2.0 mg/M^3 dust level.

C. Relationship to Airborne Respirable Dust and Cutting Parameters

One additional facet of the specific energy which offers promise in aiding future dust research is the close relationship of dust and specific energy to rotational speed and depth of cut. This suggests a cross correlation between airborne respirable dust and specific energy. Figure 8 is a plot of Respirable Dust - Air Return vs. Total Specific Energy. The results are based on calculating corresponding paired points of rotational speed and depth of cut from the MR equations for dust in the air return and total specific energy.[2]

The implications of these results are that specific energy may be used as a guide to changes in airborne respirable dust level. Certainly the results in Figure 8 are true at the Joanne Mine for changes in depth of cut. What isn't known are the effects of alternative bit types or lacing patterns on airborne respirable dust and specific energy, if depth of cut was constant. This suggests that further research in the above described areas should be considered.

The possibilities are very attractive from the researcher's viewpoint, as specific energy can easily and accurately be measured in an underground mine. The test could be of short duration and limited to several repetitive sumps at each test condition. The number of repetitions would be based on the detection of the minimum percent change which would be acceptable. The procedure would not be an absolute measuring system, but certainly very useful in detecting improved dust conditions. Because the procedure is short, a large number of parameters could be compared in a reasonable period of time.

Implications of Coal Sizing Results

The coal sizing results are disappointing in that they failed to indicate any correlation between coal size, and either rotational speed or depth of cut. This, however, was true only at the 4-inch bit spacing. Earlier preliminary results comparing high potential speed, shallow cutting and 2-inch spacing to low rotational speed, deep cutting and 8-inch spacing found a significant reduction in fines and dust for the latter set of conditions. This may have been an effect due to spacing and/or depth of cut. The effect of depth of cut at a constant 4-inch spacing may be too small to detect in a 3^2 factorial experiment with only 4 replications.

The coal sizing procedures for all tests followed similar procedures. A post test review has failed to indicate any basis for suspecting the results of the coal sizing data. It is possible that sampling procedures and coal sizing may be a source of error. Particularly in need of refinement is the size of samples and the splitting of coal samples by factors of 128 to 256 times to analyze the fines. It is believed that the procedures are correct because the nominal values agree with values previously published for continuous miners in the Pittsburgh Seam.[7,8]

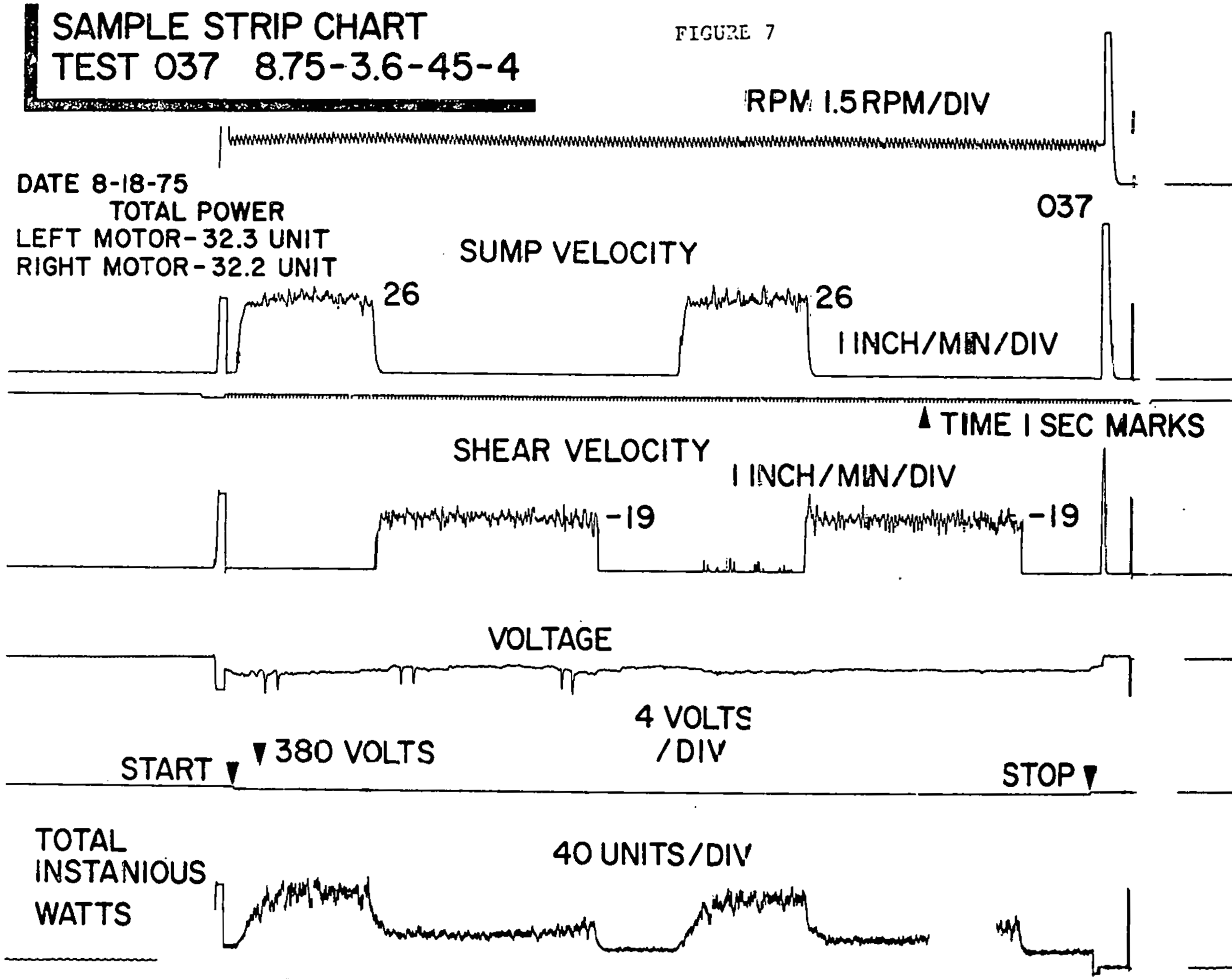
Considering the results, it must be concluded that the coal sizing method is not a proven tool for predicting airborne respirable dust levels. The fact that airborne respirable dust decreased with increased depth of cut without any detectable change in coal size distribution adds impetus to the concept that lengths of pick tip contact may be an important parameter.

NOISE

Accurate noise level measurements originally proved difficult to obtain. It was necessary to rely on the machine operator to record the noise level during each test--a method that proved to be very high in human error. Midway in the test series, the noise levels were recorded using the strip chart (Figure 7).

Recording the data on the strip chart reveals that the sump and shear noise levels do not coincide with sump and shear cutting cycles (see Figure 7). The following analysis ignores this lack of coincidence and indicates:

FIGURE 7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TEST 037 8.75-3.6-45-4
RPM 1.5 RPM/DIV
DATE 8-18-75
TOTAL POWER
LEFT MOTOR-32.3 UNIT
RIGHT MOTOR-32.2 UNIT
037
SUMP VELOCITY
26
26
1 INCH/MIN/DIV
TIME 1 SEC MARKS
SHEAR VELOCITY
1 INCH/MIN/DIV
-19
-19
VOLTAGE
4 VOLTS /DIV
380 VOLTS
START
STOP
TOTAL INSTANIOUS WATTS
40 UNITS/DIV

RESPIRABLE DUST VS. TOTAL S_pE

45° BIT ANGLE— 4" SPACING

DUST: AIR RETURN—30' FROM FACE

S_pE: BUTT COAL FACE — CORRECTED FOR ELECTRICAL AND FRICTIONAL MACHINE LOSSES

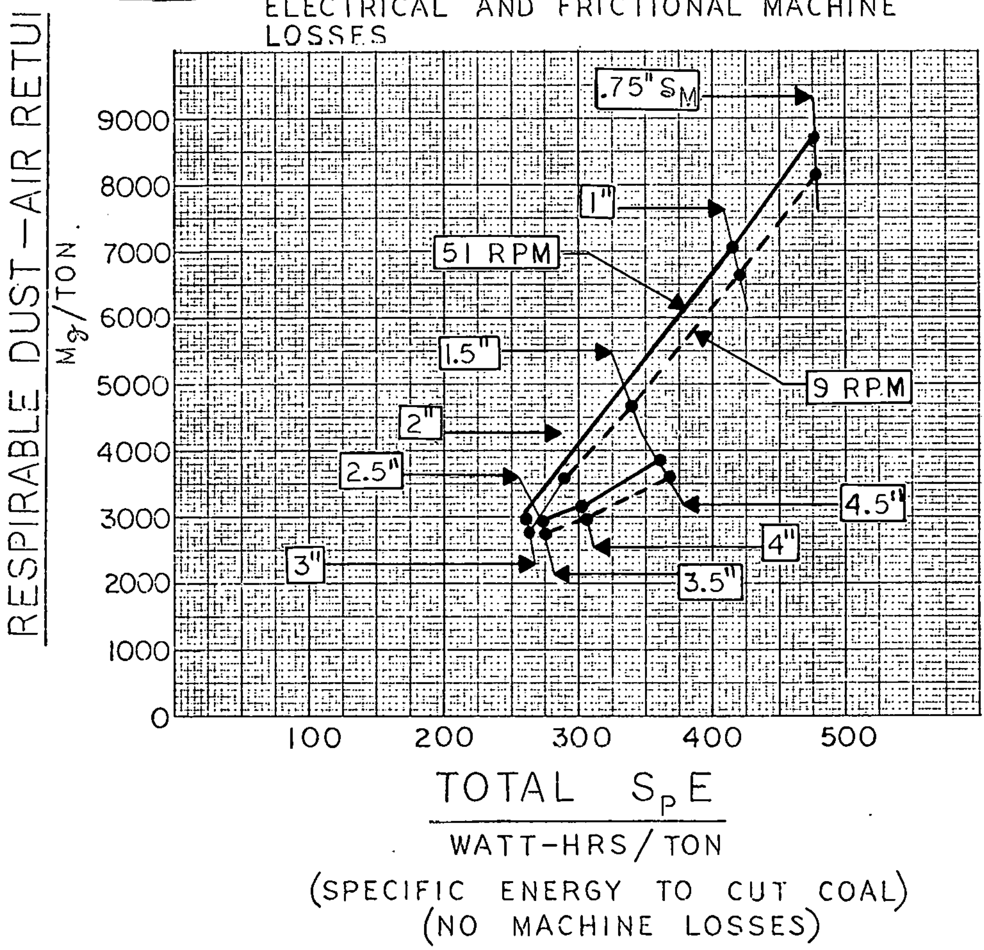

FIGURE 8

A. No correlation between sump and shear noise levels to rotational speed of depth of cut.

B. The shear noise level was 3 db-A lower than sump.

Considering that the noise level was measured immediately in front of the machine operator, it is quite possible that the rotational speed and depth of cut effects, if they exist, are masked by the pump, head motor and conveyor noises.

It is also possible that the lowering of the noise level in shear is attributable to the loading of coal on the conveyor. Note in Figure 7, that after the first shear, the noise level reached its maximum during the lifting of the boom when the conveyor was emptying.

It is judged that the results of the noise measurements are insufficient to warrant making any conclusions.

SUMMATION

The Joanne Mine tests on the HH456 continuous mining machine were a significant undertaking by the Bureau of Mines. The results on airborne respirable dust and specific energy have provided very strong evidence that invites industry to adopt lower rotational speed, deep cutting continuous mining machines.

Based on statistical analysis, the results show stepwise changes in airborne respirable dust and specific energy. No doubt future researchers will more fully develop relationships of a more continuous form as indicated by the multiple regression equations.

It is also true that the data uncovered questions that have not been resolved regarding spacing effects, spacing ratio, coal sizing, noise and specific energy. The presentation in this report is as complete as possible in order to stimulate interest in resolving these unanswered questions.

The overwhelming results of deep cutting at low rotational speed are best finalized by presenting the options that are now open to mine operators.

A. Increase production while holding the airborne respirable dust levels constant.

B. Hold the production constant but reduce the airborne respirable dust level.

C. Compromise on a slight increase in production with a small reduction in airborne respirable dust level.

The impetus for changing to deep cutting goes beyond health reasons in that the new machines can have higher production rates, lower maintenance costs and increased bit life.

Without doubt, the overwhelming conclusion which the Joanne Mine tests susta is that the mine industry immediately consider adopting the deep cutting, low rot tional speed machines to cut coal.

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OPTIMUM NEW MINE MAINTENANCE

By Bennie E. Morgan
Superintendent-Maintenance Services

U.S.S., Cumberland Coal
Waynesburg, Pennsylvania

Growing out of a rather unique case of international cooperation between the Ontario Hydro Company of Canada and USSteel, one of the world's most modern and sophisticated coal producing operations began production of steam coal in July of this year.

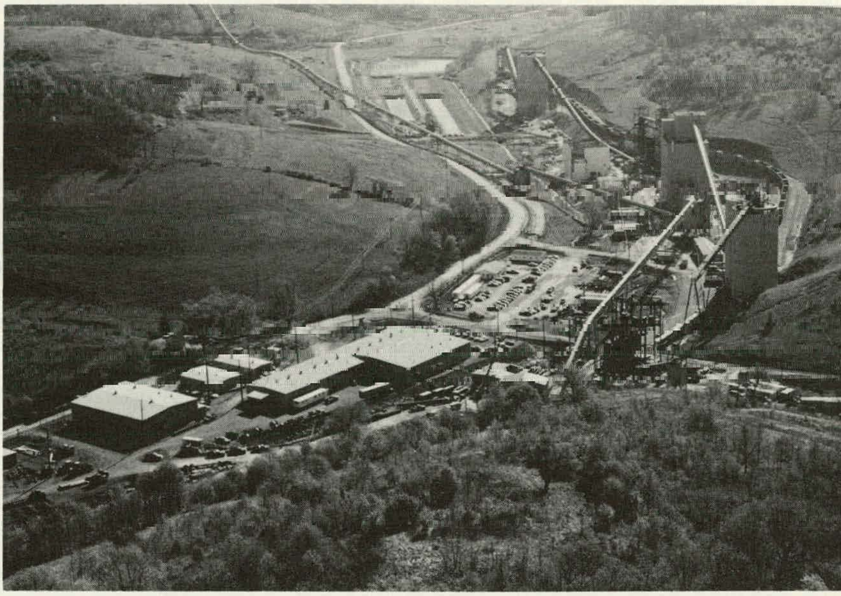
Cumberland Coal Mine is situated in Southwestern Pennsylvania's hilly sheep and livestock country in Greene County which has a population of 45,000 and its county seat at Waynesburg. Principal occupations in this area are farming and underground coal mining, with very little surface mining being done. Greene County borders the northern panhandle of West Virginia to the South and West, joins Washington County to the North and is divided from Fayette County to the East by the Monongahela River which flows North to Pittsburgh, where it joins the Allegheny to form the Ohio River. Extensive underground mining, primarily of metallurgical coal, is done all along the Monongahela to provide coke for Pittsburgh steelmaking. The area has four mineable seams of which the Pittsburgh vein is the most consistent becoming more suited for steam coal as it dips toward the West.

Mining at Cumberland is in the Pittsburgh coal vein, which lies 600 to 1200 feet below the surface. The coal averages about 6½ feet in thickness; the vein dips slightly to the West, is relatively dry, has a fair roof and is gassy. Coal grindability is 55. At full operation 3 million tons of steam coal per year will be produced.

The mine is presently opened by a track slope, a belt slope and a two compartment air shaft. Future air shafts are now being sunk to aid in degassing the coal seam before mining.

Coal is mined by continuous miner into battery and cable reel shuttle cars. Twin boom roof bolters are provided on the section and a battery scoop utility vehicle is used. The scoop is made truly utilitarian by providing a skid-mounted cable and belt winder and an oil dispenser which are transported and used in the scoop, and by also carrying a cable winch and removable jib crane. All but the jib crane are operated off the hydraulic system of the scoop. The loaded shuttle cars discharge into a belt feeder/breaker which feeds onto the conveyor belt system to the surface. Section belts are 36-inches wide, intermediate belts - 42 inches and the slope belt is 48 inches. All belts are roof hung. Coal arriving at the surface passes through a rotary breaker into two 7000-ton silos and thence to the cleaning plant which is of the metallurgical coal type employing heavy-media wet cleaning throughout. Discharge slurry from the plant is passed through a pressure filter to recover the water, which is recycled, thus forming a closed water circuit and eliminating the need for impoundment. The residue filter cake is placed on conveyor belt along with all other waste material for transportation to the disposal area.

A thermal dryer dries the coal before storage in a 9000-ton silo preparatory to shipment. Cumberland Mine operates its own unit train over 18 miles of new railroad track to a barge loading station on the Monongahela River.



Cumberland Mine Surface Layout

In addition to the facilities directly related to coal production, a 60-pound, 44-inch gage truck is provided underground for movement of men, materials and machines. All track equipment is battery powered. An electric powered cable and belt winder is mounted on a flat car and operates off the electric circuit of a battery locomotive.

A bulk rock dusting system is employed using a master and slave cars with dust piped to the faces.

The communication system provides private phone service, remote paging, monitoring and function control. The monitoring and control systems are centered in a surface control station.

Power is received from West Penn Power, the district utility company, at 3,000 volts. The underground distribution system is 7200 volts and is stepped down to 950 volts for the miner and 575 volts for other equipment on the sections. Silicon diode type chargers are provided for battery charging. The surface power distribution system is 4160 volts reduced to 2300 and 480 for individual equipment operation.

Dual axial-flow fans are installed exhausting, with the primary unit being electric and with a stand-by diesel to be used in case of electrical failure. Face ventilation is provided by two 50 H.P. exhaust fans on each section.

A sewage treatment plant is located at the mine and a water treatment plant is located at the river with water pumped through a 12-inch line buried along the 18 mile railroad track to the mine.

We consider the two most vital factors to production are men and machines if a good coal seam and mine are assumed, and that maximum production cannot occur without good maintenance. The best coal seam, plus the most capable operating crews using the best machines available cannot do the job if machines do not operate correctly. Unfortunately, there have been many cases where maintenance and other support groups such as engineering, personnel, training, and supply are considered as nothing more than unnecessary costs to be cut when convenient; also the operation of face machines has frequently been over emphasized without corresponding consideration of adequate support systems. At Cumberland we are maintenance and support oriented from top management on down the line.

The goal of our maintenance group is simply to make machines available for production, the maximum amount of time at the least cost, with a prevailing attitude of preventative maintenance.

We believe optimum maintenance begins with management philosophy, continues through machine selection, people selection, training and development, preventative and emergency maintenance systems, parts availability, rebuild systems and ends with scheduling that allows men time to work on machines.

The first factor to consider in maintaining machines is maintainability. The simplest machine with the fewest parts which will do the job safely is the one to buy, and maintenance people should have a voice in specifying and selecting machines. We believe the following guidelines to be most important in this regard:

- The trend toward solid state electrical circuits with few moving parts is a step in the right direction if the application is right. Simpler solid state controls with infinite speed control can replace functions such as tramming hydraulically or through mechanical clutches and transmissions.

- Direct coupled motors which eliminate gear cases and drive shafts are desirable.

- Spur gears should be used instead of high friction worm and bronze gear sets; and non-adjusting type bearings are easier to maintain.

- Any time the human element is taken out of adjustments an improvement has been made.

- Shafts entering gear cases should enter above the fluid level wherever possible.

- Metal faced seals will give better sealing and longer life.

- The hydraulic system should be simple and as low pressure as possible. Hydraulic trams require high pressures and volumes which are hard to live with.

- Adequate filtration and control of leaks are very important. Filtration should be at least 10 microns and the staple lock no thread hydraulic fittings are usually best.

- Low speed-high torque hydraulic motors also simplify machines by allowing the elimination of gear reducers.

- Easy access and changing of machine parts is essential as well as is easy access for inspection of vital items such as filters, fluid levels, greasing components, chains, hinge points, wear strips, permissible enclosures and meters.

- The best method of doing the job on machines should be carefully researched in the light of prevailing conditions and the mining system employed.

The preceding basic criteria were followed in selecting major items of underground equipment and developing facilities as follows:

Continuous Miners	-	Drum Type, 950 Volts A.C. with D.C. Tram and Remote Control
Roof Drills	-	Dual and Single with Hydraulic Temporary Roof Supports
Shuttle Cars	-	Cable Reel and Battery Types
Feeder Breakers	-	Equipped with Hydraulically Driven Conveyor
Auxiliary Face Fans	-	Exhaust Type, 50 H.P.
Section Load Centers	-	Equipped with Tone Type Cable Ground Wire Monitors and "Q" Type High Voltage Couplers
Section Service Centers	-	Incorporates all Safety, Maintenance and Convenience Needs for Section Support
Battery Chargers	-	Self Regulating Type
Conveyor Belts	-	Roof Hung, Cable Type
Conveyor Load Centers	-	Equipped with High Voltage Feed-Through and "Q" Type High Voltage Couplers
High Voltage Switches	-	Vacuum Breaker Type
Mantrip Locomotives	-	Dual Purpose for Carrying Section Crews or Moving Rail Cars
Locomotives	-	10-Ton Primarily for Moving Equipment
Rock Dusters	-	Master and Slave Car Bulk System
Communications	-	Multi-Channel, Employing Coaxial Cable with T.V. Transmission Capability

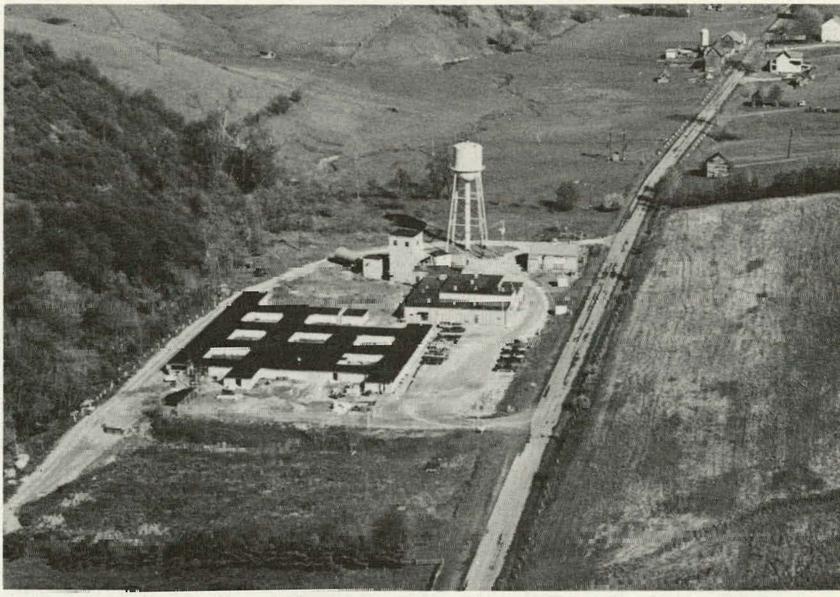
Our maintenance program is designed around the military concept of first echelon maintenance. Our personnel are trained to troubleshoot machines, identify the problem and fix it immediately if minor repairs are needed. Eaching working section has a mechanic with the section crew. His responsibility includes specified Preventative Maintenance jobs and quick repairs such as hose replacement, splicing cables, fixing broken chains and minor electrical repairs. General mine mechanics maintain rolling stock and construction equipment and are on call to assist in the sections as required. Electricians maintain the electrical distribution system, communications and monitoring systems and conveyor power equipment.

MAINTENANCE TRAINING

Effective training must be tailored to the maintenance system. Our plan at Cumberland is to rely heavily on preventative maintenance. Therefore, our people must be trained to watch for developing problems. Our system also is designed for speedy and effective emergency maintenance so our people must be trained this way as well. Specifically, our goal is to make maintenance workers as self-sufficient as possible so that they can reason out and correct their maintenance problems without relying on a foreman to troubleshoot for them. This means that they must be trainable, that is to say, willing and able to learn. They must then be taught the basics of electricity, mechanics and hydraulics so they will be able to understand prints and other machine information.

All maintenance people underground receive the same basic training at our above-ground simulated mine training center which is equipped with a full complement of underground section machines and electrical and hydraulic panels. Trainees are introduced to the machines during this period. The latest training materials have been purchased and developed and the latest teaching methods are used, including movies, video slides, overhead projectors as well as conventional classroom methods. Full-time instructors are employed along with foreman and staff personnel on a part-time basis to provide needed expertise in their particular work fields. Equipment manufacturers' trainers are brought in when deemed appropriate.

Basic training is comprised of a maximum six week course including Introduction to Cumberland Coal Maintenance Procedures, Maintenance Fundamentals, Electrical Maintenance, Mechanical Maintenance, Permissibility and Laws and Fire Protection. If electrical experience requirements are met, trainees who complete this basic course are deemed Qualified Electricians by the Mining Enforcement and Safety Administration (MESA). Testing is a key element in both basic and advanced training. Pre-tests are given before each subject to determine training needs of each individual. Those who show advanced knowledge on particular subjects may skip that part of the training. Candidates must pass post-training tests to show their proficiency before advancing to the next subject. Trainees in the basic course are classified Grade 2 and advance to Grade 3 on completion.



Regional Training Center

Advanced training is done at the surface mine shop, at the training center and on the job and is directed toward the work to be done. Electricians receive only electrical training and surface mechanics receive different courses from underground mechanics. All courses are segmented so that trainees are in class for one machine at a time. They work as trainees on the job between training periods. Pre-testing and post-testing are used in the advanced training on machines in the same way as in basic training. It should be emphasized that our testing program is aimed to qualify rather than disqualify trainees. Advanced training may be spread out over a year's period or until the trainee is deemed competent through testing and work proficiency. They are then promoted to electrician or mechanic, Grade 4 (surface) or Grade 5 (underground).

Retraining is custom designed as needs are identified.

MAINTENANCE SYSTEMS

The lubrication system must be easy, convenient and fast and above all else, it must be clean. The choice of as few lubricants as possible will help prevent mistakes in application. Special fittings for special lubricants, color coding of containers and coloring lubricants are all worthwhile. We use one grease, one gear oil and one hydraulic oil in all machines underground except belt drive hydraulic tensioning devices which must use fire resistant fluid. Several special lubricants are currently being used on the surface but we are working to consolidate these. Grease is packaged in one gallon buckets which are lightweight and throw away type. The lids are removed for use and a dispenser unit with hose and and hand pump are attached. A diaphragm is placed on top of the grease in the can so that the follower plate of the dispenser unit does not contact the grease. This helps prevent contamination when changing cans. The hand pump is a pistol grip type with settings for high and low pressures. The grease cans and lids are used for waste paper and garbage disposal when expended.

A bulk system is being installed for hydraulic and gear oils. Oils are delivered in 6000 gallon tank trucks and pumped into our 15,000 gallon storage tanks. The bulk system features economy, ease of handling and cleanliness. Each tank truck of oil is sampled and analyzed for quality and cleanliness and oil is filtered when pumped into machines on the job. Special skid mounted dispensing units are used for transport oils underground and to service machines. Each working section has two of these units so that one can be kept on the section while the other is in transport. The skid mounted units contain 200 gallons of hydraulic oil and 50 gallons of gear oil, each provided with its own in-tank pump and dispensing hose. The pumps are powered hydraulically by the battery utility scoop or from other hydraulic system and dispensed oil is filtered and metered into machines. Different mated fittings are used for hydraulic and gear oils to prevent accidental switching. Dispensing units are filled from the surface bulk tanks as they pass the fill station on track supply cars. A car spotter is employed to advance the cars past the fill station and up to the slope incline where cars are connected to the slope hoist. The oil dispensers are off loaded into the utility scoops on the sections and machines are serviced from the scoop. The dispenser is then unloaded near the face where machines can get oil as needed. Each face machine is provided with its own hand pump for emergency use. All pumps are equipped with filters and all machines are equipped with 10 micron dirt indicator filters. Machines are also equipped with sight gages on the hydraulic tanks for ease of checking. Each section of each hydraulic pump has an in line flow meter for easy monitoring of outputs and fluid aeration. We have equipped one section with Staple-Lock type hydraulic fittings looking for better leak control and hose maintenance. All other machines have thread type fittings. Oil in hydraulic systems is sampled and tested periodically for contamination control.

We consider lubrication is only the beginning of <u>Preventative Maintenance</u> and that scheduled inspection and testing of machines designed to spot approaching problems is also most essential. The computer is employed to print out reminders of P.M. jobs to be done on specific machines on a scheduled basis. The maintenance clerk operates the system and takes job instruction print-outs from the computer and distributes them to the appropriate foremen. The print-outs are then passed on to maintenance men to do the work and then return the reminders when done. If the clerk does not get reports of jobs done in a week, the computer prints duplicate reminder notices.

<u>Computer Printout</u>

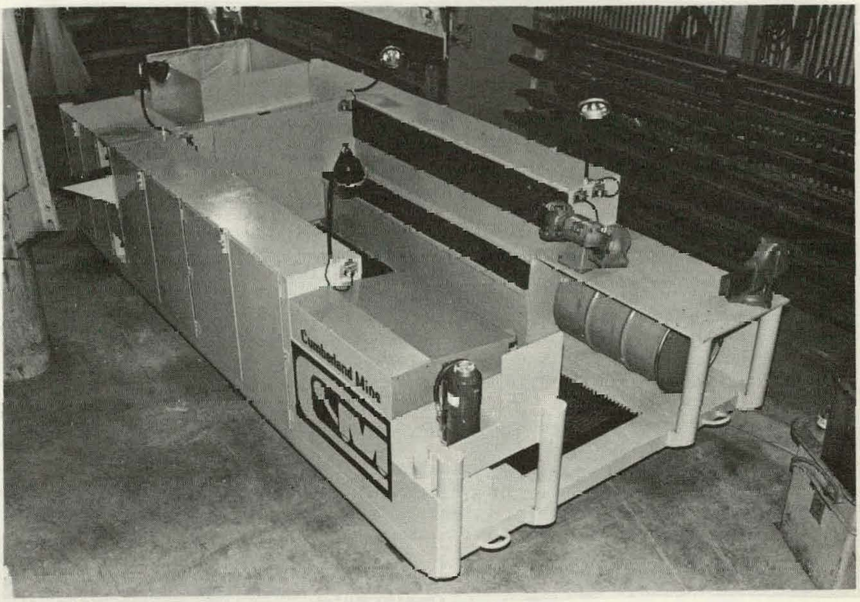
SECTION 12 EQUIP CODE CM112 SUBASSEMBLY CODE MS

1 WK INSPECTION FOR WK 11/17/75 MISCELLANEOUS

MAX 1 WK 2 DAYS ASSIGNED TO

( ) CLEAN PUMP MAGNETIC PLUG
( ) CHECK FILTER
( ) CLEAN ENTIRE MACHINE
( ) INSPECT FRAME
( ) CHECK PERMISSIBILITY
( ) CHECK FIRE SUPPRESSION
( ) CHECK ALL SPRAYS

Data such as pressures, flows, termperatures, vibrations, voltages, amperages and any notable items which the maintenance people feel to be significant are entered into the computer. All parts used are also entered by tons mined or cumulative hours on each machine so that a complete history is compiled by machine. Use of fluids and bits is also entered and any information will be printed out by the computer on demand or by periodic summary report. When we have gained sufficient history of parts used to predict impending failures, we will change parts just before their effective service life. This is our ultimate goal and can be accomplished only if maintenance is effective and uniform.



Section Service Center

Emergency maintenance depends upon ready availability of maintenance people and parts, communication, transportation and quick, safe, and easy component handling.

Each section at Cumberland is equipped with a Section Service Center which contains tools, small parts and other quick use maintenance items.

Large assemblies such as gear cases, electric motors, spare tires and hydraulic components will be kept in a separate underground storage area. This storage area will be manned by maintenance personnel and equipped with a power hoist for quick loading of components onto track jitneys. The jitneys are specified to be equipped with special jib cranes for off loading into battery utility scoops on the sections. The scoop's jib crane is used to aid in lifting components from their machines and replacing them with new ones.

Effective communications between all sections and other work areas and with the surface facilitate timely ordering and delivery of maintenance items. An assembly can be delivered to a working section by the time the section mechanic and the production crew can have the defective one removed. The defective assembly is then brought to the storage area, charged to the machine from which it came and promptly sent to the surface for rebuild-exchange. Previous arrangements have been made with rebuild shops so that they will deliver newly rebuilt assemblies on call.

In most cases they will have the rebuilt item at the mine by the time the defective one arrives at the surface. The outside shop then bills us for whatever parts and labor are required to rebuild the defective assembly. All seals and bearing are replaced any time an assembly is torn down and other parts such as gears and shafts are replaced when close inspection by x-ray techniques or equivalent shows the need. The rebuild shop is required to furnish a report to us of the condition of all components as found during their inspection, thus helping us prevent future failures. The computer also has a hand in preventing recurrence of break-downs. Failures are reported to the computer for each machine so that a complete history by assembly and by machine is compiled. The computer prints out a summary report on any failure showing at what times and tonnages that part failed previously and will also give the average service life of that part on other machines. We expect to eventually predict impending component failures before they occur and to install replacements on a routine basis.

Our surface warehouse contains parts to back up the section service centers and seldom used large assemblies are also kept in storage on the surface.

Machine overhauls will be done by outside shops when performance shows a need. We have spare machines to replace those being overhauled. We hope manufacturers will ultimately build coal production machines which will produce one million tons before overhaul. We think this can be done if frames and major hinge points and wearing surfaces are designed to last longer or can be easily replaced. Modular assemblies permit continual rehabilitation of machines.

The best trained and equipped maintenance group cannot get the job done unless sufficient time is allowed to work on machines. Down time scheduling must allow time for P.M. work especially lubrication. Some down time should be provided during each 24 hour period to take care of impending problems such as wear adjustments, fixing leaks, welding bit blocks, etc. Time must also be allowed for keeping fresh bits in cutting machines. Matched sets of bits will produce better results than replacement of only the worst worn bits. Maintenance of an entire machine is proportional to bit usage except in extremely easy cutting.

Maintenance scheduling is another vital factor if good machine performance is expected.

Production and maintenance scheduling depends on type of machines, severity of conditions, grindability of coal, impurities in coal, travel time and efficiency of the maintenance program and people. If travel time is extremely long, adequate maintenance can possibly be accomplished between shifts while producing coal three shifts per day. Another plan calls for overall coal production on all three shifts while allowing each section two shifts in production and the third idle for maintenance. A popular plan has coal production on all sections on two shifts with the entire mine idle for maintenance on the third. We feel it is essential that the same crews man the same sections each day. This is an important consideration for safety as well as for maintenance. Conditions and machines often vary widely between sections in the same mine, and most people will take better care of machines and the mine when they know they will return to that place the next day. Spare sections used as back-ups when any section goes down usually operate inefficiently si such spare sections become orphans for which nobody is specifically responsible a maintenance is therefore neglected. The same applies to back-up machines on a section. They are usually found to be in a poor state of repair when they are needed and their presence tends to give a false sense of security prompting relaxed maintenance on regular machines.

Machine improvement and replacement cannot be done intelligently without good record keeping. This is the only way to know exactly how machines are doing and what the weaknesses are. If there is a specific problem such as the use of too many hydraulic pumps on a given machine, a record must first be made as to how many pumps are used over a given period of time or tonnage. It must also be determined whether this problem is peculiar to a particular machine or whether mining conditions are unusual, operators are different, the pressure is set right, hydraulic fluid is clean, reservoirs are kept full, etc. Armed with data such as this we are ready to talk to the manufacturer about improvements; otherwise it's only a matter of opinion. This is merely the scientific approach to problem solving and applies equally well to problems on present machines, evaluating prototype machines or specifying new ones.

Most maintenance departments tend to be so caught up in current problems that they do not have the time, or often the facilities or ability, to devote to improvements and new developments. We are exceedingly fortunate at Cumberland to have the USS Monroeville Research Group to assist us on special problems. Their efforts are not limited to maintenance, as the following summary of their 1976 efforts show:

During 1976, considerable design effort has been applied to productivity and safety-related projects for USS Coal Mining. The following is a listing of projects for which significant Research design effort has been undertaken by the Design Technology Divisions:

Antifreezing systems for coal, belts, coal cars, etc.
Bureau of Mines development contract work for a "Flex Drill" and for a "Bolter Transfer Machine".
Canopy designs for coal-mining equipment as required by mining districts.
Hydraulic fluid leakage reduction systems through the use of newly designed fittings.
Hydraulic motor and pump failure reduction through the use of improved oil-filtration systems.
Mine roof and rib stabilization by sealants and structural adhesives.
Silicone application systems to reduce conveyor-belt carryback and to provide complete dumping from coal cars.
Stopping wall designs to reduce costs and provide anticrushing features.
Section Service Center - a mobile unit that provides each mining section with human comforts, safety and emergency equipment, and service parts and tools.
Ventilation systems including low-profile large-volume ducting and extendable face ventilation systems.
Venturi systems for dust suppression and methane dilution.

The operating computer systems at Cumberland including the Preventative Maintenance and parts systems were conceived by the maintenance group working with the headquarters Industrial Engineering group. Working together, we have arrived at a useful, practical system.

To summarize, let me say that attitude is the most vital ingredient to attain optimum maintenance in a new large mine such as Cumberland. This includes attitudes of officials toward maintenance and other services, attitudes of maintenance people towards the machines for which they are responsible and attitudes of operators toward their machines. The prevailing attitude of all concerned must be Preventative Maintenance.

The ingredients of good maintenance are:

- proper machine selection
- people selection and development
- effective systems
- adequate parts availability and proper location
- careful production and maintenance scheduling and,
- accurate and complete records keeping.

Long term success depends on a good rebuild program and scientific research and development but most of all on unceasing vigilance.

EXPERIENCE OF CLINCHFIELD'S NEW CENTRAL MAINTENANCE AND REBUILDING CENTER

Fred T. Wilson
Manager, Central Rebuild Center

Clinchfield Coal Company
Dante, Virginia

First lets get an understanding of the relative size of Clinchfield Coal Company, thus the demand for such a facility as the Rebuild Center can be readily seen. Clinchfield Coal Company, under the leadership of Vice President C. Max Bailes is the largest division of the Pittston Coal Group, under the leadership of President Gerald Swanson It consists of seventeen underground operations, with 54 continuous miner sections; three preparation plants; and two surface mines. The division covers approximately 400 square miles, located in Wise, Russell, Buchanan, and Dickenson counties in Southwest Virginia. The annual projected production for the year 1977 will be approximately four million tons.

In 1974, Mr. I. C. Spotte, then President of The Pittston Coal Group, saw the need for a larger rebuild center. Justifications for constructing the facility were as follows:

1. Higher cost of rebuilding equipment
2. Time interval involved
3. Quality of workmanship

Studies were made to determine the cost saving features of a new rebuild center. It was estimated by a waste cost analysis that the Rebuild Center payback would be approximately two and one-half years. This figure is realized through an economic savings occurring from less loss of production due to machine down time, thus a more efficient operation. Also, rebuild capacity could be tripled with the construction of the new facility by allowing six rebuilds to proceed simultaneously with the potential for seven.

Numerous sites were given consideration for the location of the new Rebuild Center. The existing location at South Clinchfield was chosen, since it offered more room for future expansion, a good access road and an obtainable water supply.

In December of 1975, contracts were awarded and construction began. The contract was awarded to Rentenback and Wright, Inc. Contractors to construct an Armco metal building designed by David Leonard Associates.

The Rebuild Center covers a total of eleven acres. The main shc building is approximately 43,000 sq. ft. In addition, there is a metal storage building, washbay, and a flammable storage building. All scrap material including copper is stored in locked bins until it can be sold.

Started the move from Clinchfield Central Shop at McClure, 30 miles from Rebuild Center, on July, 1976. Fourty-three employees were transferred. The move was completed on October, 1976. Total work force to date, 88 employees with an anticipated 100.

In the following paragraphs some of the safety and environmental consideration in the design of the Center will be discussed briefly, such as the sprinkler system, exhaust fans, lighting, heating system, emission controls, doors, parts cleaning and cranes. To allow for a flow of material through the main shop, a total of 14 - 14' industrial roll-up doors were installed. They allow direct access into any area of the Rebuild Center, thus making it a safer and more efficient operation in the moving of material. There is a 14' industrial roll-up door for each work bay in the overhaul department. This allows components to be moved directly into the work area without interfering with men working on other jobs.

A series of eleven overhead cranes travel through all work areas. They are distributed as follows:

Overhaul shop	(3)-10 ton
Mechanical repair shop	(2)-6 ton
Machine shop	(2)-6 ton
Hydraulic shop	(1)-3 ton
Metal storage building	(1)-5 ton
Electrical reapir shop	(2)-3 ton

All work benches in the mechanical repair and electrical repair shop are equipped with 11' boom cranes.

The fire suppression system consists of a sprinkler system, which is fed by an eight-inch line connected to a 275,000 gallon storage tank. The storage tank is filled from an eight-inch line traveling to a nearby lake.

A system of exhaust fans are used to remove fumes and dust from the work area. Four exhaust fans producing a change in air current of 22,000 cfm each are located between the mechanical repair shop and machine shop. Two exhaust fans producing a change in air current of 27,500 cfm each are located in the heavy welding area. Also, the heavy welding area is partitioned from the overhaul shop by a flame-proof wall with a 14' industrial roll-up door for entrance into the overhaul shop.

The lighting system is complimented by sky lights throughout the shop. One hundred foot candles light all work areas except for the hydraulic shop, where there is two hundred foot candles available in the work area.

To control particulants from the bake-out oven in the electrical department, an emission control stack was designed into the system.

The Rebuild Center was designed to be the nucleus of Clinchfield Coal Company's maintenance program. The Center has the potential to rebuild any component or equipment that is being used at Clinchfield or any of our sister companies in The Pittston Coal Group. The mainstay of the Center is the rebuild program. It was anticipated that e Center could more than triple rebuild capacity. As many as six jor pieces of equipment can be in the process of being overhauled multaneously. The overhaul department employs 24 UMWA employees and two foremen. The main objective of the overhaul department is to redesign existing machines with the maintenance man in mind and to strengthen where parts have failed in the past. The chief electricians are called in from the mines whenever a major component from one of their mines comes in for overhaul to give comments on problems that they have encountered and to give possible solutions. Hydraulic hosing is simplified by using Stecho hose in only five lengths. Many components are being standardized such as cat chains, conveyor chains, gearing, bit patterns, canopies, water systems, methane monitors, line starters, remote control cables and hydraulic components. We found that many main frames on continuous miners were not designed heavy enough to withstand cutting rock, which is a daily routine at some of our operations. Design criteria for such miners is to calculate design requirements, then double for a safety factor, if feasible to do so. From only using standard size Stecho hoses, we simplify our hosing problems tremendously. All fittings and hoses are kept on carts and are wheeled to wherever needed. Not as many hoses have to be kept in stock when only standard lengths are used, thus making assembly and maintenance work more efficient.

Quality of workmanship, pride in ones' job is continually stressed. A thorough check is made of all systems of a piece of equipment before it is released back to the mines. Each rebuild is firmly tested for at least four hours. All components are brought to their operating temperature, all relief valves are set, all pumps rechecke for out put and all functions are checked for proper operation.

Some features have been brought back to the basics. Hand pumps are installed on all hydraulic equipment. The hand pump goes directly to a filter. This has been found to keep a cleaner hydraulic system underground when compared to automatic oil fill pump. All access holes are welded up to the hydraulic tank. All oil tanks are cleaned and pressure checked before filling. A continuous miner can be overhauled at the Rebuild Center in approximately eight weeks vs sixteen weeks elsewhere. Cost savings of rebuilds can be seen below.

Rebuild Center Cost	O.E.M. Price
94,295.00	152,863.00
132,457.00	
122,740.31	
115,804.00	
80,818.00	

The overhaul department is supported by a hydraulic shop, a machine shop, an electrical repair shop, mechanical repair shop, heavy welding shop and a fully-stocked warehouse.

The following will give a brief description of each of the supporting departments listed above.

Hydraulic Shop

The work force consists of eight UMWA employees and one foreman. This department operates only on the day shift. The work load can readily be handled by the one shift operation. The hydraulic shop is separated from the main shop by a flame-proof, dust-proof partition. The strive for utmost quality in hydraulics was due to the poor quality of work being done by other shops in our area. In hydraulics, we saw mix match of parts, payment for parts not installed, wrong type of packing used, motors and pumps only 70 to 75% of capacity whe rebuilt. Cost of our repairs versus outside vendors can be seen in Table 1.

Table 1

HYDRAULIC COST
REBUILD CENTER VS
OUTSIDE SHOPS

Component	Rebuild Center	Outside Shop	Difference RC-/OS
Gear motor	246.01	901.50	-655.49
Vane pump	126.40	280.50	-154.10

(continued)

Component	Rebuild Center	Outside Shop	Difference RC-/OS
Floor jack	177.18	210.00	-132.82
Ripper jack	240.14	453.47	-213.33
Gathering head jack	140.60	415.71	-275.11
Stab jack	111.66	364.60	-252.94
Boom swing jack	213.23	393.17	-179.94
SC reel motor	68.59	106.90	-38.31
Steering jack	73.94	199.25	-125.31
Pinner pump	318.66	397.00	-78.34
Pinner floor jack	123.51	502.00	-378.49

A considerable savings was realized with a better final product. Reasons why components fail are investigated to determine if other problems exist that could cause hydraulic components to fail. Thus we can maintain a quality control within Clinchfield on all of our hydraulic components.

Electrical Shop

The electrical repair shop consists of 21 UMWA employees operating on a three-shift basis. They are divided as follows:

Classification	Employees
AC Motor Winders	7
Trainees	2
DC Motor Winders	4
Trainees	1
Electrician Mechanic	1
DC Panel Boards	2
Trainees	2
AC Panel Boards	2
TOTAL DEPARTMENT	21

In our experience with electric motors that have been rebuilt at outside shops, we found shafts knurled, end bells center punched, no regard for permissibility, poor insulation, payment for work not done, wrong wire size and out and out poor quality of workmanship.

A quality product is the ultimate goal at the Rebuild Center. This pertains to every product that we repair. Every motor that we rebuild receives three dips and bakes of Class H epoxy insulation. Every component is brought back to factory specifications. In Table 2 can be seen Rebuild Center cost versus Outside Shops' Cost.

Table 2

ELECTRIC MOTOR COST
REBUILD CENTER VS
OUTSIDE SHOPS

Components	Rebuild Center	Outside Shop	Difference RC-/OS
100 hp miner motor	781.28	1,982.00	-1,200.72
40 hp motor	602.00	1,115.00	- 513.00
30 hp	627.00	640.00	- 13.00
10 hp	364.00	385.00	- 21.00
3 hp	127.00	164.00	- 37.00
15 J	539.00	684.00	- 145.00

(continued)

Components	Rebuild Center	Outside Shop	Difference RC-/OS
24 J	197.00	322.00	- 125.00
40 J	289.00	340.00	- 50.91
Jeep motor	263.14	764.50	- 501.36

A substantial savings can be realized in electric motor repair cost when you take into consideration the quality of repair work that you receive and the cost savings.

Machine Shop

The machine shop employs 11 UMWA personnel. We have the larges horizontal boring mill in the coal industry. This boring mill makes it capable for us to machine the largest continuous miner chasis wit one set up. The mill is equipped with a rotary table, capacity 22 tons. It was recently purchased at a cost of $330,000.

Other major equipment located in the machine shop consists of: 6 lathes, 2 milling machines, keyway cutter, radial arm drill press, table type horizontal boring mill, surface grinder and an automatic precision cut off saw.

Mechanical Component Rebuild Shop

All mechanical components received from the mines in addition to the units for the overhaul department are rebuilt by 15 UMWA employees.

Major equipment used in this area consists of: a 150-ton vertical press, a 300 and 600-ton horizontal press, submerged arc wheel welder, wheel lathe, and a mig welder.

Washdown Bay and Parts Cleaning Systems

All parts cleaning is done in a separate building from the main shop. A high pressure steam and soap system is used for initial cleaning of parts. The washdown bay has a sludge collection system; therefore, no contamination from cleaning operations gets into streams. Once parts are thoroughly cleaned, they are brought into the shop for disassembly. When disassembled, if needed, the parts are taken to a vapor degreaser for final cleaning before reassembly. All departments use both the washdown bay and the degreaser unit.

We do all of our own trucking. Two trucks run regular routes during the day shift, five days a week, getting to each Clinchfield mine at least two times per week. There is one truck driver on evening shift; he is used to make emergency pick ups and deliveries.

TOTAL COMPLETED OVERHAULS TO DATE

9	Continuous miners
7	Belt drives or tail pieces
2	Portal buses
1	Coal drill
1	Ratio feeder
1	Eichkoff longwall shearer
1	Rectifier
1	Ventilating mine fan

Plus numerous large components; cutter booms, gathering devices, discharge booms, conveyor sections, bull dozer blade.

JOB FLOW

Mo.	Hyd.	Motors	Mech.	Elect.
Jan.	427	86	282	81
Feb.	505	113	410	123
Mar.	475	158	411	112
Apr.	323	67	293	81
May	432	121	375	105
June	371	90	287	87

AVERAGE MONTHLY TOTAL 871

By receiving all components from our 17 mines, we can now see what must be redesigned to improve them or alert the mines of abuse or lack of lubrication, or failure of a component due to failure of another part. Our seven shop foremen have very close ties with all mines for problem solving or any emergency situation.

In closing, I believe we have realized our major objectives that we originally set out to accomplish -- in that we have had numerous visits from OEM engineering staffs to incorporate some of our experience in the design of new mining machinery. Our rebuilds are accepted at the mines as a good final product instead of just another paint job. Our redesign parameters have been proven under actual working conditions. Quality rebuilding is done at an acceptable cost.

MINE PRODUCTION EQUIPMENT MAINTENANCE PROGRAM PERFORMANCE EVALUATION

R. D. Herron
Corp. Vice-President - Maintenance

Island Creek Coal Company
Lexington, Kentucky

There is probably no industry in which the maintenance of equipment is more difficult to accomplish or more important for efficient and effective production needs than that in underground coal mining. The equipment is very expensive and the conditions under which it is used and maintained are extremely difficult when compared to most any other industry.

Maintenance's basic objective is generally understood to be the administration and efficient accomplishment of all inspection, preventive maintenance, repair, overhaul, remodeling, improving and evaluation for salvage or replacement of facilities and equipment. Preventive maintenance is probably the area in which the maintenance function can effect the greatest overall improvement in operating and production cost per unit of performance. Equipment frequently and suddenly rendered unserviceable because of breakdown is not only costly in terms of lost production but also the total cost or result may be much greater than the apparent loss due to the effect on shipment schedules, customer quantity and quality, dissatisfaction and possible contractual legal suits or cost adjustments.

It is the task of effective and efficient management and administration of good preventive maintenance through scheduled inspections, work planning, routine maintenance and complete overhaul to ensure that equipment failures should not occur. Obviously, it is impracticable if not impossible to establish a plan that will provide preventive maintenance that will reduce the occurence of failure to zero though early detection and application of maintenance and corrective action. This however, should be the goal.

During the past three years, Island Creek Coal Company Management has given special attention and efforts to all phases of maintenance; starting with maintenance personnel training, central shop improvement, repair parts availability, equipment improvement - both overhaul and modification, and an overall maintenance system upgrading with concentrated emphasis on preventive maintenance.

Each phase of this attention and work has been "geared" to the individual type of equipment, physical condition of the mine, general ability of maintenance management and labor personnel skills at each mine and individual facility. This maintenance program has advanced at this time, to the stage that we are now in the process of the initial adaptation of it for installation at each mine on the computer. It is not intended here to discuss the program or systems in detail but rather to briefly present those portions which are used in obtaining the data for performance evaluation and work planning of the entire maintenance program. This evaluation with resulting performance indicators and how these indicators are used are the main topics of this presentation.

Two major factors that should control the extent of a preventative maintenance program are, first the labor cost of maintenance compared with final results of production material tons per unit shift; second the percent of production time available for proper utilization of the equipment (delays per unit shift). Establishment of a comprehensive preventive maintenance program just for its own sake should be approached with caution. It is possible for the cost of the Preventive Maintenance Program to exceed the total maintenance cost of using the breakdown or emergency repair maintenance approach. A shutdown of some equipment for no other reason than periodic inspections and adjustments may be and usually is intolerable from a production point of need. The cost of a good program that results in improvement to production is very easily supported by the lower cost per ton when proper allowances are made for all other cost changes that have some effect on the total.

We know of no easy way to pin down maintenance performance in underground mining, but trends of results in production tied to maintenance indicators of manpower, delays, level of equipment overall condition, parts cost and major component repair cost are each part of an overall system; of these many indicators each measures a specific element, but rarely does one alone produce a complete picture but rather a combination of various indicators is required to yield an overview of maintenance performance. The mine management and maintenance management must develop a history of necessary but basic maintenance information so as to establish a relationship etween maintenance performance and production results. (Fig. 1)

n this evaluation of maintenance performance at each mine we deal mainly with the final production result or improvement trends in production due to improvement in equipment availability or equipment delay trends and maintenance manpower requirements to improve cost and production. Each mine is considered on its own past history rather than some other mine or engineered standard.

The essential data is obtained from various records and portions of the mine maintenance program along with Industrial Engineering mine performance standards and pertinent cost data from each mine cost accounting record. These daily records along with actual production are the indicators.

The following is a list of those main indicators as collected and used:

A. Sectional Maintenance Production Cost Summary (Fig. 2)
   1. Unit or sectional shifts worked.
   2. Material or Raw Coal production per unit shift.
   3. Maintenance related delays to production per unit shift.
   4. All delays to production, both maintenance and operation related per unit shift.
   5. Sectional or face production equipment cost by the following divisions, each per unit shift.
      a. Maintenance labor.
      b. Maintenance parts and supplies.
      c. Complete overhaul of equipment in outside facilities.
      d. Complete repairs to parts, units & components in outside facilities.
      e. Lubricant cost.
      f. Total maintenance cost.
   6. Mine cost total per unit shift.
   7. Percent maintenance cost of mine cost.

B. Sectional or Face Production Equipment Maintenance and Production Delays Summary (Fig. 3)
   1. Continuous miner or loader.
   2. Shuttle car, tractor or face haulage equipment.
   3. Coal cutter.
   4. Coal drill.
   5. Roofbolter, stopper or roof support equipment.
   6. Belt feeder - breaker or elevator.
   7. Total maintenance related delays.
   8. Belts or conveyors.
   9. Trailing cables, miner, loaders and shuttle cars.
   10. Production related miscellaneous and other delays.
   11. All delays to production.

C. Longwall System Maintenance Related Delays.
D. Longwall System Total Production Delays.

In order to have a common and a practicable comparison indicator or denominator without special timers, record keeping, etc., the equipment operating shifts is used as the equalizer. In addition, in the delay catagories, the percent of total available production time is used and determined by using a compromise time of 390 minutes as a normal unit shift. This allows out of an eight hour shift; one hour for travel time and one-half hour for lunch. As we go into the use of the computer we expect to use actual time available for production.

The use of these indicators at each mine is individual as well as at each section or Longwall system and thus is not compared with any other operation. The management action that is taken from the trends, as developed and resulting from the overall maintenance program with its effect on production and cost at each mine is the main return or performance evaluation of the program. Over a period of time the results of this actual experience will result in the establishing of a normal or standard cost budget level for maintenance labor and all other maintenance cost as concerns the people and conditions at each individual mine. After one and a ha years of experience we are now starting to establish these levels but expect additional improvement not only in the maintenance program proceedures but especially in the maintenance personnel long range training.

As we look at the indicators and their summary reports we have arranged for our presentation a representative example of the thirty-six mines we now have involved in the maintenance program. (Fig. 4)

This example represent in summary, for an individual mine, the basic operating performance and maintenance cost history. All the data in this example, as well as all other examples to be presented, is not actual but is representative so as to illustrate the actual result in performance and overall trends.

You will note that this data as to maintenance cost is limited to the section equipment operation and is equated to unit shifts operated.

Last years (1976) averages or totals are used as a base of starting for each evaluation. The maintenance function performance trend resulting from each principal indicator is as follows:

1. Material tons (Raw Coal) per unit shift is the base for results or performance comparison. This production figure, converted to clean coal and then compared to actual mine cost per unit shift of operation is the most complete overall performance indicator. All the other indicators are management "tools" to lead us into the area with the most potential for maintenance and production effort improvement.

2. Maintenance delays in minutes per unit shift is the maintenance program effectiviness indicator. This is the main purpose or goal of any good maintenance program along with cost control of the program. If not policed and controlled it is possible for the cost of the program to exceed the total maintenance cost, using the breakdown or emergency repair maintenance approach. Again, sufficient production improvement is necessary to more than offset these additional program costs.

3. The total delay (from all causes) to production is an indicator for comparison against maintenance delays for proper consideration in trends of production changes. All delays to production are important but we must know what type and how much so as to give needed attention when evaluation the maintenance program.

4. Sectional maintenance labor cost (face maintenance work manpower available or used) is one of the most necessary and important indicators. In order to accomplish anything we need people. How

well we manage along with the individual skill of each person involved will usually determine how many people we need to do the work and how effective and efficiently it is accomplished. This means more or less people, cost and production or final results. We find that this is undoubtly the most important item in our operation and one that has to be developed to achieve any goal.

5. Sectional maintenance supply and parts cost per unit shift of equipment operation is an indicator used for trends of the preventive maintenance portion of our program. As the cost trend goes down or "holds-it's-own" we see good performance in our equipment overhaul and Preventive Maintenance Program. If the cost trend goes up we see that we are behind in either or both overhaul and preventive maintenance work.

6. Sectional maintenance - overhaul of equipment in cost per unit shift of operation is an indicator used in its effect on all other cost of maintenance but most important it indicates whether or not we are accomplishing the normal overhaul needs of the program. This cost in the program is easily budgeted, both in a normal level as well as in a "catch-up" situation. This overhaul cost has a very large impact on both the parts and component repair cost which are also closely related to the performance of preventive maintenance activities.

7. Sectional maintenance - repair of units and component assemblies in cost per unit shift of operation is a major indicator of overall equipment condition (overhauls needed) along with how well the preventive maintenance program is being accomplished. This is normally a very difficult item to budget when overhauling of equipment is behind noticable needs and when the maintenance system is about 50/50 breakdown and preventive.

8. Lubricant cost per unit shift is used as an indicator for hydraulic system leaks and handling methods. Once a good level has been established it is a very positive indicator.

In order to further improve on production equipment maintenance delays per unit shift, a maintenance and production delay summary by equipment type (or catagory) has resulted from this overall indicator. (Fig. 5) This delay summary is compared in minutes per unit shift and maintenance delays in percent of total available production time. The year 1976 is again used as a base for starting this evaluation in each catagory of face or sectional equipment.

This continuous comparison of equipment delay results is a key maintenance management indicator of the major type equipment that has an increase or decrease trend in delays. With this information maintenance program records can easily "pin-point" the individual pieces of equipment giving delay problem trends. The actual delay items can then be more completely programmed in preventive maintenance frequency or other appropriate action such as overhau ahead of schedule or other special maintenance attention.

As mentioned earlier the total clean coal cost and production figures are the best indicators of an effective and efficient maintenance program. In order to achieve suitable levels of both production and cost, the most important need is the development of people, both labor and management, who must match their abilities with the operation needs. These two areas, production and maintenance manpower, when compared for any mine in our company show a very definite resulting trend and performance evaluation that correspond with the development of a more complete maintenance program. There are many more day by day items that affect production and cost but our maintenance efforts over the past three years show up significantly in the records and results.

These are two most important items of maintenance program performance evaluation, production results and maintenance people level, can be more vividly protrayed in graphic form. (Fig. 6) This graph is very representative of all our mines and covers approximately one and a half years period, starting Jan. 1, 1976 to June, 1977. It is very noticable that within a month lag, production changes closely follow sectional maintenance manpower changes. Even though the monthly maintenance manpower level follows with similiar production levels the most significant indicator is that the trend of one year, June, 1976 to June, 1977 shows as the manpower level is increased so is the tonnage. If we charted the maintenance delyas they almost exactly follow the manpower curve. In looking at this curve, and knowing the overall progress that has been achieved in the maintenance program at this mine, this manpower curve and production curve trends indicate that we have about reached the proper manpower level and that our program is now becoming more effective.

In closing we would like to note that we think each mine, even in the same company is different in many ways such as mining conditions, equipment, facilities, management and labor just to mention a few. Each and all of these differences need to be recoginized and the maintenance program and its evaluation should be tailored to proper levels of operation and cost to best satisfy and efficiently meet the best production level at each mine.

Fig. 1

CATEGORY________

EQUIPMENT CASE HISTORY RECORD

TYPE-MODEL-MANUF. & DESCRIPTION________

S/N________ USBM NO.________ CO. NO.________
MINE OR LOCATION________ DATE________
OVERHAUL DATES________ ________ ________ ____

DATE	MAJOR ITEM, PART, UNIT, COMPONENT ETC. REPAIRED OR CHANGED	WORK CODE	MINUTES DELAY TO PRODUCTION

Fig. 2

________DIVISION
________COMPANY
SECTIONAL MAINTENANCE--------PRODUCTION-----COST SUMMARY

MINE________ YEAR 1976 VS MONTHS TO ________ MEETING DATE________

	1	2	3	4	5	6	7	8	9	10	11	12	13
PERIODS OR MONTHS	UNIT SHIFTS WORKED	MATERIAL AVG. TONS/U.S.	MATERIAL TONS/DAY	MAINT. DELAY M/U.S.	TOTAL DELAYS M/U.S.	SECTIONAL MAINTENANCE DOLLARS AVERAGE *PER UNIT SHIFT OPERATED*						PARENT MINE COST /U.S.	% MAINT. OF MINE COST
								OUTSIDE SERVICE					
						LABOR	SUPPLY	OVERHAUL	REPAIR	LUB.	TOT.		
Yr. '76													
Jan. '77													
Feb.													
Mar.													
1st 3 Mos.													
Apr.													
May													
June													
1st 6 Mos.													
July													

Fig. 3

__________ DIVISION
__________ COMPANY
MAINTENANCE & PRODUCTION DELAY SUMMARY - MINUTES/UNIT SHIFT
& PERCENT OF TOTAL AVAILABLE PRODUCTION TIME

MINE__________ YEAR 1976 VS MONTH TO ________ MEETING DATE________

PERIOD OR MONTH	MINER OR LOADER	SHUTTLE CAR OR TRACTOR	COAL CUTTER	COAL DRILL	ROOF BOLTER OR AIR STOPPER	BELT FEED OR ELEVATOR	TOTAL MAINT. 1+2+3+4+5+6	BELTS OR CONVEYOR	TRAILING CABLES MINERS	TRAILING CABLES S. CARS	MISC. & OTHER	ALL PROD. TOTAL
Yr.'76												
% Tot. Avail. Time												
Jan.'77												
% Tot. Avail. Time												
Feb.												
% Tot. Avail. Time												
Mar.												
% Tot. Avail. Time												
1st 3 Month												
% Tot. Avail. Time												
Apr.												
% Tot. Avail. Time												
May												
% Tot. Avail. Time												
June												
% Tot. Avail. Time												
1st 6 Months												
% Tot. Avail. Time												
July												
% Tot. Avail. Time												

Fig. 4

SEPERATE DIVISION
SAMPLE COAL COMPANY
SECTIONAL MAINTENANCE--------PRODUCTION-----COST SUMMARY

MINE INDIVIDUAL YEAR 1976 VS MONTHS TO ______ MEETING DATE 7/1/77

PERIODS OR MONTHS	1 UNIT SHIFTS WORKED	2 MATERIAL AVG. TONS/U.S.	3 MATERIAL TONS/DAY	4 MAINT. DELAY M/U.S.	5 TOTAL DELAYS M/U.S.	6 LABOR	7 SUPPLY	8 OUTSIDE SERVICE OVERHAUL	9 OUTSIDE SERVICE REPAIR	10 LUB.	11 TOT.	12 PARENT MINE COST /U.S.	13 % MAINT. OF MINE COST
						SECTIONAL MAINTENANCE DOLLARS AVERAGE PER UNIT SHIFT OPERATED							
Yr.'76		670		28		100	270	275	200	35	880		
Jan.'77		690		21		125	220	275	175	25	820		
Feb.		660		25		90	200	200	190	20	700		
Mar.	DATA FOR CALCULATIONS	675		22		130	220	280	160	20	810	DATA FOR CALCULATIONS	DATA FOR CALCULATIONS
1st 3 Mos.		675		23		115	216	250	175	22	776		
Apr.		730		15		145	190	200	145	10	690		
May		715		18		160	175	220	130	15	700		
June		700		20		155	170	210	100	20	655		
1st 6 Mos.		695		20		134	196	230	150	18	728		
July													

Fig. 5

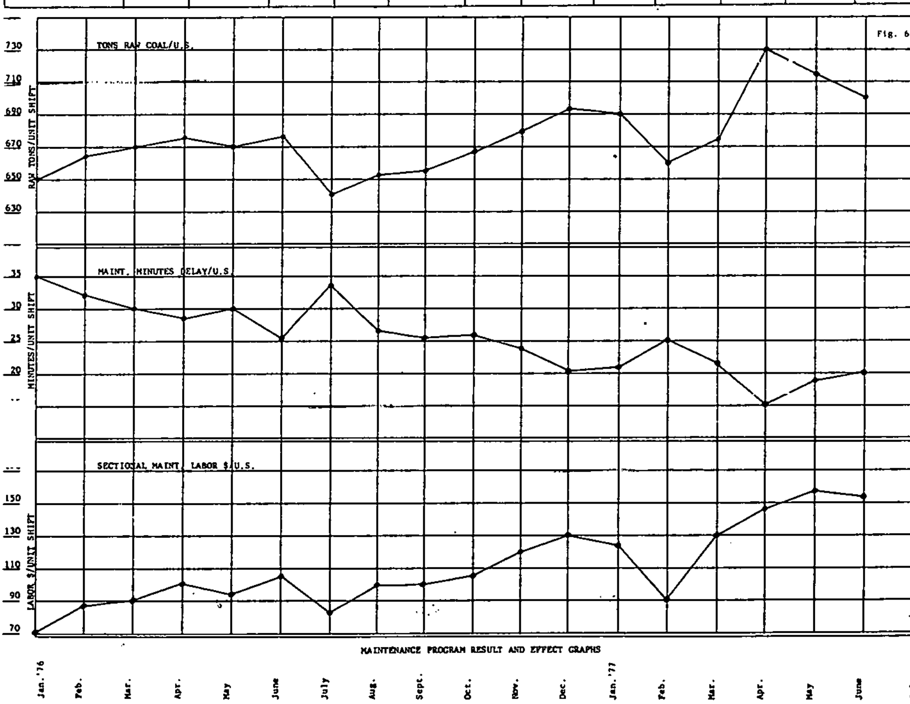
SEPERATE DIVISION
SAMPLE COAL COMPANY
MAINTENANCE & PRODUCTION DELAY SUMMARY - MINUTES/UNIT SHIFT
& PERCENT OF TOTAL AVAILABLE PRODUCTION TIME

MINE INDIVIDUAL

YEAR 1976 VS MONTH TO ________

MEETING DATE 7/1/77

PERIOD OR MONTH	MINER OR LOADER	SHUTTLE CAR OR TRACTOR	COAL CUTTER	COAL DRILL	ROOF BOLTER OR AIR STOPPER	BELT FEED OR ELEVATOR	TOTAL MAINT. 1+2+3+4+5+6	BELTS OR CONVEYOR	TRAILING CABLES MINERS	TRAILING CABLES S. CARS	MISC. & OTHER	ALL PROD. TOTAL
Yr.'76	12	8	4	-	2	2	28					
% Tot. Avail. Time	3%	2%	1%	-	1/2%	1/2%	7%					
Jan.'77	10	10	-	-	-	1	21					
% Tot. Avail. Time	3%	3%	-	-	-	-	5%					
Feb.	12	8	-	-	-	5	25					
Avail. Time	3%	2%	-	-	-	1%	6%					
	10	6	4	-	-	2	22	RECORD DATA ONLY	RECORD DATA ONLY	RECORD DATA ONLY	RECORD DATA ONLY	RECORD DATA ONLY
Avail. Time	3%	1 1/2%	1%	-	-	1/2%	6%					
1st 3 Months	11	8	1	-	-	3	23					
% Tot. Avail. Time	3%	2%	-	-	-	3/4%	6%					
Apr.	7	6	-	-	-	2	15					
% Tot. Avail. Time	2%	1 1/2%	-	-	-	1/2%	4%					
May	7	5	4	-	-	2	18					
% Tot. Avail. Time	2%	1%	1%	-	-	1/2%	5%					
June	8	5	2	-	3	2	20					
% Tot. Avail. Time	2%	1%	1/2%	-	3/4%	1/2%	5%					
1st 6 Months	9	7	2	-	-	2	20					
% Tot. Avail. Time	2%	2%	1/2%	-	-	1/2%	5%					
July												
% Tot. Avail. Time												



MAINTENANCE PROGRAM RESULT AND EFFECT GRAPHS

THE MOSS NO. 2 LONGWALL MOVE

C. MAX BAILES
VICE PRESIDENT

CLINCHFIELD COAL COMPANY
DIVISION OF THE PITTSTON COAL GROUP
LEBANON, VIRGINIA

The conditions of each mine govern how you set up your longwall move. At Moss #2 Mine, we have very fragile top conditions which make access to the tail impossible. Our present moves are done entirely from the head at the old face.

STEP I - PREPARATION:

A. One of the most important parts of a longwall move is preparation. The first thing we do is to establish a stopping point before we cut out, and work to that stopping point. The point should be directly in line with the entry (X-Cut) that our move is to take place through. This is essential so that we will not have to do any maneuvering around corners to load our longwall equipment.

B. Preparation of:

(1) Haulways - Haulways should be thoroughly cleaned and a loading point established where all material is to be loaded.

(2) Hoists - A hoist will be set in the No. 2 entry. The hoist line will be taken through a shive wheel and then to the face.

(3) Transport and Supply Cars

(4) Scoops and Little Henry on Section

(5) Battery Station (easy access and close)

(6) Crib Block

(7) Tools (slate bars, wrenches, chains, socket sets, etc.)

(8) Preparation of the New Face: As soon as we develop the new face, we dowel the face with six-foot dowel rods and resin. We start in the coal and dowel up into the upper strata (about 45° angle).

All of the above should be done before we cut out, and we are ready to start immediate disassembly as soon as we cut out.

STEP II - LAST SHEAR:

When it is time to make our last shear, we ram the panline over to the face, make the last shear and leave the jacks back. Do not pull the jacks up.

After the last shear is made, position the miner on the ram pan (head). This will enable us to remove the miner without having to use power.

STEP III:

A very important step is clean up of the longwall face. As soon as we cut out, we start people cleaning up all loose coal (face, panline, in-between jacks). The cleaning up of loose coal is important because it will save us problems later on when we start to pull off equipment. The jacks will hang up on loose coal, also the panline is hard to drag if it has to be dragged through piles of loose coal. We are now ready to start dismantling of the old face.

'EP IV:

While we have men cleaning up, we start two-four men disconnecting T-Bars from the panline and retracting ram cylinders back into the jacks. Also, we start, immediately, two men at the tail roof-bolting the face (we don't quit bolting the face until it is completely bolted).

Note: After all T-Bars are disconnected and ram cylinders are pulled back, we are then ready to kill the power on the face.

STEP V:

After the power is cut off (hydraulic and electrical), we start two men unhosing jacks.

How to Unhose:

(1) Leave leg hoses connected to legs.

(2) All other hoses (pressure, return, down pressure and ram pressure) are disconnected and hooked back into the valve chest on each individual jack. This will keep dirt and debris from getting in the lines.

STEP VI:

Start four (4) men disconnecting main power lines, hoses and control cables. Roll hoses up from the power pack to the head (label and tape). Also, do cables and hoses along the panline the same way. Load all hoses, cables and lines on supply cars as soon as they are rolled up and labeled. These are ready to transport now.

STEP VII:

After hoses and lines from the power pack are out of the way, we :e ready to disconnect the pig from the head drive (leave the pig ›oked up to the walking tailpiece).

Note: We still haven't disconnected the transformer from the walking tailpiece, so this will enable us to tram the walking tailpiece and pig out of the way. As soon as they are in the clear, disconnect the two and cut the power on the tailpiece. We will use a scoop to load the tailpiece on a Phillips Carrier and transport to the new face by rail. Also, we can transport the transformer and power pack to the new face.

(Make sure not to leave equipment all over the place at the new face. The equipment that is transported to the new face should be placed in position).

While the pig, tailpiece, transformer and power pack are being transported, immediately start the disassembling of the head drive after the pig is out of the way. Disconnect the head drive from the panline, pull it out of the way and transport the head drive to a location on the head drive car so that it can be set off and picked up at a later time.

STEP VIII:

Now that the head drive is gone, the miner is next. If you remember, we located the miner on the ram pan. So, now, all we ha to do is pull the miner off the panline, load it up and either tak it outside to be overhauled (if needed) or it can be set off at a place close to the new face.

STEP IX - REMOVAL OF CHAIN CONVEYOR:

The miner is now removed, so we can remove the chain conveyor from the panline. This is accomplished by first disconnecting the chain conveyor on the head and tail. We then run a wire rope from the hoist to the top section of the chain conveyor. This is connected to the two loose conveyor chain ends on each side of the face conveyor chain. We then drag the top section of the chain conveyor over to the loading point where it is loaded onto the supply car with the hoist. The chain conveyor is labeled so that we can put it back in the panline at the new face exactly the same way we took it off the old face. The bottom section of the chain conveyor is removed in the same manner as we removed the top section of the chain conveyor. We generally put the bottom section of the chain conveyor on first at the new face. The two sections are not reconnected until the head drive is attached to the panline.

STEP X - DISCONNECTION OF PANLINE:

We disconnect our pans in groups of three, and transport in groups of three. We use a hoist to drag each group of pans down the face to the head. A scoop will then load each group (three pans) on the pan car. We transport the pans as soon as they are loaded. Upon reaching the new face, we unload and position the pans. We do not connect the pan groups until the tailpiece is in place, since this is the last item off the old face. As soon as the last group of pans and the tailpiece are moved to the new face, we start at the tailpiece and work back to the head reconnecting the groups of pans.

While all this is being done, we are starting to pull jacks off the line at the old face.

Note: As we are connecting the groups of pans, a 3/4" wire r is placed under the panline. This will be used for remounting our chain conveyor.

STEP XI:

After the panline is removed from the old face, we will only need 5-6 men there to pull chocks off the line. This is done with a machine that we call "Little Henry"*.

The Way We Pull Our Jacks:

"Little Henry" is loaded up with crib block (piece of belt dragged behind the machine is loaded with enough crib block for three cribs)

"Little Henry" then goes to the first jack and is positioned with the hoist boom two feet inby the ram cylinder. The base of the jack should be cleaned thoroughly so we can connect a 3/4" chain down under the ram plate on both sides of the jack. The hoist rope from "Little Henry" is connected to the center of the 3/4" chain. The hoist rope should be tightened up until the "Little Henry" starts to move due to the tension. As soon as the rope is good and tight, foul the machine ("Little Henry") with either a jack stem (steel rail) or crib block. We then break the jack by dehosing the lag hoses (we put the jack in the down position to remove the pressure and then disconnect the leg hoses). As soon as the pressure is released, the jack should start to pull. We keep on pulling the jack with the "Little Henry" until the canopy of the jack is about eight inches from the face. The jack will usually collapse as soon as we start to pull it off the line. The crib block that was hauled up to the jack should be used now to build three cribs (two for each jack and one for the panline). It is a must that we build the cribs immediately to support the roof. Now that the jack is positioned next to the face, we then rehook the hoist to the face side of the chain (3/4" chain that is attached to the jack) and spin the jack toward the head. With the jack now aligned parallel to the face, we rehook the hoist rope to the center of the chain and tram the jack to the head with the "Little Henry". Upon reaching the head, we disconnect the chain and load the jack on a jack car with a scoop. This is done by positioning the scoop in front of the jack, ramming out the blade and connecting the (still attached) 3/4" chain to the center of the blade, and retracting the blade thus pulling the jack up into the bucket of the scoop. It is then lifted up and loaded onto the jack car. As each jack is transported to the new face, it is unloaded with a Joy 14BU Loader. We drag the jack to its position with the loader, where it is rehosed by the same people who did the unhosing at the old face. In the meantime, everything is being reconnected at the face.

(1) Chain conveyor is reinstalled.

(2) The miner is put on the panline and the electrical lines are reconnected.

(3) The haulage chain is reinstalled along the panline and run through the miner.

(4) The slack is then taken out of the chain conveyor and haulage chain.

(5) The pig and tailpiece are pushed into position and reconnected.

"Little Henry" - is a cut down Joy 14BU Loader. We removed the head and tail section leaving just the track vehicle. We then added a small boom to one end of the track vehicle, which has a hoist rope feeding through it. We mounted a hoist (Long-Airdox Super 500 take-up, 5 H.P.) in the center of the track vehicle. This machine is really versatile and a lot of help in moving a longwall.

STEP XII:

As the jacks are being repositioned and rehosed, we run a temporary line from the power pack down the face so that we can power up our jacks. This is usually done in groups of 5 to 10 jacks at a time. The reason for doing this is so that we can support the top and check for leaks. When the last jack is reinstalled, powered and checked out, the move should be complete.

If our move is done properly and no major problems arise during the move, it will usually take us about five days to complete a full move. The first move at the Moss #2 Mine took approximately 15 days. Over the past nine years, we have worked out procedures and cut the time to five days.

FINAL COMMENT:

The most important thing in moving a longwall is to draw up your plan and stick to it. Line up your people and make sure they stick to the plan. This is a must to have a move go smoothly and quickly.

AUTOMATION IN MINING

Mr. Ross Forney
Director - Coal Services Division
Director - Institute of Mining Automation

EBASCO SERVICES INC.
P.O. Box 986
Golden, Colorado
80401

COLORADO SCHOOL OF MINES
1500 Illinois
Golden, Colorado
80401

Introduction

The name of this session is MINING AUTOMATION. Interestingly enough, the word *automation* conjures up a different image in each of us, depending upon your own background. So that we may all approach this subject from the same vantage point, I think we should start by defining *automation* as it applies to mining.

Automation does not mean replacing people with computers or taking men out of the mine. That is about the last thing that will happen. What it does mean is that automation will:

1) Upgrade the human operator to a *hands off* position.
2) Minimize machinery damage due to human error.
3) Minimize production variations based upon operator decisions.
4) Protect men and machinery from catastrophic damage triggered as a result of a minor mechanical failure.
5) Improve operational safety by limiting excessive and dangerous machine excursions.
6) Allow the operation of machines in hostile environments with operating personnel removed to a safe distance.

Now, I know this all sounds attractive, but somewhat nebulous. Past experience in other industries indicates that with automation, you should be able to double and triple production with little or no increase in operating personnel. This doesn't mean you can increase productivity with the same personnel. It does mean that you must have a higher caliber operator to oversee an automated production system than is required to drive a continuous mining system.

What will all of this cost? I can't say for sure, but I am not aware of other industries abandoning their automated process to go back to a *hands on* operation. Past experience would indicate that automation requires an additional 10% increase in cost, plus another 10% to cover necessary improved tool and ıchinery designs to meet the greater demands of an automation program.

ɜst I be accused of deceit, I must also point out that there are some extra costs involved in development and the start up of new automated systems.

Now that we have a broad brush picture of what we mean by *automation* and its expected costs; let's get more specific. Our target areas for automation are:

1) Coal winning at the face
2) Main and secondary haulage from face to surface
3) Preparation plant quantity and quality control

Let's start with the winning operation at the face:

In the United States, we are very fortunate to have our coal near the surface. This has allowed *room and pillar* operations to predominate with little incentive for our coal industry to develop along the European lines of *longwalling*. Thus, we have evolved into the post OPEC era with continuous miners, while our wordly associates have migrated to high performance *longwalling* systems.

> Note: Any production device which is confined to a track with repetitive operations is inherently more automatable than a free running, trackless, steerable, continuous miner which seldom repeats any cycle in the same sequence or time frame. For this reason, we will devote our attention today on automation of longwalls.

But do not despair, you friend of the continuous miner; both longwalling, as wel as room and piller, suffer considerable production losses because of unreliable coal haulage. With coal haulage; we have a system with repeatable cycles, confined on a fixed track, so why is this system subject to such poor reliability? As we all know, one of Mr. Peter Tregelles' recent papers has directed the National Coal Board's attention to this inconsistency. I hope Peter will give us some insight to this problem today.

Preparation plant automation is a third category. In the past, we in this country have had little need for tight quality control over our wash plant product stream. There has been little demand to maintain close controls on sulphur, ash or coal reject to pond tailings. Those carefree days are rapidly drawing to an end with the advent of strict sulphur emission standards, and with more to come. The current market price paid for coal today puts an end to excessive reject streams. In short, American design and built preparation plants will soon be equipped with highly sensitive transducers, plus sophisticated process instrumentation, controls, and computers.

You know, we can talk and theorize all day about why automation may or may not be suited to a particular mining or preparation scheme, but the cold hard facts are: Today in America, we have a mining machinery capacity to production ratio of ten to one. That means we have the capacity to produce ten times more than we are actually producing. This appalling capacity to produce ratio can not all be blamed on hostile mine environment. Some of this loss in production is due to system incompatibility - and that is what automation is all about!

Once again; I am looking forward to hearing from our British and German friends who are advancing the art of mining automation above and below ground.

MONITORING AND REMOTE CONTROL PROGRESS TOWARDS AUTOMATION IN BRITISH COAL MINES - 1977 *

V M Thomas
Deputy Director (Electrical Engineering)

K W Chandler
Head of Computer Systems Branch

National Coal Board, Mining Research and Development Establishment,
Burton-on-Trent, England

nopsis

After a significant rise in the 1960's, productivity has been stubbornly static for the last six years. This has been so in spite of heavy investment in machines and plant and the high potential of this equipment.

Remote control can make a direct contribution to improved productivity in some cases by the elimination or reduction of manshifts involved in operating equipment. The automation of bunkers is a good example.

However, a significant characteristic of poor performance more generally is the high incidence of delays, breakdowns and lost working time. This applies not only at the face but also in the elsewhere-below-ground category especially in the field of transport of all types: mineral, men and materials. There is no easy solution to these problems - that is clear. A persistent attack on unreliability, higher standards of engineering and better control and organisation of operations are important factors in any solution. The role of comprehensive monitoring in support of management in tackling these problems is examined.

If productivity is the major concern,safety is of equal importance. Developments in environmental monitoring instruments and systems have been rapid during the last few years. The major improvements are presented, and the continuing work outlined.

The field of comprehensive monitoring and automation is one which borrows heavily from innovation elsewhere with microcircuit electronics and computers the key technologies. They also have a major role to play in our own industry. The Board has pioneered a powerful minicomputer-based system for colliery applications. This has been given the name MINOS. This system is outlined and the facilities and opportunities it offers are described.

Introduction

In several papers dealing with the future patterm of the British Coal Industry(1-3) one of the major themes of a research and development programme for proved performance was that of monitoring, remote control and automation. iefly, the importance of these is based on the following:

- There is no new mining machine or technique in sight which will produce an improvement in productivity comparable with that which resulted from mechanisation at the face in the 1950's and 1960's.

- The machines we already possess have potential of an order of magnitude greater than that currently being achieved.

- The application of monitoring and control techniques backed by vigorous managerial action can make a major contribution to the improved performance which is sought.

* Presented by Peter G. Tregelles, Director of Mining Research and Development, National Coal Board

An illustration will suffice to emphasise the problem. Table 1 presents the face delays for a sample of 70 faces in the last quarter of 1976. (Extracted from Reference No 6).

Table 1. Face Delay Analysis, 1976

CAUSE	AVERAGE DELAYS PER SHIFT		PERCENTAGE OF DELAY
	No of Occurrences	Minutes	
AFC	3.7	19	16
Coal Clearance	4.1	28	24
Machine	2.5	29	25
Supports	0.9	7	5.8
Face Ends	0.6	12	9.8
Remainder	2.1	23	19
TOTAL		118	

With a machine-available time of 103 minutes and a total delay of 118 minutes in a shift time of 296 minutes there can be no clearer example of the need for reliable and consistent production. Delay is the norm!

A useful if occasionally hesitant start had already been made towards remote monitoring in the 1960's [4,5]. The development of computer technology has now made the widespread application a practical proposition. Comprehensive monitoring is the phrase often used to cover this field. As members of MRDE directly involved in the formulation and conduct of the programme the authors take as the theme for their paper the progress being made towards its application.

Evolutionary Stages To Automatic Monitoring And Control

There is a progression through several evolutionary stages along the path from manually performed tasks to automatic control. This is shown in Table 2.

Table 2. Evolutionary Stages

Evolutionary Phase	Advantages/Benefits	Disadvantages
Manual task	Low capital cost. Improvisation/adaptability of man.	Severely constrained by manual effort. Labour intensive. Low working rate. Exposure to hazard. Limited to man's senses
Mechanisation (manual control)	Much less manual effort. Less constrained by manual effort. Less men, lower labour cost. Less exposure to hazard.	Large capital and installation cost. More maintenance, training. Greater technical breadth and competence needed to run efficiently.
Remote monitoring and control	Less operator constraint. Less men.	Large capital cost. Maintenance, training.

Table 2 (continued)

Evolutionary Phase	Advantages/Benefits	Disadvantages
	Less exposure of men to hazard. Better data.	Greater technical competence needed. No operator to improvise way out of trouble.
Automatic control with central monitoring.	No operator constraint. Optimised operation within design parameters. Data acquisition and storage.	Moderate extra cost. No improvisation.

The points to note in addition to the obvious exchange of large capital costs for reduced labour content are:

- Each phase reduces the constraint of the man. Initially this is due simply to the manual effort involved, but in the later stages is due to his inability to react with sufficient speed and accuracy when performing a complex task.
- Substitution for the man is successful only when the operation is thoroughly understood. The unexpected must be met very infrequently since when he is removed there can be no improvisation by a man to circumvent it.
- Reliability of equipment and system is essential in the later phases.
- Very considerable improvement in safety would accompany the introduction of a wide measure of monitoring and remote/automatic control techniques. This derives from three sources: the reduction of men, their removal from places of hazard and the increased awareness of hazards through an improved measurement capability.
- Design of the equipment to operate within its surroundings and its converse, control of the conditions to ensure operation of equipment within its design parameters are essential to the achievement of high reliability.
- A steadily increasing technical competence is required, in a variety of disciplines. Leadership of a multi-disciplined team is also of importance.

In selecting the fields of activity for the introduction of monitoring and control, the relevant factors to be considered are the evolutionary phase already tained, the value of the potential contribution of a remotely controlled/ tomated operation and the availability of the technical solution or the resources cessary to achieve it. Based on these considerations the prime attack has been switched from the face, as was the case in the attempts at ROLF[5], to the elsewhere-below-ground category and the surface. The face is without doubt the major prize to be won but the task is also one of great complexity and difficulty.

Monitoring And Control Technology

Two most significant developments have occurred since the first steps towards comprehensive monitoring and control were taken in the early 1960's. These are the very rapid expansion of semiconductor electronics and secondly the equally rapid emergence of very powerful digital computers. The latter is of course one of the major consequences of the former.

The basic elements of any monitoring and control system are:

- Sensors to measure the relevant variables, eg, to detect the running of a machine, monitor position, temperature, etc.

- A data transmission system to transmit the sensor and actuator signals to and from a processor.

- A signal processor, which may operate in a wide variety of ways; amplifying, selecting, comparing, counting, performing logical operations.

- An actuator (for control) or some indication of the output (for monitoring).

Sensors are required for a wide range of variables appropriate to the plant or activity involved. Taking as an example the remotely controlled belt conveyor driveheads, detectors have been in use for many years to monitor belt slip, alignment, tear, blocked chute, motor temperature, bearing temperature and ambient temperature.

For bunker automation, optimisation of the coal clearance system, and production monitoring, however, additional transducers have been required. These include:

Horizontal Bunker Contents Monitor

- Angular rotation (shaft encoder) of the chain bed or moving car drive mechanism.

Vertical Bunker Level (several alternatives under development)

- A nucleonic discrete level detector, multi-level if required.

- 'Bindicator' motorised plumb-bob depth sounding.

- Ultrasonic or electromagnetic pulse-echo radar.

Coal Flow Measurement on Belt Conveyors (under development)

- Belt idler weigher system based upon spring movement or strain gauge force measurement.

- Ultrasonic coal profile monitor.

- Optical shadow system (for coal profile).

Similarly for envionmental monitoring a range of instruments has been, or is being, developed covering:

- General body methane concentration 0 - 3% - BM1.

- Roadway airflow - BA2 and BA4.

- Drainage pipe high concentration methane 100% - BM2H.

- Drainage flow - orifice plate and differential pressure meter.

- Suction - pressure transducer.

More detail of these and other sensors and their use may be found in papers by Cooper (1976)[7] and Thomas and Cooper (1976)[8].

Data transmission systems had been developed by manufacturers in the 1960's for conveyor control[15] and these have been updated in the last few years to take advantage of a whole new generation of semiconductor components of greatly increased capability[13, 14]. They also have been designed to meet new more stringent standards for intrinsic safety.

Signal processing has been radically affected by the emergence of extremely powerful small digital computers over the last ten years. A key project activity has been the absorption of the minicomputer technology (and more recently micro- puter technology), following its explosive expansion and proliferation, and the ation of a computer system by which the coal industry can utilise this powerful l to best advantage[10, 11, 12]. The outcome of this effort is the MINOS minicomputer system described below.

The Control Loop

Control technology is the engineering discipline which relates to automatic control of a mechanism or system. In its simplest form this involves:

(i) Measurement of the difference between the actual and desired value of the controlled quantity, eg, force, quantity, position.

(ii) Use of this difference, or error signal, suitably amplified, to adjust the quantity closer to the desired value. But activities of all types including managerial control involving human operators as well as machines, can be viewed in an analogue way. Objectives can be set analogous to 'desired values', and measurements can be made in the form of performance figures - face delays, daily output per face, tons per manshift, etc - which require action, managerial in this case, to bring performance closer to the desired value. The command loop in such a case has considerable similarity to the 'feedback loop' of the control engineer and the analogy can be illuminating and useful in emphasising key activities: the setting of objectives (desired values), the measurement of actual performance for comparison with required performance and the need for action when these differ. The computer offers a tool which for the first time can make a real contribution to management especially through the data collection, storage and analysis to provide a 'measure' of activities. Experiments in management information systems based on computers are now being mounted at a few collieries at which the managers will 'try their hand' with technical support supplied by the MRDE.

MINOS Computer Systems

MINOS stands for Mine Operating Systems. It provides an industry standard for mining control computer systems but uses traditional makes of data transmission equipment, each with a range of outstations providing standard facilities. MINOS is an approach to mine monitoring and control which it is believed will attain a number of desirable objectives:

- It provides a modular computer control system for collieries which can be installed now and yet fit into the pattern of future developments.

- It will bring a common standard throughout the industry while maintaining utmost flexibility to accommodate local variety and preference.

- Computer software developed for one mine can be applied to all mines on the MINOS system.

- Training is simplified and development costs are reduced by having a common standard.

Many application packages are already available. Each uses the same basic computer control technology, and several applications may be incorporated in the same computer. Each of the packages links to a common MINOS secondary computer which provides long term storage and information processing for the colliery management. The separation of the control and data processing functions imparts a high degree of reliability to the control system.

The applications include:

* Coal Clearance Control.

* Bunker Management.

* Stone/Coal Segregation and Blending.

* Environmental Monitoring.

* Main Pump Supervisory Control.

* Production Monitoring.

* Switchgear Monitoring and Control.

Figure 1 illustrates the composition of a system.

The basis is a Digital Equipment Corporation PDP 11/34 computer linked to a common form of control console, and driving underground data transmission equipment manufactured by the systems contractor. The surface station includes a printer for report generation and the system accommodates optional remote display terminals for special use by officials and management away from the control centre.

Normally operation is automatic with the computer controlling the remote equipment linked to the system. It monitors the actions and values and stores away data and events in its memory or sends them to a secondary system. Only when anything abnormal occurs is operator action required.

The system has been designed for simplicity of operation. All communications with the operator are in English with pit orientated words: reference to code books is not necessary.

An operator's console is shown in Figure 2.

Information is displayed on two screens forming part of the console. One screen is for messages to the operator, advising him of departures from the norm, warnings that action is needed, or alarms. The other screen shows operating values, tables of performance, mimic diagrams or other displays selected by the operator.

When the operator needs to take action either to change the control paramet or start or stop plant, he uses the special keyboard associated with the consol Each item of plant, conveyor, bunker or power pack for example, is identified by single key. Dedicated keys are also allocated to meaningful function controls, for example, STOP, LOCAL, FORWARD DRIVE.

The MINOS system provides many useful facilities as standard. These include:

- Automatic setting up or adjustment by question and answer of plant parameters such as warning levels, calibration factors, and protection without interrupting the operation of unaffected plant.

- Automatic diagnostics. Computer assisted fault finding, testing and maintenance are built in. Provision of on line test programs for plant, and automatic logging of system and data transmission faults is provided.

A standard structure is used throughout all systems to facilitate the collection, storage and access information. Means are provided for looking back over the shift on primary control systems and examining information more extensively on secondary management systems. The standard structure of the system is the key to interchangeability of programs from one colliery to another, and of the ability to add on new developments as they are generated.

Automation Of Coal Clearance At Bagworth

The principles employed on the MINOS coal clearance systems were first :ablished on an experimental system at Bagworth Colliery begun in 1973(16). : work at Bagworth had three broad objectives

- To automate the operation of a coal transport network comprising belt conveyors and bunkers.

- To develop and apply techniques for optimising the utilisation of the transport system, including stone/coal segregation and automatically implementing bunker management strategies.

- To encourage the management's use of performance data.

The complete transport system of some 26 belt conveyors and two bunkers is now automatically controlled by the computer system under the supervision of an operator in the control centre. More belts from the adjacent Ellistown Colliery are being added to make a total of about 50.

Management Control Action

All events which occur in the transport system during each shift are stored in the computer's memory. Within minutes of the end of the shift a range of printed reports (see Figure 3) summarising the performance of the coal transport system is produced for engineering and production personnel. These reports have rapidly gained acceptance as a source of objective, unbiased information, and have now become an integral part of the management procedures at Bagworth.

A single figure known as the gate belt index represents the average percentage stop time of the gate belts. Nationally outbye delays represent a significant part of the lost time at the face. The same was true at Bagworth. Before the introduction of automatic control (monitoring was implemented first) typical figures for outbye delays were 40 mins per shift. Introduction of automatic control brought about an immediate improvement in running time of the gate belts, conservatively estimated at 10 minutes per face per shift. A persistent attack on outbye delays was mounted by the colliery management team using the reports as a guide. They have now brought the normal lost time due to outbye delays down to less than 15 minutes.

These successes are due to the determined action of the management, who use information on delay causes to tackle the problems: layout of transfer points re it is shown that blocked chutes or spillage contribute significantly to delays; the alignment of belts where that is shown to be significant; and all the detailed information to ensure a smooth running transport system. It is now standard practice at Bagworth to use the reporting information to measure and ensure the satisfactory performance of belt transport systems from new faces before they are finally got away and put on to full production. The gate belt index gives management a yardstick with which to judge the performance of the transport system and against which to set targets for improvement. This illustrates how data collection and good management practice go hand in hand to bring about a sound and reliable transport system.

The automatic controls themselves provide some improvement to the system. More rapid restarting saves minutes on each stoppage; block and reverse sequence stops prevent over-runs and have significantly reduced blocked chutes and spillage at some junctions; while the more uniform loading of belts provided by the automatic outloading features of the bunker controls has made a valuable contribution to less spillage, less trouble from electrical faults and belt slip due to overloading, and consequently permitted a reduction in the number of patrol men.

Improvements

The figures from Bagworth (Table 3) show the gate belt running time to be improved by 25 minutes per shift or say 6%. A proportionate increase in production would give an additional 1400 tons per week or £25,000 per week. On that basis such a scheme would pay for itself very rapidly.

Table 3. Bagworth Improvement

YEAR	1972	1974	1976
	REMOTE CONTROL	AUTOMATIC CONTROL	MANAGEMENT ACTION
OUTBYE DELAYS (Mins per shift)	40	30	15
NO. OF FACES	5	5	3
MACHINE SHIFTS (Per week)	35	35	30
OUTPUT (Tons per week)	20,000	23,000	23,000
OMS	94	100	114

There is corroborative evidence that improved production is possible. In 1972/73 the OMS averaged 94.4. It now averages 114 with a record value of 142.4 about a year ago. The pit achieves the same production now from 6 face shifts as it previously did from 7. It is not claimed that the automatic system caused these improvements. It permitted them to be achieved.

Many coal clearance automation schemes may be justified on manpower savings alone. It is not practical to compare the manning at Bagworth now with the entirely different pit that existed 14 years ago, when remote control first began to be introduced. Instead Table 4 indicates how the present system would have been manned if it were simply remotely controlled and also if it had no remote control equipment. Substantial reductions are possible.

Table 4. Bagworth Manning

	Local Control	Remote Control	Auto Control
Control Room (Including telephone exchange)	3	7	4

Table 4 (continued)

	Local Control	Remote Control	Auto Control
Patrol Men	-	16	14
Maintenance Team	10	15	15
Conveyor Operators	56	-	-
Bunker Operators	10	3	-
	79	41	33

Environmental Monitoring At Brodsworth

The safety of the mine is of foremost importance to the management and a prime area of concern is the ventilation system. In order to form an experimental on-site base for environmental studies a computer based monitoring system, using the same MINOS computer system design that was emerging for coal clearance, was set up in 1975 at Brodsworth Main Colliery, Doncaster(16). Underground data transmission outstations and instrumentation are specific to the environmental monitoring applications. The following instruments are available:

* BM1 - General body air percentage methane.
* BA2 - General body air velocity.
* BM2H - Methane drainage (0-100%).
* Drainage range suction.
* Duct velocity trips.
* Smoke detectors.
* Pressure switches.

The important measurements are from BM1's and BA2's outbye of the faces in the return airway, although the siting of instruments is very much a matter of local choice. A tube bundle sampling system with infra-red and other analysers at the surface(17) is also being linked up to the computer system at Brodsworth. The analysers measure methane and carbon monoxide. Oxygen deficiency and Graham Ratio are calculated by the computer. Trends in carbon monoxide are examined by the computer to give early warnings of heatings or fires and to distinguish from short peaks due to shot firing or changes in barometric pressure.

Further instruments are currently under development at MRDE and they will be able to be added to the systems as soon as their use becomes established. These include a solid state air flow meter based upon vortex shedding, a pressure transducer, a temperature and humidity transducer and in the slightly longer term, transducers for oxygen and carbon monoxide.

Use Of The Information

The most outstanding feature of the installation at Brodsworth is the way in which the information on both airflow and methane make has been seen to be of value and used by management and officials. Graphical type representations of the parameters are available with three sets of variables displayed side by side (Figure 4). The displays may be of current shift, previous shift or trend over the previous seven days. Changes in the ventilation pattern are immediately apparent even to the untrained. The displays quickly high-lighted bad air door discipline, and now special rules and exercises have been introduced to ensure proper control of air doors. Similarly the relation between general body methane and the face production cycle has been driven home to all who come into contact with the system.

This is all leading to a better understanding of the behaviour of the environment and will surely lead to better control and more economic use of production facilities.

Further Developments

The experience gained at Bagworth and Brodsworth is being transferred to commercial systems suppliers by commissioning future pit schemes using the same systems designs. Six coal clearance schemes are being installed and a similar number of environmental monitoring schemes are in various stages of planning. It is now possible to use MRDE resources to extend the development and use of computer systems into other fields.

Management of Coal Clearance

In the immediate future it is proposed to carry out field experiments with programs for optimising the use of bunkers and for batching stone through a complex system. The Rawdon-Donisthorpe complex has been chosen for this exercise. A similar solution is applicable to optimising the use of bunkers in a restricted situation. Automatic implementation of bunker management algorithms will be tried out during the summer at Eppleton and other collieries supplied with MINOS coal clearance systems.

Face and Production Monitoring

A system for the collection and storage of accurate minute to minute information about face performance is one of the keys to effective colliery management and planning. An experimental installation providing automatic facilities for recording and displaying this information, with the ability to supplement this with manually input records of delay causes and other relevant subjective information, is being installed at Bagworth. This system will form the basis for the development of a more comprehensive package for monitoring the operations on the coal face, and the state of health of the cutter loaders, roof support systems and other equipment contributing to the face performance.

Coal Preparation

Similar system developments are taking place in Coal Preparation, with a major experimental scheme being planned for Lea Hall Colliery. In this case there is less expectation of a substantial manpower saving than a considerable improvement in plant performance by the application of closed loop process control techniques and accurate monitoring and control of plant parameters. Management information will also be made available to a secondary computer.

Fixed Plant

Standard systems for monitoring and controlling pumps, methane drainage extraction plant and other similar installations are being developed. These may be installed in any of the standard MINOS systems. Another major area where significant changes to present techniques may result is that of monitoring and controlling the electrical distribution network of a colliery. This can be extended to include every item of pit equipment and could form the nucleus of a comprehensive system for monitoring plant health and recording maintenance.

The experimental installation is planned for Manton Colliery and will form part of the comprehensive computer automation installation there (9).

Processing Information for Management

A major effort has been directed to the on-line collection of data for immediate operational control at collieries. The potential in the use of this data for longer term analysis, planning and control is recognised to be great. It is also recognised that the introduction of such techniques in the industry must be in evolutionary stages, intimately involving colliery management in the development and use of data processing systems to secure their acceptance of these techniques. The possible application of computers to information analysis overs a wide range of activities at the colliery from production at the face to istribution from the washery; and from monitoring the operation of plant to ssisting colliery services in the preparation of wages.

Secondary Computer Systems

The concept of secondary computer systems for storing and processing the information produced in primary control systems has evolved from the initial data collection experiments on the Bagworth coal clearance system. Initially it is intended to concentrate on the provision and analysis of production information from coal clearance primary control systems, supplemented by information on face characteristics, manning and performance manually input or obtained from the experimental face information system. As confidence and experience are gained the scope of the reporting will be widened and the use of the information will be exploited further with the management at the colliery to cover not only information and reports for immediate use but the storage and manipulation of historic data for in-depth analysis and long term planning.

Particular attention is being paid to the terminal and display equipment to meet the varying requirements at collieries. The importance of encouraging managers to play a part in the evolution of systems is recognised. System design is being aimed towards flexibility and the ability of pits to determine their own requirements. Standardisation and interchangeability between systems is also important, as is compatibility with the more powerful data processing facilities available to the National Coal Board through Compower.

Face Automation

While the major automation effort has been directed elsewhere, effort continues towards the solution of two important problems which must be overcome before a substantial measure of face automation becomes practicable. One of these problems, that of automatic horizon control of the cutting machine, has been a stubborn problem for many years and although partial success has rewarded a considerable effort, one further stage of development is seen to be necessary to reach a general solution applicable to a variety of machines and conditions. The second problem is that of measuring, and subsequently controlling, face alignment.

Automatic Horizon Control

Work on automatic horizon control has been mainly confined to steering fixed drum Anderton shearers, though more recently attention has been directed to ranging drum machines. The control system, as outlined in general terms in an earlier paragraph, includes sensors, processor and an actuating/steering mechanism.

A most important sensor, which has required considerable research and development effort, is that for measuring roof coal thickness left by the machine. Several stages of improvement have been necessary culminating in the current Type 709 gamma radiation back-scatter probe. The increased thickness range of this version over earlier types has been obtained, however, at the price of a much larger and heavier construction. A transducer measuring the roll angle of the machine towards the face has also been successfully developed with a roll-type steering underframe as the actuating mechanism.

The technique was successfully demonstrated in 1968, following which a batch of 20 prototype machines was installed and, since 1973, a further 20 or so installations of production machines. Only partial success can be claimed, however, since exploitation remains at a modest level of 10 to 15 installations at any one time. The main reasons for this are:

- The large probe and trailing mounting arm is a considerable encumbrance, is liable to damage, and interferes with the use of the shearer for stable elimination.

- It has tended to be a 'last resort' solution for difficult conditions rather than a means for making normal conditions better.

- Roof coal must be left within a range of 2 to 20 cms.

The fixed head machine is in any case a diminishing market, with ranging machines now occupying the major share.

Trials of a single-ended ranging shearer with a control system similar to that on the fixed head machine have shown conclusive evidence of steering instability.

A further programme of development is now being launched. This programme is on a broader front, including several additional transducers, to ensure that, on completion, a wide range of machines can be automatically steered in a stable manner in most mining conditions.

Face Alignment

A means of measuring and, finally, controlling face alignment is the second major target. Two solutions, both currently in the 'principle proving' stage are being pursued.

One system is based on the reel-and-cord method. In this, a cord is trapped in the goaf, and a reel from which the cord is drawn off is attached to a chock or conveyor. The length of cord drawn off as the face advances is measured, and by mounting such devices periodically along the face a rough profile can be obtained from the series of lengths.

The second method is an optical technique using a machine-mounted light source and detector unit and a series of passive reflectors attached to chocks along the face. A microcomputer processes the data contained in the positions of the reflections obtained and generates a profile of the face line as an output.

Substantial further development and field trials are required before the next step, the control of face line, can be tackled.

Conclusions

A new industry standard computer system - MINOS - has been evolved, which will enable interchangeability of developments across the whole industry, while making use of the traditional Board suppliers' proven data transmission and remote control expertise. This should aid the rapid introduction and development of computer monitoring technology at collieries. The benefits of such technological developments are now beginning to be seen in the new remote control and monitoring systems being installed. Taking the coal clearance application as an example, outbye delays have been reduced to less than 15 minutes per shift. This results not only from automatic operation but as significantly from corrective action based upon performance reports. In addition to the manpower savings achieved by central remote control a further 20% reduction has resulted from automatic operation by computer.

Control technology is being applied not only to machines, but also, in computer based information retrieval and display, to colliery management enabling them to become part of the control loop. To achieve the full benefits it is not sufficient to provide new equipment, although even that step may be justified by manpower savings alone; management must take determined action to use the new tools; they must set targets, measure performance, compare results with objectives and apply corrective action. Attention is becoming focussed on the management fuction to achieve the continuing improvement the industry needs.

While the first steps have only recently been taken, evidence now accumulating shows that a development programme of automation and comprehensive monitoring will be an important contribution to improving the performance of the industry.

Acknowledgements

In preparing this paper the authors have drawn on the work of many of the Engineers and Scientists at MRDE. This work would not have proceeded had it not been for the co-operation of their colleagues at Collieries where much of the practical demonstration of Remote Control and Monitoring is in progress. The authors gratefully acknowledge this co-operation.

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End of Shift Report - Log Period from 7.00 to 13.30 Friday 18 February 1977

Performance Summary

Unit	Del. No.	3 Mins < Mins (cum)	Del. No.	3 Mins > Mins (cum)	Delay Mins	Run Mins	% Lost
East Trunk	3	5	0	0	5	385	1
Output Conveyor	4	8	1	5	13	377	3
Input Conveyor	1	2	0	0	2	388	0
North Trunk	5	7	0	0	7	383	1
6's Main Gate	9	14	1	4	18	373	4*
Sth Trunk Conveyor	2	3	0	0	3	388	0
8's Main Gate	2	2	2	15	17	374	4*
outh 5 Foot	2	2	1	6	8	383	2
0's Main Gate	13	24	2	11	35	356	8*

Gate Belt (*) Delay Index = 5

Delay Analysis

By Time	LOC STP	SUR STP	CHU	LIN	SLP	SEQ	TOT
Shuttle Belt							0
East Trunk	3	2					5
East Trunk Tandem	1					5	6
New Main Conveyor	1					10	11
Output Conveyor						13	13
Input Conveyor	2						2
North Trunk	4		1			1	6
28's Trunk C/V						7	7
28's x C/V						8	8
6's Main Gate	2					6	8
Sth Trunk Conveyor					2	1	3
38's Main Gate	12					5	17
Sth Trunk Tandem	16	6				3	25
Sth Minge Return Dev	91	2				27	120
South 5 Foot	7					1	8
14's Main Gate	144	25				7	176
12's Main Gate	80			71		6	340
10's Main Gate	20		4		2	8	34

By Number	LOC STP	SUR STP	CHU	LIN	SLP	SEQ	TOT
Shuttle Belt							0
East Trunk	2	1					3
ast Trunk Tandem	1					3	4
ew Main Conveyor	1					4	5
utput Conveyor						5	5
Input Conveyor	1						1
North Trunk	2		1			1	4
28's Trunk C/V						4	4
28's x C/V						4	4
6's Main Gate	1					3	4
Sth Trunk Conveyor					1	1	2
38's Main Gate	2					2	4
Sth Trunk Tandem	5	1				1	7
Sth Minge Return Dev	6	1				7	14

By Number (cont'd)	LOC STP	SUR STP	CHU	LIN	SLP	SEQ	TOT
South 5 Foot	2					1	3
14's Main Gate	4	1				2	7
12's Main Gate	7			6		1	14
10's Main Gate	8		2		1	2	13

Delay Log

Unit Delayed (*)	FROM	TO	DURATION MINUTES	CAUSE	UNIT RESPONSIBLE
Input Conveyor	8.09	8.11	2	LOC STP	*
North Trunk	7.56	7.57	1	SEQ	*
	8.10	8.11	1	LOC STP	Input Conveyor
	8.19	8.20	1	CHU	*
	9.28	9.29	1	LOC STP	*
	10.06	10.09	3	LOC STP	*
6's Main Gate	7.56	7.57	1	SEQ	North Trunk
	8.00	8.01	1	SEQ	*
	8.09	8.11	2	SEQ	*
	8.19	8.20	1	CHU	North Trunk
	9.28	9.30	2	LOC STP	North Trunk
	10.06	10.09	3	LOC STP	North Trunk
	10.53	10.57	4	SEQ	*
	11.12	11.13	1	SEQ	*
	12.36	12.37	1	SEQ	*
	12.55	12.57	2	LOC STP	*
38's Main Gate	7.58	8.09	11	LOC STP	*
10's Main Gate	8.09	8.11	2	SEQ	*
	8.20	8.21	1	LOC STP	South 5 Foot
	8.22	8.29	7	LOC STP	South 5 Foot
	8.55	8.59	4	LOC STP	*
	9.06	9.09	3	LOC STP	*
	9.16	9.17	1	SEQ	*
	9.32	9.33	1	SEQ	*
	9.47	9.50		LOC STP	*
	9.47	9.50	3	CHU	*
	9.51	9.52	1	LOC STP	*
	9.55	9.56	1	CHU	*
	11.15	11.17	2	LOC STP	*
	11.20	11.22	2	SLP	*
	12.23	12.25	2	LOC STP	*
	12.28	12.31	3	LOC STP	*
	13.17	13.19	2	LOC STP	*

Fault Log

Unit	FROM	TO	DURATION MINUTES	FAULT
North Trunk	8.10	8.11	1	CHU
	8.19	8.20	1	CHU
Sth Trunk Conveyor	12.03	12.05	2	SLP
South 5 Foot	8.10	8.11	1	CHU
2's Main Gate	7.00	8.20	80	LIN
	8.23	12.52	269	LIN
	8.33	8.37	4	CHU
	8.57	8.58	1	CHU
10's Main Gate	9.36	9.37	1	SLP
	9.47	9.48	1	CHU
	9.49	9.50	1	CHU
	9.55	9.56	1	CHU
	11.20	11.21	1	SLP

FIGURE 3 SHIFT REPORTS

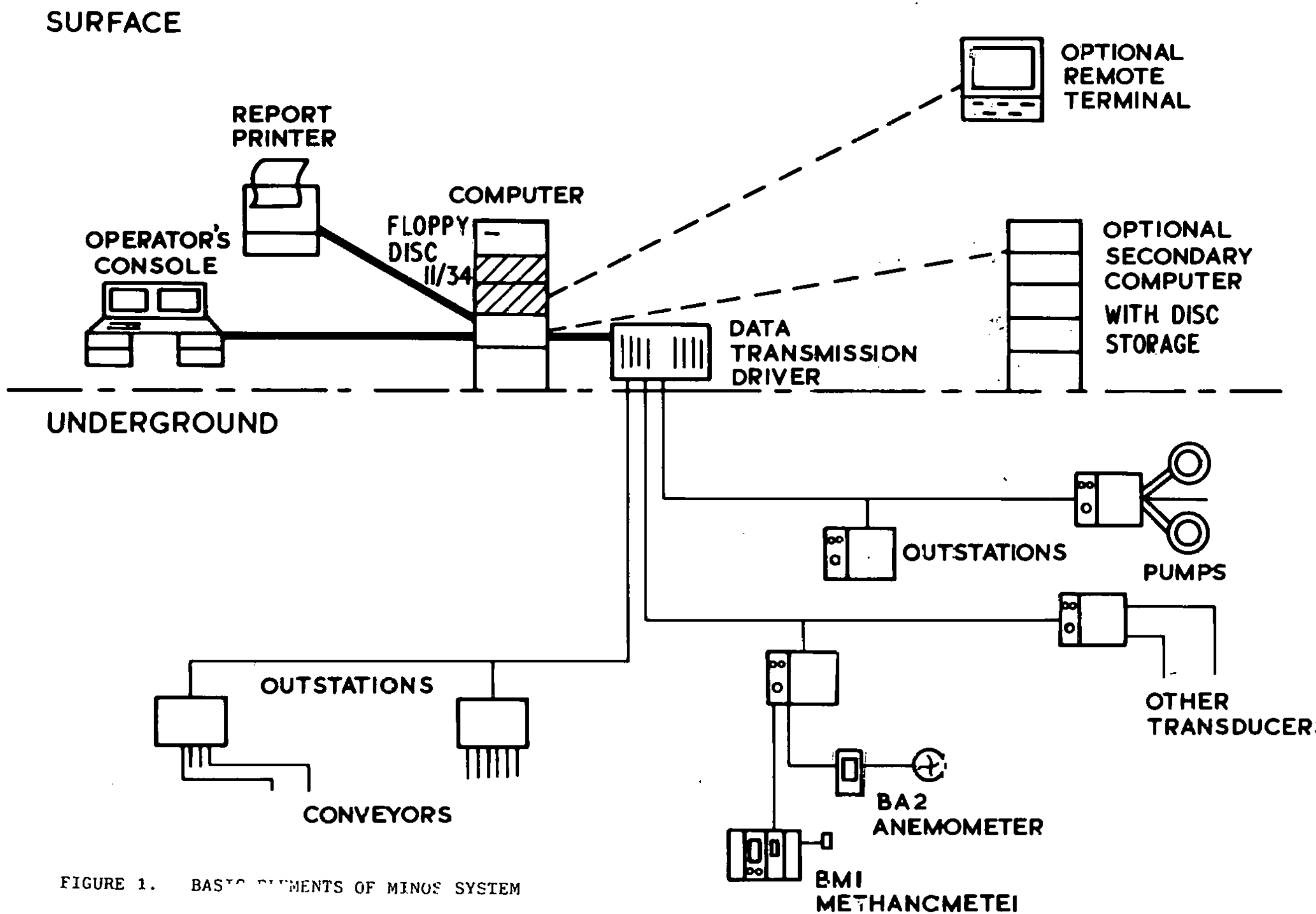


FIGURE 1. BASIC ELEMENTS OF MINOS SYSTEM

228.

FIGURE 2.   OPERATOR'S CONSOLE

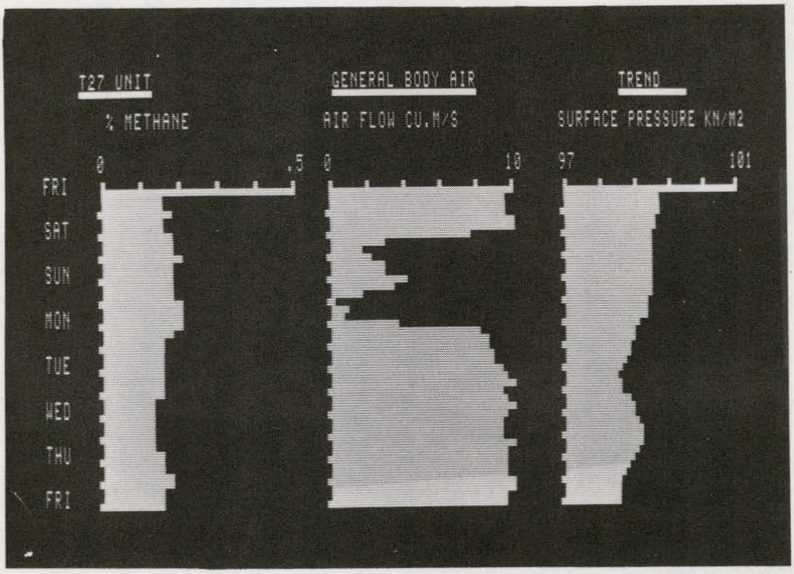


FIGURE 4. ENVIRONMENTAL DISPLAY - TRIPLE HISTOGRAM

"Automating Longwall Processes"

Anthony Sharkey
Staff Engineer

U. S Bureau of Mines
Washington, D. C. 20241

The major objective of the Advancing Coal Mining Technology (ACMT) Program of the United States Bureau of Mines can be defined as follows:

"Through research, development, and demonstration to provide the coal-mining industry with the technology to minimize the national average cost per ton of coal produced to meet the targets set by the Federal Government and consistent with the need to improve coal miner Health and Safety, resource recovery and environmental impacts."

A considerable part of the Program effort is devoted to longwall mining and related activities and seeks to exploit longwall's known advantages over alternative mining systems; to removing constraints to its application where these can be identified and to further improve the production potential of the system by the development and application of advanced technology to the process.

Among the many advantages of the longwall system is its relatively simple, repetitive production process which is conceptually capable of being adapted to a high level of remote control and possibly to automation. The automation of longwall processes offers a number of operational advantages; these are detailed later. Part of the funding of the ACMT program is therefore devoted to the development of the means for achieving partial or total automation of the longwall mining system. The degree of automation eventually achieved will be dicated by the cost-effectiveness of the developed technology in terms of its acceptance by the coal mining industry. Any new technology must be evaluated against that which it is expected to supersede and in the progression to automation a succession of steps can be conceived with each step representing a technological advance. In order to cost-effectively evaluate the progression, a baseline of performance must be established against which "improvements" can be measured and the major objectives re-evaluated.

Stereotype Longwall Mine

To quantitatively measure the expected benefits from various system improvements, a model has been constructed of a longwall mine. Available data has been used but at this time insufficient is available to claim that the model is "typical" and the description "stereotypical" is preferred. As more data on performance and costs becomes available, it will be used to progressively improve the model. The stereotypical mine produces one million tons per year from two longwall faces and two continuous miner sections, the latter being used primarily for longwall panel development. The "mine" is worked on two shifts per day, each longwall producing 924 tons per shift and each C.M. section 300 tons per shift. Overall cost per ton is calculated to be $16.01 and cost details of the mine are given in Table 1. The performance of the longwall faces is based upon:

(a) a shearer utilization factor per shift of 37% (utilization in this sense is the time during which the shearer produces coal, expressed as a percentage of shift available time)
(b) a seam thickness of 79 inches (6.6 feet)
(c) a face length of 500 feet
(d) a shearer haulage speed averaging 10 feet/minute
(e) a web of 30 inches
(f) 20 days per panel spent in tranferring equipment to a new face or 19 days per year per panel.

Table 1

General Cost* Details of Stereotypical Mine

CATEGORY	COST PER LW SECTION $	COST PER CM SECTION $	GENERAL MINE COST $	WHOLE MINE COST $
Equipment Costs	2,398,500	674,700	30,426,000	36,572,400
Depreciation	327,938	92,770	1,876,180	2,717,596
Taxes and Insurance	48,060	13,480	608,520	731,600
Materials $/Ton	2.60 $/T	2.60 $/T		2.60 $/T
Power	.09 $/T	.09 $/T	189,912	189,912 + .09 $/T
Equipment Replacement	___	___	200,000	200,000
Labor	350,851	308,259	2,347,821	3,666,041
Payroll Overhead	140,340	123,304	939,128	1,466,416
Indirect	52,628 + .39 $/T	46,239 + .39 $/T	352,173	549,907 + .39 $/T
Union Welfare	55,700 + .78 $/T	50,130 + .78 $/T	297,067	508,723 + .78 $/T
Land $/Ton	.39 $/T	.48 $/T	___	.39 $/T
Interest	118,934	33,428	1,509,036	1,813,760
Annual Operating Costs	1,094,451 + 4.25 $/T	667,611 + 4.34 $/T	8,319,837	11,843,961 + 4.25 $/T
Average Shift Production Tons	924	300	___	2,448
Annual Production Days	201	220	___	201/220
Section Annual Production Tons	371,448	132,000	___	1,006,896
Total Cost/Ton	___	___	___	16.01

* Costs are based largely on those available in "Minimum Cost Strategies for Longwall Equipment Moves," NTIS PB 267 764/AS, and Bureau of Mines Report 1C 8715, "Basic Estimated Capital Investment and Operating Costs for Underground Bituminous Coal Mines Developed for Longwall Mining." Longwall Supports cost based on $3,000 per foot of face.
Wage costs are based on current agreements.

Any changes in the longwall system must be evaluated in terms of its effect on the whole mine operating costs. To be considered cost-effective any increase in the cost of face equipment must increase the tons produced (by increasing effective machine utilization) so that whole mine costs are maintained constant or improved. The 1977 cost of equipping a "normal" longwall face varies approximately from $2.2 million to $4 million and progress towards automation will almost inevitably lead to higher initial costs. For example, assuming the face supports equipment to be capable of modification (which may or may not be the case) it is estimated that a control system for the supports based on a headgate console will cost almost $1 million and to this must be added the cost of numerous sensing devices to determine the system's status if safe and efficient operation is to be achieved. No firm estimate can be given at this time of the eventual increase in face capital costs necessary to achieve full automation. Figure 1 has been prepared to illustrate the effect of increased face capitalization on the minimum required face output (or machine utilization) if whole mine costs are to be maintained at the present level of $16.01 per ton and mine profitability is to be maintained or improved. The graph indicates that for every additional $500,000 of expenditure on face equipment there must be an increase in production of 26.2 tons per shift or a 2.84% improvement in machine utilization. The model must be refined to indicate in addition to output requirements from the faces, how the effect of such increases in output impact on panel life and hence on development requirements and haulage capacity. These factors will in their turn require increased capitalization and a further consequent rise in the required tonnage to maintain existing cost standards. The model can of course be used to indicate the level of justifiable expenditure to eliminate or reduce identifiable delays. Development of the model is proceeding.

Potential Advantages of Longwall Automation

Neglecting any improvements in gross tonnage and costs which have yet to be demonstrated and discounting Health and Safety advantages in taking personnel out of the potentially hazardous face area, remote or automatic control of the longwall processes would offer,

A. In the case of the shearer

1. Speed of machine operation would not be limited by an operator's walking or crawling speed, a factor which gets progressively more important as the extracted height gets less.

2. Taking the man out of a "hands on" control situation frees the system design engineer from the constraint of man's restricted mobility. Shearer cutting speeds in excess of 30 feet/minute are currently possible but in present circumstances (i.e. with direct manual control) little purpose would be served in seeking to achieve higher speeds even though no technical reasons appear to exist why machines cannot be developed to cut and load at much higher speeds.

3. Automatic controls will provide more precise cutting and a potentially cleaner product. Better face alignment and more consistent roof and floor horizons will be obtained, these in turn contributing to more effective strata control and less delays due to falls of ground.

4. Accurate sensing of the thickness of coal left to form a stronger roof will lead to considerable improvements in coal recovery. Present manual methods of steering in such conditions involve a degree of estimation. Often extra inches of coal are left because of the imprecise method and in order to provide a safety factor against error. Every vertical inch of coal represents 5,000 tons in a longwall panel 3,000 feet x 500 feet and unnecessary losses of 4"/6" are not unknown.

B. In the case of Supports

1. Speed of operation of the supports along the face would be governed by the transmission of an hydraulic or electrical impulse and not by an operator's walking or crawling speed; an important factor, in any case, but one which gets progressively more important as the extracted height gets less. The manufacturers of the Dowty Electro-Hydraulic Supports Control System claim a total cycle time per support (lower, advance, reset) of 8 seconds. This, with 5 feet center spacing of supports equates to an advance rate along the face of 37.5 feet/minute when operating one support at a time. Speeds of advance greatly in excess of this can be obtained by multiple advance which is technically possible within present "state of the art" knowledge.

2. Push button controls would provide closer control over setting pressures and ram-stroke, leading to more consistent roof and floor loading, better face alignment, consequently better strata control and reduced delays from falls of ground.

3. Bi-directional control of the supports would offer improvements in machine utilization by making possible the application of bi-directional cutting in situations where it cannot now be practiced because of respirable dust production from the cutting machine.

C. In general

1. Remote or automatic control should lead to improved results by enabling equipment to be operated more closely to designed performance. Increases in efficiency (i.e. higher average production rates) and a reduction in mechanical breakdowns will result.

2. Valuable "spin off" technology is likely to result from the work. The value and amount of such technology cannot be predicted and arguments for the pursuit of advanced technology based solely on probable "spin off" have little strength. Results can be used "after the event" to lend weight to the positive side of the accounting when examining cost/benefits. N.A.S.A.'s Aerospace R&D program is perhaps the best known example of valuable technological "spin off". The B1 Bomber project is another example but in mining terms the "Remotely Operated Longwall Faces" (R.O.L.F.) of the National Coal Board in Britain in the mid 60's although never producing coal in any large quantity resulted in the development of,

   (i) Automatic cable handling systems which since their introduction as standard longwall equipment have saved countless hours of down time caused by damaged electrical cables in addition to savings in replacement cable costs and the cost of cable transportation.

   (ii) Static ramp plates which improve face-side clean up and gain valuable extra inches of advance and increased production on the many faces on which they are now standard equipment.

   (iii) The development of pre-start warning devices for face equipment which have proved to be a considerable safety improvement.

   (iv) The development of face signalling, communications, and lock-out systems which are now standard equipment on all but a few of the world's mechanized longwall faces. By the use of such equipment, delays and losses have been minimized and safety standards substantially improved.

Achieving Cost-Effectiveness with Longwall Automation

The output from the automated face must be increased at least to a degree consistent with the cost of investment if automation is to be justified. Since output from the face is directly related to effective utilization of the cutter loader this latter factor can be used as the measure of "effectiveness." In general, machine utilization can be improved in any or all of the following ways,

(a) By better utilizing available time

(b) By making more time available

(c) By better utilizing machine haulage speed capability

(d) By improving machine haulage speed capability

[The output from a face might also be improved by the use of additional shearers but two or more machines on a longwall face will compound the already difficult problem of remote control or automation of the face operations. This method for potential improvement although conceptually possible is not considered further in this paper]

Considering each of the ways for effecting an increase in machine utilization,

(a) Better utilizing available time

Table 2 is a summary of available data on longwall shearer utilization in terms of time.

Table 2. Longwall Shift Utilization

			Minutes	Percentage
Cutting Time			139	37
Turn Around Time			20	5
Cleaning Time			35	9
Maintenance Time			27	7
Down Time -	External Haulage	43		
	External Power	8		
	Cutter/Loader	34		
	Face Conveyor	36		
	Supports	8		
	Geologic	8		
	Others	20		
	Total Down Time	157	157	42
Available Working Time			378	100

Considerable improvements can be effected by direct management action and t is difficult to quantify what additional reductions in non-productive time ight be achieved by automation. Some of the delays caused by improper setting f supports, imprecise cutting and misalignment of the conveyors will probably be reduced but to an unpredictable extent. Taking the operators out of the "control loop" should also prove to be beneficial and result in some reduction in non-productive time but once again the degree of improvement cannot be predicted. Bi-directional cutting which becomes possible with remote control will save time but against the saving must be set the increased "end" time associated with the method. Because of the large increases in instrumentation and control sophistication it is indeed possible that automation will cause an increase in the incidence of delays which might collectively be sufficient to offset the improvements otherwise gained in the utilization of time. The development of reliability will need at least as much attention in the field of automation as it does elsewhere.

(b) Making more time available for production,

Means for achieving this objective are well-known and include working more shifts per day, more shifts per week, reducing the effect of equipment transfers between worked out and replacement panels and reducing travel time at the start and finish of shifts. These improvements can be made by direct management action and do not depend on the development of new technology, including face automation.

(c) Better utilization of machine haulage speed capacity,

The output from the longwall faces in the "stereotypical mine" as shown in Table 1 is based on the utilization factors listed in Table 2, but is also conditioned by the assumption of an average shearer haulage speed of only 10 feet/minute when coal is being produced. There are many factors which influence the cutting speed of a longwall shearer, including coal hardness, conveyor capacity, and geological anomalies, but the major controlling influence is the machine operator. No data is available to indicate the extent to which designed haulage speed capability might be more nearly achieved by the elimination of "hands on" control, but it is probably considerable. Studies are currently being conducted for the Bureau in an attempt to quantify this effect. Increasing the average cutting speed of the shearer from 10 feet/minute to 11 feet/minute (an increase of 10%) is perhaps easier to visualize than reducing face down time from 157 minutes per shift to 144 minutes per shift (a reduction of 8%). A 3.3% improvement in face production would theoretically result from either improvement.

(d) Improving machine haulage speed capability,

Except in thin seam conditions where space restriction limits the size of equipment, there appears to be no technical reason preventing the development of shearers with greatly increased power and cutting capability. There will be constraints against the full exploitation of machine design capability with the most immediate being probably the maximum haulage capacity of the present generation of face conveyors. Figure 2 relates seam height, web depth, and shearer haulage speed with output. The constraint of the face conveyor is immediately apparent, particularly as web depth and extracted heights increase. However, the exploitation of cutting speed capability up to maximum conveyor capacity would represent a tremendous improvement on present performance (except in isolated cases). Work is to be done on behalf of the Bureau to properly identify the constraints against full exploitation of shearer design capability and the measures which will be necessary to remove the constraints.

Summarizing the ways whereby effective longwall machine utilization might be improved by automation, it is probable that a more significant impact will be obtained by exploiting shearer cutting capability than by reducing "down time" and this appears to be the situation in both the long and short term. Efforts to reduce production losses must of course continue, but the means to do this are, by and large, already available to management and the impact of automation in this area is difficult to predict. It would be very difficult currently to justify a major Research and Development effort into longwall automation on the basis of a speculated reduction in down time.

An early appreciation of the probable impact of face automation on mining systems and labor requirements will be needed by management, manufacturers, and research and development organizations if full advantage is to be taken of increased production potential as it becomes available. Answers must be sought to problems arising out of the use of more sophisticated and complex control equipment, reduced panel life, increased peak haulage demand, and in many cases increases in rates of methane emission. Such problems, together with those posed by the different skills required of systems control personnel, will probably be neither quickly nor easily solved.

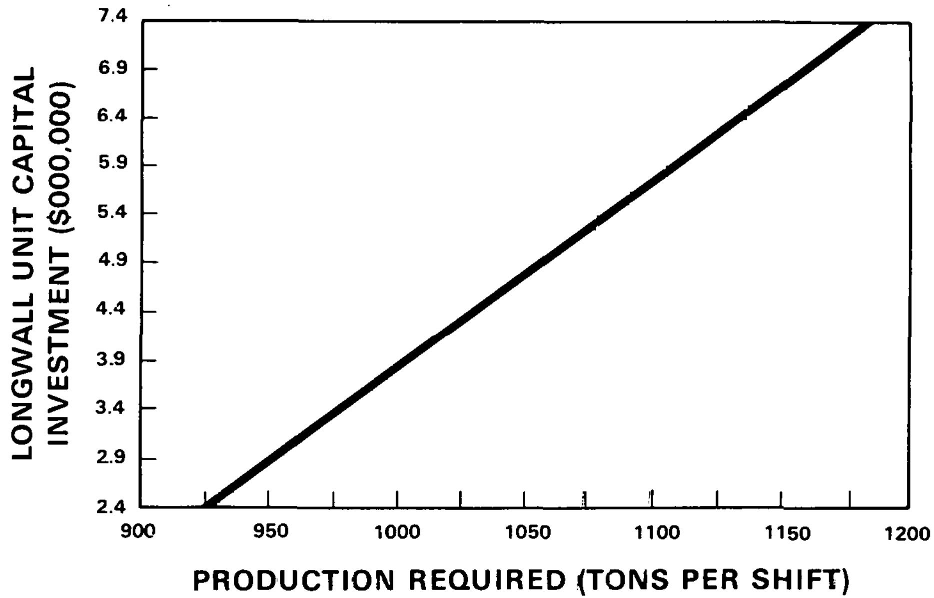
A precise and accurate definition is needed of the nature of potential problems and their probable constraining effect on system performance. The levels of performance at which they do become constraints to further improvements must also be defined. By such definition a clear order of the relative importance of the constraints can be established and a systematic approach to solutions formulated. For example, it can be seen from Figure 2 that the haulage capacity of current designs of face conveyor becomes a constraint at a production rate of about 20 tons per minute; if methane emission is unlikely in given circumstances to constrain production at rates below 25 tons per minute there is little immediate value in pursuing solutions to the methane problem unless known improvements to conveyor design will potentially allow this rate of production to be exceeded.

Figure 3 illustrates a conceptual succession of events leading to a demonstration of a Very High Output Face. Dates have been deliberately excluded because of the numerous uncertainties along the path but it is confidently hoped that such a demonstration can take place by the end of 1985. The achievement of this objective depends on it being demonstrably contributive to the stated major objective of the Advancing Coal Mining Technology Program. The means whereby the benefits of automation can be exploited on the way towards the Very High Output Face are now apparent and they will become a reality only if accompanied by acceptable cost. The development of the mine cost model by the collection and use of further basic data is considered to be a suitable tool for monitoring these events.

Acknowledgement

The author wishes to record his appreciation of the help given in the development of material for this paper by Dr. Lewis V. Wade and his staff in the Roof Support Group of the Bureau's Mining and Safety Research Center at Pittsburgh, Pennsylvania.

FIGURE 1. OUTPUT PER FACE RELATED TO INCREASED INVESTMENT ON EQUIPMENT



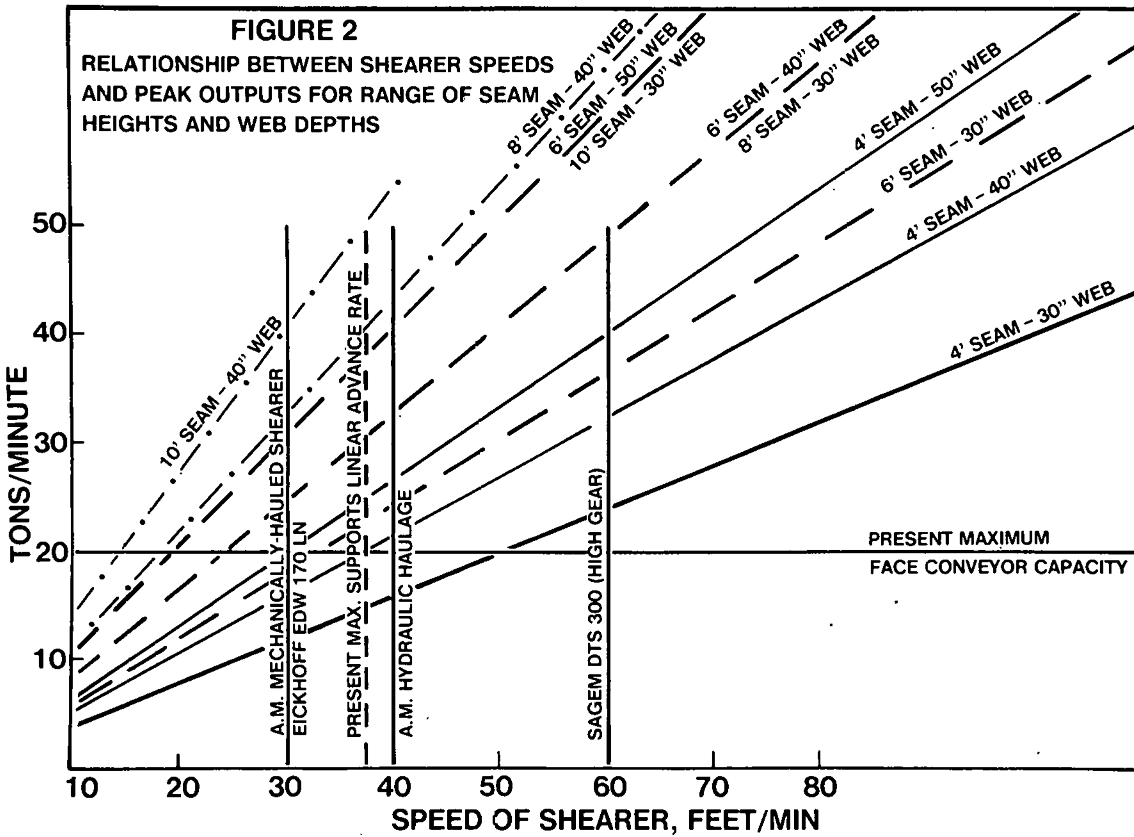


FIGURE 2
RELATIONSHIP BETWEEN SHEARER SPEEDS AND PEAK OUTPUTS FOR RANGE OF SEAM HEIGHTS AND WEB DEPTHS

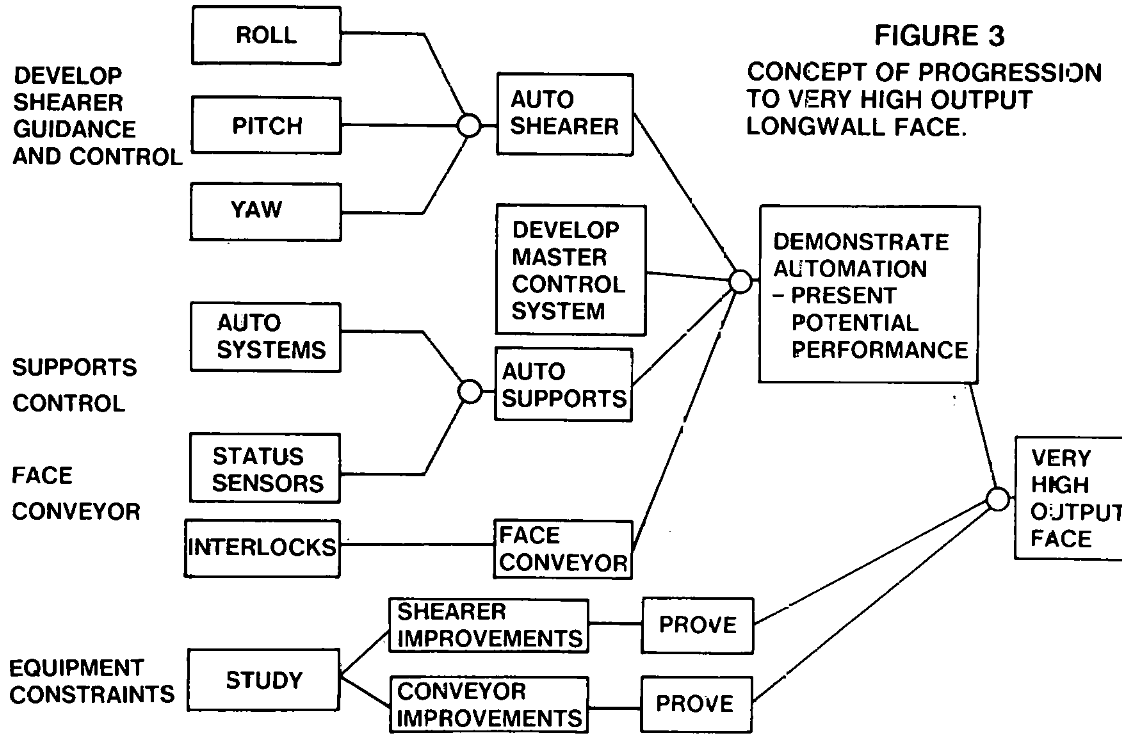


FIGURE 3

CONCEPT OF PROGRESSION TO VERY HIGH OUTPUT LONGWALL FACE.

DEVELOPMENTS FOR AUTOMATION IN UNDERGROUND MINING IN GERMANY

Jörn Olaf
Dr. rer. nat.
Bergbau-Forschung GmbH, 43 Essen, Germany

Automation of coal winning systems

The winning system with coal plough has priority especially in the German mines, because it is working very well in seams of 3 to 5 feet thickness - the main seam thickness in Germany. The coal plough system is characterized by the remote pulled iron plough with a travelling speed up to 2.5 m/sec. (about 8 feet/sec) and a cutting depth depending on the hardness of coal and varies therefore in the same seam from one point to another. Therefore you can lose the prefixed direction of the face very easily. For moving the face support in steps of 40 to 50 cm (1.5 to 3.0 feet) we need some cuts of the plough and a following very fast moving of the support to avoid the break down of the roof.

The automation of the sequence of operations is the main work for an increase of the production time of the coal plough and the increase of face-output.

Therefore the steering-system comprises the working sequence of the coal plough:

- the precise approach of the face ends
- the exact reversing of the travelling direction at pre-fixed points
- the monitoring of the power input in order to avoid chain raptures or failures by overloading
- the interconnection between coal plough and face conveyor and

- the control and maintenance of the determined direction of the coalface which can be done by a local or sectional operation of the coal plough and regulation of the hydraulic flow through the rams. As a further problem, there is still the control of the cutting horizon of the coal plough in the seam.

The effort to integrate these control and monitoring functions into a compact control unit, led to the development of a control pannel for the coalface. It is built up in three system groups for steering and monitoring functions in consideration of the ergonomic conditions (figure 1).

The necessary instrinsically safe electronics required for an automatic sequence of operations are built up modular.

For monitoring of the actual directions of the face mechanical systems are mounted at different points to the face conveyor to measure the advance of the conveyor. Each of this systems containes a drum with about 2 000 feet of a wire, the end of which is fixed at the stone behind the face support. Figure 2 shows the mounting of this device on the face conveyor. In the upper part of this picture a switching device for the coal plough with a high-frequency radio transmitter at the plough and a receiver fixed at the conveyor near the face end are shown.

For indication of the actual position of the plough pulses from the movement of the plough are used. Fluid crystal displays are used for a digital indication and a great number of luminescence diodes are mounted for the analog indication of the plough position. Actually seven of these control installations are working underground in German mines.

The considerable extent of the electronic equipment led to the idea to control the entire sequence of operations by a process computer, which, however, has still to be on the surface with regard to the present regulations for mine safety. The use of a process computer on the surface demands an additional control and monitoring panel underground for an emergency case of a break-down of the transmission lines.

The present development of the computer industry allowes to hope to solve the entire problem within the next future by means of micro-computer to be installed underground. Figure 3 shows an intrinsically safe micro-computer developped by Bergbau-Forschung.

<u>Optimation of Mining Operations Underground</u>

Beyond any automation of individual operations, there is a need to ensure that all of these operations as coal winning, road conveying and haulage, shaft winding and bunkering smoothly match with one another, taking also into account certain aspects of underground environment, e.g.

- development and drainage of methane
- mine temperature and mine climate
- mine ventilation
- and energy supply

The purpose of optimising all these operations is to arrive at a maximum working time of the coal winning equipment together with an optimum loading of coal without interruption of the means of conveying and haulage. The technical system required for accomplishing this task can only consist of an as comprehensive as possible monitoring of all the operational sectors and of a rapid systematic processing of the data obtained. To this end will be indispensable the use of both a process control computer as a master computer for the optimation of all operational sectors, as well as "mini"-or "micro-computers" which are a kind of "satellites" collecting data and accomplishing specific control functions underground.

<u>Datatransmission systems</u>

An efficient data transmitting system is the necessary link between these underground substations or satellites and the master computer at the surface. The great number of data, information, and measuring values to be transmitted from underground has initiated some time ago a wide use of cable-saving transmission techniques. In view of practical requirements the mining companies have preferred to recur to the audio frequency multiplex technique which consists of allocating a potential of a specific frequency to each information to be transmitted. In this way a multitude of audio frequency transmitters and receivers located in the substations are able to transmit in parallel - i. e. simultaneously and independently - say 24 informations through one single two-core cable. The main pros of the audio frequency multiplex technique are that the systems are economic already with relatively few signals per substation and, secondly, that planning and servicing of the equipment is facilitated thanks to the transparency of functions.

As far as the time multiplex technique is concerned, the informations are transmitted as pulse telegrams with sequential transmission of the various information units. The main advantages of the time multiplex technique are its special suitability for transmitting a multitude of signals per substation and a transmission free of disturbances thanks to additional transferring of check characters.

The selection of a specific transmission technique is essentially dependent on the volume of information to be transmitted, to which add certain other criteria so that one may come to the following conclusions:

1) It cannot be reasonable, neither from the technical nor from the economical point of view, to use more than one frequency multiplex system for parallel data transmission from a collecting substation to some master station

2) Not all of the collected operational data is useful to the master station

3) Some informations are required at more than one working point underground.

4) Some data are important enough to require priority and prompt transmission

A transmission system based on time multiplex makes it difficult to meet the requirements under item 4) in as much as the regular scanning cycle may get interrupted unacceptably often by priority recordings.Meeting the requirements of item 3) is not without problems either because of the need to retransmit certain data from the master station to any of the substations.

With the frequency multiplex technique, the requirements of items 3) and 4) can be met easily. In view of the big data volumes to be dealt with, however, only high frequency systems with ample channel capacity will be suitable. Although, on the other hand, these may possibly create new difficulties due to the need of laying coaxial cables into the face areas.

Trying to find a solution for item 2) we thought of processing the data as close as possible to the point of their measurement. The result of our considerations was a two-part data transmission system. The first part covers the section between data collection and substation. The second part covers the data transmission between the substations and thence to the master station. Since in this case greater data volumes have to be transmitted simultaneously, time multiplex systems (under the reservations already mentioned) or the high frequency system with a sufficient number of channels are the appropriate solutions. The latter system also allows an additional input of specific data along the transmission line.

One has to bear in mind, too, that with "active substation" systems - i.e. substations accomplishing process control functions by themselves - there is an increasing need for transmittung control commands from the master station to the underground stations, e.g. for switching on/off pumps, cable phases, mine fans etc. As these are nearly always specific commands which have to go to specific places, the use of a time multiplex system seems to be less appropriate here, too.

A summary of the evolution of remote monitoring devices from 1972 through 1976 is given in the table, with the respective numbers of monitoring channels being in service in the German

Coal Mining Industry. It should pointed out to the above-average increase of remote control functions, - a tendency which is liable to continue.

The evaluation of data gathered in the pit control centers covers a great number of informations in the form of status reports, meter indications, recorder graphs, and indications of measuring data. This kind of data display allows, in fact, some influencing of operations which is the more effective the more rapidly data are evaluated.

With on-line data collecting, the recording device is substituted for by the terminal multiplexer of a data processing unit. The real-time system docs not only collect the data on the spot and at the moment of their measurement, but processes them immediately. The most recent underground resulte are ready to be called-off and may eventually be used as an aid to management decision. This is, in fact, the basis for any optimation of mining operations.

The first section of the report dealt already with the details of an automation of coal faces, especially of coal ploughs. Here the first so-called substations, i.e. the micro-computers mentioned earlier, have been installed which receive their commands from a master station and in turn have to retransmit the significant characteristic data to the master station. Similar systems are being developed for coal faces with shearer-loaders.

In the immediate context with coal winning, the optimation of underground coal transport seems to be rather promising. With regard to main road haulage, systems for optimising the haulage performance have already been tested successfully.

Optimation of Continuous Conveying

Continuous conveying systems consist of a multitude of successive and branching conveyor belts. Relatively small bunkers, e.g. spiral chutes or automatic horizontal bunkers, serve for homogenizing the coal streams coming from the different face areas. The coal is then withdrawn from these intermediate storage bunkers a. o. by means of vibrating chutes and thence led on to trunk conveyors. In the event of a badly coordinated transport flow conveyors may become overloaded or bunkers may flow over which would mean transport standstills. On the other hand poor utilization of the conveying capacity may occur as well which would lead to shaft winding standstills due to lack of coal. Rationalization by means of concentrating smaller mines into a few big compound mines creates an increasing need for optimised control of the coal flow. To meet these requirements, the Remote Control and Automation Department of Bergbau-Forschung GmbH, in cooperation with the Ruhrkohle mine "Haus Aden", has set up the programming for a master control system.

The only solution which is generally applicable and at the same time promising from an economical point of view is a central control of the whole conveying system by one pit control center. For this purpose the previous monitoring and remote control systems have to be completed by a system selecting automatically the parameters to be controlled. Especially in the long run, continuous and persistent changes of the main conveying streams will have to be dealt with in every mine, so that a flexible and optimum control is an absolute must.

One has to bear in mind that the coal flow never is continuous. The quantity of coal production is rather subjected to fluctuation by standstills in the coal faces due to equipment break-downs or geological disturbances. At present, control starts only beginning from an intermediate storage bunker which homogenizes the coal flow to some extent before it reaches the main conveying system.

As simulation calculations have shown, the rate of conveyor utilization has improved along with a certain improved smoothing-out of the coal flow in the face areas, although the winning operations are still monitoring and controlled by the pit control center.

If, moreover, conveyor load rates can be measured already shortly behind the face, the coal volume to be received by the conveyor system may be determined before reaching the intermediate storage bunker. Controlling will thus become easier and more effective since the computer is able to determine presumable bunker filling levels. Thus a further improvement of control can be expected.

Structure of the Main Conveying Programming System*)

Fig. 4 shows the programming structure; collection of measuring data and recording are not represented as they are self-evident.

To obtain an optimum utilization of conveyor capacities one has to know not only the bunker filling levels but also the conveyor belt loads prior to calculating the volumes to be withdrawn from the bunkers. It is thus recommended to simulate the conveyor belts in the computer. For such simulation of the coal flow one may recur either to time scanning or to section scanning (fig. 5). For time scanning, conveyor belts are subdivided into sections whose lengths correspond to constant lengths of time, whereas for section scanning belts are subdivided into sections of constant lengths.

The second mode of simulation should be preferred to time scanning since it eliminates speed variations during transitional phases, e.g. starting and after-running of conveyor belts

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*) Prozeßsteuerung von Fließfördersystemen (Process Control of Continuous Conveying Systems), by H. Libuda, D. Sill, J. Steudel, Glückauf-Forschungshefte 35 (1974) 145-49

The realistic simulation of a conveyor belt involves a following up of the real belt movement by the "computer belt image". With the real conveyor belt, this movement is auto-controlled by section pulses. Simultaneously the conveyor load rates determined by the measuring instruments located at the conveyor loading points are allocated to the different belt sections. Now, the "computer image" of the conveyor belt system and of its load rates allows a control of the volumes to be withdrawn from the storage bunkers, on condition that face priorities and possible disturbances are also taken into account and overload checking provided for. The optimation properly speaking consists of predetermining by a paramount strategy the volumes to be withdrawn from the bunkers in a way so as to utilize shaft and bunkering capacities also fully as possible.

When calculating free conveyor belt capacity, one has to take into account besides conveyor load rates also the capacity to be reserved to subsequent bunkers of higher priority and, as the case may be, the load rate of parallel conveyor belts.

A control system is furthermore required for the vibrating chutes through which the coal is loaded out from the bunkers. Triggering is caused by rated value changings. Firstly, the computer sets a quantity allocated to the rated value, as a function of the vibration amplitude. The load-out regulator at the bunker then checks whether the quantity withdrawn corresponds to the calculation: This checking is based on the average of several belt sections. Should the admissible value be exceeded for several successive belt sections, the effective rated value of the volumes to be withdrawn would be reduced by the average of all excess values. If belt overloading at bunker outlets can no longer be avoided by restricting the outlet valves, the feed conveyors will be stopped. The programming schedule is shown on fig. 6. All the conveyor belt loading points are computer-controlled.

The purpose of the storage bunker next to the shaft is to compensate both production peaks and unsufficient belt load rates.

An optimum condition is reached if the total belt load rates equal the shaft winding rate. If this is not the case, the shaft bunker is used as a storage facility for any coal volumes in excess of the shaft winding capacity, up to a given filling level; only when this level is reached the volume of coal on the conveyors must be reduced.

The upper filling level has to be set so as to allow any coal remaining on the feeding belts to be still received by the bunker.

Strategy of transporting different coal types separately on one conveyor belt system

The increasing use of continuous conveyor belt systems involves the problem of transporting several coal types simultaneously and yet separately. A possible solution for automatic belt loading could be some charging method where only defined quantities are withdrawn from the bunkers, thus allowing an integration of any free belt capacities into the overall system. Having reached a defined filling level, the bunkers would automatically request allocation of belt capacity. Then, following up the coal flow, as described earlier, the computer scans the "conveyor belt image" for free capacities and allocates them to the requesting bunker by identifying the appropriate belt sections. If several bunkers request at the same time, allocations are made according to priorities or, if the conveyor belt is fully loaded, by means of a "waiting queue" (fig. 7).

As soon as a feeding belt section with the identification mark of the requesting bunker arrives at the bunker, the predefined coal volume will be loaded out. Another belt section may be allocated to the same bunker as a function both of its remaining storage capacity as well as of free belt capacities.

<u>Process control of trains underground</u>

Similarly to the aforementioned solution, an optimation system*) has been developed for underground rail haulage. The objective is to optimise the train circulation in order to avoid standstills. As a first step towards a solution a prototype was installed which included all functions required for a control system, i. e. track lay-out, bunkers representing the loading points, and the shaft as discharging point. Bunker filling levels, switching positions, levels of coal output at the individual underground districts and the shaft are indicated by pilot lights (fig. 8).

Programming has been set up in a way as to cover any number of trains. The track lay-out is subdivided into sections where pilot lamps indicate the position of trains. Besides, the operational condition, destination, and loaded coal volume of every train are displayed by lamp and digital indications on a master board.

The programming required for controlling the prototype railway system consists of a simulation part and a control part. The simulation part comprehends train circulation, bunker filling levels, train charging and discharging operations. The control part deals with the evaluation of data collected by the simulation part and with the resulting control functions.

The control programme consists of several partial programmes which - although dependent on each other - do not correspond but through a data field. This allows easy exchanging of partial programmes without any need to redesign the whole system in case of changing conditions. The various partial programmes are shown in fig. 9.

The prototype makes ist possible to develop and test programming systems which, due to their modular structure, are adaptable to widely varying underground requirements.

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*) Prozeßsteuerung der gleisgebundenen Hauptstreckenförderung (Process Control of Track-Mounted Main Road Conveying), by D. Sill, Glückauf-Sonderdruck Nr. 15, S. 718

<u>Automatic Monitoring of the environment</u>

One of the most important reasons for remote monitoring in coal mines is the supervision of the environmental conditions underground. Important steps have been done for predetection of mine-fires by automatic monitoring of the Carbonmonoxide. Up to 50 sensor-heads, working by infrared absorption, are installed around the working points and mine gates underground. The data are scanned every minute and presented on a display unit as shown in fig. 10. Together with the presentati... of the values of the last 60 minutes the process computer calculates continuously the tendency of the values. So it is possible to differe between Carbonmonoxid contents generated by mine-fire or by diesel engine or shot-firing.

All these described systems have been developed in the last years and are actually in installation in single mines in Germany. Fig. 11 shows as example the pit control center of the "Haus Aden" mine. The excellent economic result we got with the first prototype allow us to hope to get the first complete control system in the near future.

Development of Remote Control in the Coal Mining Industry of the Federal Republic of Germany

Sector	Measured values		Binary conditions		Control		Total		Increase in %
	1972	1976	1972	1976	1972	1976	1972	1976	
Ventilation	1195	1997	1009	1173	69	137	2273	3307	45
Energy supply	149	299	1150	1199	263	261	1562	1759	13
Mining operations	216	572	3908	6671	126	801	4250	8044	89
Total	1560	2868	6067	9043	458	1199	8085	13110	62
Increase in %	83		49		162		62		

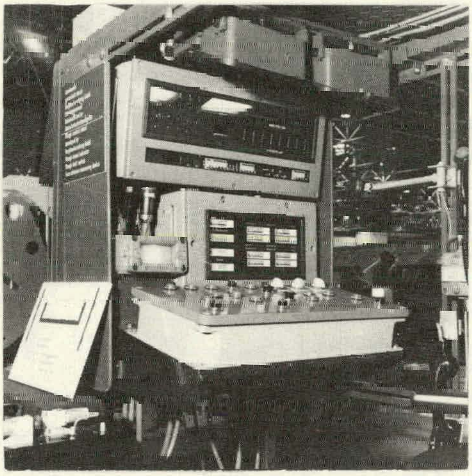


Fig. 1 Face control and monitoring Unit

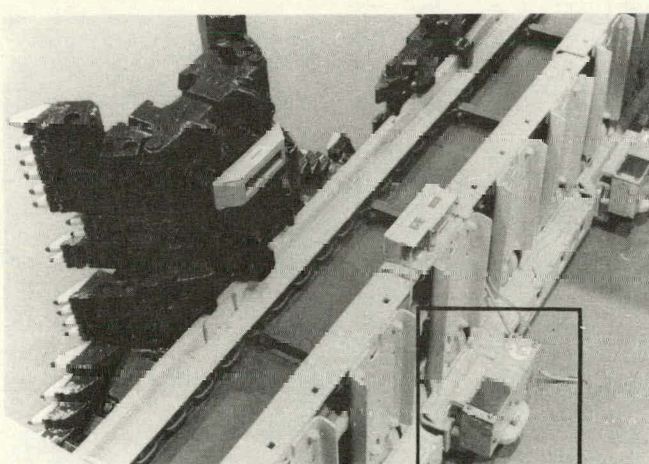


Fig. 2 Advance sensor and position indicator at the plough

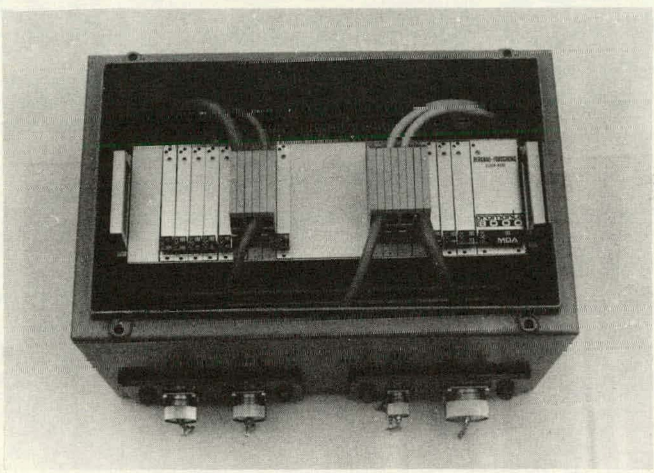


Fig. 3 Microprocessor Promonta 8000

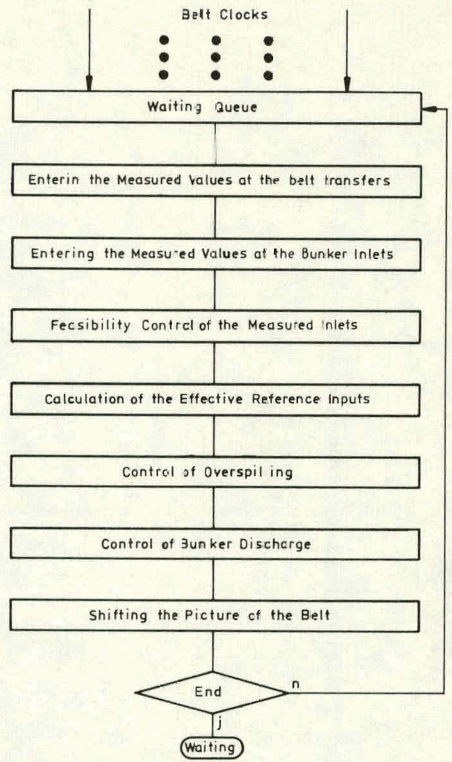


Fig. 4 Structure of :he Program System

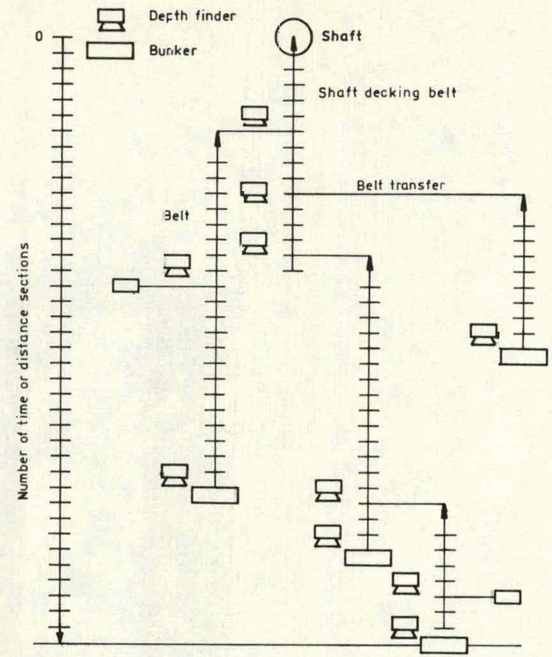
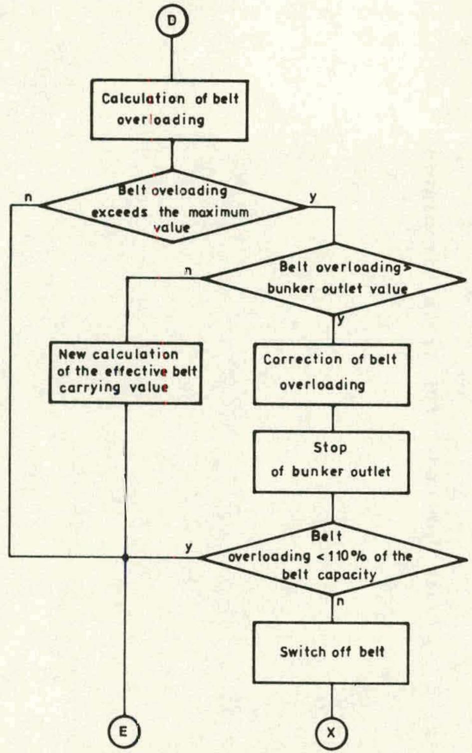
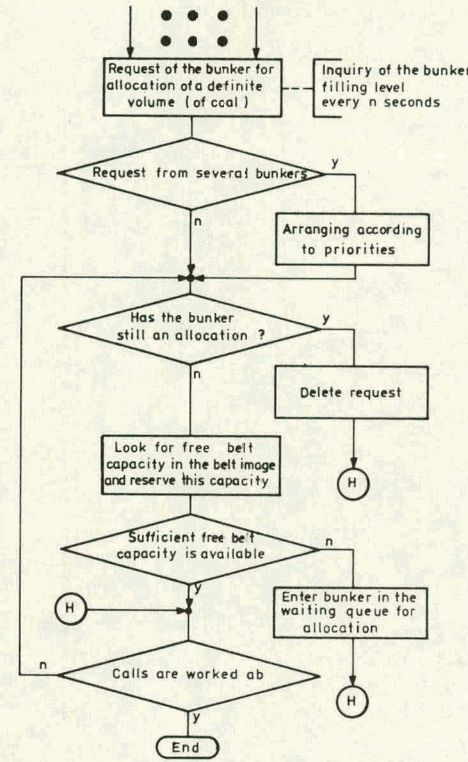


Fig. 5 D stribution of Belt Conveyors in the Time or in the Distance le



D From the control of technical feasibility
E To the control of bunker discharge
X Work up next bunker feed or next belt

Fig. 6 Control of Overloading



H Change to re-entry into the program

Fig. 7 Separate Conveying of Several Types of Coal on the same Belt Conveyor System

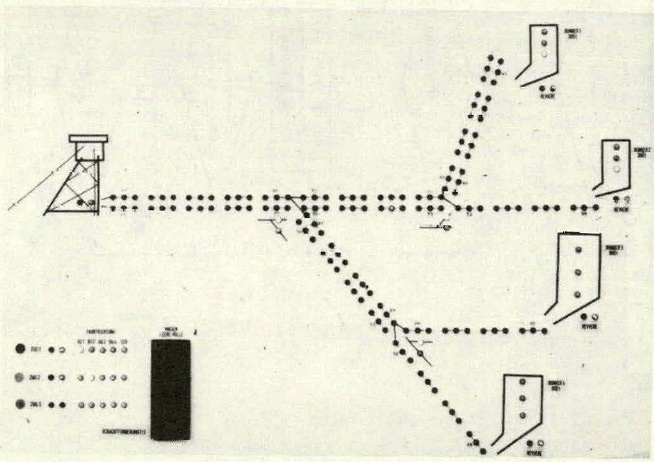


Fig. 8 Simulation model for train circulation

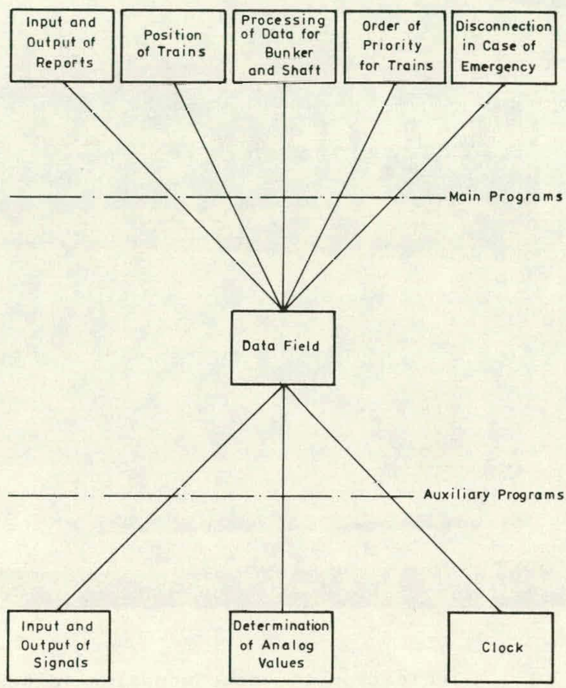


Fig. 9 Program System for Locomotive Haulage

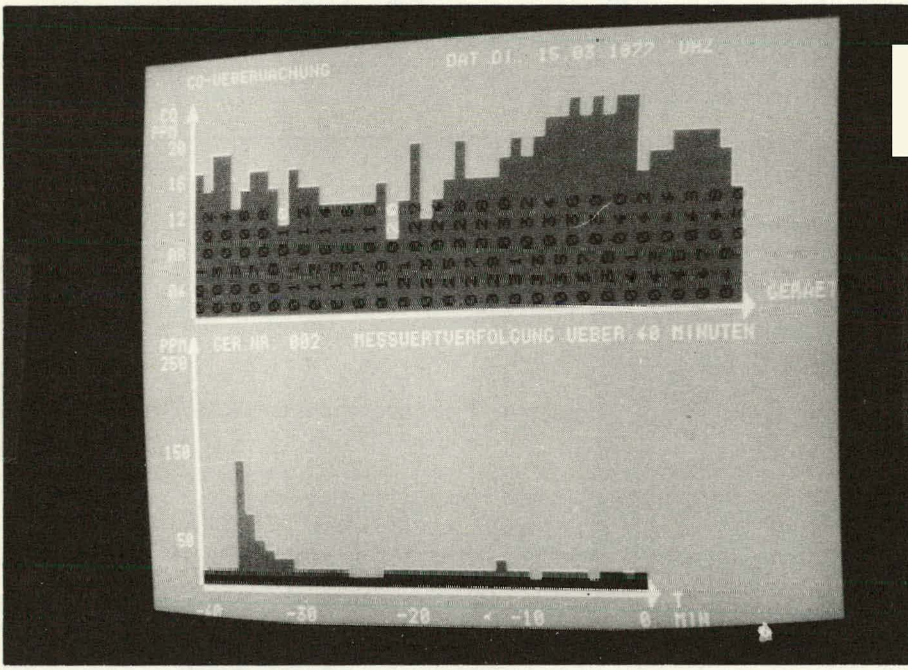


Fig. 10 Indication of carbonmonoxide values on a display unit

AUTOMATED EXTRACTION SYSTEM USING A CONTINUOUS MINER OPERATING RESULTS

Jerry Karlovsky, Jr.
Manager - Development Engineering

National Mine Service Company
Nashville, Illinois

The manuscript of this paper was not received in time to permit its inclusion in the book of preprints. Copies of the paper will be available at the Conference or can be secured by writing directly to the author at the following address:

Jerry Karlovsky, Jr.
Manager - Development Engineering
National Mine Service Company
Mine Road
Nashville, Illinois 62263

SYNOPSIS

ENTRY LAYOUT FOR LONGWALL TOP SLICING OF THICK COAL SEAMS

R. Kenneth Dunham,[1] David O. Wisecarver,[2] and Richard D. Ellison[3]

Introduction

Future demands for coal have been predicted to be for 700 million tons in 1980 to between 1 and 3 billion tons by 2000.[4] The United States presently has a demonstrated reserve of 434 billion tons, the majority of which (54 percent) is west of the Mississippi River. Although current emphasis in the West is on surface mining, more than half of the western reserves can only be recovered by undergr methods. Much of this coal (45 billion tons) occurs in seams greater than 10 f in thickness, often pitching to a greater extent than is common in the East. A___, much of the thick coal lies adjacent to coal planned for surface mining, so that their extension can only be by underground methods.

Recognizing the limited technology and lack of experience in exploiting the thicker seams by underground methods, the United States Bureau of Mines (USBM) is implementing a "Mining Systems for Western Coal" research program designed to develop and demonstrate methods and technology necessary to develop these seams. This synopsis summarizes the results of one investigation carried out as part of this USBM-sponsored program by D'Appolonia Consulting Engineers, Inc. The main objective of the investigation was to develop an entry system to extract coal seams up to 100 feet thick by the longwall top slicing method. More detailed discussion is presented in a paper provided as a handout at the NCA/BCR Coal Conference and Expo IV. A complete discussion of our entire study will be available from NTIS in the near future.

Design Constraints

The basic hypothetical design constraints established by the USBM to satisfy a broad range of western conditions are:

- A 100-foot-thick coal seam, as shown in Figure 2;
- A 10-degree seam dip;
- An overburden depth from 200 to 2,000 feet;
- A four-square-mile area;
- Surface subsidence should be bowl shaped over the majority of the mined area, avoiding sharp surface breaks;
- The coal is young in geological age, of low rank and liable to spontaneous combustion;

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(1) Mining Project Supervisor, D'Appolonia Consulting Engineers, Inc., Denver, Colorado.

(2) Technical Project Officer, United States Bureau of Mines, Denver, Colorado.

(3) Executive Vice President, D'Appolonia Consulting Engineers, Inc., Pittsburgh, Pennsylvania.

(4) Murphy, Z. E., et al., June 1976, Demonstrated Coal Reserve Base of the United States on January 1, 1976, U.S. Bureau of Mines Mineral Industry Survey, U.S. Department of the Interior, Washington, D.C.

- The combined coal recovery from the panels and developments must exceed 60 percent;

- The coal has a low gas emission; and

- The mining method will be longwall top slicing with a slice height of 10 feet.

Basic Design Concept Philosophy

The very thick seam condition provides the opportunity for many possible basic mining arrangements when compared to alternatives available for single slice operations. On the other hand, the thick seam adds considerable complexity that places severe constraints on some important planning factors. A major part of the study was to develop the rationale for the selection of the best candidate entry system for this new and complex condition. Many alternative combinations of entries and their subsystems were evaluated. The optimization procedure centered on three critical entry components:

- Mains. The network of roads considered for the extraction of the whole seam included main entries:

  - At several levels in the coal seams;

  - At the base of the seam;

  - In the measure rock beneath the seam; and

  - At both sides of the property and/or along the center of the property, extending downdip.

- Submains. Options considered for the development of roadways from the mains to each panel in each slice included:

  - Short horizontal entries from multiple level mains;

  - Single inclines from common mains to panel entries in all slices;

  - Single inclines to each panel in each slice; and

  - Double inclines for each panel in each slice.

- Panel Entries. The entry possibilities considered for the longwall extraction in each slice included:

  - Multiple entry systems;

  - Dual entry systems, with and without pillar extraction; and

  - A single entry system.

The section of a final candidate system as introduced below requires careful analyses of many factors, varying from ventilation to rock mechanics. Several of the most notable controlling factors are:

- The long life of a thick seam mine;

- The importance of isolation without major interference with other current and future working areas;

- Remnant pillar effects on subsequent lifts;
- Spontaneous combustion problems with coal in the gob and crushed pillars;
- The need for a uniform subsidence profile; and
- The current health and safety regulations imposed by MESA and/or states; and probable variances that will be required for thick seam mining.

Geometric Design of the Candidate System

Major features of the feasible candidate system developed by "optimizing" and analyzing the operations in detail are summarized in Figure 3 and include:

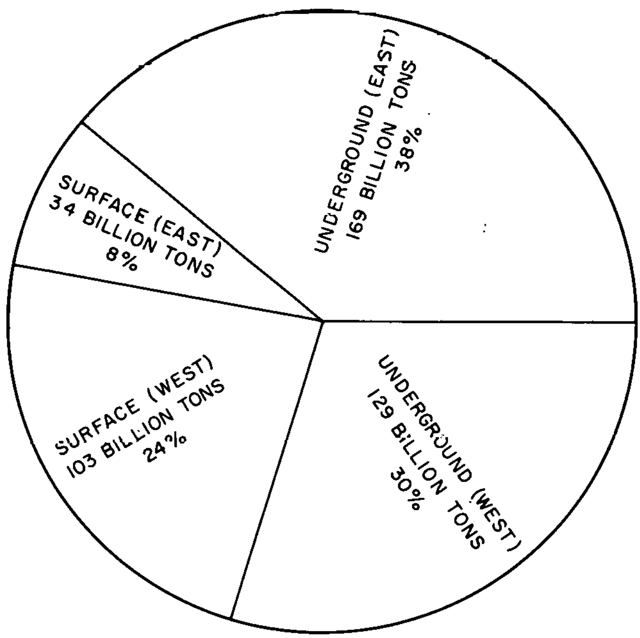
- A centralized set of main entries, driven downdip in rock below the seam beneath a large central barrier pillar, which are used for the extraction of all recoverable coal within the four-square-mile area;
- Submain inclines developed in pairs into each slice from pairs of laterals driven off the mains along strike;
- Panel entry development based on single entries with each entry being reused for the extraction of adjacent panels in any particular slice;
- An offset 50 feet in an updip direction for entries to panels in successively descending slices, except as required to satisfy boundary conditions, as shown in Figure 4;
- Longwall mining of 600-foot faces by retreat methods along strike from the edges of the property to the central barrier pillar; and
- Staggered panels at each edge of the property and both sides of the barrier pillar to produce a uniform subsidence trough.

Operational Design and Costs

The paper discusses alternatives considered and selected systems for:

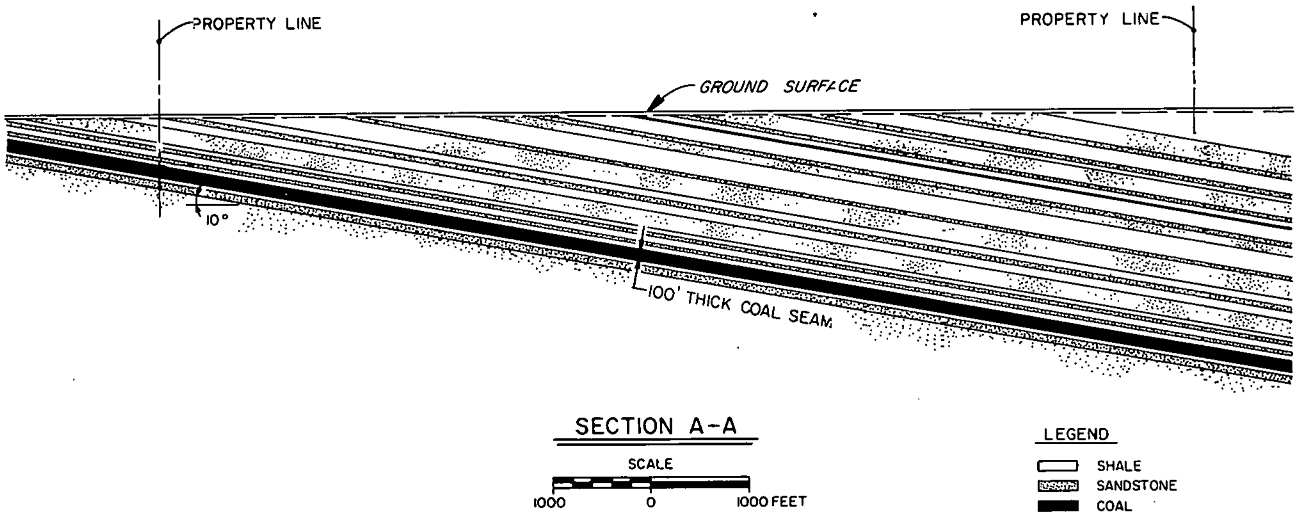
- Panel Sequencing
- Mining Methods and Equipment
- Materials Handling
- Ventilation
- Spontaneous Heating Control

Detailed economic analyses show small but significant increases in the mining cost when compared to conventional thin seam mining. However, economic feasibility of the method in the foreseeable future is apparent. Lowest costs will be realized by mechanizing the more labor-intensive aspects of mining the thick coal.



I Short Ton = 0.907 Metric Tons

FIGURE I DEMONSTRATED COAL RESERVE BASE OF THE U.S.A.



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FIGUF HYPOTHET:CAL STRATIGRAGHY NO OUTCROP ON PROPERT'

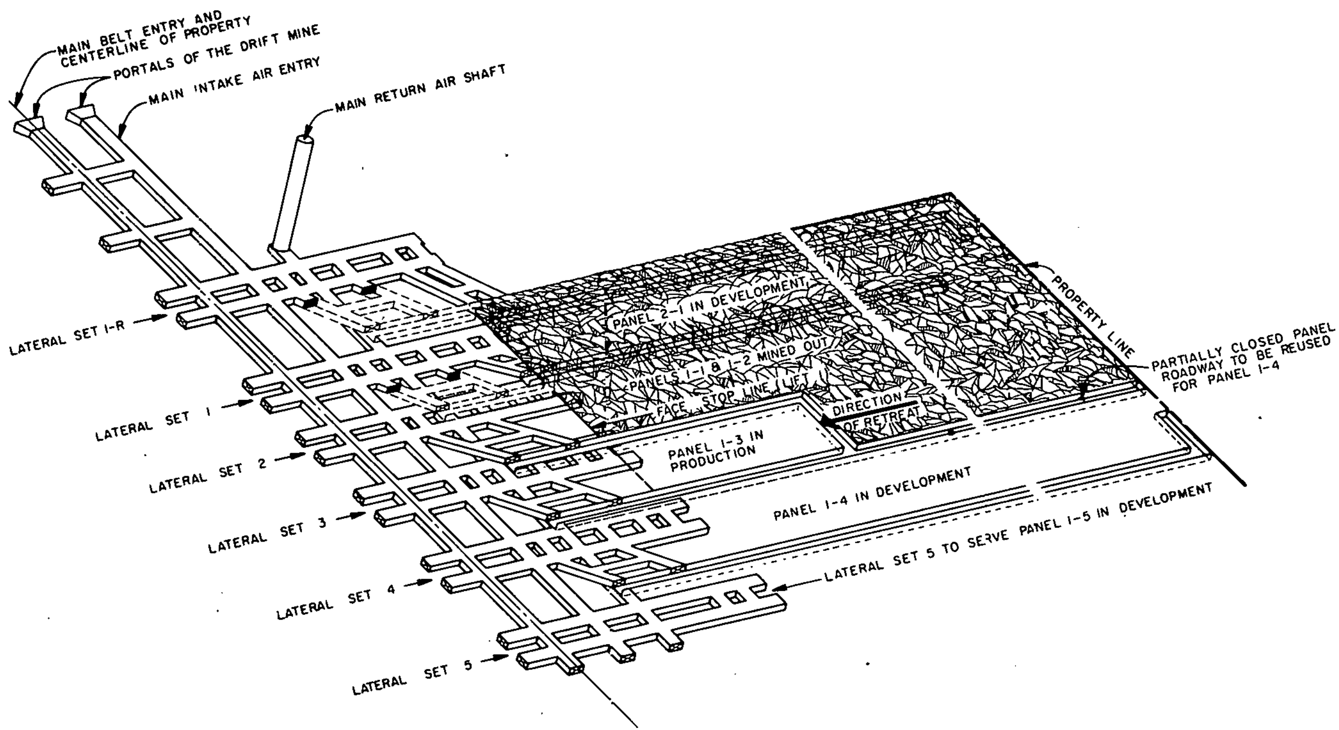

FIGURE 3 PANEL (1-3) IN PRODUCTION AFTER 5 AND ONE HALF YEARS

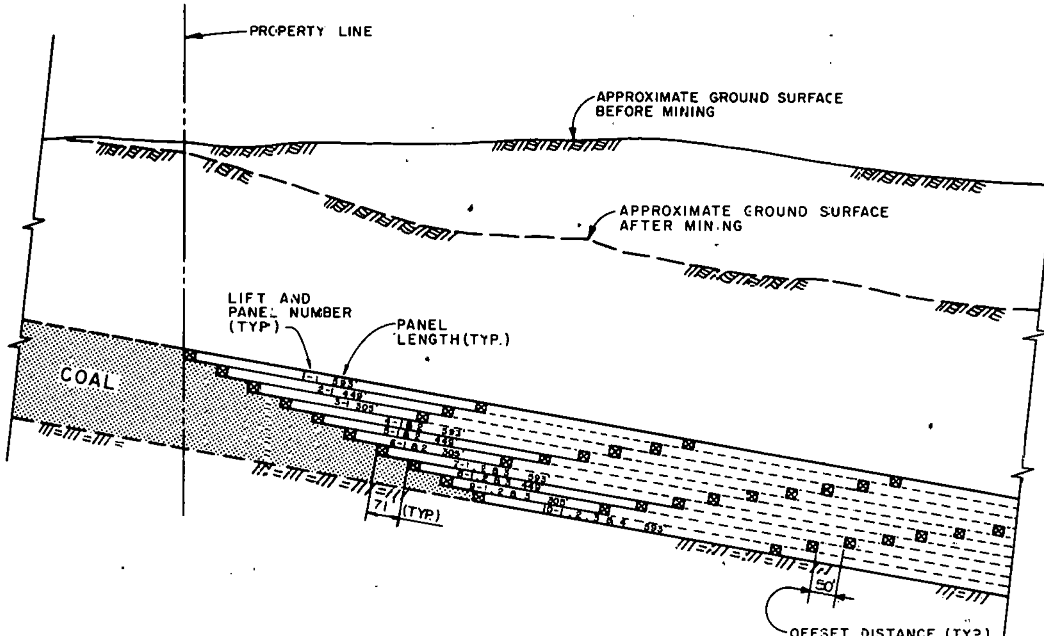

FIGURE 4 LOCATION OF PANEL ENTRIES—OFFSETTING OF THE FACES IN SUCCESSIVE SLICES AT THE UPDIP EDGE OF THE PROPERTY

SHORTWALL AND ROOM-AND-PILLAR MINING COSTS

Larry E. Green
Industrial Engineer

and

Eugene R. Palowitch
Deputy Research Director

Pittsburgh Mining and Safety Research Center
Pittsburgh, Pennsylvania

›stract

For shortwall mining to be profitable, the additional capital investment of about $1 million for shortwall roof supports must be offset by decreased total operation costs and/or increased coal production. The Bureau of Mines, therefore, conducted a demonstration project to compare shortwall and room-and-pillar mining.

Daily two-shift production from the Bureau of Mines-Beth-Elkhorn Corp. shortwall demonstration project averaged 975 tons of raw coal, compared with 905 tons from a room-and-pillar unit operating under similar conditions. The estimated direct operating costs for the shortwall and room-and-pillar units were $4.21 and $4.03 per ton, respectively.

The profitability of the shortwall system was found to depend not only on the increased depreciation cost of the powered roof supports and the decreased cost of supplies and materials but also on such factors as the fixed and variable indirect costs and the selling price of the coal. Shortwall mining was demonstrated to be a viable alternative to room-and-pillar mining.

Introduction

The Bureau of Mines and the Beth-Elkhorn Corp. under a cooperative agreement have been demonstrating shortwall mining since January 1973 in the 2 West section of Beth-Elkhorn's No. 22 mine in Letcher County, near Deane, Ky. (fig. 1). Mining is in the Elkhorn No. 3 coalbed, which is 48 to 54 inches thick and lies under 150 to 800 feet of cover.

This study compares the cost of mining coal by two different mining systems--shortwall and room-and-pillar. Because both systems are operating in close proximity with each other in the 2 West section of the mine, in the same coalbed, and under similar geologic conditions, the indirect costs for general overhead, general mine depreciation, transportation, preparation, waste disposal, taxes, insurance, and all other costs outby the operating section and common to both systems were not separately identified but were added directly to the direct operating costs (direct labor, material, utilities, and face equipment depreciation). Production, supervision and maintenance labor, face equipment depreciation, materials, and utilities costs were obtained from the operator and from pertinent literature (1), (2), (3).

Irect Mining Costs

The two systems being compared are adjacently located in the 2 West section of Hendrix No. 22 mine. D unit, the shortwall unit, develops 150-foot wide panels by driving three entries 2,400 feet deep and extracting these shortwall panels with a continuous miner and shuttle cars under powered roof supports; B unit, a standard room-and-pillar unit, develops sets of eight entries and subsequently extracts these pillars on retreat. Both units normally work two production shifts and one maintenance shift per day, five days per week. Both employ 10-man production crews and essentially the same coal cutting, loading and haulage equipment. On the average, the raw coal produced contains 22 percent reject.

Room-and-Pillar System

The standard room-and-pillar system (fig. 2) was used by B unit. The unit consisted of a 10-man production crew working two production shifts per day. The equipment was maintained by a four-man maintenance crew on the nonproduction shift. The face production equipment included a continuous miner, a roof-bolting machine, two shuttle cars, one battery-operated scoop tractor, and a section belt ratio feeder supplying a 36-inch belt conveyor.

The production crew initially drove eight entries, cutting the coal with the continuous miner and hauling it with the two shuttle cars to the ratio feeder. The entries and crosscuts are driven 20 feet wide on 50- and 75-foot centers, respectively (fig. 2). During development, belt conveyor, track, trolley wire, high-voltage cable, and waterline are installed in one of the middle entries. Concret block stoppings are installed in the crosscuts. Roof bolts are used throughout t section to support the roof. The MESA-approved roof support plan prescribes the use of 5/8-inch diameter bolts 5 feet long placed on 4-1/2-foot centers across the width of the entry (not to exceed 22 feet) and on 4-foot centers in the direction of advance.

At the completion of the eight-entry development, the pillars are extracted on retreat. The same equipment and personnel used for development are used for retreat. During retreat the installed track, trolley wire, conveyor, high-voltage cable, and waterlines are recovered. Although the roof control plan for the Hendrix No. 22 mine indicates virtually complete pillar extraction, actual room-and-pillar recovery averages about 85 percent.

During a 16-month period prior to the installation of the shortwall system, B unit's production averaged 888 tons per day; during the shortwall demonstration period (January 2, 1973, to January 19, 1977) B unit's production while developing 1 Right averaged 905 tons per day (4).

The 10-man production crews mining coal on both the development and retreat phases of the room-and-pillar mining system represent only the personnel working at the face. The same is true of the four-man maintenance crew. The personnel cost necessary for all general inside labor not performed by the direct operating crews is included in the indirect cost. This includes such functions as installing and recovering track, trolley wire, waterline, and concrete block stoppings.

The wage rates used were those in effect November 1976 per the National Bituminous Coal Wage Agreement of 1974. The direct labor cost per shift for the 10-man production crew is $579, or $1.28 per ton. The maintenance shift direct labor cost is $229, for a unit cost per ton of $0.25.

Another direct operating cost is depreciation of the face equipment. Straight-line depreciation with zero salvage value for all equipment was assumed. The initial equipment costs reflect 1976 prices. Based on an average production rate of 905 tons per day and 220 production days per year, the depreciation charge is $121,770 per year, equivalent to $0.61 per ton.

Material costs were estimated by charging the materials actually consumed during a 150-foot advance of the eight-entry room-and-pillar section (B unit). A distinction was made between those materials used during development and those used during retreat.

The materials can be classified into two broad categories: (1) Materials that are completely consumed during mining, such as roof bolts, rock dust, and timber; and (2) materials that are recovered and reused, such as trolley wire and track. The recoverable materials are capital in nature; hence their depreciation cost was included in the fixed depreciation costs.

The material costs for both development and retreat, weighted in proportion to the coal production in a normal eight-entry production section, is estimated at $1.75 per ton.

Face utility costs for water and electricity vary with the type of equipment used for mining. Estimated water required for a typical room-and-pillar section is 3,000 gallons per shift; at $0.10 per 1,000 gallons, this amounts to approximately $1 per day. Electrical power costs are directly related to the rated horsepower and duty cycle of the face equipment. Continuous miner and shuttle car operations were time studied at the shortwall demonstration. Based on a $0.02-per-kilowatt-hour electric rate, the daily face power cost is $127. The combined water and electric utility cost is thus $0.14 per ton.

The total direct operating costs calculated for an eight-entry room-and-pillar section were $3,644 per day and $4.03 per ton. Table 1 summarizes these costs, which include direct production and maintenance labor, equipment depreciation, materials, and utilities.

Table 1. - Direct operating costs, B unit

Cost category	Daily	Per ton
Production labor.................	$1,158	$1.28
Maintenance labor................	229	.25
Depreciation.....................	554	.61
Materials........................	1,575	1.75
Utilities........................	128	.14
Total.......	3,644	4.03

Shortwall System

The shortwall operation (D unit) at Hendrix No. 22 is depicted in figure 3. D unit is manned by a 10-man production crew, working two production shifts per day, five days per week. A five-man crew maintains the equipment on the evening shift of each workday.

The primary mining equipment employed by D unit consists of a continuous miner, two shuttle cars, a battery-powered scoop tractor, a section belt, a ratio feeder, a roof-bolting machine, and a shortwall powered roof support system. The coal is cut and loaded with a continuous miner and hauled to a section belt ratio feeder by two shuttle cars.

The shortwall panels are developed by driving a set of three entries 20 feet wide on 50-foot centers with 20-foot-wide crosscuts on 75-foot centers to a depth of 2,400 feet to create a 150-foot wide by 2,400-foot-long shortwall panel (5). The development entries serve first as headgate entries and subsequently as tailgate entries for the next panel. Approximately 200 feet of the developed shortwall panel is left standing as a barrier pillar to protect the main entries. Roof bolts are used throughout the development entries to support the roof. Concrete block stoppings are installed in the crosscuts as development progresses. A belt conveyor, trolley wire, high-voltage cable, waterlines, and track are installed in the center entry.

At the inby end of the 2,400-foot-deep development entries, 150 feet of additional three-entry bleeders are driven perpendicular to establish a shortwall face (fig. 4). No track, conveyor, cable, or waterline is installed in these entries.

The same equipment and face crews used to develop the entries are used to mine the coal and haul it from the shortwall face to the panel conveyor. The average lift across the face is 4.5 feet high and 10.5 feet wide. It takes approximately

1.7 hours to make a complete lift and requires 69 to 70 shuttle car trips at a payload of 4.3 tons to transport the coal from the face to the breaker-feeder. The average time to clean up and advance the face after each lift is 33 minutes. A battery-powered scoop crosses the face after each lift to collect loose coal and haul it to the headgate.

Once the panel of coal is mined, the production crews pull the roof supports off the face and store them in an entry. The production crews move the mining equipment to begin development of another panel. A separate three-man crew moves and sets up the stored roof supports onto the next, already developed, shortwall face. This crew also performs necessary maintenance on the chocks as well as general cleanup and face preparation for the shortwall panel.

No production is lost between panels in the normal mining cycle because the shortwall supports are set up on the new panel during the development of the succeeding panel.

During the period January 2, 1973, to January 19, 1977, six full shortwall panels were developed and extracted; the development of a seventh panel was started but had to be abandoned after driving only about 900 feet because of adverse roof conditions (fig. 3).

Total raw coal production from the seven and one-third sets of panel development entries and from the six panels during the period January 2, 1973, to January 19, 1977 was 691,749 tons, of which 306,791 tons was development coal and 384,958 tons was retreat (shortwall) coal. Average daily production from the shortwall ranged from a low of 856 tons in the first panel to a high of 1,166 tons in the fourth panel, while average daily development production ranged from 714 on 1 Left to 1,032 on 7 Left. Productivity ranged from 29 to 47 tons per face-manshift. The highest daily production was 2,027 tons.

Overall, D unit's daily performance showed 913 tons on development, and 1,031 tons on retreat (shortwall extraction) for a weighted average of 975 tons.

Because the chain pillars between the development entries are not recovered, coalbed recovery with the shortwall system is approximately 82 percent.

The 10-man production crew for D unit is costed at $581 per shift, or $1.19 per ton. The maintenance crew cost is $278 per shift, or $0.29 per ton. The roof support moving crew shift and unit cost per ton are $198 and $0.07, respectively. The moving labor cost is based on an average 36 production days for the crew, and 108,000 tons of coal for one complete shortwall cycle of development and panel extraction.

The equipment used in shortwall mining is identical to that used for room-and-pillar operation with one significant addition--the powered roof supports. The support system used in this particular case consisted of 42, four-leg, 500-ton-capacity, 42-inch-wide chocks. The chocks were set on 48-inch centers and attached to an articulated reference rail to maintain proper spacing and alinement across the face. For purposes of this study, the chocks were costed at an installed 1976 value of $1 million. A life expectancy of 10 years was assumed. Although a roof bolter normally will be used only occasionally while mining a shortwall panel, its depreciation was fully charged to the shortwall system.

Annual straight-line depreciation cost for the shortwall roof supports is $100,000. This raises the total annual depreciation from $121,770 per year, or $0.61 per ton for a room-and-pillar system, to $221,770 per year, or $1.03 per ton for the shortwall system.

Material costs were estimated in the same manner as for the continuous room-and-pillar system. Materials used during panel development and panel extraction were individually identified because of the distinct difference in production and costs between the two mining phases.

As a complete cycle of the shortwall system includes developing 2,670 feet of a three-entry system and extracting a 2,200-foot-long by 150-foot-wide shortwall panel, the individual material unit costs were weighted in proportion to the extracted coal tonnages attributable to each of these two phases. The material cost for D unit is $1.50 per ton.

The shortwall system daily utility costs are assumed to be the same as for the room-and-pillar system. The roof bolter is used sparingly while mining the shortwall panel, but this cost saving is offset by the cost to operate the hydraulic system for the chocks. The utility cost is estimated at $128 per day, or $0.13 per ton.

A summary of the shortwall direct operating costs is given in table 2.

TABLE 2. Direct operating costs, D unit

Cost category	Daily	Per ton
Production labor.................	$1,162	$1.19
Maintenance labor................	278	.29
Chock moving labor...............	68	.07
Depreciation.....................	1,008	1.03
Materials........................	1,463	1.50
Utilities........................	128	.13
Total........	4,107	4.21

Total Mining Costs

The total costs of mining includes the direct operating costs and the indirect costs. Direct operating costs determined for the room-and-pillar system (table 1) and for the shortwall system (table 2) at average raw coal production levels of 905 tons per day and 975 tons per day respectively are given in table 3.

TABLE 3. - Summary of total direct operating costs per ton

Cost category	Room-and-pillar	Shortwall
Production labor................	$1.28	$1.19
Maintenance labor...............	.25	[1].36
Face depreciation...............	.61	1.03
Materials.......................	1.75	1.50
Utilities.......................	.14	.13
Total unit cost per ton of raw coal..............	4.03	4.21

[1]Includes chock moves.

The indirect costs incurred in the production of coal must be included in determining the total mining cost. Specific costs include general mine and payroll overhead, transportation, depreciation, labor, insurance, preparation, waste disposal, and taxes. These indirect costs consist of both fixed and variable elements with respect to production. Insurance, depreciation, and property taxes are examples of fixed indirect costs, which remain essentially constant regardless of the amount of coal produced. The variable indirect costs include expenses for such items as coal royalties, excise taxes, and contributions to the United Mine Workers of America Trust Fund. These costs are a direct function of the level of coal production. In general, only a small proportion of total indirect costs is variable in nature.

Conventional accounting practice spreads the indirect costs of production equally over each unit produced to arrive at a total unit cost of production. As a major portion of the indirect costs are fixed, any increase in annual production will spread this fixed annual cost over more units and hence will provide a lower total cost per unit.

Although indirect costs vary appreciably between mines, an assumption was made of these costs to allow comparing the total costs of the two mining systems. A fixed indirect cost of operating a hypothetical mine with a single coal producing section operating two production shifts per day was estimated at $4 million annually. The variable portion of the total indirect costs was estimated at $3 per ton of clean coal.

The production and cost experience gained at Hendrix No. 22 was used as a basis for comparing the economics of the two systems: the room-and-pillar system produced an average of 905 tons of raw coal daily at a direct cost of $4.03 per ton (706 tons of clean coal per day at a direct cost of $5.17 per ton); the shortwall system produced an average of 975 tons of raw coal per day at a direct cost of $4.21 per ton (761 tons of clean coal daily at a direct cost of $5.39 per ton). The total operating costs of the two mining systems, based on clean coal production, are depicted in table 4.

TABLE 4. Operating costs summary, hypothetical mine

Mining system	Annual clean coal production, tons	Operating costs: Direct		Variable indirect		Fixed annual indirect	Total	
		per ton	Annual	per ton	Annual	millions	Annual	per ton
Room-and-pillar	155,320	$5.17	$802,373	$3.00	$465,960	$4	$5,268,333	$33.92
Shortwall	167,420	5.39	903,045	3.00	502,260	4	5,405,305	32.28

Because the shortwall system produces an average of 12,100 tons of clean coal per year more than the room-and-pillar system (167,420 tons versus 155,320 tons), the reduction of $1.86 per ton in indirect costs ($25.75 versus $23.89) more than offsets the increased direct cost of $0.22 per ton ($5.39 versus $5.17) for the shortwall coal. Thus, under the above assumptions, the total cost of shortwall coal at this hypothetical mine is $1.64 per ton less than the cost of room-and-pillar coal ($33.92 versus $32.28).

Profitability

Using the operating costs developed above, the economics of the two systems can be compared to determine the profitability of the shortwall investments. The lower total operating costs per ton for the shortwall result in higher gross profit. If the selling price of clean coal is assumed to be $40 per ton, this cost benefit results in an additional annual gross profit of $347,028 per year. This increased profitability of the shortwall system in the hypothetical mine is illustrated in table 5.

TABLE 5. - Comparative profitability, hypothetical mine

Mining system	Total revenues		Total operating costs		Gross profits	
	per ton	annual	per ton	annual	per ton	annual
Room-and-pillar	$40.0	$6,212,800	$32.92	$5,268,333	$6.08	$944,467
Shortwall	40.0	6,696,800	32.28	5.405,305	7.72	1,291,495

Some businessmen consider all fixed indirect costs as prepaid or recovered in an existing profitable enterprise. Under these circumstances only the variable portion of the indirect costs and the direct operating costs would be attributed to any increased production in evaluating a proposed capital investment. Under this philosophy, the extra daily shortwall production of 55 tons of clean coal would be costed at $8.39 per ton. With coal selling at $40 per ton, this represents a gross profit, before taxes, of $31.61 per ton. Comparing the increased annual gross profit to an initial shortwall investment of $1 million, the investment would be recovered in 2.6 years.

Annual Cost and Capital Recovery Analysis

An annual cost analysis was performed with the following assumptions: (1) As the run-of-mine coal averaged 22 percent reject during the demonstration period, raw coal production was adjusted accordingly to reflect salable clean coal tonnages when determining gross income from sales; (2) the salable clean coal was valued at $40 per ton; (3) depletion was estimated at 10 percent of sales in accordance with Internal Revenue Service procedures; (4) a 7-percent investment tax credit for the capital cost of the shortwall supports was claimed (As the cost comparison is made on an annual basis, this one-time tax credit was converted to an equivalent uniform annual tax savings); (5) a 48-percent tax rate was used to determine the rate of return after taxes; (6) a 15-percent rate of return was selected as the minimum acceptable to any prospective mine operator evaluating an initial capital investment in a shortwall system; and (7) the mine work schedule was set at two production shifts per day and 220 workdays per year.

Based on these assumptions, the shortwall system generates an additional positive annual cash flow of $366,087. This represents a capital recovery period of 4.3 years to recover the required $1 million estimated current price of a set of shortwall roof supports at 15-percent rate of return. The capital recovery period for the actual initial investment of $430,000 for the shortwall roof supports in 1973 is 1.7 years.

Discussion

Shortwall mining is relatively new in the United States. The first shortwall face was installed in the Federal No. 1 mine in February 1973, and by the end of 1975, five shortwall faces had been installed (6). Since then at least three additional shortwall faces have begun operating, and several more are in the planning stages. It is felt that as more experience is gained, better methods will evolve, face equipment will improve, and coal production will increase--all of which will improve the economics of shortwall mining.

Significant operational improvements in the system also are possible. A recent study of the availability of the shortwall system showed that it was operational only 68.2 percent of the available working time (7). Haulage and other non-face-related delays represent an additional 15.7 percent of the available working time. Simulation studies of the system indicated, and experience proved, that the continuous miner was idle approximately half of the available face time waiting for shuttle-car haulage (8). If any of the continuous haulage techniques currently under study by the coal industry, the mining machine manufacturers, or the Bureau prove successful, substantial increases in production might be achieved.

The development and extraction of shortwall panels by a single production unit was the more convenient approach for the purposes of this demonstration. This strategy eliminated the loss of coal production at the completion of a shortwall panel. Because no coal production was being lost, there was ample time to repair and maintain the roof supports between panel moves. Thus, the cost incurred through idle capital equipment was at least partially offset by the high availability of the well-maintained roof support system.

Conclusion

Experience gained during this demonstration supports the following conclusions:

1. The shortwall unit produced 70 tons of raw coal per day more than an adjacent room-and-pillar unit operating under essentially identical conditions.

2. The estimated direct cost of mining a ton of raw coal by the shortwall method is 18 cents higher than the cost of mining by the room-and-pillar method. The increased cost of depreciating the shortwall roof support system was only partially offset by reductions in the cost of supplies and materials.

3. The profitability of a system that produces an additional 70 tons of coa per day but at an 18-cent-per-ton higher direct cost is a function of the direct operating costs, the fixed and variable indirect costs, the quality of the raw coal, and the selling price of the clean coal.

4. Shortwall mining has been proven a viable alternative to room-and-pillar mining.

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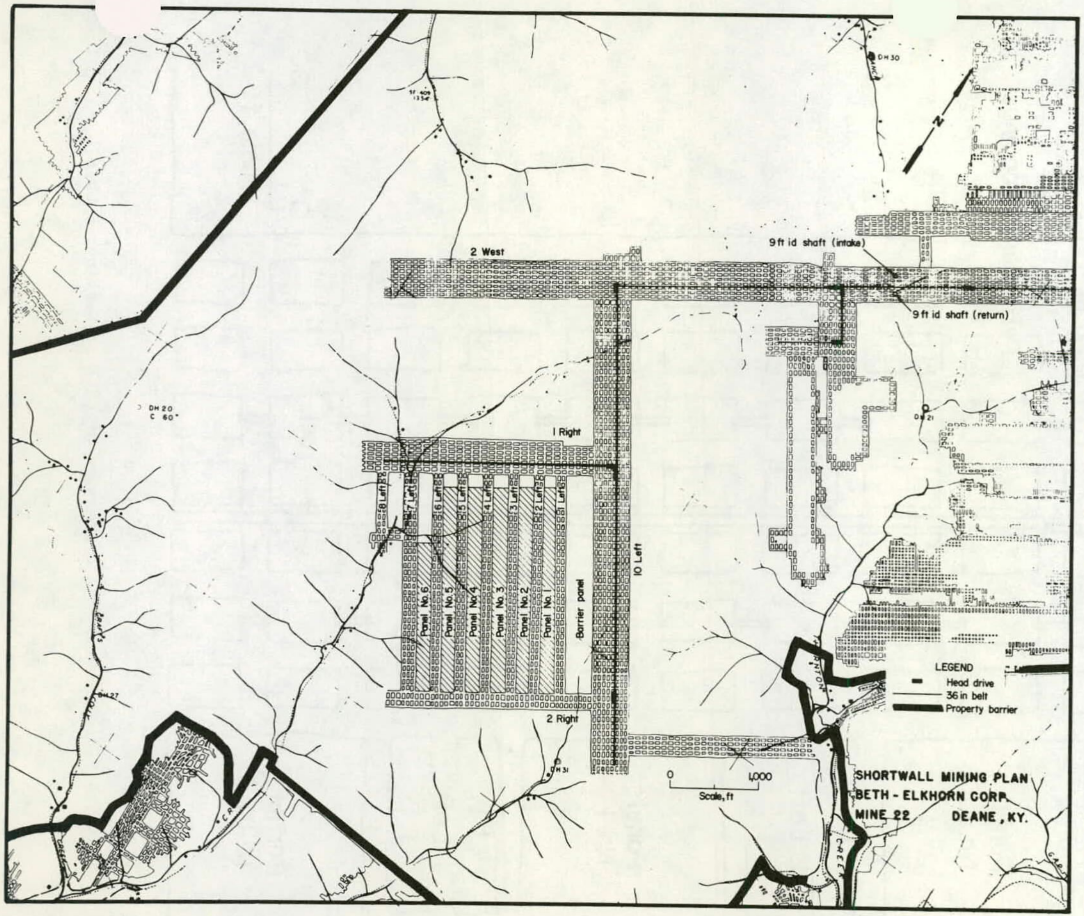

FIGURE 1. - Hendrix No. 22 Mine, 2 West Section.

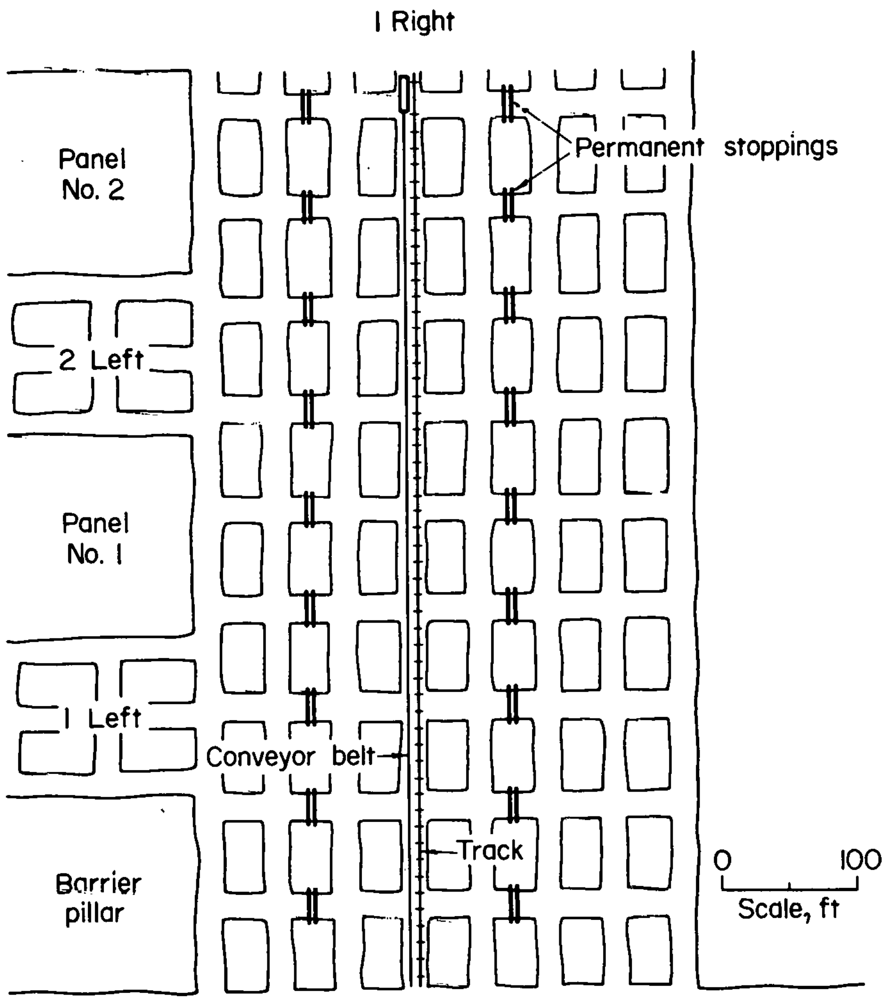

FIGURE 2. - Room-and-Pillar Arrangement, 2 West Section, B Unit.

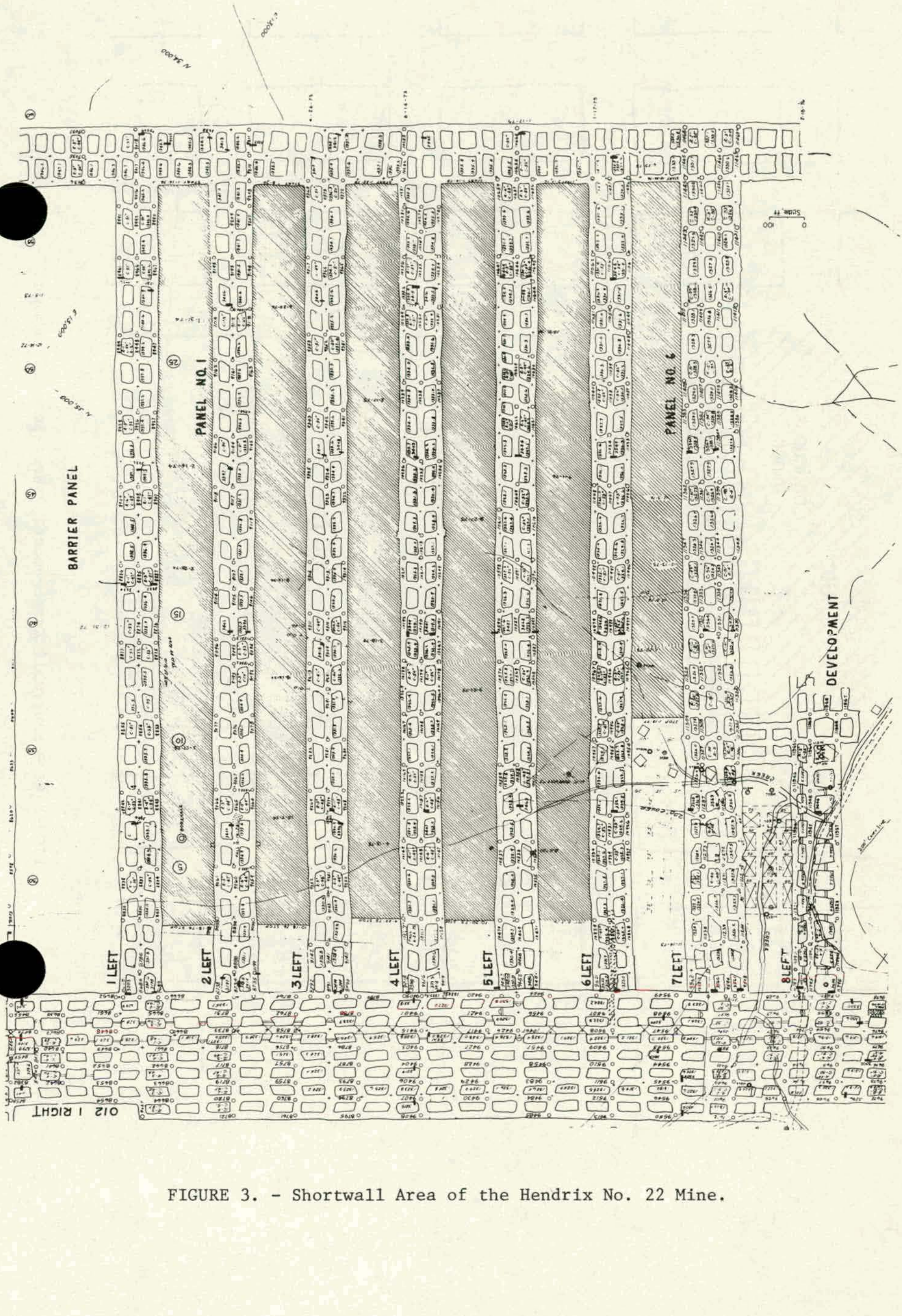

FIGURE 3. - Shortwall Area of the Hendrix No. 22 Mine.

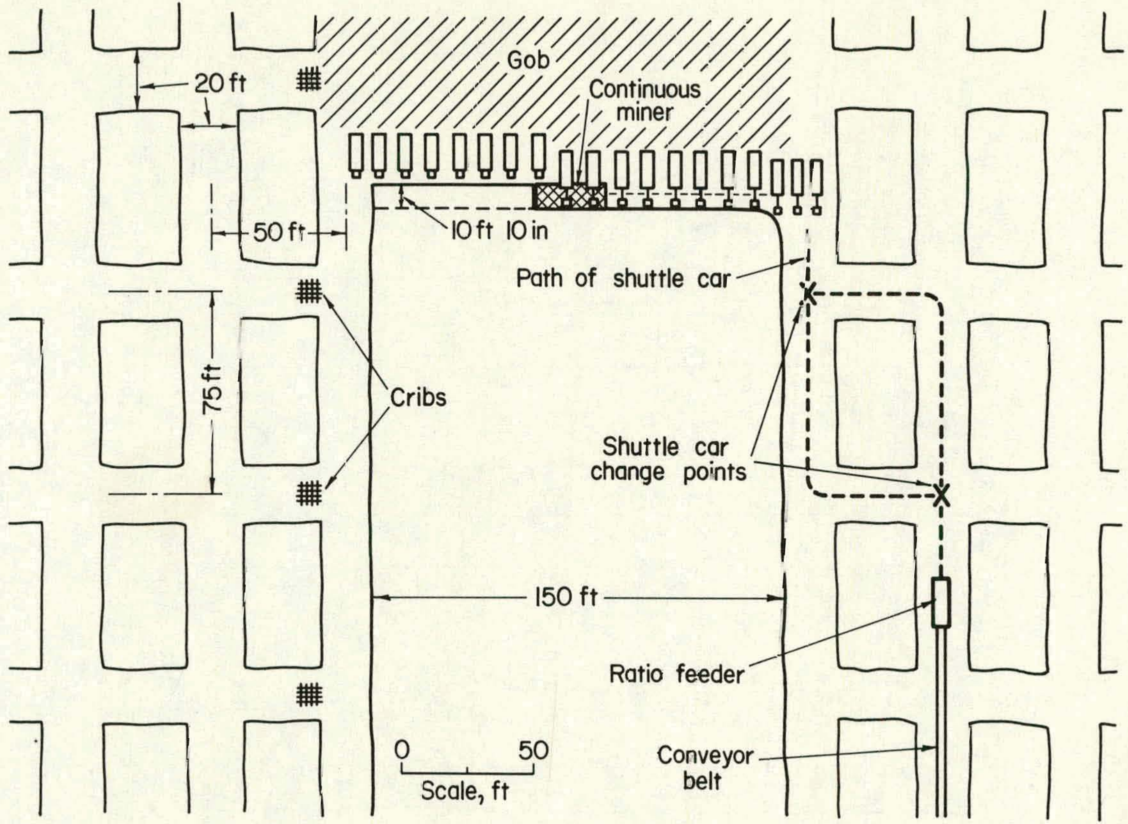

FIGURE 4. - Shortwall Arrangement, 2 West Section, D Unit.

A NOVEL APPROACH TO LONGWALL MINING

A. J. Hargraves
Superintendent Rock Mechanics & Strata Control

The Broken Hill Proprietary Co. Ltd./
Australian Iron & Steel Pty. Ltd.
Wollongong, Australia.

Abstract

With the progressively increasing depth of mining operations in ustralia, and with increasing problems with room and pillar mining, onsideration of the alternative of longwall has resulted in a number of self advancing retreating longwalls and shortwalls in the N.S.W. mining areas, with some successes and some failures. Even with drivage at depth being thus minimised, problems are still experienced in keeping sufficient development ahead of extraction, with consequent hold-up of extraction schedules.

The development in Australia of the shortwall method using trackless equipment and self advancing supports as an alternative to pillar extraction has generally met with success and has reduced drivages and improved recovery. If shortwall extraction can be done on the advance using existing mining equipment it could provide roadways for subsequent retreating longwalls or shortwalls and thus obviate drivage completely in areas destined for extraction. The particular problems in such a system lie in support of roadways during the advance stage and the retreating stage. There is promise in modern pack building methods of obviating the tedious chock building or pack building of the past and at the same time of providing effective seals against air leakages through goaves during advance work. In the case of certain Australian areas where outbursts of coal and gas occur in development the change to development by shortwall advancing should reduce the proneness one category and should thereby reduce the extent of preventive measures. The slower rate of advance of the "development" front,yet with perhaps augmented tonnage, should be a further factor in reducing proneness. So far the method remains as a proposal and has not been tested.

Background

The two major underground coalmining areas of Australia are the Newcastle and Illawarra areas of New South Wales. The flat-lying Permian seams outcrop and since the commencement of mining in about 1800, work has progressed towards deeper cover, more quickly in the Illawarra area where the seams outcrop below an escarpment, and more quickly with the passage of time and mounting production, especially n the last decade due to a significant export market. Now, the orkings have penetrated to depths of cover up to 500 m, still shallow y some overseas standards.

Elsewhere in Australia, and particularly in the Bowen Basin Queensland, deposits are being opened up, also in flat-lying seams of Permian age, and in seam thicknesses up to 6 m. Already one mine has reached 380 m depth of cover, and significant problems are experienced. The great productive potential of this area, at depth, is presently jeopardised by development problems.

For many years the standard coal mining method in Australia has been room and pillar. This has suited the thick flat deposits, usually single seams, usually at shallow depth. With increasing depths and with more problems in the support of roadways, thoughts have turned to retreating shortwalls and retreating longwalls with self advancing supports. A few installations already exist. These systems reduce development drivages, increase recoveries and minimise expendable items. Now, deeper mines are experiencing difficulties in keeping development ahead of extraction with both shortwalls and longwalls because of problems in supporting roofs, of high gas makes, and, in some instances tendencies to instantaneous outbursts of coal and gas. Additionally, the recently exploited thick seams of hard coking coal provide the challenge of sustaining recoveries under su(difficult physical conditions. The worked seam thicknesses in Australia are usually 2 m or more, providing adequate working height without brushing, and so longwall advancing has been used only rarely, in thin seams, and only in the past with timber and packwall supports. The use of advancing methods in thicker seams, to minimise development drivages, has been deterred by the need for expendables for gateroad support and the manual work needed to establish such supports, as well as the possibility of geological unknowns which naturally overshadow any systematic plan of development. Such advancing walls could provide gateroads which, if stable, could be used in adjoining retreating walls, thus minimising development. The slower face advance and the simpler ventilation circuits could possibly reduce ventilation problems, from a point of view of gas. When mining upper seams on the advance, the goaves in between the advancing roadways might alleviate the stress problems of virgin drivages and the change from heading to face work could reduce proneness to instantaneous outburst of coal and gas by one category (Hargraves, 1967). The more prone zones - ahead of the gateroads on the faces - should be the only zones where special precautions would be needed. If such advance roadways could be eliminated, so much the better. If the gateroads could be adequately sealed from the goaf, then the dangers of spontaneous combustion could be markedly reduced.

For these and other reasons the use of advancing shortwall with self-advancing support is seen as a means of developing for retreating longwalls, and the possibility of monolithic concrete packwalling is seen as a stronger alternative to the conventional steel and/or timber supports. (Fig. 1). Modern methods of slip forming and concrete pumping could minimise labour requirements.

Apart from the obvious economic considerations, the concrete support should be designed to develop support strength in the minimum period for a complete face cycle, unless the web were small and pouring could be done in multiples of webs.

Already some deeper mines in the Illawarra area are benefitin from stress relief by using the caving of sacrificial headings in some circumstances. Also, initial retreating longwalls with powere supports designed on English and Continental practice failed, and a local basis for design was indicated (Smith, 1970; Reed, 1970). This independent approach has been met with considerable success (McCoy, 1976). Subsidence observations have confirmed the unique behaviour of superincumbent strata in Australia (Kapp, 1973). For these reasons and despite maximum pressure arch theory and the relatively short spans of the shortwall principle it is felt that a more experimental approach to shortwall advancing should be taken. The record of shortwall retreating experience is further encouragement to do this. No less encouragement is the saving of say 15% of

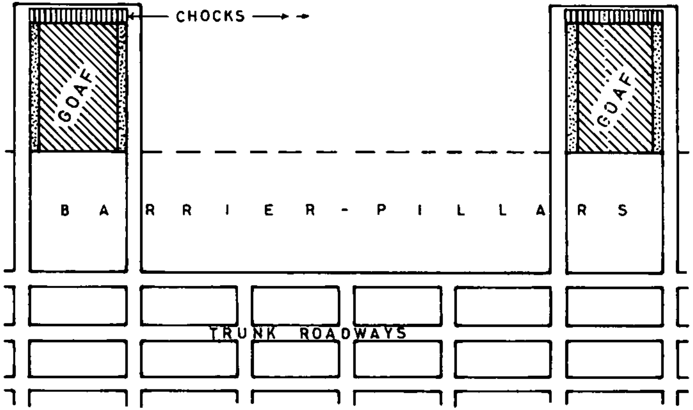

Fig. 1,b. Development by Shortwall advancing.

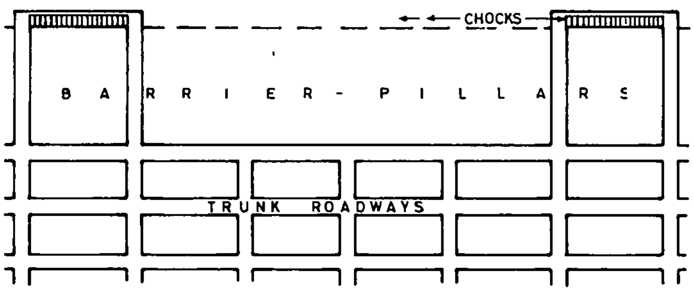

Fig. 1,a. Heading development through Barrier-pillar and installation of supports.

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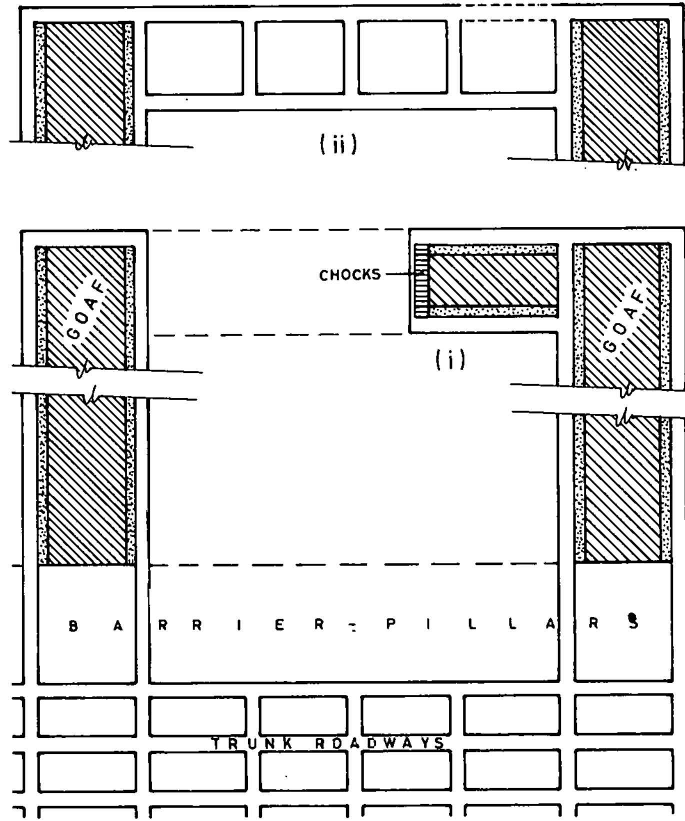

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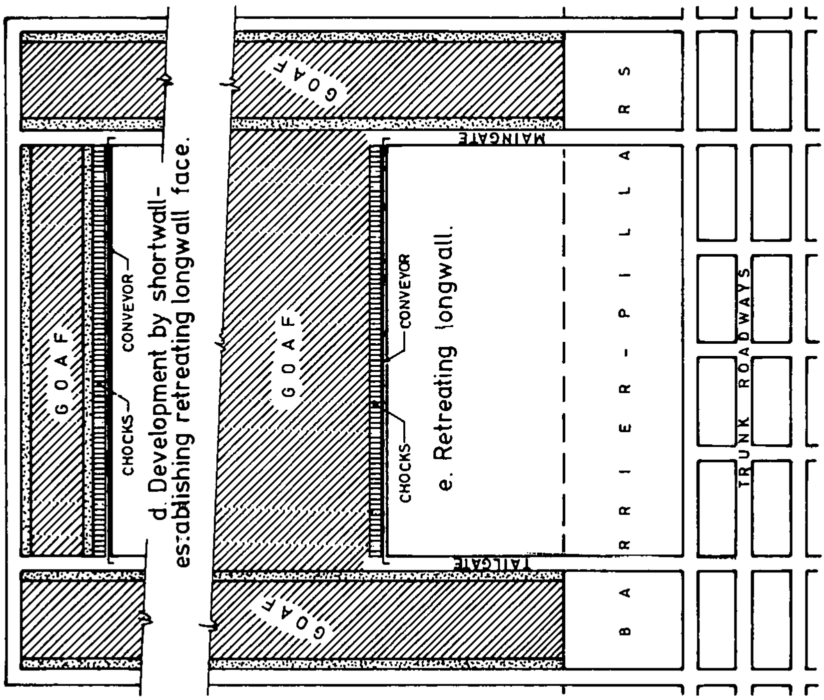
Fig. 1,c. Development of Longwall cross heading by starting (i)-shortwall or (ii)-roadways.



Figs. 1,d.

reserves which would be lost in two heading development for retreating longwall. Due to copious gas during development the number of headings has been as many as four, with commensurately greater losses, although some of this development pillar coal is sometimes extracted. In the subsequent mining of adjoining retreating longwall blocks the progressive extension of the cave over the (incompleted caving) shortwall goaf could be expected. The shorter the shortwall, the less would be this extension.

History of Longwalls and Shortwalls

The history of mechanised retreating longwalls in Australia has been one of initial disappointments of a deterrent nature (Smith, 1970) with a few intrepid experiments (Reed, 1970) and now with well established and highly productive modern longwalls in two mines and the prospect of extension of the method to other mines (Pearce, 1976). The early longwalls were dogged with mechanical problems - conveyors particularly - and with inadequacy of support resistances of European type supports for the strong rather inflexible roofs experienced under Australian conditions. Improvement of control and performance of longwalls has followed demand for higher resistance supports and more rugged face conveyors and crushers and presently maximum daily face output has been 6,394 tonnes and monthly output (23 working days) 86,565 tonnes. Unfortunately no similar improvement has been experienced with development drivage rates in spite of machines of higher productivity, early roof bolting and strongest discipline on widths of roadways, and because of mounting gas problems and roof support problems as depths of working exceed, say, 350 m.

The experience with shortwall, in time, has been quite different from longwall. The initial retreating shortwalls, at comparatively shallow depth - from 200 m - were very satisfactory and encouragingly productive (Marsden, 1970). Only when subsided ground over an incompletely extracted lower seam was experienced did the method fail (Martin and Hargraves, 1972). This experience led to increase of support resistances for shortwalls at greater depths (Hargraves, 1972), with even greater increases as experiences at 400 m met with support, steering and alignment problems. The initial webs of up to 4 m were reduced to 3 m, with some benefits (Hargraves, 1972). At this stage it is clear that the greatest application for the shortwall system is in the shallower workings of the Newcastle Coalfield - maximum depth ?00 m. Whilst in the deeper Illawarra coal measures some shortwalls continue, the shorter face length and higher proportion of development appear to make longwall more attractive. Under poor roof conditions the large area turning from the maingate onto the face has given some problems. The stone floor rolls in some parts of the Illawarra area appear to cause problems with shortwalls also.

There have been several examples of hand longwall advancing in Australia, but always in seams of thickness 1.5 m or less and generally in high grade coal or in areas remote from coal deposits of greater thickness (Brisbane, 1970). Generally face lengths have been short. Apparently roof control problems have not been severe.

History of Development Drivage

All development in Australia is in coal and almost all is by continuous mining. Because of the thickness of the seams, development is normally flat-back at the roof of the seam. Heading machines are rarely used in drivage except where faulting requires traversing stone and in general close faulting is rare. The sparseness of faulting generally lends itself to longwall and shortwall operation and likewise has minimum bar to development by a shortwall advancing method. In recent development in the Blackwater field in Queensland, with a seam 6 m thick, development near the top of the seam has been conducted in part by heading machine with the object of keeping development places narrow and cutting an arched roof. There and elsewhere in deeper developments problems with stability of headings have been experienced and presently a general programme of reduction of widths of development places is under way. The statutory maximum width of 5.5 m in practice generally results in actual widths up to 7 m and exceptionally more. Some of the problems of roof support in development have been attributed to this cause. Hargraves and Martin (1977) have described some of the current problems in development support. A particular problem experienced in development drivage is roof guttering on one side. Both arched section and guttered rectangular section can expect problems as gateroads for retreating walls.

Proposed Gateroad Supports

Principle

For some time advancing shortwalls (and perhaps longwalls) have been under consideration as a means of reducing problems experienced in development in deeper drivages, on the presumption that gateroads for advancing shortwalls would still be available for retreating longwalls adjoining. The earliest proposal for advancing walls (Hargraves, 1974) was on the basis of lower category of outburst proneness of walls relative to headings as a means of alleviating the outburst problem (Hargraves, 1966). It was presumed that relatively inflexible support would be required to sustain stable roadway roof and rib until the retreating phase. For this purpose it appeared that handbuilding chocks of steel or concrete members would be necessary, augmented by transverse steel roof bars roof bolted, and with some means of supporting the coal rib side of such bars during the subsequent retreating operation. (It was envisaged that the handbuilt chocks could be made more cheaply by forming from reject steel sections from the steelworks to which numerous mines were captive, but the labour intensive nature of chock-building appeared to be a great disadvantage.). Now the possibility of monolithic pack-walls revives the principle of advancing walls for development of retreating walls in thicker seams.

Composition

In a conservation conscious environment it seemed that alternatives to small coal as an aggregate should be considered. It was hoped to use in the concrete monolithic pack walls waste products from steel manufacture and particularly washery refuse and blast furnace slags. Almost all coal washing is done at the steel works. The possibility was seen of back loading these materials to the collieries for aggregate. At the same time the cementing property of finely pulverized slag was known as well as the cementing properties of pozzolana, a by-product from the power stations sited on the coalfields. Early enquiries indicated that the cost of anhydrous gypsum plaster would be prohibitive as well as the strength of plaster being less than that of Portland and other cements and of Portland cement concretes. The investigations of concrete compositions should take into account the need to pump the aggregate and the cement slurry as guided by slump tests, the possible need for accelerators, fluidisers etc. as well as necessary mixing times and strengths of pumped concrete slurry relative to the time since mixing. Sufficient strength should be attained according to roof loadings expected with time. Initial tests indicate that a pumpable concrete mix can reach measurable strength in 6 hours and reach 1.5 MPa in 24 hours. Current tests aim to minimise expensive constituents such as bentonite and commercial cements and to promote industry by-products. Minus 9.5 mm run of mine coal for aggregate is being tested also. The costs and transport costs to and into collieries of cementing and aggregate materials, related to the early and 28 day strength, may make a smaller pack of higher cost material of higher early and 28 day strength desirable on an overall economic basis.

So far no inexpensive concrete mixture, including accelerators, provides the high early strength apparently demanded of the monolithic pack, and very wide packs of very weak material with consequent large quantities of constituents are undesirable. Adequate strength of course is attainable after a period of days. For this reason it was considered possibly economic to investigate exotic cements as alternatives. If the use of aggregate from freshly mined coal is to be considered, the possibility of subsequent mining of and use of such "coal concrete" might be a consideration. However, any Bulli Seam undersize coal from a screening process would contain a disproportionately higher content of bright coal, weaker as an aggregate, and more valuable in coke making than the seam average.

Proposed Advance Rates

(a) Shortwall Advancing

The length of shortwall retreating faces has been 40-70 m of solid coal, with maximum length dictated, more than any other factor, by the length of cable accommodated by the cablereel of a shuttlecar.

Rates of advance have been up to 12 m per 24 hour day. With similar gateroad centre distances in shortwall advancing, the amount of coal won per cut would be increased by the widths of the two gateroads, say 9 m overall relative to shortwall retreating. The required strength of packwall should be relatively independent of length of face, and hence the longer the face the amount of pack building per ton of coal won should be reduced. But there are some reservations about increasing unduly the length of such a wide span of prop free front. On these bases, for shortwall advancing, expected maximum rates of advance should be one 3 m web per 6 to 8 hours. Any lengthening of the face would tend to increase this period and with the shortwall advancing system, with the conveyor boot possibly closer to the maingate face corner than in retreating, it could be possible. However, for design purposes a concrete should be sought to develop an acceptable strength in 6 hours, the period between 3 m advances of the supports, less placing time for concrete. It is envisaged that because of dust and other reasons, packwall pouring, at least at the tailgate end could not be concurrent with production.

(b) Longwall Advancing

The length of retreating longwall faces has been 110 to 180 m of solid coal, with maximum length dictated by need to minimise development as a compromise with capital cost of equipment and magnification of problems by scale and complexity. Rates of advance on the most successful faces, 130 m long, have been up to 23 cuts of 0.53 m per day - again, about 12 m per 24 hour day. As with shortwall advancing, with similar gateroad centre distances the length of advancing longwalls is say 9 m greater than retreating longwalls, with equivalent increase of coal per cut, and probably increased time per cut. As with shortwall advancing also, the longer the advancing longwall face, the smaller the amount of packbuilding per ton of coal won. Perhaps on these bases and on general overseas trends, the lengths of coal on the face should be assumed to be 200 m, with maximum rate of advance one 0.53 m shear every 2 hours. Perhaps it would be possible to pour concrete once every two advances, say, this requiring a concrete developing an acceptable strength in 3 hours, say, the period between each two successive 0.53 m advances of supports, less placing time for concrete. In the case of the pouring at the end of each shear, the "setting" time should be 80 minutes. Neither of these setting rates appears to be readily attainable with the common mix constituents tested so far.

(c) Longwall Retreating

The success of the retreating longwalls subsequent to development by shortwall, as compared with present retreating longwalls with conventional development, will depend upon the success of the monolithic packwalls in the new support system. If gateroad problems in retreating longwalls do not differ or are less than current problems in retreating longwalls, then productions should be similar. Whether the present tendency to guttering of gateroads - especially on the tailgate - along the rib of the longwall block - and the frequent deterioration of the gateroads generally about 40 m outbye the face are avoided remains to be seen. The possibility of widening of the span of goaf bridging the gateroads has been mentioned above. In a critical tailgate situation the face could be shortened and a small pillar left, to be breached at intervals, to support the tailgate provided gas was not a great problem. In the past this method has been used with retreating shortwalls with considerable success under poor conditions.

Design of Gateroad Supports

Basis

The basis for the support design is a slip form for rectangular section concrete dragged by the supports immediately adjoining the gateroads. The form would provide, perhaps with expendable additions complete casting of concrete from floor to roof and between the inside surfaces of the form. The width of the form would be determined by the strength of concrete relative to time. A steel form is envisaged, with a porthole to allow entry within the form prior to pouring and with some basis for constraining the sides from spreading during the pouring process. Although the concrete would be placed at atmospheric pressure, the sheer height of the pour, nominally 2.5 m, a normal seam height worked in Australia, would require significant constraint of the formwork. Constraint is required in two aspects, the first during the slurry condition of the concrete when particularly at the higher pressure bottom there will be a tendency to spread and, the second, for the form to align alongside the gateroad. The type of constraint for alignment could be perhaps added progressively on the gateroad and on the goaf side as the face advanced and pours were made - for the roadway side hydraulic props to be withdrawn outbye towards the end of the form after setting and replaced inbye along the future path of the form, maybe with rubbing plates as the form will need to skid past the supports. On the goaf side expendable slender timber supports could be placed between the last traction chock for the form work and the first normal chock along the face. The bottom of the form would skid along the floor and could clean the floor as it is moved forward. On the face and the gateroad side the form would have two components, a fixed larger base component and a relatively smaller upper component moved up to the roof and retracted from the roof by hydraulic cylinders. In the side of the top moveable section would be appropriately placed removable ports for introducing the concrete slurry hose. On the goaf side there would be a single fixed base portion of the moveable form and the cover in the top section would be by expendable thin board timber or other suitable sheeting attached to the prop line installed to constrain the goaf side form and seal to the roof. Some assistance in sealing should be obtained by any lowering effects of roof due to shrinkage of freshly exposed coal.

To allow installation of the constraining prop line on the goaf side the gap between the traction chocks and the next normal face chock would be increased either totally or outbye the forward canopie of the chocks to allow room for installing the timber props between the chocks. If shield type chocks are planned for the face then these should commence at the second or third chock adjoining the traction chocks so that adequate cover is provided for installation of the goaf side props within chocks of square back. The slip forms could be further constrained from spreading by sectionalised bolts in sleeves to be fixed through the form prior to pouring, particularly near the construction joint. Prior to moving the form forward after setting of the concrete the bolts would need to be withdrawn in sections abandoning the sleeves in the concrete mass.

At the gateroad entry to the face the first chocks would be staggered to give maximum support, with clearance. If this end of the face is made square, there would be hydraulic prop support in an arc protecting this corner. The cutting of the 3 m web along the face would leave the tailgate end square.

Additional gateroad supports would comprise roofbolted cross straps and/or steel or timber bars. These would be set square in the tailgate and square with a square maingate face-end, or with a turned maingate face-end could be set at an angle to maximise support at the maingate end. Some of these details are shown in Fig. 2. The boot-end of the conveyor would be outbye along the maingate, and would be advanced periodically. Apart from the continuous miner and shuttle-car, a second shuttlecar could be used as a surge car. Concrete pouring, roofbolting and moving chocks (apart from face ends) could be done concurrently.

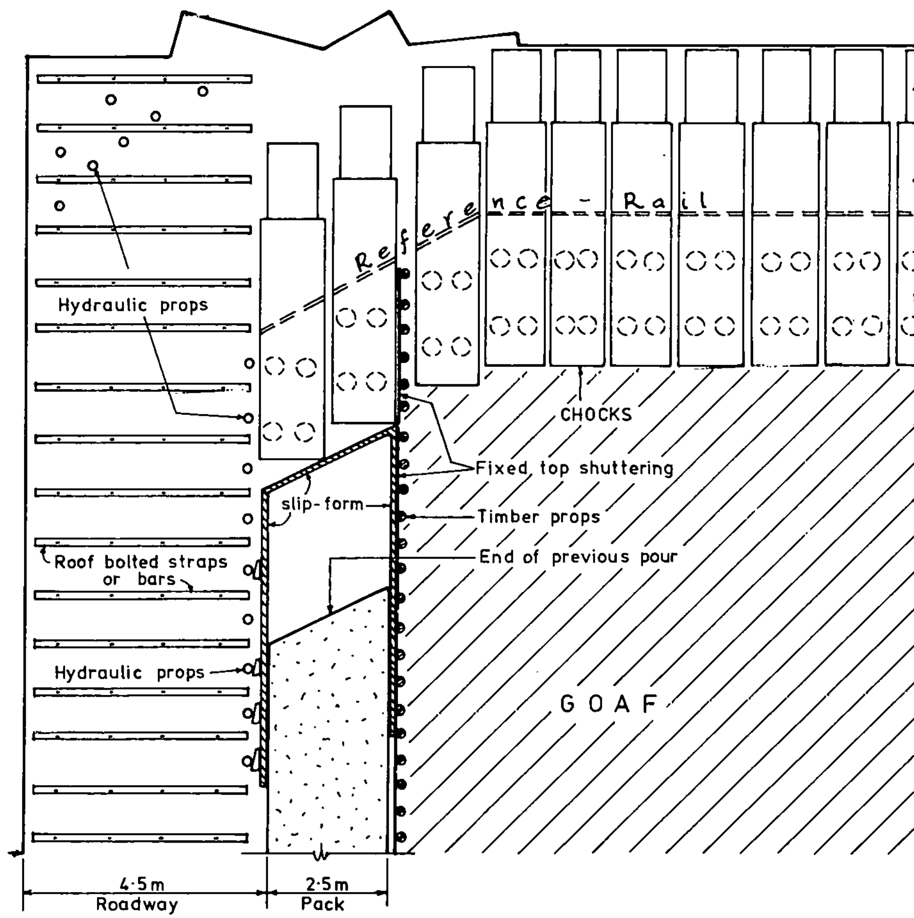


Fig. 2 Detail of maingate supports and gate-end of face.

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Design

The usual design of pillars in coal mining does not take into account pillars of the width to height ratio envisaged in the gate-road supports. However, in estimating the strength of the supports relative to their width for the assumed height of 2.5 m the pillar design formulae of several authorities have been used to establish an average criterion (Fig. 3). This clearly is an extrapolation to lower width to height ratios of largely empirical formulae from higher ratios and as such must be no more than an approximation. It is also an extrapolation from in situ coal to the packwall material. However, assuming the weaker concrete is with coal aggregate, the physical properties of such pack material should be closest to those of coal pillars, of the compositions considered. With these reservations, it appears that the strength per unit area of pillar markedly reduces with reduction of the minimum lateral dimension and that a pillar of width equal to 2.5 m is as narrow as should be attempted.

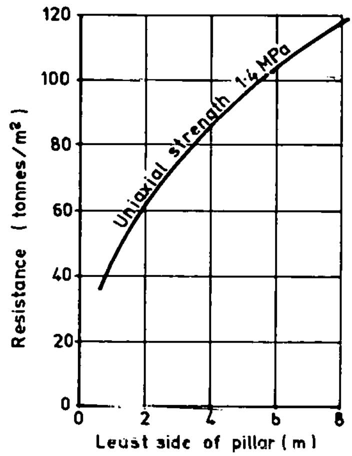

Fig.3 Relationship of least pillar dimension to-supporting capacity.

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Composition

The strength of the shortwall gate road support material should be such after a period of 6 hours that sufficient resistance is developed by the poured material. It is reasonable to assume that greater strength will be demanded in an increment of pour of greater length as loadings increase in the direction of the goaf. But the reduced convergences expected from shortwall work should reduce early loadings. Outbye in the vicinity of the rear abutment the 7 day strength will be more appropriate provided no failure has occurred.

Various attempts have been made to design a mix from Australian source materials which would achieve some strength in the 6 hour period and greater strength on curing. The aggregates considered were basically granulated slag and undersized coal. Fluidity for pumping was borne in mind. Various additives and cements were considered and undesirable materials for undergorund use were avoided. Higher cost materials were not excluded from consideration where they markedly increased early strength and gave promise of marked reduction of volume of material and time in placement. At this stage strengths apparently possible are 0.35 MPa in 6 hours, 1.5 MPa in 24 hours and continuing to increase markedly up to 28 days.

Flexibility of the Method

The aspects in which flexibility might be required are:-

1. To accommodate faulting which is more possible to experience with an advancing method. The fault, if normal and of significant throw may require stopping of the face, drifting the gateroads through and re-establishment of a faceline on the other side. If convenient, the shortwall facelines for abandonment and for recommencement need not be at right angles to the gates, thus conserving some coal by swinging them more parallel to the faults.

   Without the conveyor of longwall, the removal of chocks from a face and re-establishing on a new faceline is less complex. Whilst protecting the old faceline with timber supports, the supports in ones or twos are winched, or towed by shuttlecar or tractor off the face towards their new position. (Martin and Hargraves, 1972). The position and nature of any fault so encountered could influence the integrity of the subsequent retreating longwall block as a single extraction area, and plans for the retreating work could be modified accordingly.

2. The possibility of monitoring strengths of gateroad supports to confirm design and in the light of experience (i) to allow variations of mixes, or (ii) to widen or narrow the gateroad supports. The monolithic nature of the packs would make more meaningful any load cell measurements of pack loadings. Flat jacks could be cast at any position within the pack and perhaps used in conjunction with gateroad convergence and coal stress change measurements to understand the strata control aspects of the method. The pack loadings could be used to design more economic mixes and dimensions of the packs.

3. As the ultimate requirement in the case of failure, the ability to be able to abandon the advancing shortwall method and to replace it in the same panel with heading development methods. As with shortwall retreating, if roof and floor problems become too arduous for the supports to control, the face can be abandoned by withdrawing the supports and narrow work can be reverted to, as in the barrier pillar section of the development - Fig. 1. The ends of the concrete packs would need to be extended within manual shuttering up to the abandoned face to seal the shortwall goaf. The possibility of later recovery of the pillar so formed between headings in the extraction area would be remote, unless some method could be used of extracting it with the retreating longwall face.

Multiple Slice Work

A great problem in Australia at present is the solution to high recovery extraction of thick seams. The immediate challenge is in the Bowen Basin of Queensland where coals are friable and where roofs are tender. The work is in open grass country with little need to maintain surface level or integrity as seen at this time. Any multiple slice work should be from the top downwards and probably would involve entire completion of a top slice before commencement of the second slice, at least in early instances. In the top slice of both shortwall advancing and longwall retreating the addition of materials into the caves to assist in goaf consolidation could be an advantage. The method of advancing shortwall to provide development for subsequent retreating longwalls appears to suit this multiple slice work in thick seams. The possible need for special chocks and packwall material in the second slice to accommodate the high loadings under the roadside monolithic packs of the first slice might be a major design aspect in such a method. There would seem to be no advantage in not having the second slice immediately under the first slice as such expected high loadings under the pack would have to be negotiated somewhere along a face underlying the completed upper slice. There is some possibility however that if coal is used as the aggregate for the work in the upper slice it may be possible to remove this support during the retreating of the longwall between two shortwall advance goafs in the top slice and thus obviate any stress concentrations in the lower slice. To achieve this a low ultimate strength for the top support material would be desirable to allow its machining or shattering.

Gas

In advancing walls, gas is derived from coal being won and from virgin ribs. Gas is also derived from extraction areas from seams above and below in amount which is a function of the minimum lateral dimension of the extracted area, and the distance to and thickness of virgin seams in roof and floor. For seams in the roof the amount is related also to thickness of working seam (Hargraves, 1973). The shorter the wall, the less gas is derived from adjoining seams per ton of coal won. Thus shortwalls should experience less gas/ton than longwalls. Also the assumed faster rate of face advance with shortwalls should mean that a higher proportion of gas in the working seam reaches the surface still in the broken coal.

In retreating longwall work, the retreating block should be rather drained of gas and so gas from the working seam (and the possibility of productive methane drainage) is considerably reduced. However, with the wide spans of goaf from the retreating longwalls plus "developmental" advancing shortwalls alongside, the tapping of a proportion of gas from seams in the roof and floor should be over a much greater vertical range, the range increasing with the total span.

The possibilities of concurrent methane drainage with holes in the roof and floor are therefore considered small with shortwall advancing. The possibilities of methane drainage from above and under the subsequent retreating block is higher, from holes drilled up and down from one or both gateroads. The holes would be of such geometry that they could be drilled from the pack supported gateroads and be preserved until passing into the goaf. The holes should be pre-drilled and oriented somewhat inbye. If the longwall was sufficiently long, the holes could be installed from intakes as well as returns.

The Bulli Seam is the most productive seam of metallurgical coking coal in N.S.W. and is the top seam of the Illawarra Coal measures and therefore does not lend itself to effective methane drainage from the roof. From the point of view of gas, the shortwall method appears to offer the best possibilities for advancing wall development for longwall retreating.

Experience in Australia has been that gas from surrounded - or nearly surrounded longwall retreating blocks has largely escaped by the time the retreating longwalls are to commence. The same must be said for multiple slice work - if the lower slices have similar geometry to the prior upper slices, then the gas should have largely emitted from the lower coal before the second slice is commenced and extraction should be under relatively gas-free conditions.

The nature of the packwalls is such that an adequate seal is provided between gateroads and goaf. Any surges of gas, whether caused by intermittent increases in span of roof or floor and consequent breaking or cracking to adjoining seams or by sudden drops in barometer, are no more serious than in any other extraction. In fact the relative impermeability of the monolithic packs allows better control of ventilation, and if a bleeding principle is desired to obviate the risk of high concentrations at the tailgate corner, a controlled flow through the goaf into the return can be provided by a controlled breach of the packwall at the outbye end. Except for this, the packwalls should be tight from the start to prevent movement from intake to return except through the entire maingate, and, with the possible partial exception above, around the tailgate corner.

Instantaneous Outbursts of Coal and Gas

It was stated that the use of longwall advancing has the effect of reducing any proneness to instantaneous outbursts of coal and gas by one category, relative to heading development. However the most prone areas in longwall advancing are the face ends, unless faulting is encountered, in which case a proneness extends across the face as the fault is traversed. Any such event would need treatment appropriate to the circumstances - perhaps relaxation hole boring in the vicinity of the fault.

For protecting the gate ends, advance relaxation hole boring could be undertaken, in density related to the proneness. For this purpose a mobile drill could be stationed in the tailgate and, when holes needed renewing, could be flitted up the tailgate and across the face. (Perhaps the same machine could provide the power pack for roof bolting at the tailgate end and for any drillholes in coal for tests to establish local proneness to instantaneous outbursts.) The slack coal from the relaxation hole boring would be picked up by the continuous miner on its next cutting web. The mobile relaxation drill may take up too much room in the tailgate and a stable could be prepared for this by continuing mining 2 m into solid coal across the tailgate in two successive webs; after boring appropriate relaxation holes to protect this work and then supporting the stable normally. This stable could be backfilled with weak coal aggregate concrete after abandonment to allow machining in the adjacent longwall retreat without creating local roof problems. If separate relaxation drilling capacity were required at the maingate, the conveyor and rail track outbye would rather restrict space available. But the maingate may be made wider than the tailgate for this purpose, if required.

The subsequent retreat should have low proneness to instantaneous outbursts and should be accomplished without precaution unless faulting is encountered in which case some local treatment may be required.

In multiple slice work in thick seams, with workings superimposed, there should be no risk of instantaneous outbursts in second and further slices.

Spontaneous Combustion

In room and pillar work, generally the risk of development of spontaneous combustion does not exist, with all roadways accessible, both intake and return, during development work. In pillar extraction, however, the complexity of the ventilation circuits and inaccessibility of goaf areas with incomplete extraction tends to increase the hazards. Shortwall advancing on the other hand, provides a less complex circuit and can allow extinctive atmospheres to develop in the goaf, thus suppressing the risks of spontaneous combustion. The more total extraction system, in any case, mitigates against the occurrence of spontaneous combustion, except where the coal is left, or where caving introduces coal from any upper seams into the goaf cave.

In the case of blind goaves, advancement of advancing walls to the dip should provide rapid development of extinctive atmospheres in the goaf (where the seam gas is largely CH_4, that is, lighter than air). In coals known to be subject to spontaneous combustion, a monitor on the main returns and any bleeder returns for CO will establish any development of spontaneous combustion; and bleeders may need to be closed down. With heavier than air seam gases, if still inflammable (i.e. over 23% combustibles) retreating to the rise will afford security.

With the ultimate aim of retreating normal long face longwalls between the monolithic pack-supported maintained roadways of previous advancing shortwalls, these retreating longwalls may be retreating outbye, leaving a blind goaf behind which could be subject to aeration particularly if retreating updip, with lighter than air seam gas. To retreat downdip under these circumstances should minimise the risk of spontaneous combustion. If desired, bleeder roadways could be provided to the inbye ends of goaves - both from development by shortwall, or through drillholes from the surface, adequately protected on surface, to remove high concentrations of CH_4 in the goaf. As the shortwalls advance in one direction and the longwalls retreat in the opposite direction, any choice of orientation relative to dip should be a compromise between the effect of dip in both situations.

Subsequent Retreating Longwalls

Much of the success of the proposed methods will depend on the competence of the roof and floor in the gateroads, and this will depend on the nature of roof and floor, the depth, and, on the assumption that roofs are relatively inflexible, the supporting capacity of the gateroad supports.

The ideal, from a conservation point of view would be to employ one advancing shortwall to develop each retreating longwall, after the first retreating longwall, and to continue expanding the goaf by such repetetive total extractions. The history of such extraction, without intervening pillars, is absent from Australian experience, but the somewhat analagous situation of extraction of successive panels by the local "Wongawilli System" of pillar extraction, albeit incomplete extraction, usually has the result of crippling problems developing at about the third block extracted. The aim in the shortwall advancing - longwall retreating work should be to attempt total extraction by extracting without intervening pillars but always making the intake, maingate on the side away from the goaf. If troubles are experienced due to interaction between goaves, etc., then the affected longwall retreating face could be shortened by drivage of a single separate gateroad, leaving a pillar of appropriate width, maybe 30 m, which could be regarded as indestructable at seam height of 2.5 m.

With such problems occurring, a decision should be made whether to use two shortwalls to develop each longwall, or whether it would be possible to mine one or more retreating longwalls alongside the goaf.

It is possible that gateroad deterioration could occur during longwall retreating under the influence of the front abutment of the longwall. In general longwall practice brushing might cure this problem, or cure it temporarily.

In the foregoing development of the method numerous alternatives have been suggested, including shortwalling across the end of the imminent retreating block to provide the starting roadway for the longwall. It might be possible to turn the development shortwall around a 90° angle for this purpose. Either the inbye roadway of this shortwall across the end, or the inbye roadway of a pair of headings could be used as a bleeder return for the retreating face. As the starting roadway may be wider than normal, with spread supports to allow movement of chocks and their installation, a wider road may be required, and for this purpose a pre-driven sacrificial roadway a short pillar inbye could be preferred.

Conclusion

The proposed method of development by self advancing supported advancing shortwall has not yet been tried, so far as is known. The method seems to overcome some problems in development presently experienced in Australia, provided suitable rates of advance can be obtained at reasonable overall cost, and provided that the efficiency of the subsequent retreating longwall is not greatly impaired. There seem to be benefits from the method in handling the problem of instantaneous outbursts of coal and gas, now assuming greater importance in Australia.

Because the method is new, and has a number of unknowns it seems that the best approach is to have a trial under relatively simple physical conditions - a uniform seam at relatively shallow depth, and as a single advancing shortwall. Given some promise, a block of coal for longwall retreating at greater depth could be developed by an advancing shortwall on the one side, and a pair of headings on the other, to give an immediate basis of comparison. If the test were conducted on a single shift basis, with the completion of a web as the task for each shift, cheaper packwall concrete based on some ordinary Portland cement could be used for the 16 plus hours of available setting time. In subsequent normal production, such a cheaper mix could be used at the completion of the 5-day normal production week. If serious problems occurred in the shortwall compared with the pair of headings after a reasonable trial, the shortwall could be converted to a pair of headings for continued development of the longwall. If the longwall on the retreat indicated worse conditions on the shortwall developed side than the other, the method would be thrown into doubt. If, on the other hand, the shortwall advancing proved successful, it should give a decided fillip to longwall retreating in Australia with consequent improvements in recoveries.

Acknowledgement

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MODERNIZATION AND MODIFICATION OF LONGWALL SYSTEMS AT ISLAND CREEK COAL COMPANY

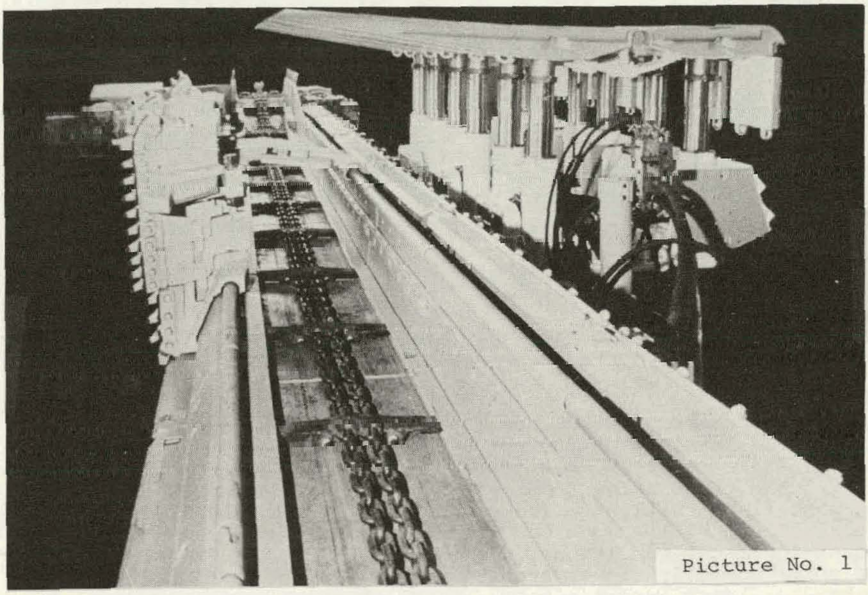
Jesse J. Curry, Jr.
Superintendent of Underground Maintenance

Richard Herron
Vice President of Maintenance

Island Creek Coal Company
Keen Mountain, Virginia

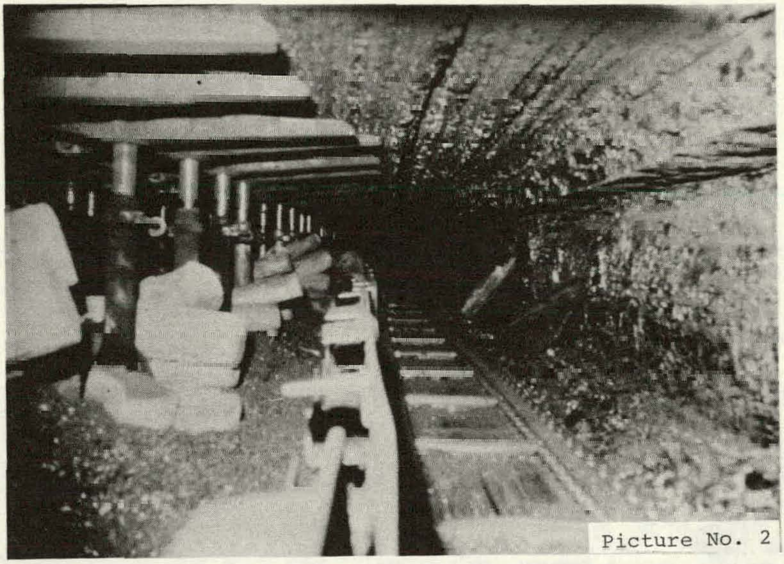
Island Creek's first experience with longwall mining began in 1953 at Holden, West Virginia, and was the second installation in the United States at that time.

The plow that was used was very small compared to the ones we use today; however, it is similar in design to our latest Gleithoble.



Picture No. 1

The roof support was of a friction type using a forepole header, and this support had to be moved by hand. The conveyor rams were compressed air operated.



Picture No. 2

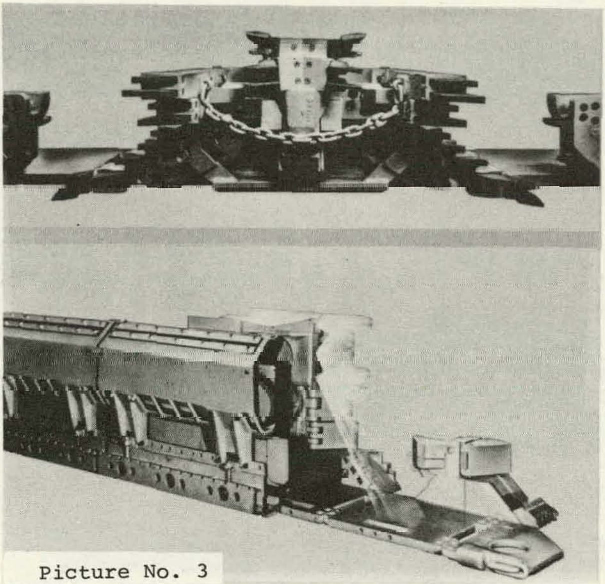
Longwall mining machinery has improved a great deal since then.

At the present time, Island Creek is operating seven longwall faces in the Virginia Division.

In September, 1966, Island Creek renewed its longwall mining by installing a plow in the Pocahontas No. 3 Seam at their Beatrice Mine.

The immediate roof at the Beatrice Mine is composed of shales, sandy shale, and sandstone which varies in thickness. Very often the immediate roof is made of laminated banks which are "squeezed out". The floor below the Pocahontas No. 3 Seam is medium to soft. The mine floor will have about 12 to 18 inches of immediate soft material and will heave on occasions when face advance is slow. Except for local conditions, the Pocahontas No. 3 Seam is flat lying. The seam normally ranges from 52 to 60 inches thick but will go higher or lower under some regional conditions. The overburden ranges from 1,200 to 2,000 feet and contributes to the large quantity of methane present. Because of high methane content in the roof rock, the gob side of any support must be open to allow adequate air flow to the active caving zone.

Due to large amounts of methane being liberated and the fact that the coal is very soft, it was decided, at this time, to use a plow instead of a shear machine. The type of plow that was selected was a hook type connected by a plow chain and pulled on the gob side.

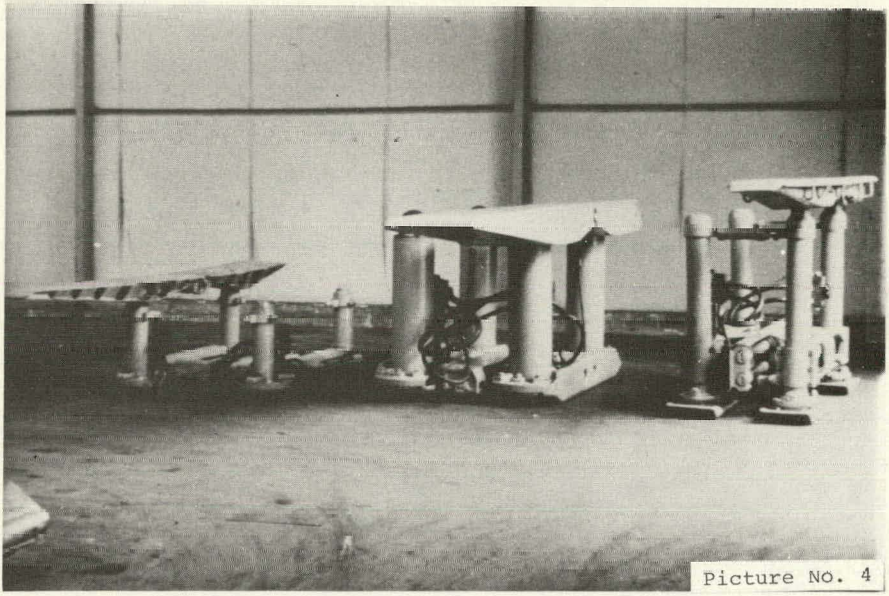


Picture No. 3

The chain was 22 millimeters in diameter and was captured by covers close to the floor level. The plow motors were two 60-horsepower, 440 volts, 3-phase. One was located at the head; the other, at the tail. The face conveyor was 650 millimeters in width and was driven by three 60-horsepower motors, two at the head, and one at the tail. The outboard driven conveyor chain was triple stranded, with an 18 millimeter diameter, and a flight spacing of 960 millimeters.

In selecting the support, a look at the strata profile showed that a massive sandstone was approximately 30 feet above the coal seam. It was decided, at that time, to have a support designed that would have some cushioning effect since this was the first longwall at this depth.

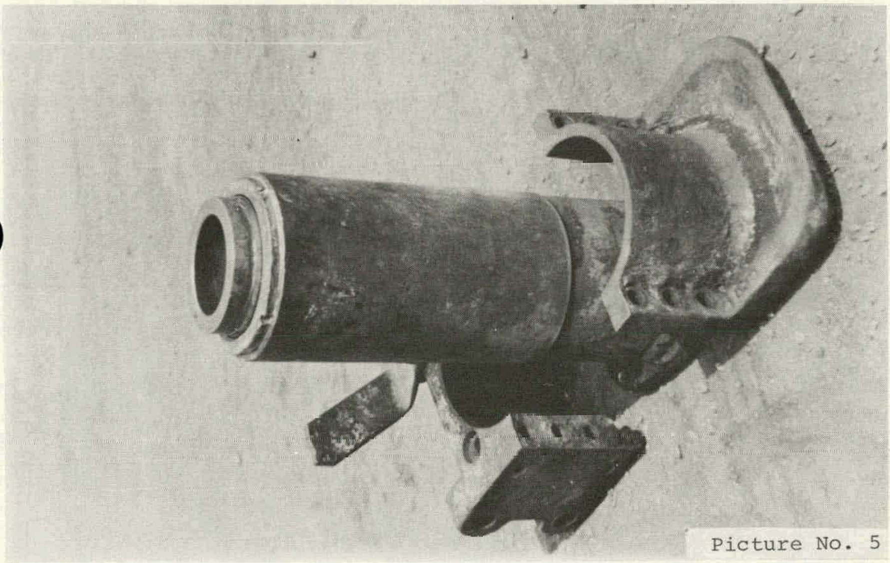
The design was such that a doughnut type cushion made of a plastic like material was sandwiched between the bottom of the cylinder and the support base followed by a flange made of the same plastic material. This was retained by a metal ring and by lugs and bolts similar to a truck tire.



Picture No. 4

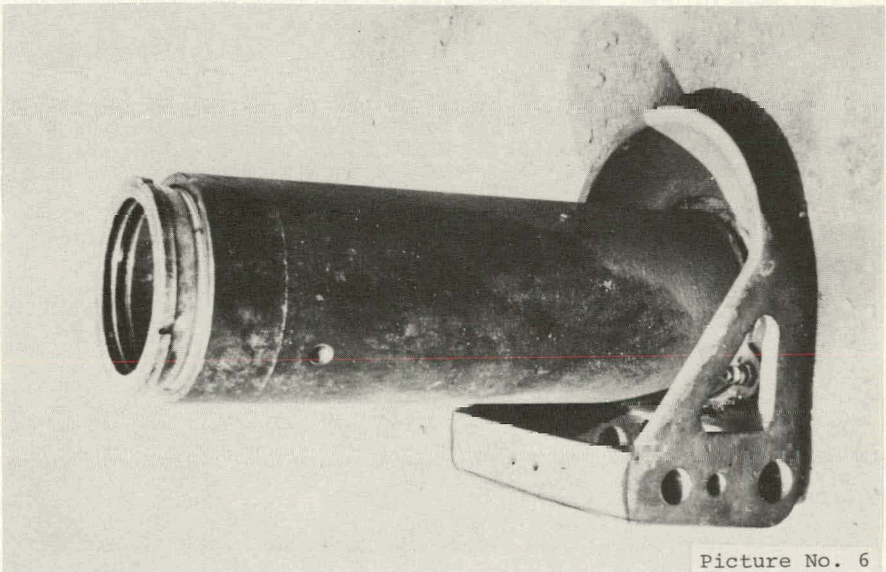
Two cylinders each were mounted on two cast steel bases held together by an advancing cylinder. However, once the support experienced yield loads, it would not retain its vertical shape causing the flange and doughnut to become distorted due to lateral loading. This resulted in a very serious safety problem and a high maintenance cost, not to mention the loss in production. We also experienced other problems with this support; mainly, in the one piece cast steel base, it was not designed to follow undulations.

The second type of support was of a different design. It was a combination of a cast steel base and spring steel hinged at four points allowing it to follow undulations better. The design of the cylinder itself was machined to accommodate a retaining plate and held on by a series of socket head bolts. The cylinder became part of the base and had direct contact with the floor. This proved to be the best design from a maintenance standpoint of any support that we have any experience with.



Picture No. 5

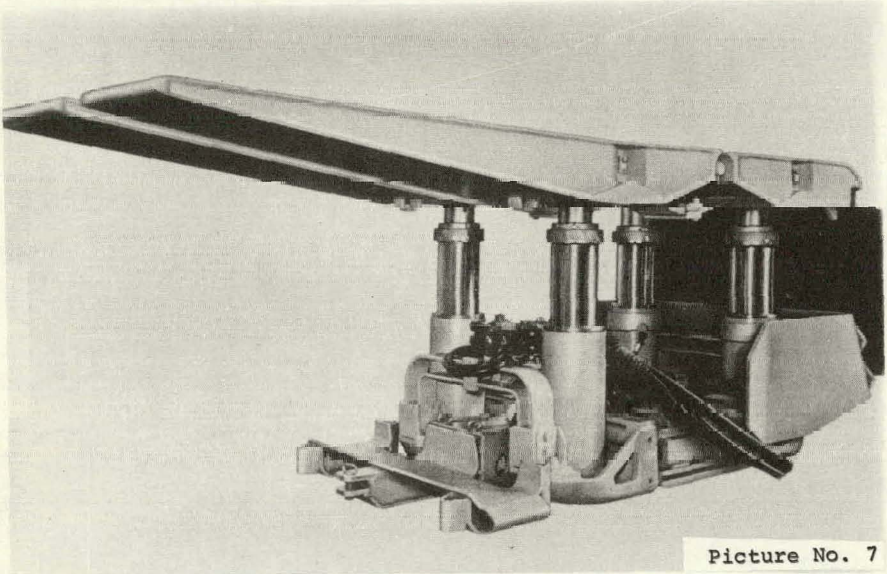
The third type of support was of the same cylinder dimensions in diameter as the rest, but the cylinder base was welded to the support shoe and again became part of the base.



Picture No. 6

In replacing these cylinders, due to leaks, etc., we had difficulty in removing the retaining roll pins due to seizing. This was due to salt deposits on the floor.

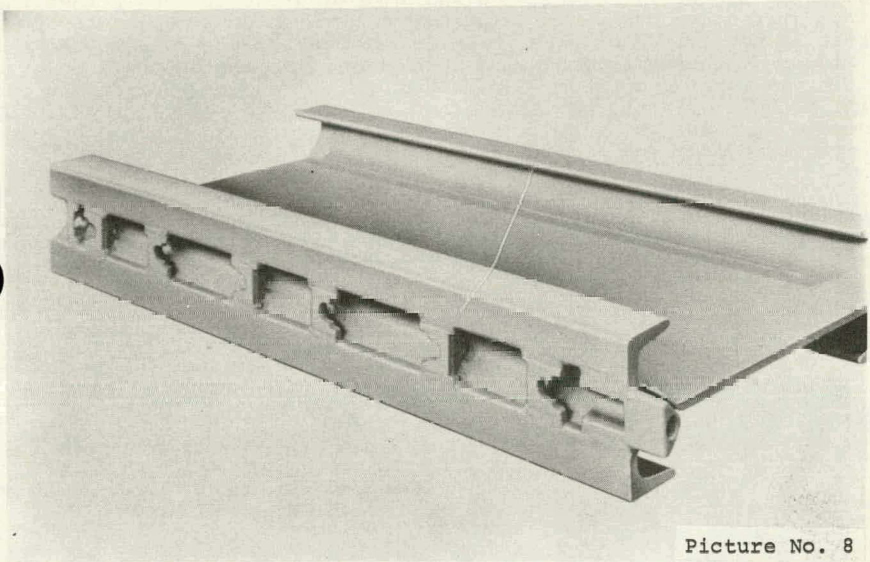
The fourth type is a chock; the cylinders being of the same design as the third type. Although either the two front or two rear pistons could be lowered separately, the complete chock had to be advanced together. This was different than the frame type where only two legs are advanced at different times.



Picture No. 7

In selecting a longwall system for the first time, you must rely on both the manufacturer to give you the best designed equipment to suit the conditions at that particular face and field experience available from other companies using longwall systems, especially European companies.

The conveyor system that was selected was 650 millimeters in width and had a 1/2 inch deck plate.



Picture No. 8

We experienced a lot of wear both in the deck plate and the chain raceway. After the first panel was pulled, it was decided to try to repair these pans. So, after selecting the best 50 percent of the used pans, they were sent off to an outside job shop for repair. After using the reconditioned pans, we found that they had to be replaced before the panel was half way out. So it was decided, at this point, not to use anymore reconditioned pans until we could establish some particular guidelines in gauging tolerances.

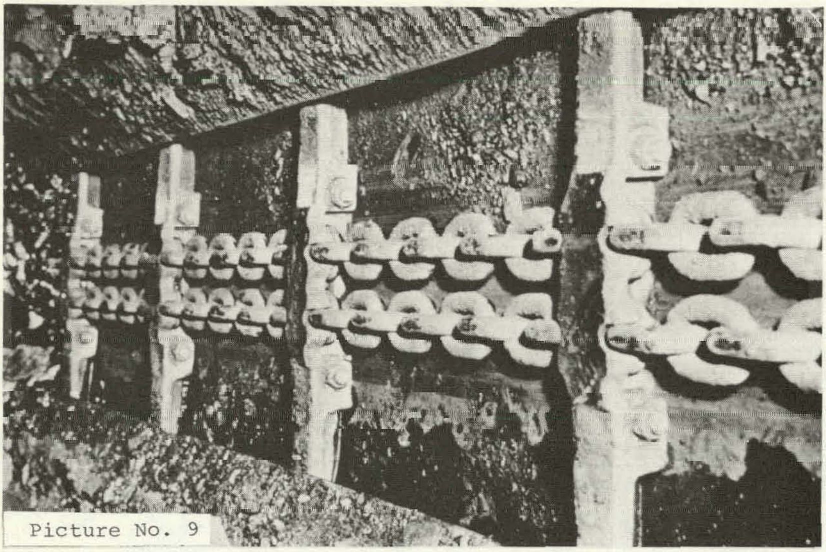
After presenting the pan wear problem to the manufacturer, they increased the deck plate to 5/8 inch. However, it later became necessary to change this to 11/16 inch, which we are now using.

Other problems were encountered on the face conveyor. The spill plates on the face conveyor also served as the back guide for the plow chain. Where the plow chain rubbed these spill plates were worn to a point that bending occurred, thereby, derailing the plow chains from its bottom guide, thus, causing the plow to wreck. This problem was also brought to the manufacturer's attention, and a new design of plow guides and spill plates was made available.

Some of the most serious problems are encountered at both the head and tail drives, especially in the advancing of the longwall. Although physical conditions are part of this, there is still room for vast improvements. We have tried several head and tail drive systems. The first one was advanced by a slide bar arrangement which served a dual role by guiding the face and acting as a stabilizer by housing two roof support cylinders attached to it. This drive frame had the plow drive speed reducer on the gob side.

The second type of drive had the same type of slide bar arrangement but had the plow drive speed reducer on the face side using a cross shaft through the drive frame. To allow for the drive shaft, the angle of the discharge end was more of an acute angle than the first system; but the return conveyor chain cut into the cross shaft, creating not only a problem in shaft damage but carry back coal has a closer opening to cope with, thereby, hanging of the conveyor chain was more prevalent. This type of drive was replaced by one similar to the first, but advancing was done by a set of chocks, to advance the drives and a tensioning cylinder set at a right angle to tension the face and plow chain.

The latest type that we now use has a twin conveyor chain, 24 millimeters in diameter and inboard center driven sprocket. This type of frame has the plow sprocket speed reducer drive mounted parallel with the face conveyor. This allows the roof support to b moved closer to the conveyor; and, thus, better roof support can b attained.



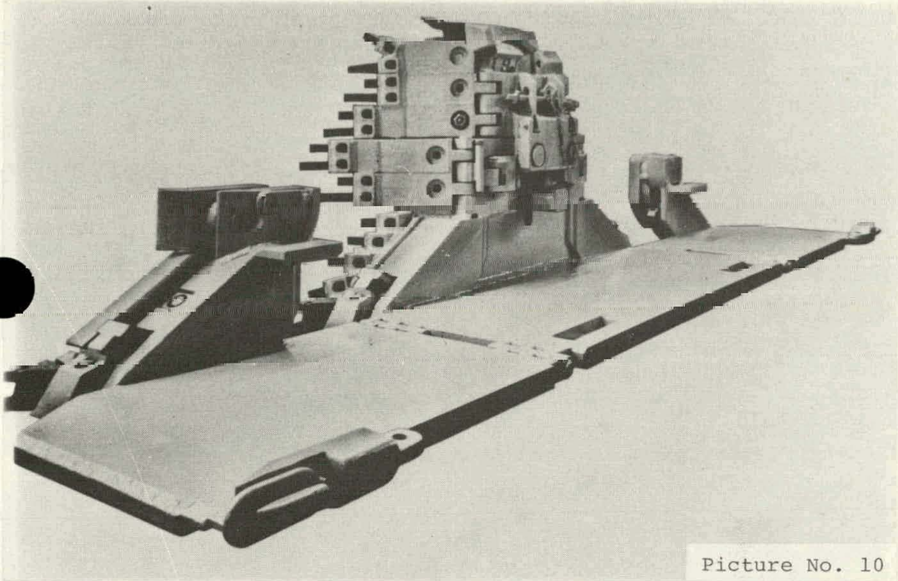
Picture No. 9

The canopies on all of our supports are hinged near the center to allow better contact to the roof by being able to more closely follow the contour of the roof.

Our first plow design was a hook type, with a three piece base very small compared to the ones we use today and much lighter i weight.

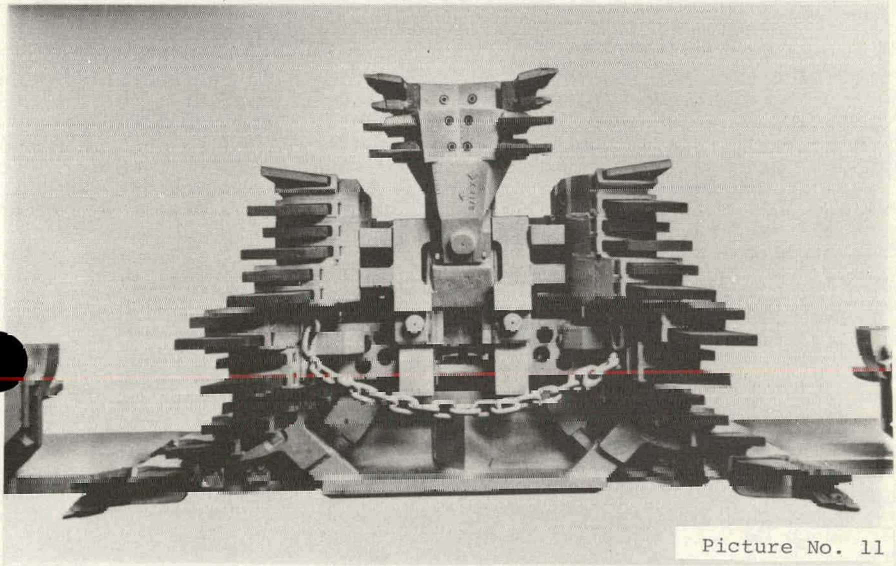
The plow chain that was first used was 22 millimeters in diameter but was later changed to 24 millimeters. Due to lengthening of the face and the added horsepower, needed for the extra distance, the size of the chain was increased to 26 millimeters. We anticipate using a 30 millimeter chain in the future.

The second plow was much the same design but heavier and had several more bits. It could be raised or lowered by a hand operated hydraulic cylinder.



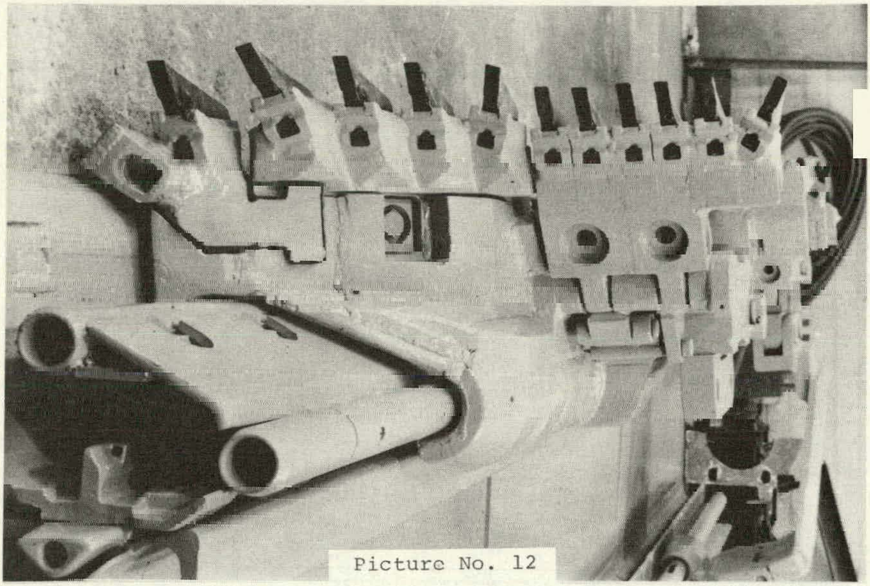
Picture No. 10

The third plow had five sections. This had two distinct improvements. It would follow bottom undulations better, and it would allow clear rib cutting at the head and tail.



Picture No. 11

Our latest plow is a Gleithoble type, using a 26 millimeter chain, captured in guides and pulled on the face side. This plow is about twice the size and weight of the previous plows and has a very good bit arrangement. This plow does not create the wear on the pans as does the hook plow, so a longer pan life can be expected. Since the plow chain is directly in line with the plow, better torque and accurate plowing can be acquired.



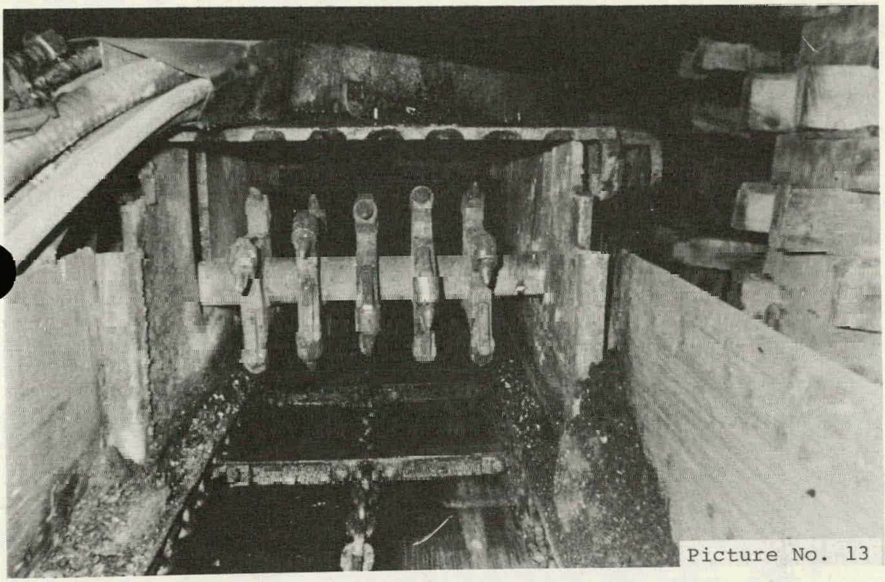
Picture No. 12

The first entry conveyor or mother line had the same dimensions as the face conveyor but with closer flight spacings. All other components such as pans, chain bolts, etc., could be interchanged. This conveyor was driven by a 60-horsepower motor through a speed reducer, but had a speed somewhat faster than the face conveyor. At times, it was found necessary to unload this conveyor by hand when it was stopped while loaded. The manufacturer came up with a two speed gear changer that would sandwich between the speed reducer and motors. This could be shifted to a lower speed to help move the load. Once this was accomplished, it could be returned to its normal speed.

We are now using a 125-horsepower motor, thereby, eliminating the need for the speed changer.

The need for a crusher became very evident during the extracti of the first panel.

Our first crusher was designed and built at our central shop. The drive consisted of a belt and pulley arrangement and used a counter-balance wheel. This was drive by a 30-horsepower motor using the same bits and holders that our continuous miners use. This proved to be very effective in coal, but somewhat less effective in rock.



Picture No. 13

Our second design had a motor speed reducer driven shaft, chain coupled, and had a lump buster or breaker type bit. It also had two hydraulic cylinders that could apply pressure against the rock when needed. We found this design to be very satisfactory, and it has become our standard crusher.

When rock is being crushed, it would sometimes break the conveyor chain. Therefore, the manufacturer designed a shear hub drive that will attach to the entry conveyor drive; and will eliminate this problem by shearing a pin.

The method we first used to convey coal from the entry conveyor to the belt was by a transfer conveyor. It was skid mounted and had no means to move other than by a chain hoist or a leverage such as a bar.

The conveyor was about 30 inches wide and had flights and chains similar to a shuttle car.

The second method tried was to eliminate the transfer conveyor and dump on the belt using the piggy-back method. It was soon discovered that a very poor relationship existed between the belt and the entry conveyor. At this time, the belt tail piece was modified to have two hydraulic cylinders for steering the belt under the conveyor. This system worked very satisfactory until the seam height would not allow piggy-backing.

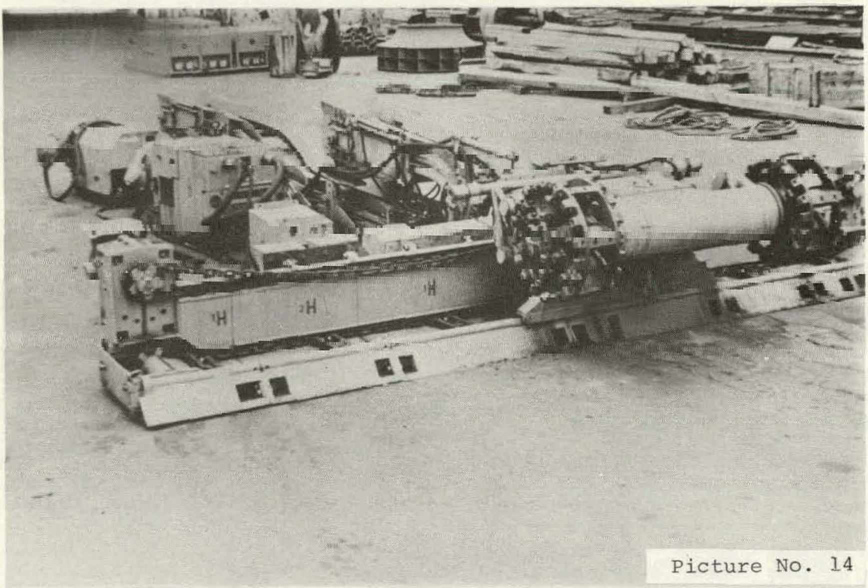
The transfer conveyor, at this time, was designed and attached to the entry conveyor in such a way that a good loading relationship could be maintained at all times. It was designed so that all movements of the transfer conveyor was done by hydraulics using the existing hydraulic system. This has proved satisfactory.

After several panels were pulled, the tail entry began to take weight and heaving occurred. It was necessary to keep this entry open so that the tail drive could pass or drive a parallel entry (stable hole), and drive it at about the same rate as you would the face.

The next method was a shop made air operated conveyor that was made for hand loading. This discharged on the face conveyor.

The third method was a stable hole machine designed by the manufacturer that was a combination electric, hydraulic, and mechanical machine. The ripper was driven by an electric motor, through a gear reduction box, and was transversed across the face by speed reducer driven by a hydraulic motor.

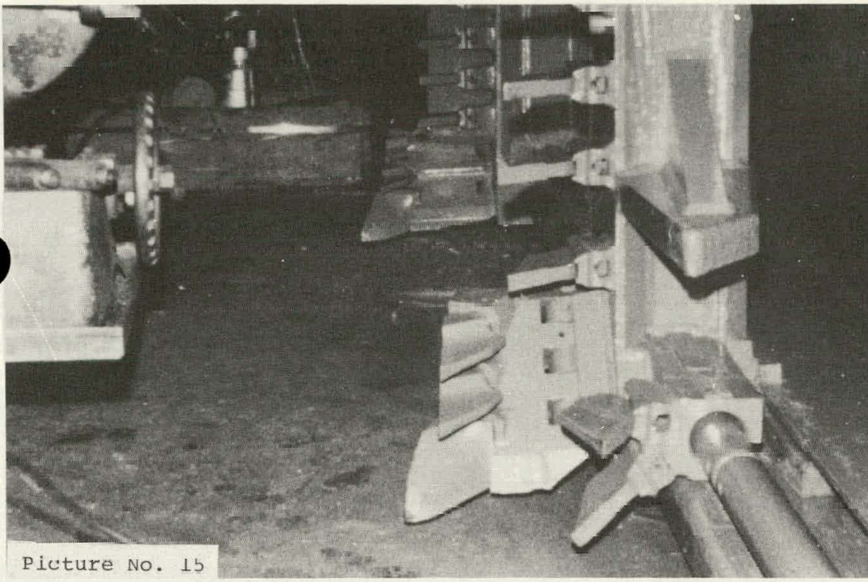
The conveyor was built in a "T" configuration using two conveyor chains linked together near the discharge point similar to a zipper.



Picture No. 14

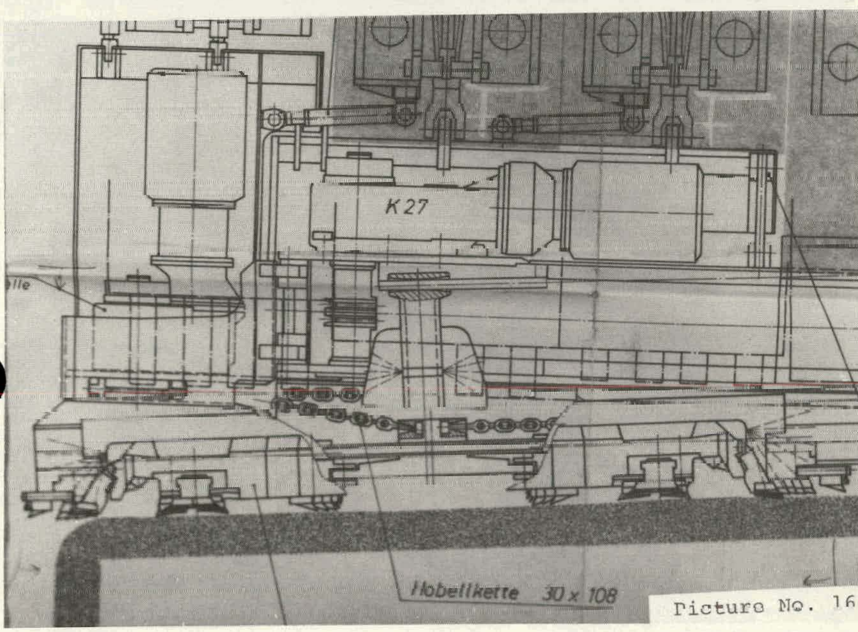
The next method was similar in design, but the ripper was be driven by a hydraulic motor without a speed reducer. This ap proved unsatisfactory due to crowding of the stable hole with supports, hydraulic pump, reservoirs, controllers, and conveyors. It became very difficult to operate. At this time, the hand loading method was again initiated.

There are two other methods we will try. One is a device that we designed and is called our mini-plow. It is operated on a hydraulic cylinder and is so arranged that it will cut in a tail end face area that the main plow cannot travel, thereby, eliminating the need for a stable hole. We are still working on improvement of this device.



Picture No. 15

The other method is a center hook plow that will allow cutting out on the tail and was designed by the manufacturer for the Gleithoble plow.



Picture No. 16

Our face lengths started at 270 feet, and we are now at 480 feet. We are currently developing for 600 foot face lengths.

When lengthening of the face becomes necessary, there are more things involved than just distance itself. More horsepower, a better designed gear box, and more powerful motors are needed to accommodate these extra loads. We have gone from 60-horsepower motors to 175- to 200-horsepower motors and from 300 KVA, 440 volts, transformers to 1,000 KVA, 950 volts, transformers and to larger electric controllers.

The Pocahontas Seam is very dry and very difficult to get in compliance in regards to respirable dust. Management and labor at our various mines worked with M.E.S.A. for about a year testi different sprays and pressures and experimenting with foam for du allaying purposes, and came up with an acceptable system using sole noids, that were controlled by a limit switch control, to operate the sprays in the most effective way. We have approximately 600 pounds static spray water pressure at the longwall face. We operate in sequence with the plow movement, 15 sprays in front, and 15 sprays in the rear. This sequence is controlled by a screw device operating micro switches which has a mechanical connection between the shear hub and the switch control. These micro switches open and close electric valves for the spray water sequence control.

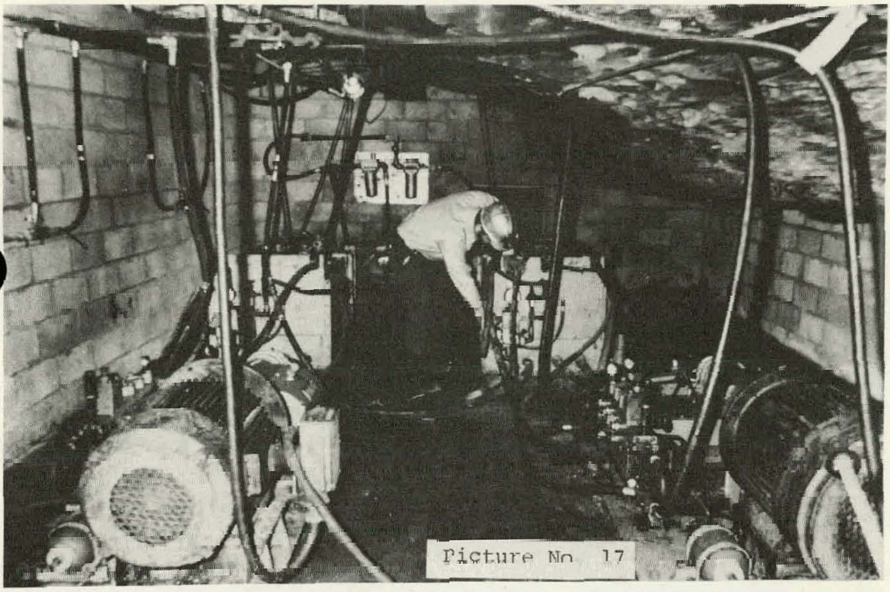
The hydraulic system is one of the most important functions of the longwall controls. The first system we used had two 7 1/2 gallons per minute pumps for the roof support operations. The tail and head drives were advanced by oil from (two) compressed air operated pumps, and the conveyor rams were operated by oil from a third air operated pump. These systems were of a low volume and high pressure.

To increase productivity, it was necessary to improve the hydraulic system. At our central shop, they designed a hydraulic system similar to a continuous miner using a variable volume pump with an output of 42 gallons per minute and at a pressure of 3,200 pressure per square inch. Since the air operated pump that was used to advance the head and tail was of a higher pressure, it was necessary to increase the size of the cylinders, thereby, getting the same effect.

We have recently begun using a water and oil emulsion with a ratio of 95 to 5, and we are also using a plug type hose with "O" ring seal, which we feel has several advantages. It prevents cross threading, hose twisting, and reduces leaks.

Since 1966, our pump stations have been part of the mule train, by being attached to the entry conveyor near the face.

In February of this year, it was decided to install the pum station at the mouth of the plow panel, approximately 4,000 fe from the face. At this time, we have had no downtime due to hydra lic pump failures.



There are several ways to benefit from this type set up. Better timbering can be had by the absence of the pump station. From the face area, it provides a less dusty atmosphere for the hydraulic system, eliminates handling of oil near the face, and eliminates the noise from the operator. We plan to pull as many as three panels without moving the pump station.

In summary, the main things that we have done to improve and modernize our original and existing longwall faces are better design in the following:

1. Roof support
2. Hydraulic systems
3. Crusher
4. Chain conveyance systems
5. Power and controls
6. Communication
7. Plow
8. Motor and speed reducer
9. Dust allaying
10. Belts and their related hardware

In conclusion, the improvement in any longwall system will depend to a large extent on mining people specifying what they want and having it custom designed to suit their needs and conditions.

Looking to the future, we will be seriously considering such things as shield supports, automated hydraulic systems, such as the back push methods, and taking advantage of electronic know-how in areas such as remote controls and wireless communication systems.

LONGWALL MINING THE HERRIN NO. 6 COALBED IN SOUTHERN ILLINOIS

Ernest A. Curth
Mining Engineer

Pittsburgh Mining and Safety Research Center
Pittsburgh, Pennsylvania

and

Mark D. Cavinder
Longwall Coordinator

Old Ben Coal Company
Benton, Illinois

Abstract

In April, 1975, the Bureau of Mines awarded a cost-sharing contract to Old Ben Coal Co. with the objective of demonstrating that the Herrin No. 6 coalbed in southern Illinois can be mined by longwall methods using shield-type roof supports. Earlier attempts at longwalling using chocks ended in failure. A minimum of three longwall panels will be mined at mine No. 24.

A premining investigation conducted by Dames & Moore laid the groundwork for mine design and equipment selection. Roof shields of the lemniscate type were chosen to control hazardous ground. The Bureau funded procurement of the roof support system, and Old Ben contributed the other longwall equipment. A ground control program included underground and subsurface instrumentation, a surface subsidence survey, and geologic mapping.

Longwall face 1 started operation on September 3, 1976, after intensive training of crews and supervisors. The major adverse condition during the course of the panel extraction was the occurrence of limestone concretions called coalballs in massive pods, which slowed the face advance and damaged the shearer and other equipment. The shields maintained adequate roof control even in faulty ground. The panel came to completion on May 7, 1977, when the face holed through into the recovery room. After the very heavy and bulky equipment was recovered and transferred to the next panel, face 2 produced coal on August 1, 1977.

Introduction

The Bureau of Mines and Old Ben Coal Co. concluded a cost-sharing agreement in 1975 with the objective of demonstrating that the Herrin No. 6 coalbed in southern Illinois can be mined safely with longwall methods. Though the first cost of a longwall system runs three to five times higher than that of a continuous miner unit, longwall mining offers the potential for improved productivity, higher resource recovery, better ground control, and generally easier compliance with safety standards. For the same reason, the promotion of longwall mining in the United States is a goal of the Bureau's research efforts under the Advanced Mining Technology Program so coal can be mined more safely and efficiently without higher social and environme costs.

Acknowledgements

The authors are grateful for the cooperation of the Old Ben Coal Co. in the preparation of this report. In particular, we wish to thank E. T. Moroni, John Janes, and Gary Gray, Dr. P. Conroy of Dames and Moore, and all the others who generously contributed so much of their time in providing invaluable help and advice.

Early Longwall Mining at Old Ben No. 21

Old Ben No. 21 mine near Sesser, Ill. was the scene of earlier attempts at longwall mining, as described by E. Moroni (5).[1] The No. 6 coalbed ranges from 7 to 9 ft. in thickness and lies under a cover of 600 to 700 ft. (Fig. 1).

The immediate roof is formed by a very strong fine-grained shale that contains 20% to 30% siltstone or fine-grained sandstone up to 70 ft in thickness. The main top is composed of sandstone, limestone, and sandy shale. The floor consists of a weak underclay 2 ft in thickness, which is underlain by limestone.

After four failures at longwall mining in the early 1960's, a fifth attempt was moderately successful during 1966-67. Figure 2 shows the location of the faces at mine No. 21 and the new panels at mine No. 24. The 600-ft-long No. 5 face was equipped with an Eickhoff shearer and conveyor. The roof was supported with Rheinstahl one-web-back chocks with stiff canopies and four legs rated to yield at 120 tons each. However, the operation was discontinued because of chock instability; productivity did not surpass or even equal that of coal extraction by continuous miner. The sixth (300-ft-long) longwall face was equipped with Gullick seven-leg chocks rated to yield at 560 tons. Periodic weighting, every 30 to 50 ft of retreat, was experienced at the face, and massive falls occurred in the head and tailgates 50 to 60 ft. ahead of the longwall face. Results did not improve after the face was shortened to 210 ft. The face was abandoned in 1970. Experience with four longwalls at Orient No. 5 and 6 mines of Freeman Coal Co. was similar, and longwalling was discontinued, too.

Mine Design for Old Ben No. 24 Longwall

In recent years, gradually decreasing productivity of full extraction with continuous miners made longwall mining again appear to be attractive if novel designs such as roof shields could solve the instability problem at the face and achieve control of a hazardous roof where conventional powered roof supports failed. Under the 1975 agreement between the Bureau of Mines and Old Ben, three panels, 460 ft by 1740 ft, are to be mined by longwall methods at Old Ben No. 24 mine near Benton, Franklin County, Ill.

Mine No. 24 is an all-belt-haulage mine with four shafts intersecting the Herrin No. 6 coalbed at a depth of 600 to 650 ft. The coal output, approximately 9,000 tons per day, is hoisted by skips, and, after passing a Bradford Breaker without any other preparation, is conveyed to a surface storage and loading facility whence unit trains take it to various utilities. Surface elevations of the gently rolling country range from 380 to 460 ft above sea level. A surface structure, the Rend Lake Dam, which impounds a large body of water, limits the longwall mining area to the north. A large fault system, the Rend Lake Fault, trending north-south, probably passes within several hundred feet of the proposed longwall panels according to projections from earlier exposures.

The panel development entries in sets of three on 60 and 100-ft centers are driven with drum-type continuous miners. The belt is in the center entry. The standard entry width is 14 ft. except for the belt entry, which is 18 ft in width to accommodate power center and power pack for the longwall (Fig. 3).

An entry height of 7 ft is maintained in the 8.5 to 9-ft No. 6 coalbed by cutting to the roof rock and leaving floor coal in order to avoid exposure of the soft underclay floor. The roof is supported by resin bolts on 5-ft centers, and a coat of sealing material is applied to roof and ribs. The longwall panel design is unique in that the longwall face includes a row of chain pillars on 60-ft centers and therefore will have a length of 460 ft on centers. Retreating a row of chain pillars is an unusual and first-to-be-tried method. Three gate roads must be maintained: the headgate, the supply gate, which is tracked for supplies and mantrips, and the tailgate. Pillar extraction by continuous miner of the panel adjacent to the tailgate preceded the longwall mining, and this activity could affect stress conditions in the longwall panel.

1/Underlined numbers in parentheses refer to items in the list of references.

Premining Investigations

Dames and Moore, Geotechnical Consultants and subcontractors to Old Ben, conducted a premining study comprising the following investigations:

1. Roof and floor bearing-capacity tests (2) (Fig. 4).

2. Bed separation in the roof strata.

3. Stress analysis of damage to chocks salvaged from No. 5 face of No. 21 mine.

Based on the results of the investigation, Dames and Moore presented the following recommendations:

1. A minimum of 6 in. of floor coal should be maintained because the soft underclay fails at 300 psi when wet.

2. Mean load density should be not less than 9 tons/sq ft considering strata separation at 60 to 70 ft above the coal plus cantilever action in very stiff roof strata (1).

3. The roof support must provide stability against lateral thrust parallel to the face and a continuous canopy overhead.

Roof Support Design

Following the above recommendations, Old Ben and Bureau engineers developed specifications for roof shields. Bids from manufacturers were invited, and the diagrams show how the mean load density requirement was evaluated for two types of shields, the caliper or single-jointed shield and the lemniscate or double-pivot bar shield (Fig. 5). The canopy tip of the caliper-type shield describes a circular arc when the shield is raised or lowered. Hence, the critical span between canopy tip and face widens with increasing height of the shield unless compensated by an extension. Caliper shields also develop a horizontal thrust towards the face when yielding. The canopy tip of the lemniscate shields moves up and down in a nearly straight line which is the center part of a lemniscate curve so that the distance between canopy tip and face remains nearly constant.

The Thyssen roof shield model RHS 18/30 L was selected because it satisfied the following requirements:

1. A mean load density in excess of 9 tons/sq ft.

2. A hydraulic travel from 6 ft to 10 ft in one stroke.

3. A yield pressure of less than 6,000 psi.

4. A span of 12 in. between canopy tip and face before the shearer pass.

5. Each half of the base can be lifted individually.

Shield Design

The shield weighs 17.5 tons, and its overall length is 16.5 ft. Figure 6 shows the shield in the one-web-back position before the shearer pass, and Figure 7 shows details. The shields are set with a pump pressure of 4,350 psi and are yielding at 5,500 psi through spring-loaded relief cartridges which open up at this pressure. The two legs are of a special design to pack great power in a relatively confined space. Each leg can be set by pump pressure to exert a thrust of 259 tons and then will sustain a load of 328 tons at yield, or 518 tons and 656 tons respectively for both legs. However, the loads are introduced to the roof at the hinge of the canopy and, due to the geometry of the shield, are subject to a mechanical disadvantage which is a function of the height of the shield and is 0.78 at 7 ft of mining height with this shield. Therefore, the shield can be set by pump pressure to a thrust of 518 x 0.78 = 404 tons and yields at 656 x 0.78 = 512 tons. The shearer with conveyor is also depicted to indicate the relative position.

The distance between the canopy tip and face is 12 inches, and the shield advances by a step of 24 in., which is also the width of the shearer drums so that, after the shearer has passed, the maximum roof exposure approximates 3 ft. The canopy extensions, called "flippers" by the miners, are 24 in. long and can be lowered so that the shield can be brought closer to the face to further reduce the area supported by the shield. In this closed position, the shield is rated to yield at 12.50 tons/sq ft before the shearer pass and 10 tons/sq ft after the shearer pass. The ratio of the front portion of the canopy to the rear portion is slightly greater than 2 with extended roof bar, and the canopy can be swung against the roof by a ram to improve roof contact with a measure of stability during the shield advance. The canopy tip load exceeds 5 tons at yield.

The base consists of two halves and exerts a floor pressure of 311 psi at yield, including the 35,000 pound weight of the shield. Each half can be lifted individually by its double-acting leg so that the advancing shield can overcome adverse floor conditions such as uneven or soft floor and crushed or loose bottom rock or coal. The base is designed to be self-cleaning in that space is provided for slack coal or debris to pass through into the gob when the shields are advanced in order to eliminate hand shoveling.

The forward edge of the base is ahead of the projection of the canopy hinge joint to counteract the tendency of the shield base to dig into the floor. For the same purpose. the advancing ram is mounted in an inclined position to raise the base slightly during the forward stroke. The advancing ram acts through reverse linkage so that the full force of the piston end exerts the pull for the shield advance.

The hydraulic controls are arranged to actuate the operation of each shield from the adjacent unit. An automated control circuit maintains a soft roof contact during the shield advance. The objective is to keep the roof intact directly after exposure. A manual override is provided to overcome roof irregularities by retracting the legs. The rationale for control automation is to make maintaining roof contact preferable to lowering the shields by manual control. The speed of shield advance is commensurate with a shearer haulage speed of 25 ft/min to minimize the delay of supporting the roof after exposure. Hydraulic pressure in each leg can be measured by a load indicator (Fig. 8). The power pack provides a setting pressure of 80% of the yield pressure. A medium-pressure supply powers following functions: in-and-out movement of side plates, conveyor push, and active lowering of legs.

A safety travelway is provided along the face. However. it is somewhat restricted after the shields are advanced to a closed position relative to the conveyor.

Sideplates serving as seals are fitted between the shields and maintained by springs and hydraulic cylinders to prevent the dust from entering the working space during shield advance. Water sprays are mounted on the canopy in a protected position but yet are readily accessible for inspection and maintenance.

Permissible lighting fixtures were being procured, and a test group of nine lights was installed. The procurement is being held up pending lifting of suspension of permissibility for the luminaires to be purchased.

Components and entire prototypes were tested at the laboratory of the Research Center for Rock Mechanics and Roof Support at Essen, Germany (Fig. 9) (4). Components also underwent the tests mandatory for application in German mines at the appropriate center.

The testing program at the Essen laboratory included the following investigations: The forces exerted on roof and floor; the support resistance and the bending moments in the floor skids; the penetration of the floor skids into the floor; the strength of the shield at one to three loading cycles; the durability of the shield under load cycling; the function and operation mode of the canopy, the canopy cylinder, and the angular positioning of the canopy; the stability of the roof support; the protection against flushing; the travelway, width, height, and behavior of shield on uneven floor; and the sideways mobility of the advancing ram.

Face Equipment

The Bureau of Mines funded procurement of the following equipment:

1. Ninety-five shields to form a shield line of 470 ft. Three of the shield are placed across the headgate and two in the tailgate. The shields are spaced at approximately 5-ft. (1.5m) centers to accommodate the 5-ft-long (1.5m) conveyor pans so that each advancing ram can be attached to the middle of each conveyor pan. Stresses at pan connections due to eccentric loading are thus minimized.

2. The power pack consisting of two high-pressure pumps, each supplying 25 gpm to a dual-pressure system, one emulsion plant to prepare the 5% oil-in-water emulsion which poweres the hydraulic system, and two filters.

3. One hundred fifty jacks with a range of 84 to 124 in. They are extended by connecting them to the hydraulic pressure system of the shields via hose and quick-connect pistol-type coupling. The jacks can be set to 30 tons of thrust and then yield at 44 tons. They are retracted by bleeding the fluid into the open through a valve.

These jacks are used for reinforcement of the three gate roads near the face as indicated on Figure 10 (3). They were placed in entries and crosscuts less than 80 ft from the face and remained in place until the face approached within 3 to 5 ft (Fig. 11). Several jacks in the supply gate were instrumented with hydraulic pressure gages to provide early warning of excessive abutment pressure ahead of the face, which never occurred; two instrumented jacks were set in the recovery room and yielded when the face cut through (Fig. 12). Figure 13 shows a view from supply gate to tailgate.

Old Ben contributed the following equipment:

1. A modified Eickhoff[2/] EDW 2 x 100 L double-drum shearer using two air-cooled 100-kw motors salvaged from an old shearer; 54 by 24 in. drums with two-start spiral vanes turning at 63 rpm; 4 in. picks, 26 on the vanes and 24 on the disc; each drum equipped with a cowl; integrated water sprays on the drums and sprays on the machine frame; 26-mm haulage chain with chain tensioners mounted on each of the conveyor terminals. Keeping the haulage chain in tension compensates for the elastic stretch of the chain regardless of the position of the shearer along the face. The shearer is trapped by guide tubes on the gobside and is guided on the face side by rollers which run on the ramp plate.

2. Halbach & Braun EKF 3-3-72 single-chain face conveyor system with two 150-hp drives; the drives are on the gobside and parallel to the conveyor; 30 by 108-mm chain; 180-ft/min speed. Emergency cutoff pushbutton stations are located every 50 ft along the panline.

3. One stageloader system with two two-speed motors, 40-hp each, and one 40-hp motor for the chunk breaker. The stageloader is made of two shuttle-car bodies and uses shuttle car conveyor chain for coal transport. It runs

2/Reference to specific equipment does not imply endorsement by the Bureau of Mines.

on four rubber tires; the inby end is attached to the face conveyor head drive; the outby end swivels on the belt tail piece, which allows retraction of the conveyor belt in 12-ft increments.

4. A 42-in. retractable belt conveyor. The original belt ropes were replaced with solid frame structures suspended from the roof by chains.

5. A power center for 440-V AC power.

6. Two motors, 75-hp each, to power the pumps of the power pack.

7. Communication by page phones located at 100-ft intervals along the shield line.

:e Installation and Training

Installation of the face equipment began in June 1976 when the armored face conveyor with its furniture of ramp plates and spill plates was assembled. A 12-ft high rigging chamber was excavated at the head of the staging area in order to assemble the shields, which were transported from the surface through the shaft as they arrived in the containers in four parts: each leg with its gob shield and skid half, the canopy, and the advancing ram. The shields were assembled by means of two overhead cranes. A track of three rails was laid along the staging room. After completion, each shield was skidded on this track by a scoop to its place in the shield line and connected to the hydraulic power circuits. The shearer arrived from Germany later in July and was taken underground in parts to be assembled in the underground shop and taken to the face (Fig. 14). The conveyor head drive and the stage-loader completed the face installation. After delays due to miners' vacations, a strike, and waiting for approval by the Mining Enforcement and Safety Administration, the longwall operation started on September 3, 1976, after intensive training of crews and supervisors on the surface and at the face.

At the nearby Rend Lake College, management officials such as the Vice President--Operations, General Manager, Corporate Safety Director, and Manager of Industrial Engineering had conducted the introductory training of face crews, union officials, and supervisors. They explained Old Ben's decisionto reintroduce longwall mining after earlier failures. Hazards peculiar to longwall mining were identified. Slides were shown, and a three-dimensional model designed as a teaching aid by the industrial engineering staff was used to simulate face operations in sequential order (Fig. 15). At the face, manufacturers' service personnel aided by industrial engineering staff initiated each crew member individually in the proper operation of each type of face equipment while slow production runs were made.

Technical training of maintenance personnel took place at the Old Ben central electric shop and at the Rend Lake College vocational facility, where the shearer manufacturer had provided a cutting motor and a haulage box wired for hands-on training under guidance of a service representative. Face supervisors had spent two days in intensive training at the longwall faces of a West Virginia mine. Job skill, safety training, and retraining classes are conducted continuously.

ce Operation

The standard crew consists of 9:2 shearer operators, three chock men, two utility men, one mechanic, and one foreman. The shearer cuts to the roof rock and leaves floor coal to achieve a mining height of 7 ft and to prevent exposure of the soft underclay floor. The Blueband, a very consistent parting, 1-1/2 in. thick and 18 in. above the floor, serves as the marker for the shearer operators, who endeavor to keep it level with the top of the conveyor pans. Figure 16 shows the unidirectional mode of operation as follows:

1. Start: Machine at tailgate.

2. Cleanup: Machine travels from tailgate to supply gate at high speed.

The conveyor is pushed up to the face as soon as the machine has flitted by.

3. Cutting: Machine cuts from supply gate to headgate. The shields are advanced as the machine cuts by.

4. Cleanup: Machine flits from headgate to supply gate. Conveyor is advanced as the machine cuts by (Fig. 17).

5. Cutting: Machine cuts from supply gate to tailgate. Shields are advanced as shearer cuts by (Fig. 18).

Unidirectional cutting has the following advantages:

1. The shields are moved forward as soon as the shearer has cut by.

2. The crew is in the intake air while coal is cut, so that exposure to dust is minimized. Intake air at a quantity of 25,000 cfm is conducted to the face through the supply gate. It splits at the face. One split of 5,000 cfm moves to the headgate, from where one subsplit reaches the bleeder via the gob and the other returns through 21 North entry. The main split of 20,000 cfm travels from the supply gate to the tailgate, where it splits again, one split going through the gob to the bleeder and the other returning through the tailgate. The sensor of the methane monitor is mounted on the tailgate conveyor drive. Power is cut off, should CH_4 concentration reach 1 percent.

3. The shearer pass is cleaned by the machine prior to ramming the conveyor to the face.

4. The tail-side drum is favored for cutting because the tail-side motor drives it exclusively. The head-side motor not only drives the head-side drum, but also the hydraulic pumps which power the haulage and actuate the ranging arms and other controls.

Little would be gained by bidirectional cutting on account of delayed turnaround at the face ends. Also, the crews would be exposed to dust. A prerequisite for bidirectional cutting would be hydraulic actuation of the cowls. Such a system failed, and the first panel was extracted with manual cowl control.

Pillar extraction by continuous miner of the panel adjacent to the longwall tailgate preceded the longwall mining; it did not affect roof control at the longwall face to any great extent, though the barrier between longwall and pillar extraction consisted of only one row of chain pillars or 86 ft of coal in width. The tailgate was subject to some pressure, which was sustained by the hydraulic jacks in addition to the two shields in the tailgate and by cribbing built across the crosscuts.

The greatest adversity during the mining of the first panel was the occurrence of coalballs, sideritic limestone concretions, ranging in size from golf balls to footballs (Fig. 19). They were first encountered in two large pods in the tail entı development and then at the longwall face, at first near the tailgate and later at the center. All through November and three weeks in December, the crews battled the massive pods with devastating effect on productivity. It was necessary to blast at spots where the coalballs displaced the entire coalbed or stuck to the floor. Illinois State Law permits blasting only when nobody is in the mine. Consumption of shearer picks at $11 per piece was very high, and numerous bit blocks on the drums were cracked. The bit blocks were welded to keep the shearer operating until a new tail drum could be delivered. Also, the stageloader with the chunk breaker and even the Bradford Breaker on the surface suffered severe damage.

Ground control in longwall depends on the adequate load-bearing capacity and reliable function of the roof support (Fig. 20, 21, 22). A good rule for bringing the full hydraulic pump pressure to bear is to hold the setting pressure for 1 minute at each shield. The shields maintained roof control even when the face struck a roll which displaced the entire coalbed and caused rock falls 20 ft in height, well above the maximum shield range of 10 ft.

Maintenance of the shields was generally satisfactory. Items reported were deformation of side plates and cylinders and of some canopy tips. The most frequent defect, which involved all shields, was breaking of shear pins in the advancing arrangement. The manufacturer came out with a new version, and Old Ben has also developed a more promising attachment.

To maintain the availability of the shields, the hydraulic system and particularly the yield cartridges have to be checked and adjusted periodically. A pump with gage is provided for such functional tests. As indicated by moisture around the valves, a few shields have been yielding during a period of idleness brought on by a wildcat strike. Numerous shields in the midsection and tailend of the face yielded when the face cut through into the recovery room at the completion of the panel on May 11, 1977.

The first panel delivered a total of 205,000 tons of coal on a length of 1,734 ft during 162 working days. The occurrence of coalballs during 7-1/2 weeks caused a serious setback on productivity (Fig. 23). The overall utilization factor was 49% with the following types of delays: shearer 17%, other longwall delays including roof control 13%, non-longwall or external delays 11%, conveyor 6%, and stageloader 4%. The cutting web averaged 21.4 in. in depth with the 24-inch wide drums, and the mean quantity per cut was 211 tons of coal. Daily averages in April were 8.2 cuts, 14.5 ft of advance, and 1,708 tons. At best, the shearer achieved 15-1/2 cuts or 3,317 tons in 24 hours, which can be considered the potential for this type of operation under constraints such as 24-in. drum width, unidirectional cutting, manual shearer haulage control, manual cowl actuation, chain pillar extraction, and dependence on the surge capacity and availablility of the main belt haulage.

Preparing Panel-to-Panel Transfer

While extraction proceeded in panel 1, the headgate development entries for panel 2, 22d, 23d, and 24th N, traversed faulty ground accompanied by water and gas feeders and then struck several pods of coalballs. When the new panel entries reached the point where the staging room for panel 2 was projected, it appeared that the chain pillar to be extracted first consisted of solid rib to rib coalballs. Therefore, the staging room was turned 80 ft outby the originally planned location.

A fault with 5-ft throw and trending N-S appeared in the staging room. Evidently, it is part of the system that intersects the main headings. This fault will stay in face 2 from the start to completion, and operators must pay attention to grade the shearer pass accordingly.

Meantime, a recovery area was readied for the move of the face equipment at the completion of the panel. Recovery of longwall roof support is always onerous and requires circumspect planning and execution. A shield move poses a particularly thorny problem because part of the structure is under the caved rock. A height of 8 ft 6 in. was provided in the recovery room by taking roof rock 6 in. in thickness. The roof was supported with 5-ft-long resin bolts and 9-ft-long conventional bolts on a 3-ft grid and reinforced with No. 9 gage chain link fence-type wire mesh (Fig. 24). Pin-rails 40 pounds in weight and 15 ft in length, were installed in the roof shale to reach over the longwall block. The inby end of these rails would come to rest on the shield canopies at face approach prior to cutting through, and the outby end was supported by a hydraulic prop set in the recovery room (Fig. 25, 26). Track was laid to facilitate removal of the shields.

The transport route for the panel-to-panel move was 21st N entry, future tailgate of face 2. The entry was graded to a height of 8 ft, drained, and tracked, and connection was made between this track and the belt-takeup track in the belt entry via a slanted breakthrough and a track switch installed in the belt take-up.

When the face was within 10 cuts, or approximately 20 ft, from cutting through into the recovery room, wire mesh of the chain link fence-type in 5-ft-wide sections was fed on top of the canopies with extensions dropped. The wire mesh sections were spliced together with connectors (hogrings) and 3/16-in. wire rope, leaving 6-inch overlaps. As planned, the pinrails, installed into the roof shale and supported by hydraulic jacks at the outby end, came to rest on the canopies immediately before the face cut through and secured the roof during this critical moments in the evening of May 11, 1977. Most shields in the midsection and tailend from No. 20 to 95 yielded a little but sustained the roof pressure, which leveled off and transferred to the recovery chain pillars and the barrier pillar as indicated by the readings on the stressmeters installed nearby. The floor heaved. Coal and rock spillage had to be loaded out, and the track in the recovery room had to be removed to provide the clearance necessary for the passage of the shields (Fig. 27).

Panel-to-Panel Transfer

The shearer, after removal of drums and ranging arms, was plucked from the conveyor by two scoops. Chains, cables, pan line in lengths of three pans, and the drives were removed from the recovery room, and the pan line was reassembled in the face 2 staging room. Shearer and conveyor drives were taken to the surface for overhaul. The track was then relaid in the recovery room for the removal of the shields. The heavy weight of each shield, 35,000 pounds, and the need for vertical clearance of 84 in. when upright and 72 in. when on its side, including the rail dolly, made the move difficult and slow. A minimum side clearance of 9 ft between canopy tip and rip had to be provided (Fig. 28, 29).

Each shield, refitted with longer hoses and under adjacent control, was moved ahead step by step under its own hydraulic power. A beam held down by hydraulic jacks, a S&S Unatrack scoop, or even a carhaul under very heavy roof provided the anchorage. Whenever a shield was lowered for recovery, five adjacent shields yielded somewhat. Once on the track in the recovery room, the shield was set on a skid and pulled by S&S Unatrack to the headgate where it was laid on its side on a rail dolly by means of rope, sheaves, and pull by Unatrack. The first six shields were removed from the tail end under the protection of three cribs built across the recovery room in place of each shield taken out. Eventually, roof control was maintained by forming a moveable bulkhead consisting of two shields advancing side by side along the rib side of the recovery room (Fig. 30). A crib was built in place of each shield removed. Extremely heavy roof pressures had to be controlled in the midsection of the recovery room. The weight was on the rib side and particularly heavy in intersections where cribs were built across breakthroughs. A few shields yielded extensively, and shield 52 yielded to the extend that only 4-1/2 in. of the leng travel was left.

Each shield lying on its side on the rail dolly was hauled on the track by Unatrack to the transfer point at the tail end of the staging area, where it was uprighted and set on the staging room track under control of another Unatrack. It was then maneuvered to take its place in the new shield line. The shields were rehosed and temporarily connected to an auxiliary hydraulic power supply pending the move of the power pack. The shearer was introduced over the tailend of the panline. Connecting the tail drive with the conveyor concluded the move. Operation started on August 1, 1977.

Accident Experience

Accident experience in the period from May 1976 to June 1977 included installation, operation, and most of the panel-to-panel transfer and compared very favorably with the overall accident experience in the mine. Intensive training apparently paid off. Only one lost-time accident occurred when a workman dropped a prop on his foot. Among 25 no-lost-time accidents, bruises by equipment and burst hoses were most frequent. The table below shows the accident analysis, comparing the longwall with the whole mine:

Accident Analysis: Longwall versus whole mine No. 24 during May 1976-June 1977

	Man-hours	Severity	Frequency
Mine No. 24	1,224,624	837.81	32.60
Longwall	57,096	262.82	17.50

Ground Control Program

The ground control program proposed and conducted by Dames & Moore was designed to provide information for evaluating the adequacy of the roof support system as ollows (Fig. 21):

1. Strata movement by measuring convergence in gate roads and differential rock and floor strata movement to detect bed separation.
2. In situ pressure measurements by installing stressmeters of the vibrating wire-gage type in coal and floor of the longwall panel and the chain pillars.
3. Loading profile in the roof support system by measuring hydraulic pressures with load indicators, gages, and recorders.
4. Vertical extent of caving by boreholes from surface to coalbed.

Convergence of roof and floor seldom reached or surpassed 2 in. before the face passed. Only in the tailgate, a station showed a 4-in. convergence at face approach. Differential roof strata movement measurement did not indicate bed separation between shale and limestone. Differential floor movement readings showed slight heaving of the underclay (up to 2 in.) relative to the siltstone base.

In situ pressures were measured by stressmeters at the indicated locations (Fig. 32). Near the staging area, the stressmeter readings showed a gradual rise in stress and a sudden drop when the face attained a distance of 150 ft and the first major roof cave occurred. Afterwards, the stress rose gradually.

In midpanel, stressmeters sharply responded to the face approach, one of them indicating a gain of 1600 psi before destruction.

Stressmeters in the siltstone floor under the longwall panel showed a small rise at face approach, a drop below setting pressure after the face rolled over then, and a slow gain back to setting pressure while the face moved away.

Stressmeters indicated a sharp rise when the face cut through into the recovery room.

Pressures in the hydraulic system of shields and props were measured to compute the roof loads. Shield loads increased during idle days (weekends) by approximately 10 psi = 0.07 ton/sq ft. A sharp rise in canopy loads occurred immediately prior to ınd following the first major roof falls, simultaneously with the stress relief ıeasured by stressmeters near the staging area (Fig. 33).

The instrumented hydraulic props were adequately blocked by wood headers and steel plates so that they could sustain high loads; consequently, the only two jacks that yielded were the instrumented ones in the recovery room when the face cut through.

The successive caving of roof strata over the mined out area was monitored by Time Domaine Reflectometry. A 7/8-in. coaxial cable was grouted into a NA size (3-in.) borehole to a depth of 670 ft. A Tectronix 1502 Time Domaine Reflectometry cable tester was used to monitor the condition of the cable. The instrument detects cable faults and breaks by radar. Prior to installation, the cable is crimped at 10-ft intervals as an aid to calibration and measurement. Figure 34 shows the progress of caving in terms of face advance until the reading stabilized at 410 ft above the coalbed.

After a license of right-of-way was secured from the Army Corps of Engineers, who hold the surface rights, Old Ben and Bureau personnel of the Denver Mining Research Center initiated surface subsidence studies over the longwall mining area to measure vertical and horizontal surface movements including tilting of monuments. Surface strains between selected monuments are measured by extensometer or strain gages. The shape of the subsidence trough and the angle of draw are to be determined to indicate whether the mining will affect surface structures such as the Rend Lake Dam. A total of 196 monuments were installed in three lines. Two of them were N-S center lines over two adjacent longwall panels, and one is an E-W crossline. The monuments consist of steel pipes driven into the ground. However, several monuments had to be installed in boggy ground and were specially designed for the purpose (Fig. 35).

The maximum vertical surface displacement was 4.39, which is equal to 62% of 7 ft of mining height. An angle of draw of 23° was measured at the N or inby boundary. Full subsidence occurred when the face distance approximated the overburden depth.

The Illinois State Geological Survey mapped the longwall panel area as it was developed and mined and explored the adjacent strata by vertical boreholes up to 40 ft in height.

Conclusions

The following milestones mark the progress of this cooperative effort:

1. Premining study as a basis for equipment selection.

2. Roof support designed to control hazardous ground.

3. Mining equipment with many safeguards.

4. Operation mode to minimize dust exposure and to provide adequate face clean-up.

5. Panel-to-panel transfer involving recovery, move, and reinstallation of heavy and bulky equipment.

6. Ground control and subsidence program to monitor the effect of mining on rock strata and surface.

This is an interim report. Time will tell whether the subject demonstration of longwall mining will be a success after the third panel comes to completion. The initial face mobilization and the panel-to-panel transfer must be included in productivity and cost account to strike a fair balance between pillar extraction and longwall mining. As of now, after the face move, the cost picture does not favor longwall. However, a variety of adverse conditions occur at this particular site, and installation and panel-to-panel move of the bulky and heavy equipment posed formidable challenges to novice crews. The long range payoff of this demonstration if successful, would be to show that Herrin No. 6 could be mined safely and efficiently with a high degree of recovery. The demonstration is an invaluable testing ground for equipment and offers a training opportunity for workmen and supervisors for future efforts in bringing longwall mining to the Illinois coalfield.

References

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2. Barry, A. J. and O. B. Nair "In-Situ Tests of Bearing Capacity of Roof and Floor in Selected Bituminous Coal Mines," BuMines RI 7406, 1970, 20 pp.

3. Curth, E. A. "Roof Support Problems in Longwall Mining. A Study in the United States and Germany in 1971," Paper in Ground Control Aspect of Coal Mine Design. (Proceedings: Bureau of Mines Technology Transfer Seminar, Lexington, Ky., March 6, 1973), BuMines IC 8630, pp. 101-104.

4. Curth, E. A. "Coal Mining Techniques in the Federal Republic of Germany in 1971," BuMines IC 8645, 1974, 52 pp.

5. Moroni, E. T. "Longwall Experiences in the Illinois No. 6 Seam," Illinois Mining Institute, Springfield, Illinois, October 11, 1973.

Illustrations

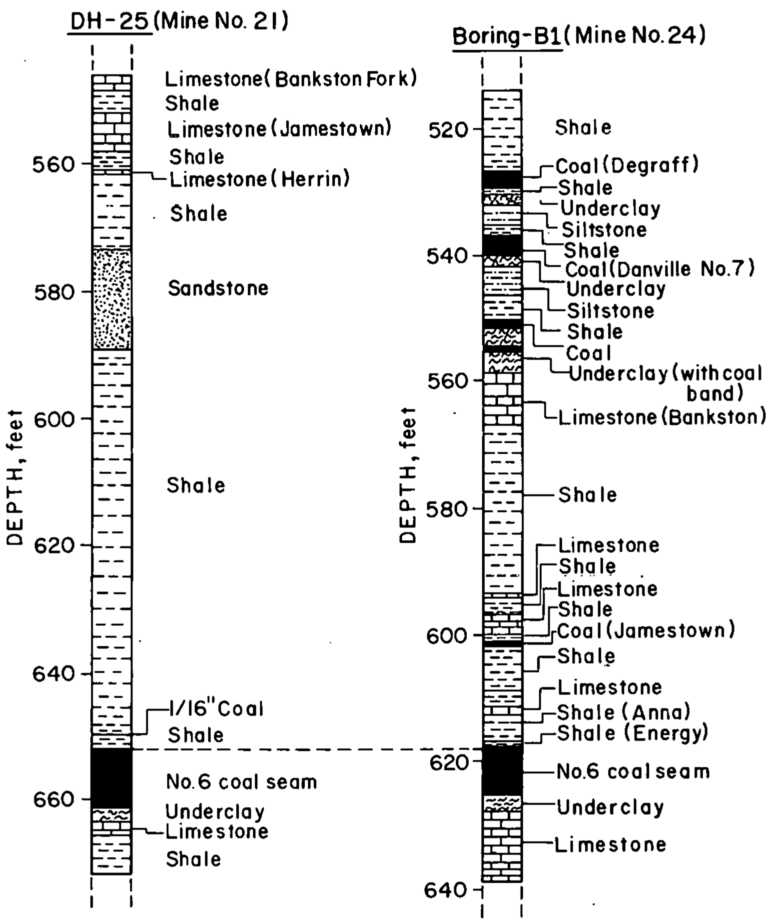


FIGURE 1. - Geologic Columns

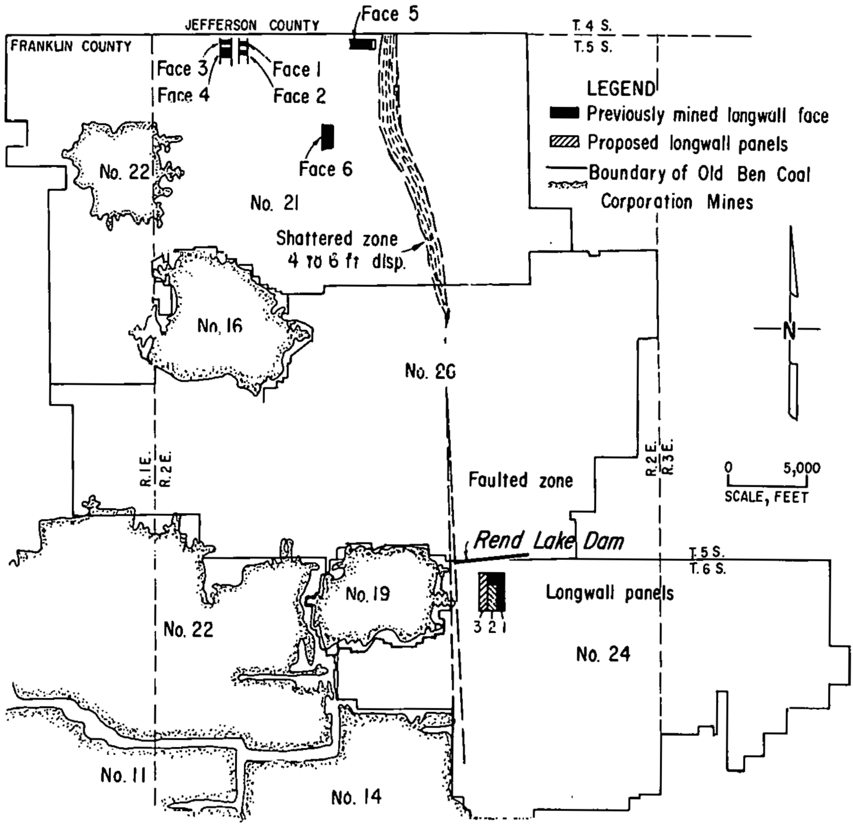


FIGURE 2. - Key Plan, Project Area

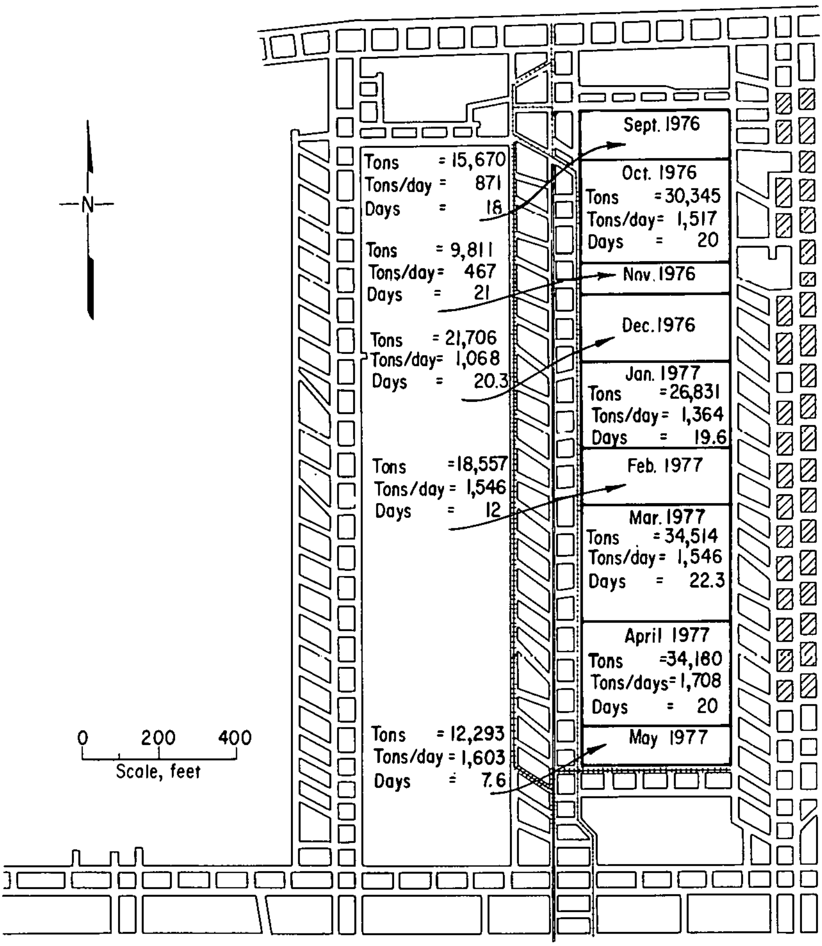


FIGURE 3. - Panel Layout

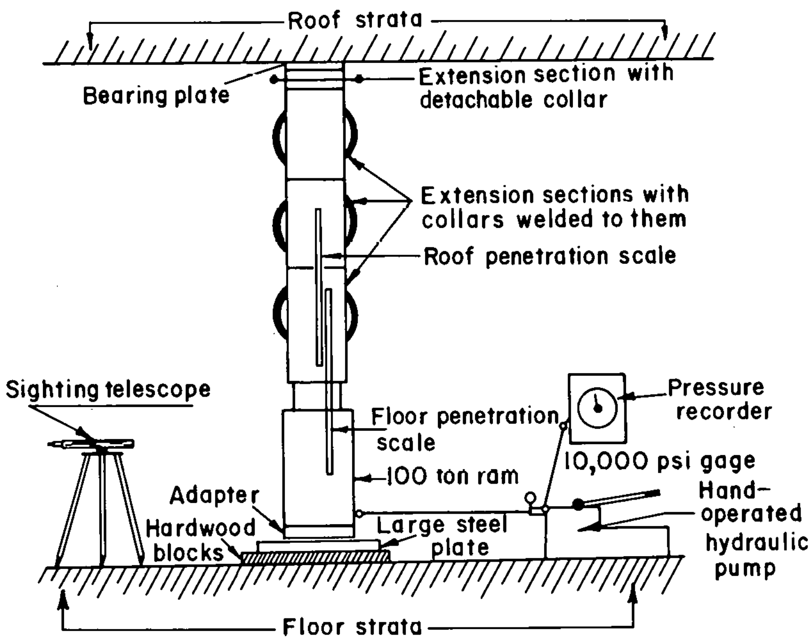


FIGURE 4. - Bearing Test Apparatus

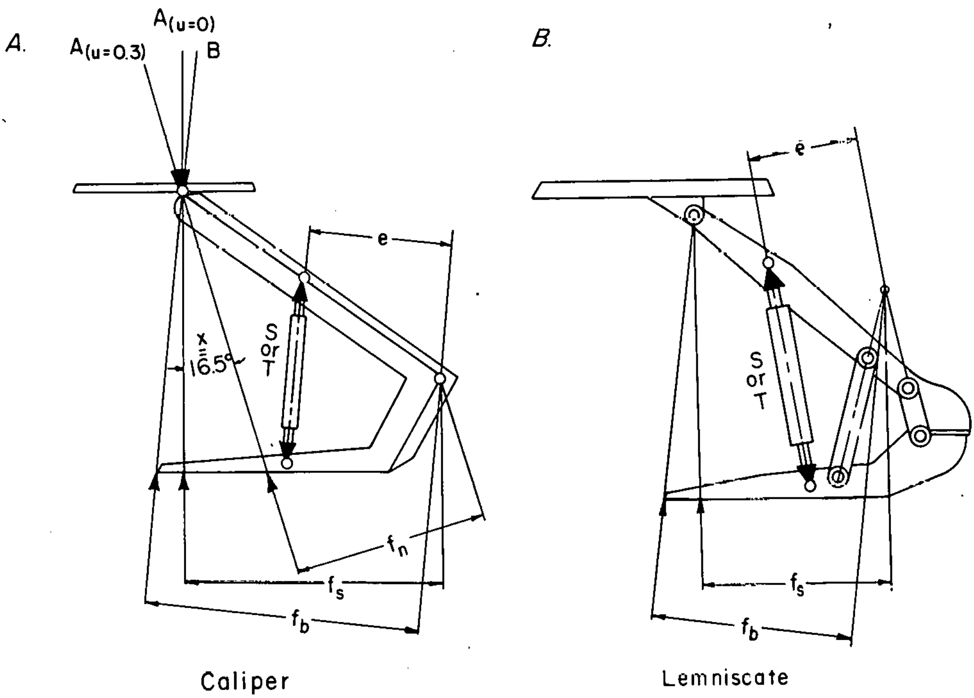


FIGURE 5. - Diagrams of Caliper and Lemniscate Shields

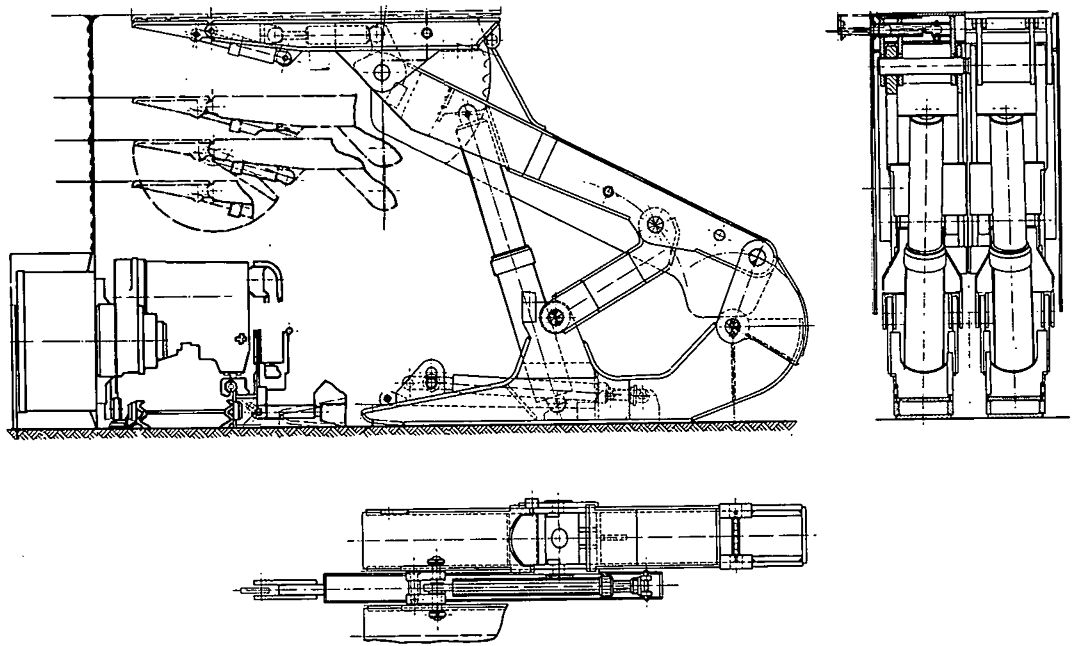


FIGURE 6. - Shield Structure

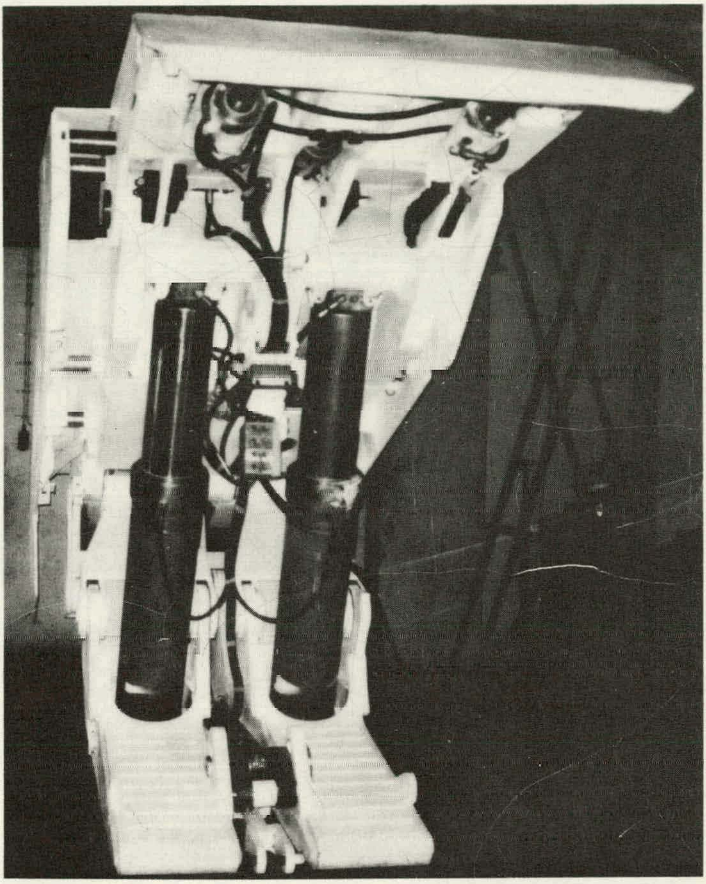
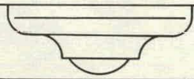
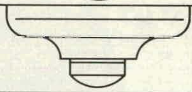
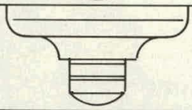
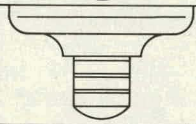
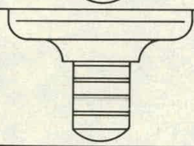


FIGURE 7. - Detail of Shield

Indication of marking pin	Nominal prop load		
	40 Mp	60 Mp	100 Mp
	Pressure indication in Mp		
	0	0	0
	10	15	22,5
	20	30	45
	30	45	67,5
	40	60	90

1 Mp = 1.1 short tons

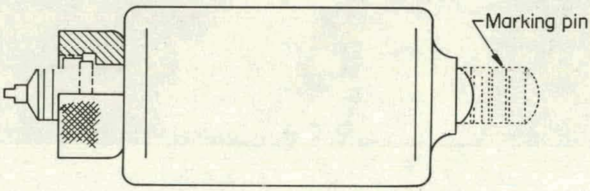


FIGURE 8. - Load Indicator

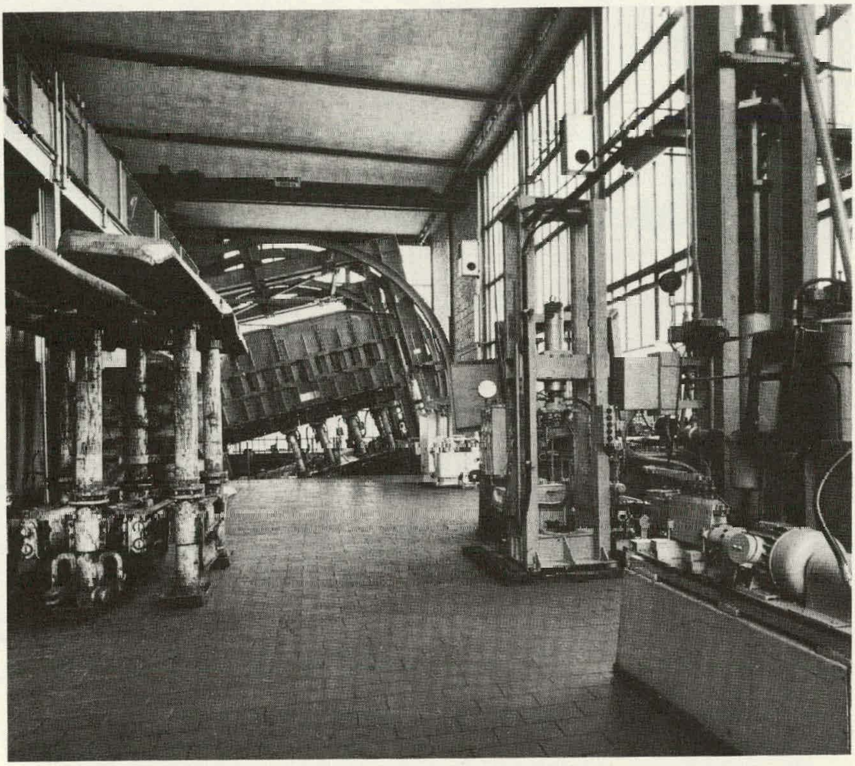


FIGURE 9. - Test Frame for Powered Supports

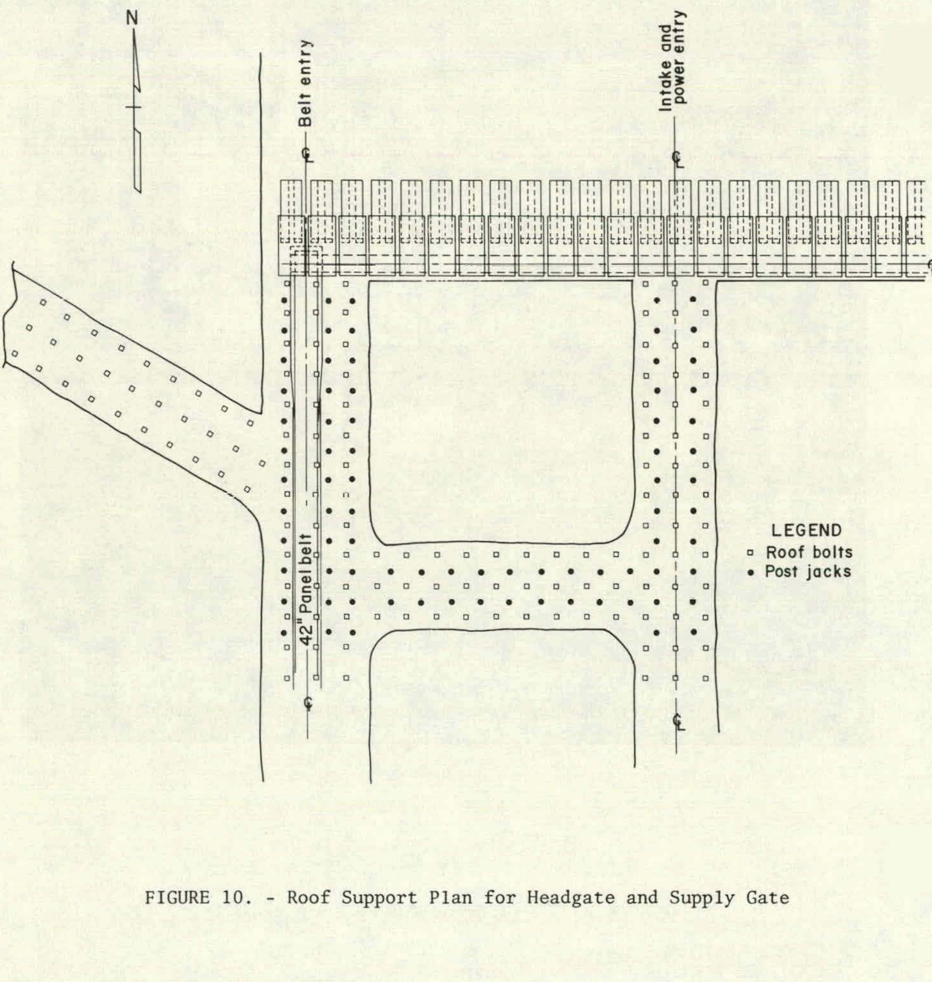


FIGURE 10. - Roof Support Plan for Headgate and Supply Gate

FIGURE 11. - Props in Supply Gate

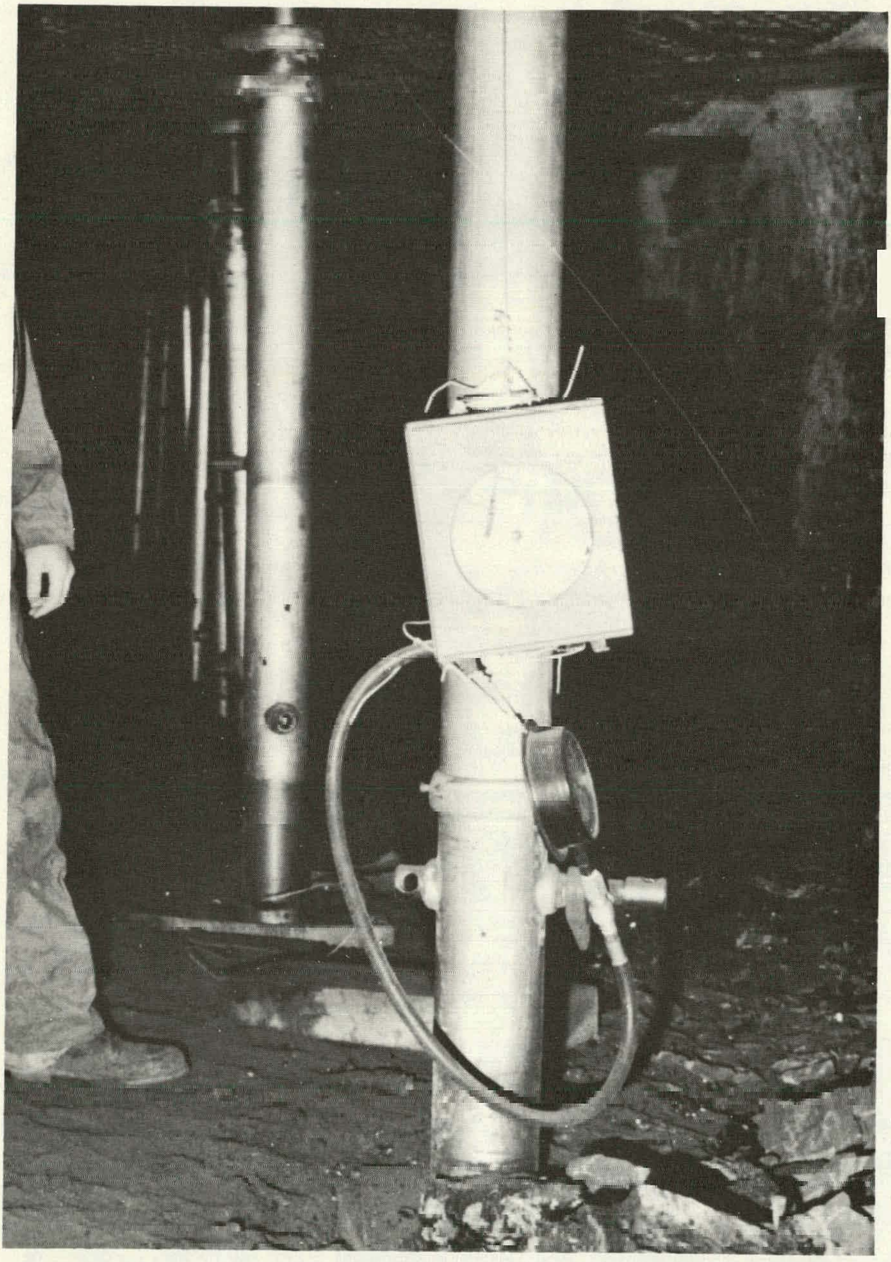


FIGURE 12. - Monitoring Loading

FIGURE 13. - View From Supply Gate to Tailgate

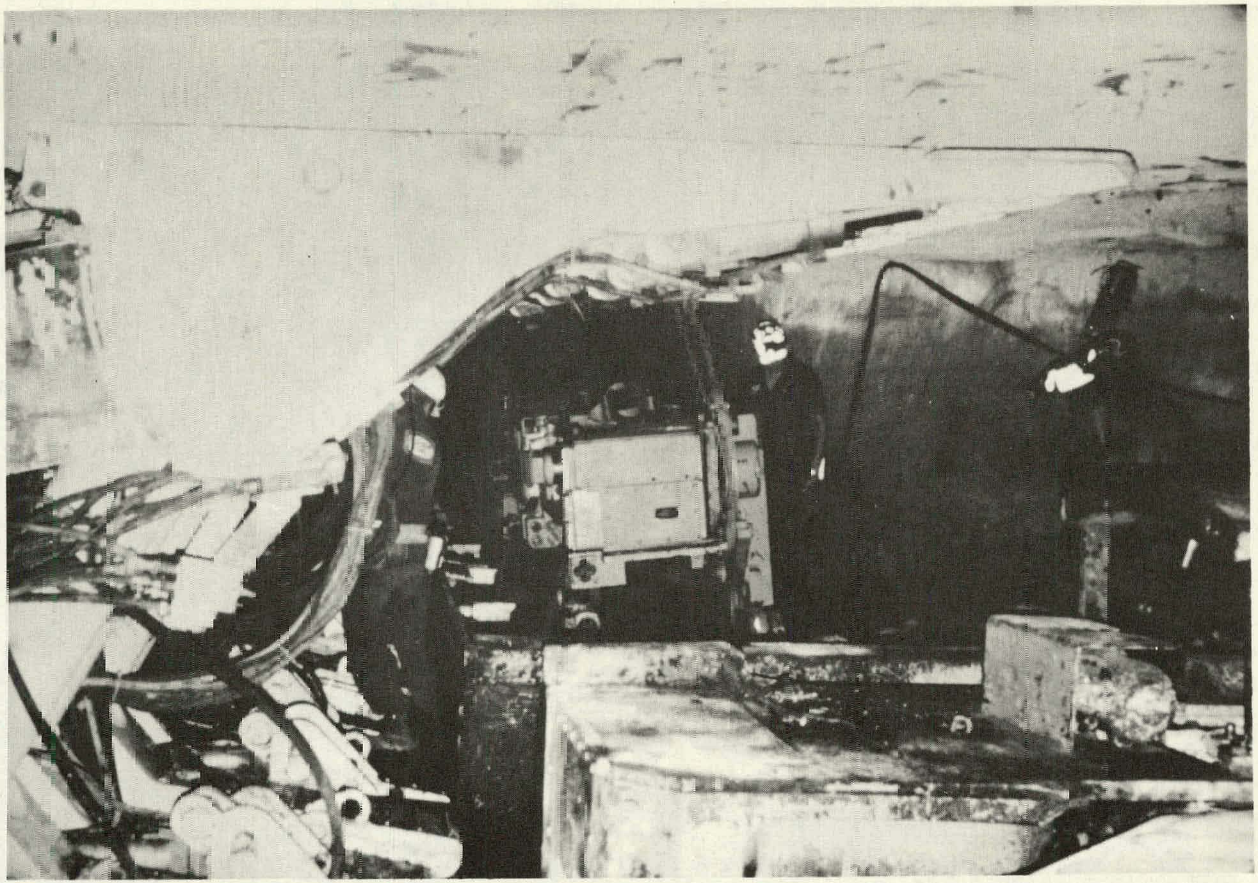


FIGURE 14 - Installation of Eickhoff Shearer

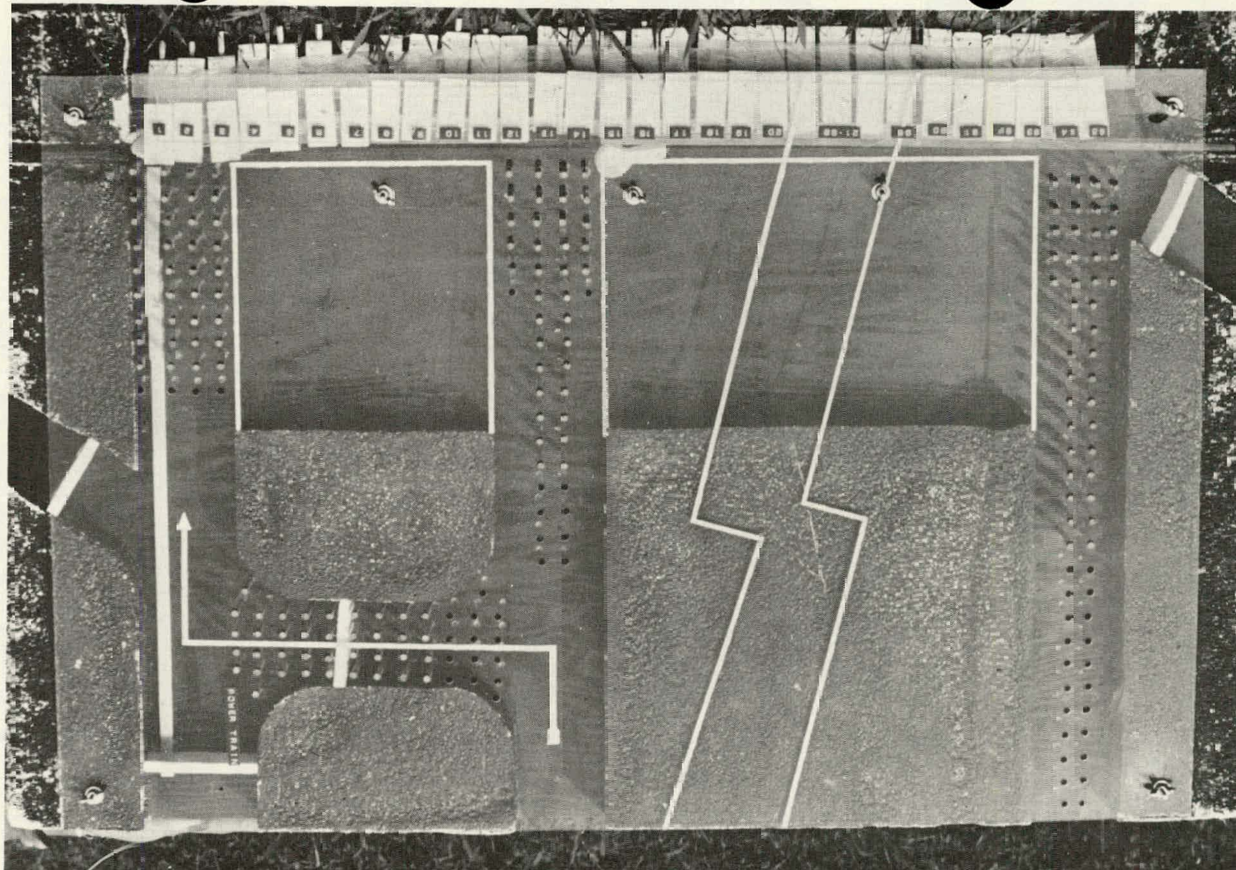


FIGURE 15. - Three-dimensional Longwall Model

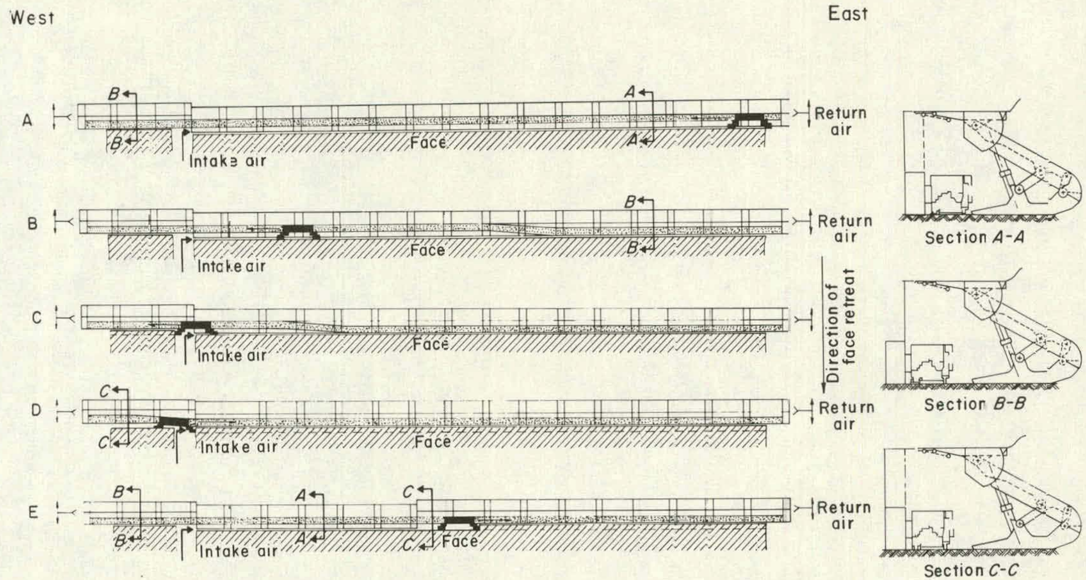


FIGURE 16. - Longwall Mode of Operation

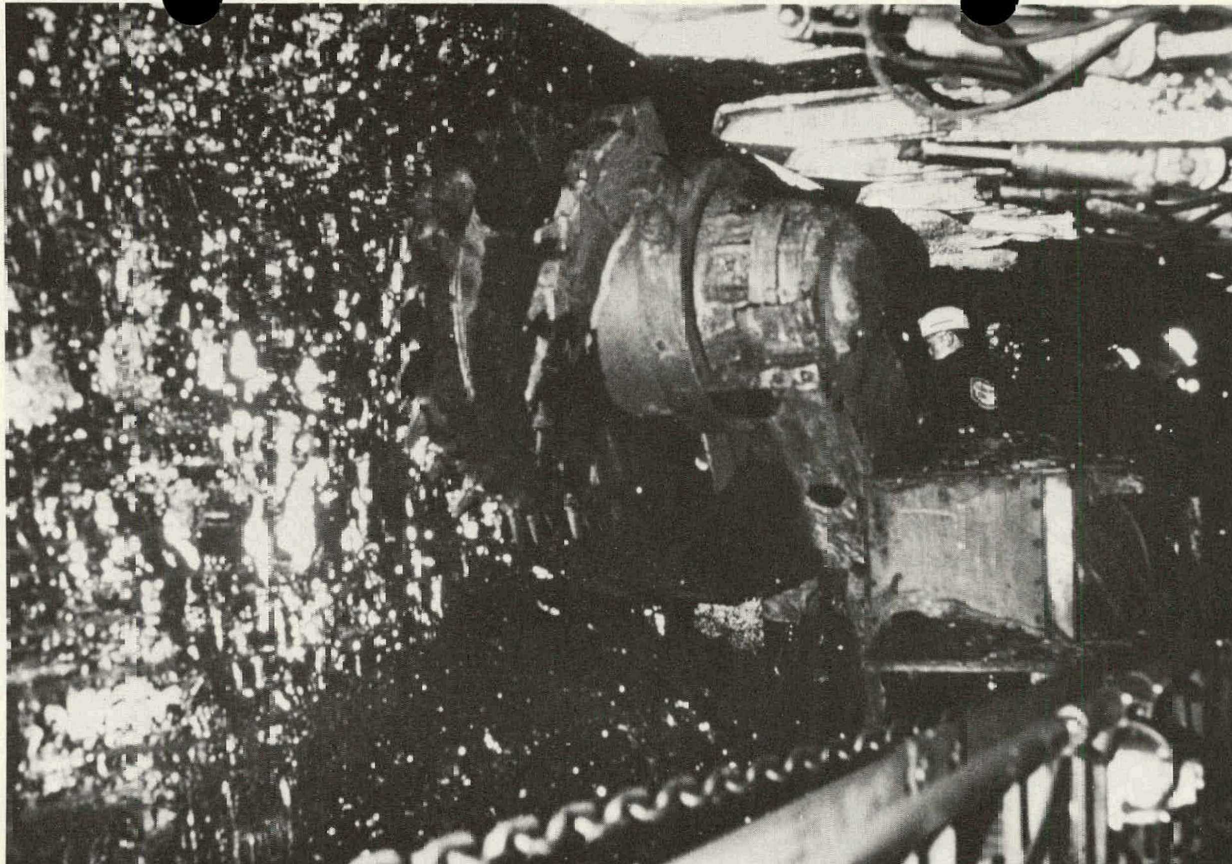


FIGURE 17. - Shearer Flitting Headgate to Supply Gate

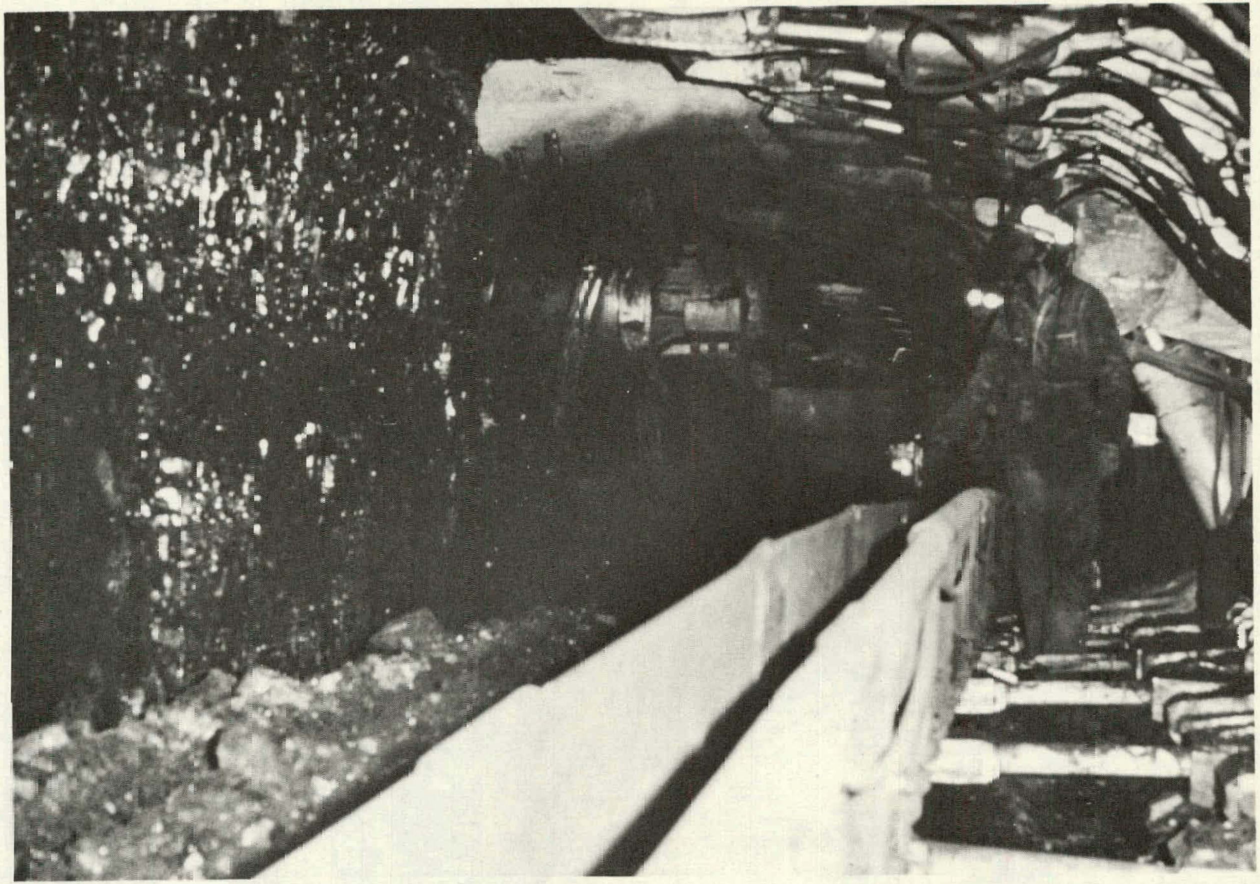


FIGURE 18. - Shearer Cutting Supply Gate to Tailgate

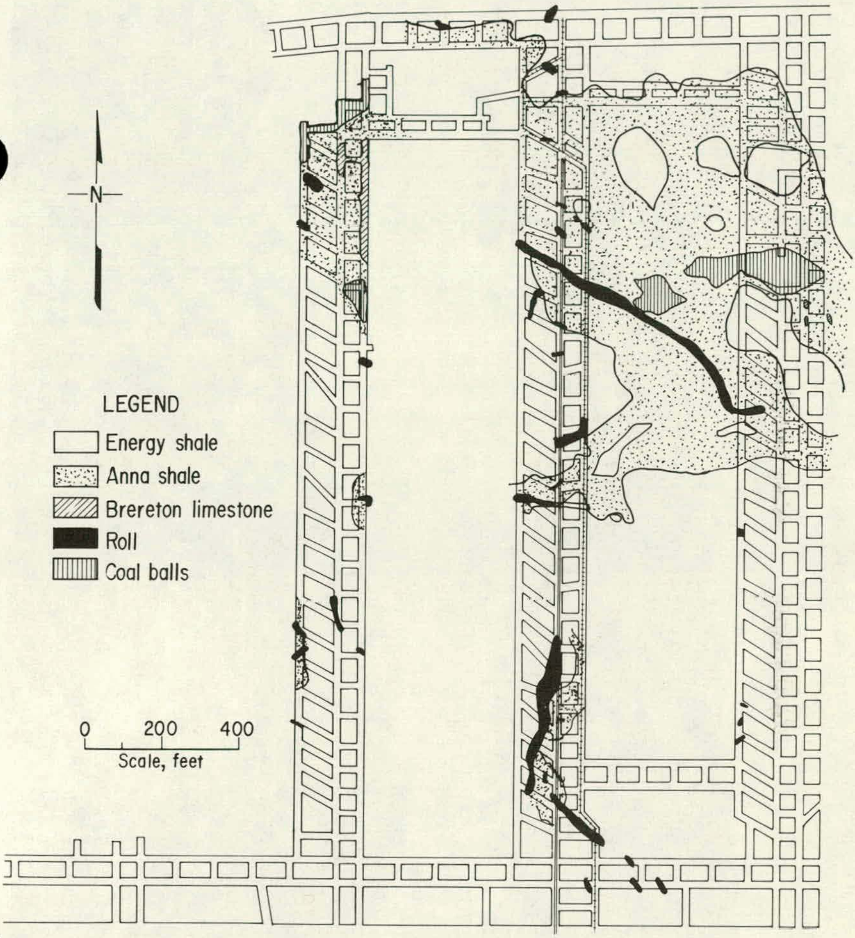


FIGURE 19. - Map by Illinois State Geologic Survey

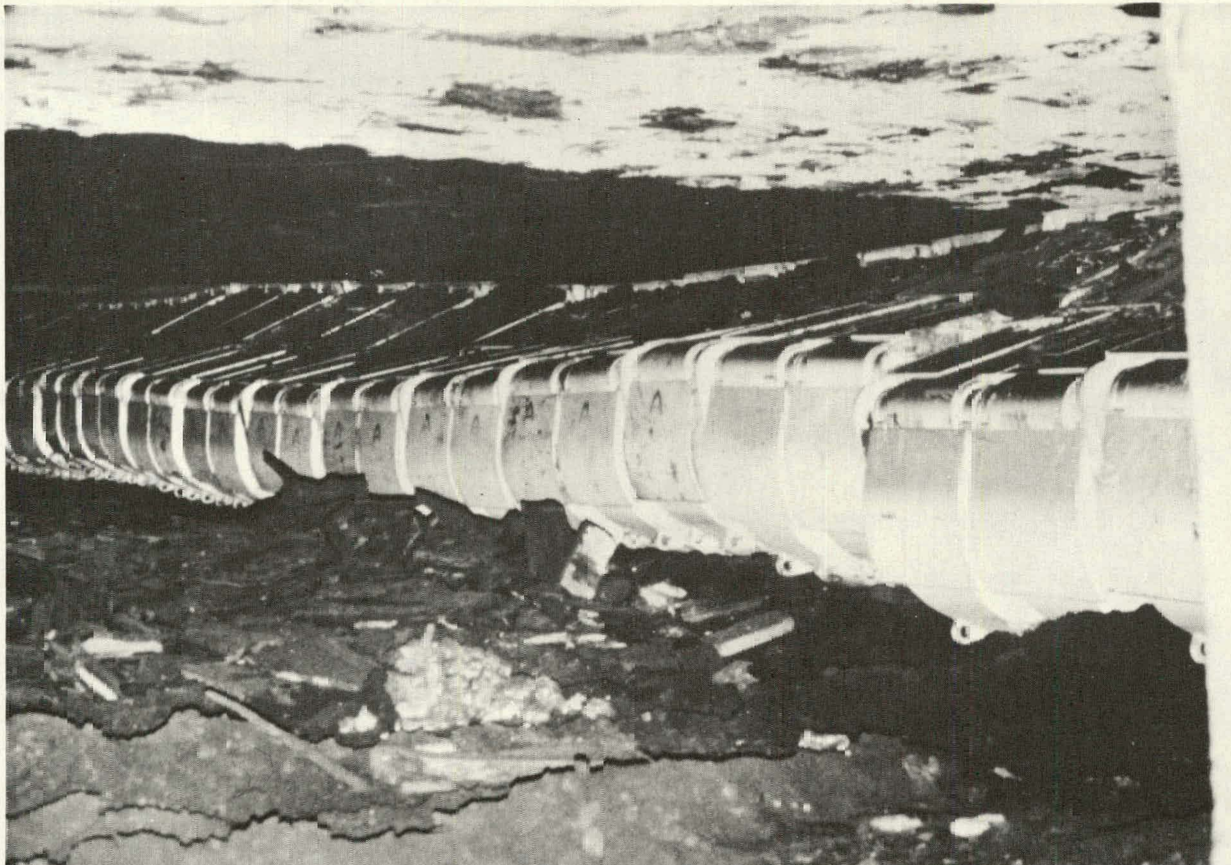


FIGURE 20. - Gob Before the First Roof Fall

FIGURE 21. - Gob at Headgate

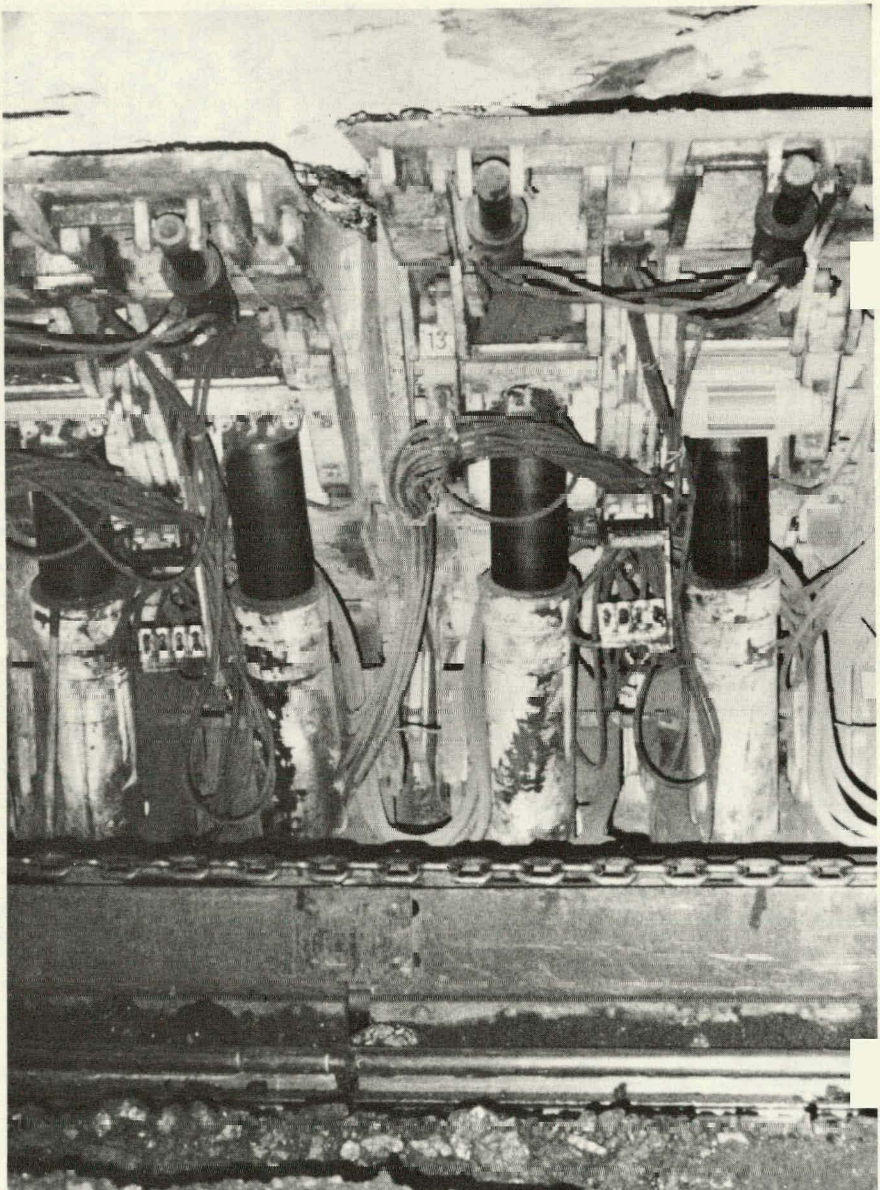


FIGURE 22. - Shield Contact with Roof

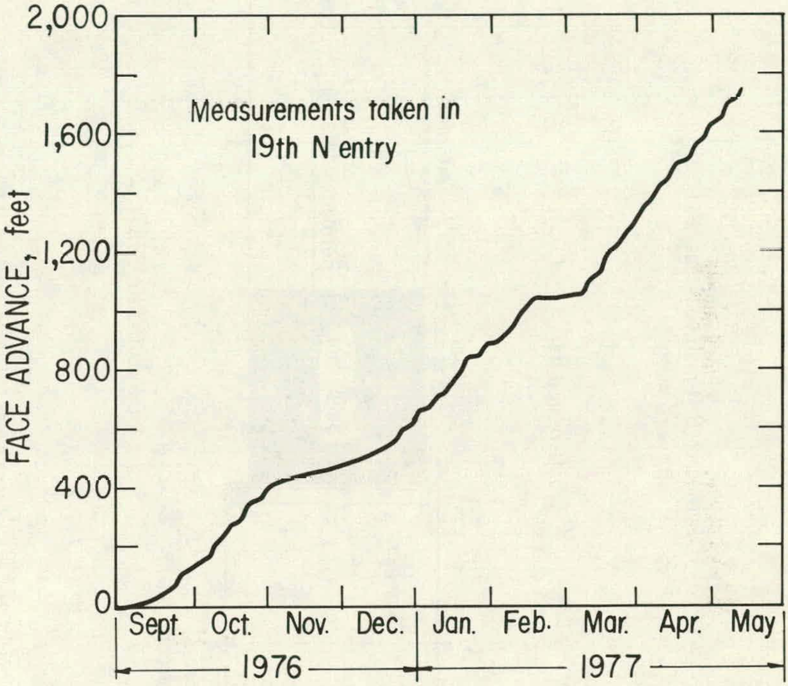


FIGURE 23. - Face Advance Versus Time

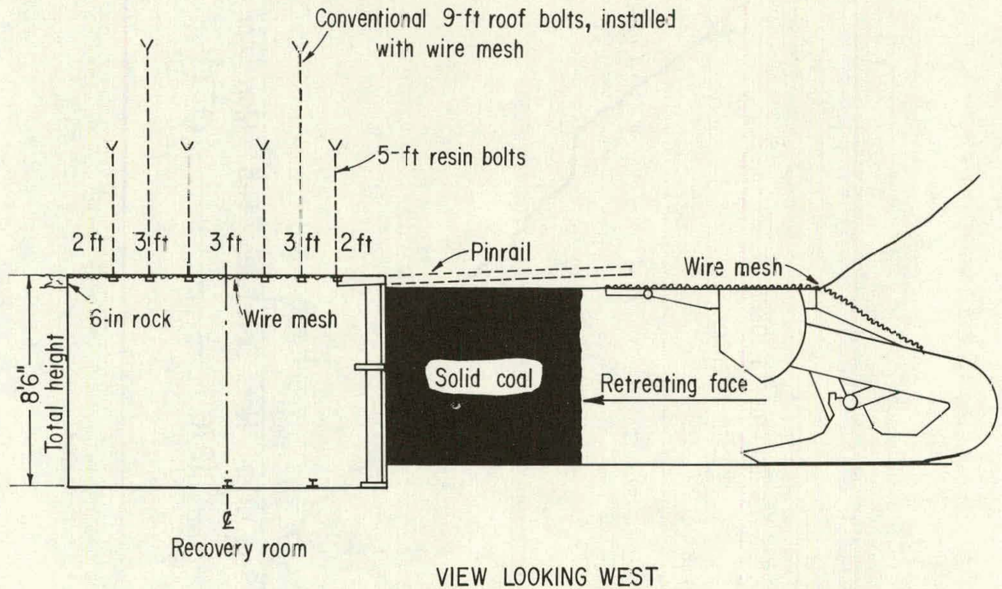


FIGURE 24. - Recovery Room Prior to Holing Through

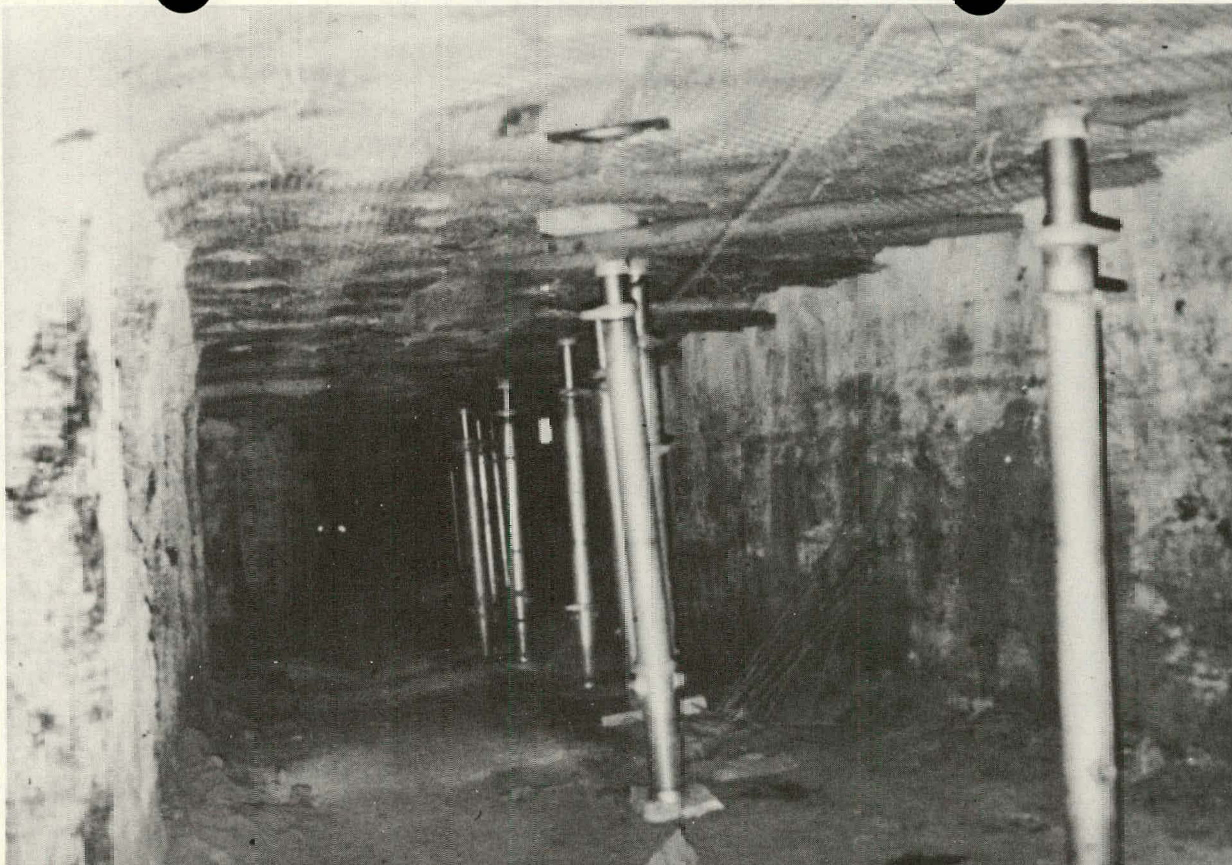


FIGURE 25. - Props in Recovery Room

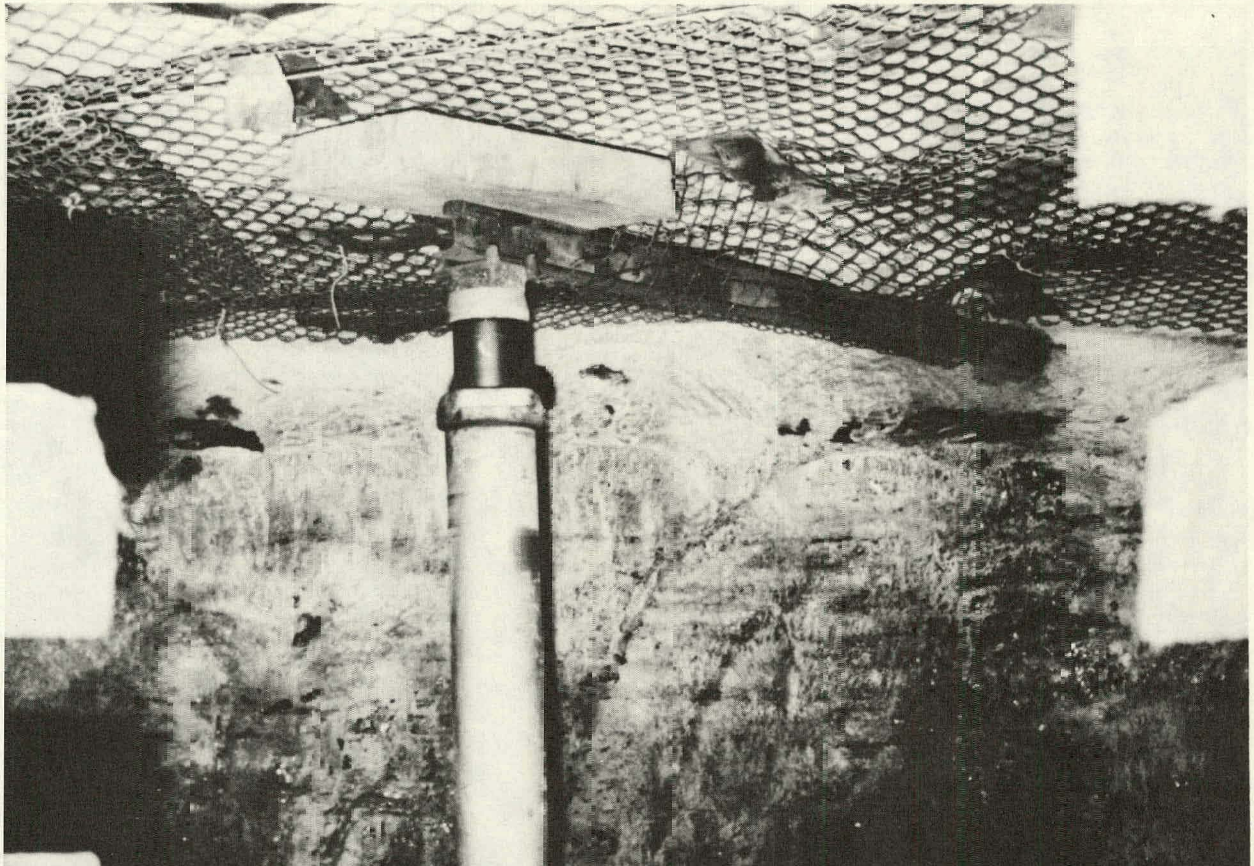


FIGURE 26. - Pinrails Placed in Shale Roof

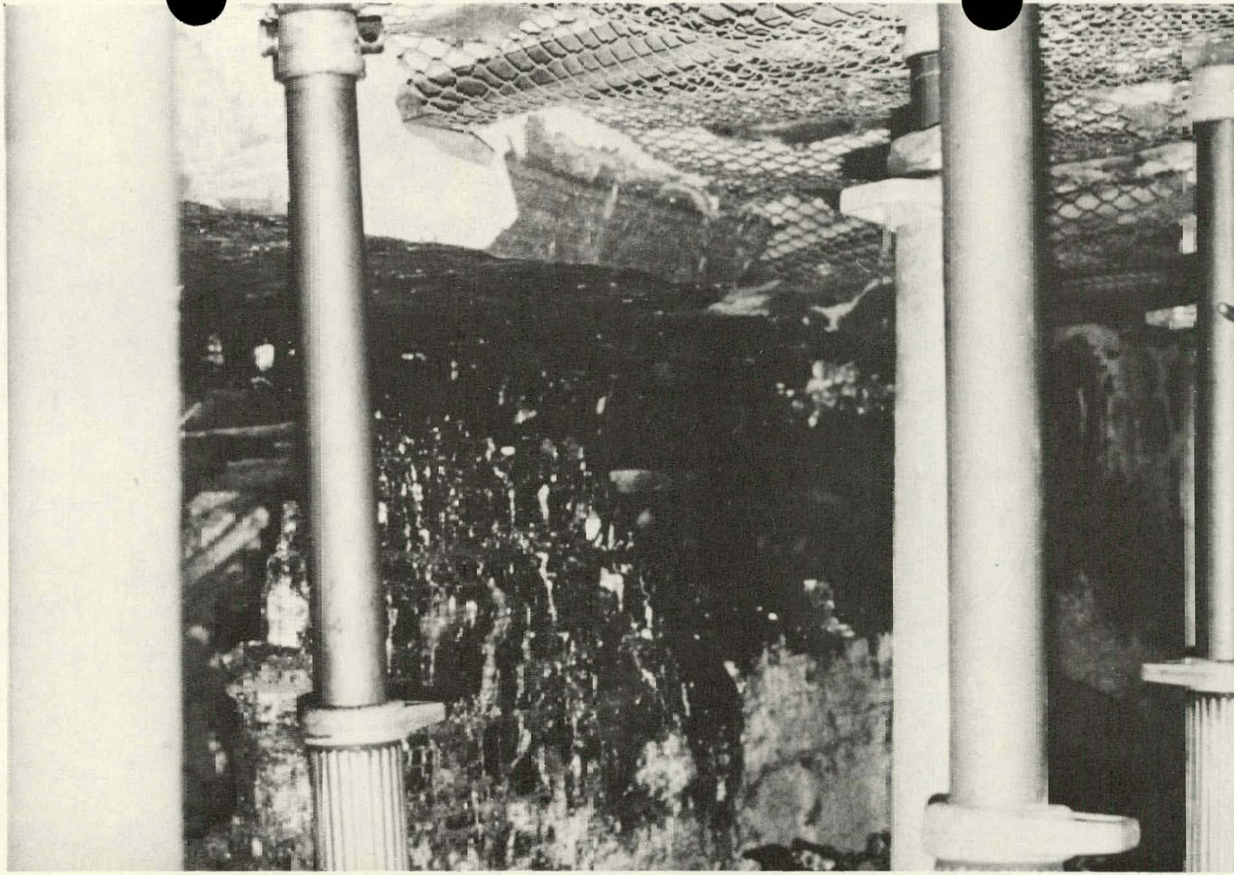


FIGURE 27. - Last Cut

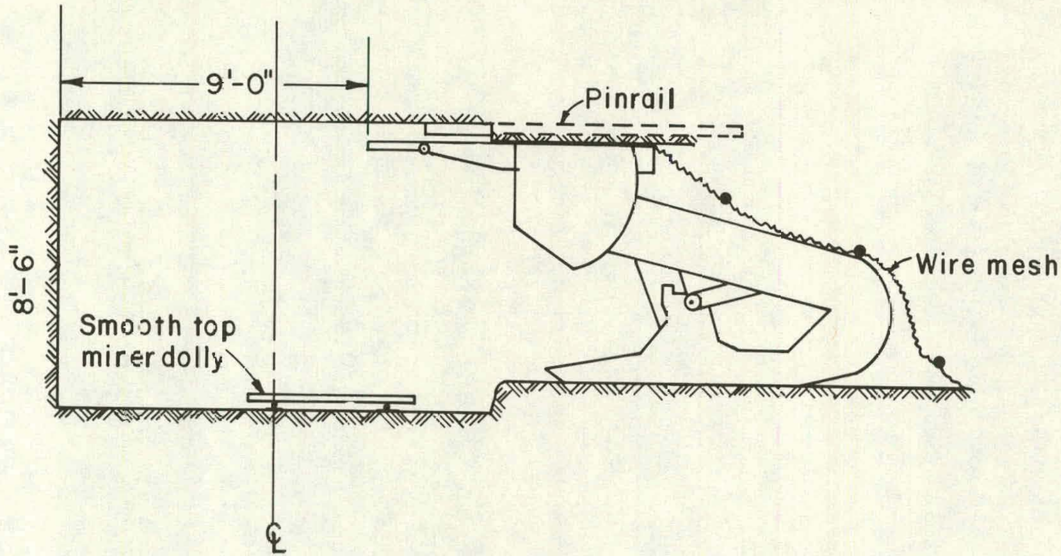


FIGURE 28. - Recovery Room Prior to Shield Recovery

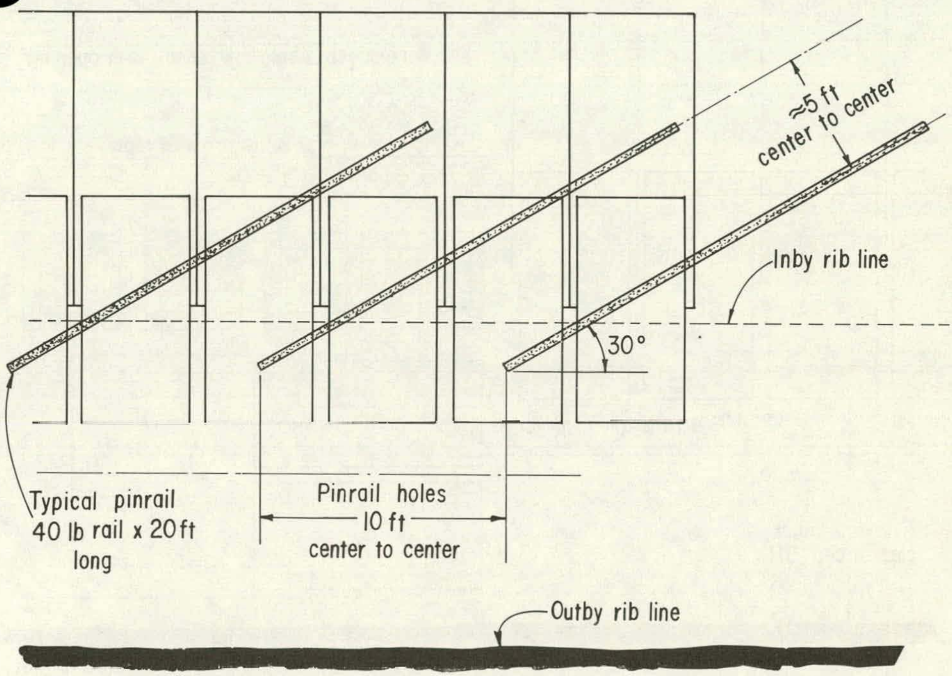


FIGURE 29. - Recovery Room Showing Pinrails Location

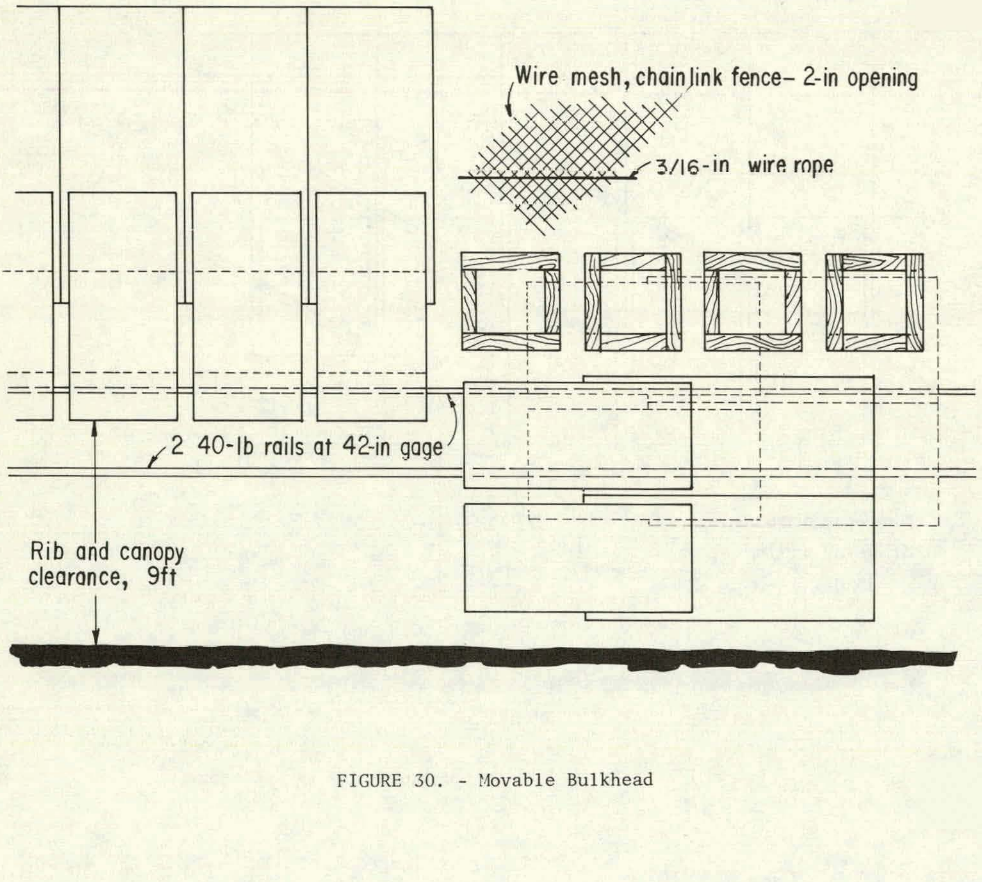


FIGURE 30. - Movable Bulkhead

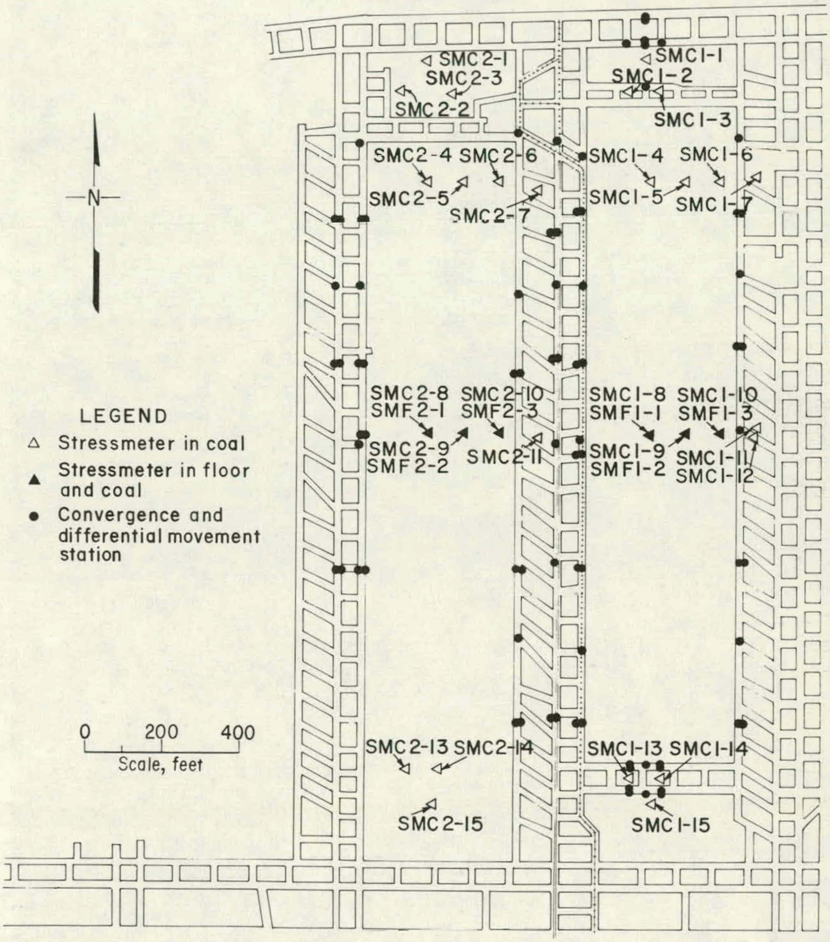


FIGURE 31. - Rock Mechanics Instrumentation

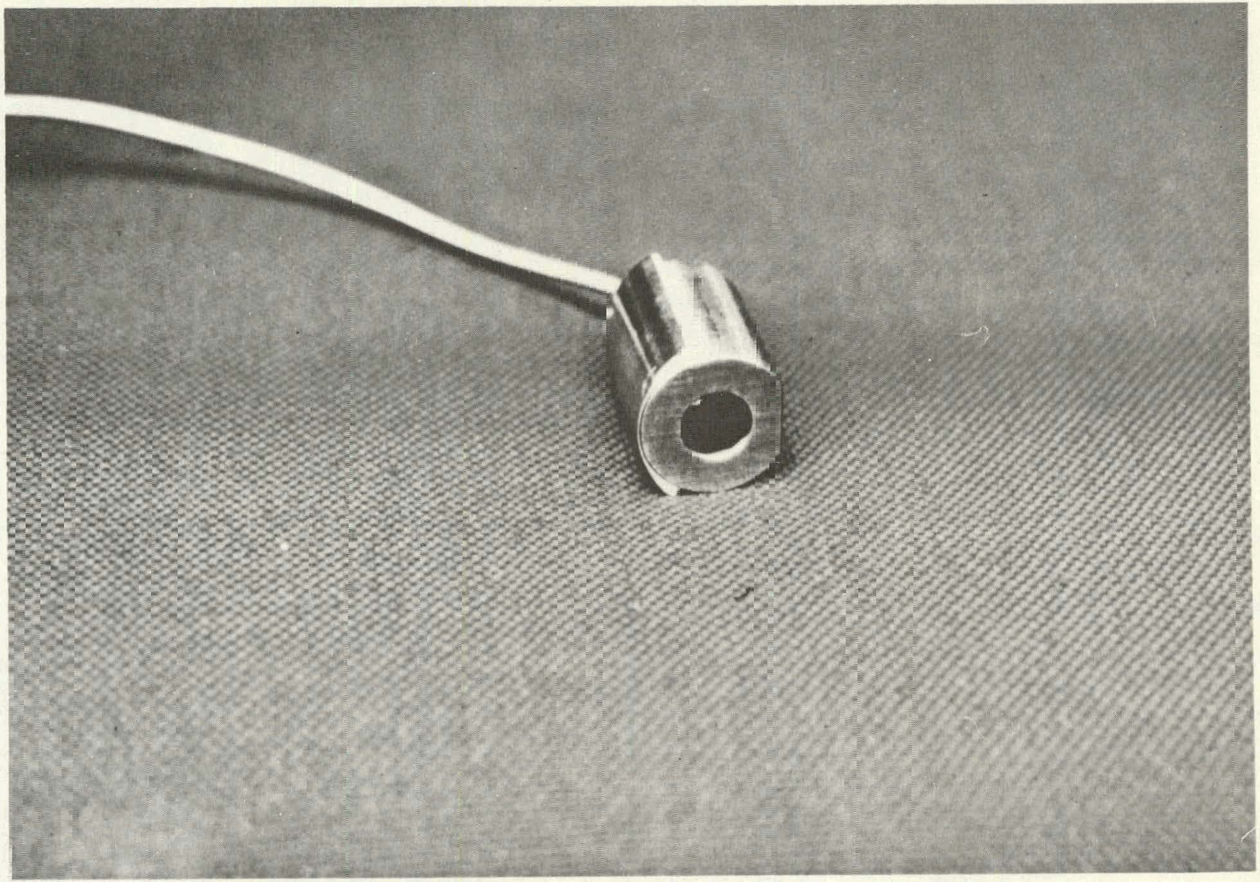


FIGURE 32. - Irad Gage Vibrating Wire Stressmeter

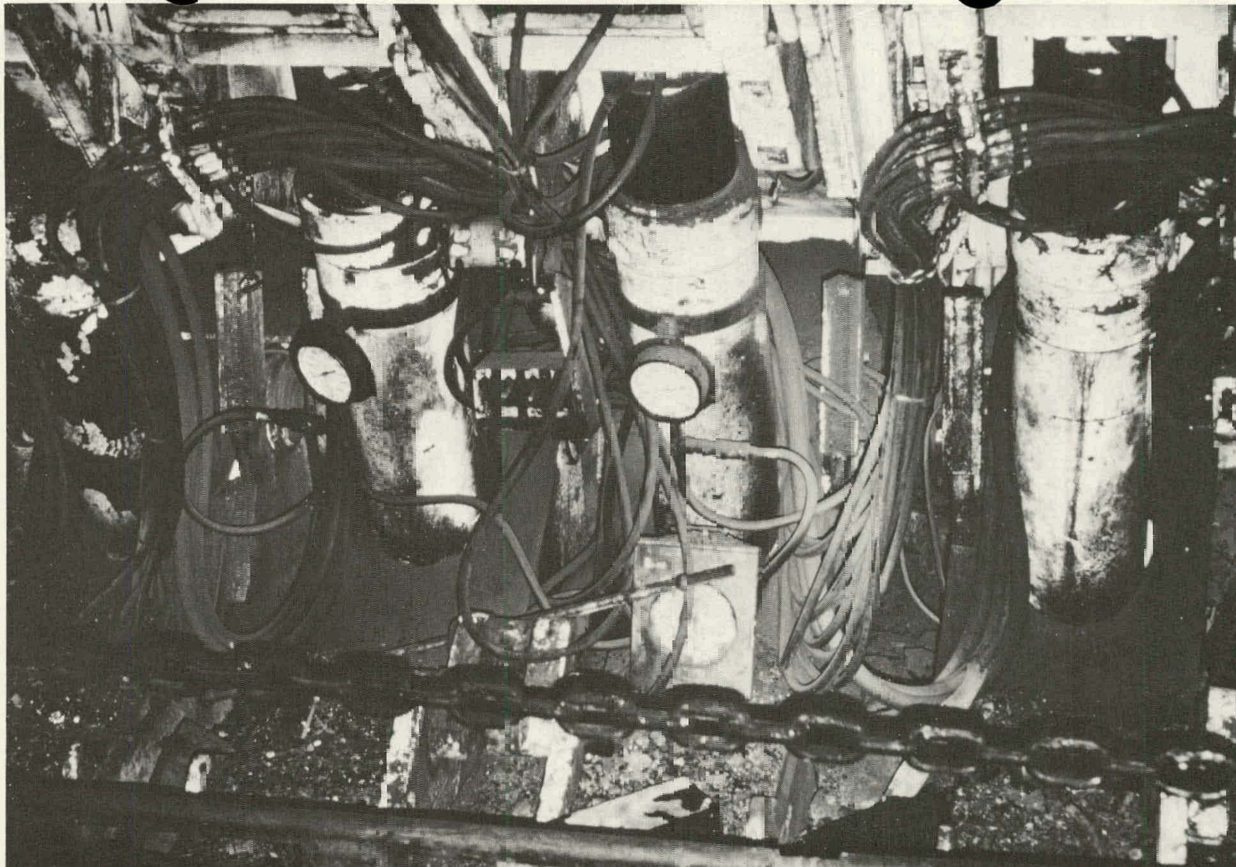


FIGURE 33. - Hydraulic Pressure Gages and Recorders on Shield

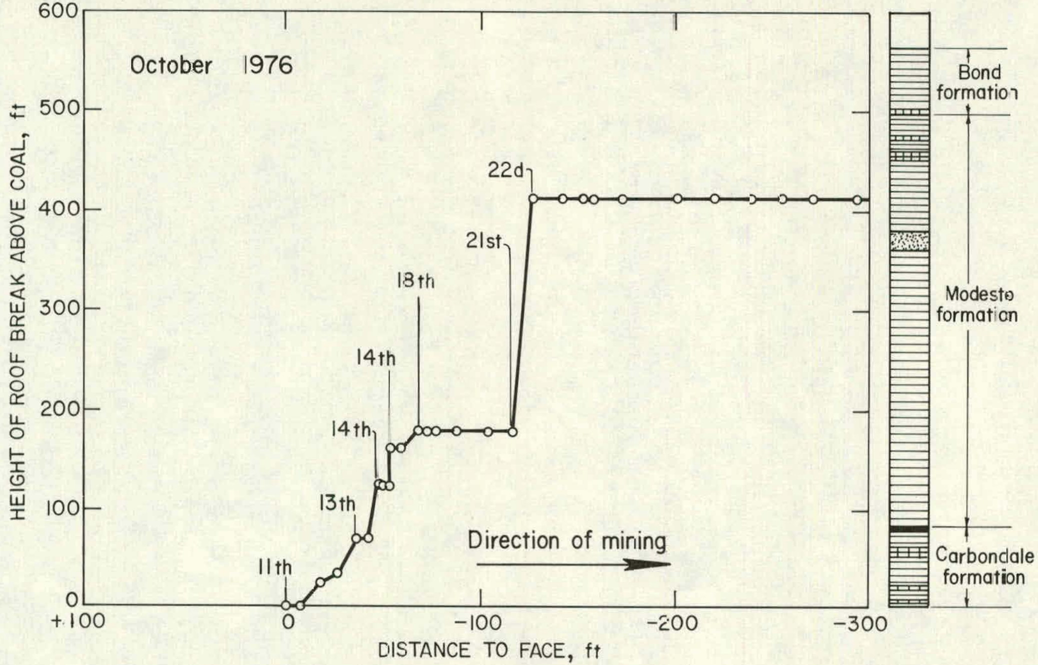


FIGURE 34. - Plot of Time Domaine Reflectometry Measurements

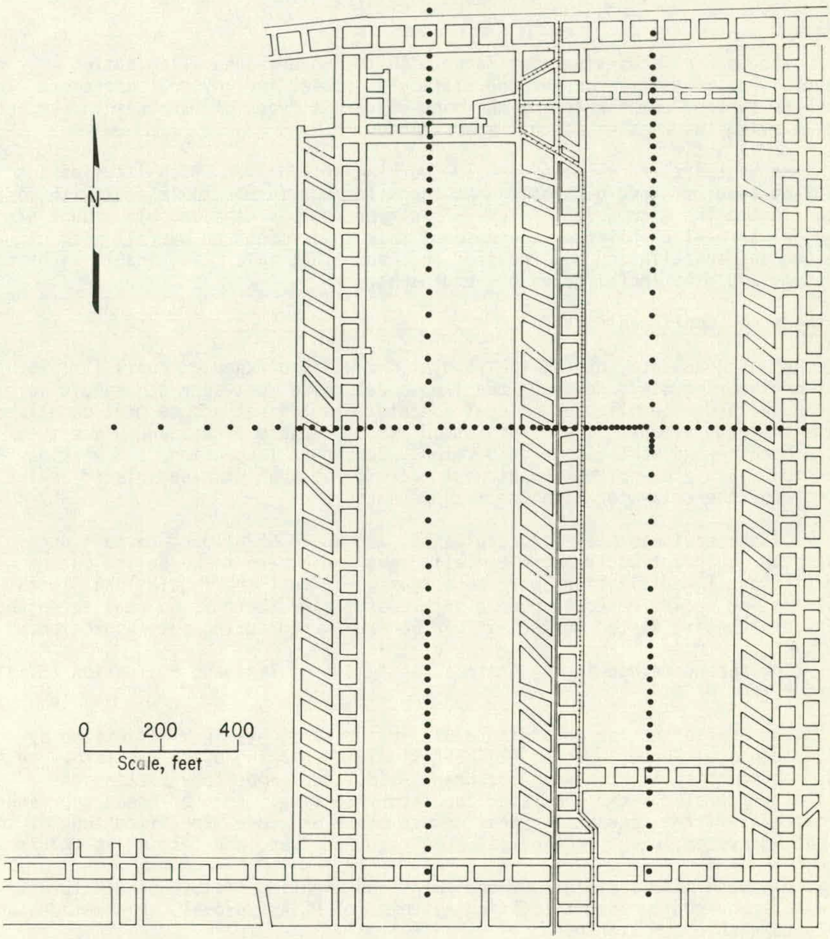


FIGURE 35. - Map of Surface Subsidence Stations

WESTMORELAND COAL COMPANY'S APPROACH TO MINE ILLUMINATION

Randolph E. Slone
Electrical Engineer

Westmoreland Coal Company
Big Stone Gap, Virginia

The task of achieving compliance with the underground illumination regulations is a very complex and difficult problem for any coal operator. This task is further complicated by the many different types of machines in use and the varying thicknesses of coal seams mined.

At Westmoreland we have over 100 working sections on which illumination will be required, and our coal seams range from 30 inches to 18 feet with most seams averaging 4 to 6 feet thick. This represents a considerable amount of capital to meet compliance. Because of this investment, an overall plan of design and installation was required to insure that safe maintainable lighting systems would be installed on our equipment.

Methods of Compliance

When an advance copy of the regulations was published on April 1, 1976, our engineering staff reviewed the law to determine what approach should be taken to insure compliance. A coal operator has two methods to meet compliance with the regulations. The first method is to purchase lights and place them on a machine and wait for an inspection. During an inspection of a working section, each piece of equipment would have to be shut down and placed in the middle of the entry for lighting measurements.

If the equipment were in compliance, you would be allowed to continue. If not, the equipment would be in violation and would have to be pulled out of service until modifications were made that would meet the regulations. With this method, downtime could become very costly, and there is no real acceptance of your lighting system in that it can be remeasured during any inspection.

The second method is to obtain a "Statement of Test and Evaluation (STE)" letter from MESA.

The approaches for obtaining a STE letter are: (1) in-mine testing by MESA Technical Support Group; (2) MESA simulated working place laboratory test; (3) equipment or lighting manufacturer's simulated laboratory testing; or (4) mine operator's own simulated laboratory testing. Each of these approaches require a written request for testing, complete drawings, specifications of the lighting system, prepayment of all MESA required fees, and testing as required.

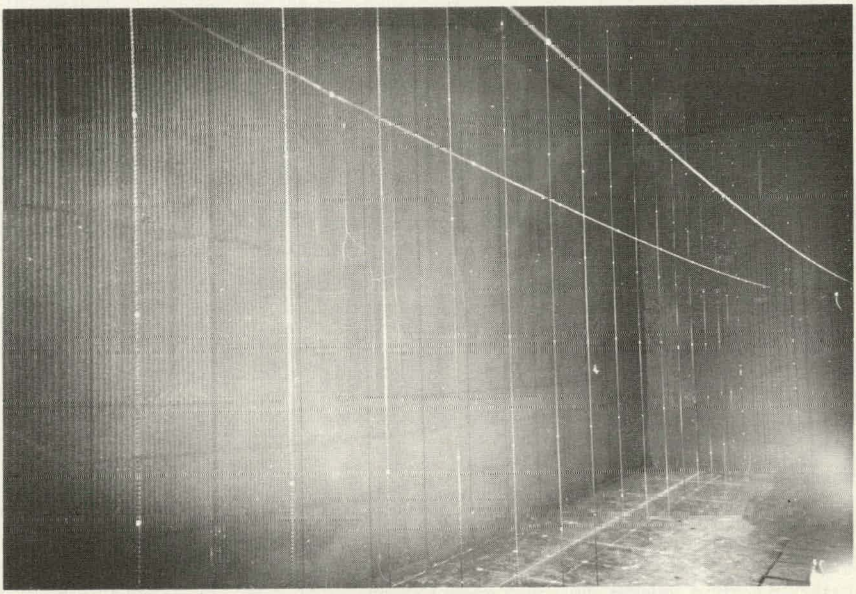
Although mine operators or equipment and lighting fixture manufacturers are not required to submit lighting systems for MESA approval, this method can help to assure compliance.

An approval plate or label is not issued by the government agency, but after examination of the design of a lighting system or a system installed on a piece of equipment, MESA will issue a Statement of Test and Evaluation. This statement, in letter form, defines the maximum height of the working place, the maximum width of the working place, the type and model of machine, and location and orientation of light fixtures, and other conditions under which the equipment may be operated to provide required light.

Lighting systems that have been tested and evaluated by MESA shall be considered as being in compliance with all requirements of the illumination regulations, provided that the lighting system is operated within the parameters specified. Normally, when MESA inspectors encounter a lighting system underground that has been tested and evaluated by MESA, no measurements will be necessary; however, the inspector will take measurements to determine compliance if conditions such as the following are found to exist:

(1) One or more lighting fixtures not operating;
(2) Lighting fixtures not properly oriented or maintained;
(3) Excessive coal dust and dirt accumulated on lighting fixtures; or,
(4) Lighting fixtures operated outside the parameters specified.

Of the approaches available, we felt that our own illumination laboratory would be the best way to comply with the regulations. This allows us complete control over the design of our lighting systems and does not tie us to any one manufacturer. This procedure is also useful in predicting performance of lighting systems and allows for changes to be made to correct deficiencies prior to installation.



Westmoreland uses a unique method to determine adequacy of lighting systems to meet federal standards for underground face illumination. Chains are suspended vertically and horizontally to simulate coal seam height and width. The 2-ft sq grids on which light measurements are taken can be seen on the laboratory floor.

Laboratory Construction

Westmoreland launched their planning and projections to institute the laboratory shortly after April 1, 1976. Company allocation of funds for the project followed immediately, and construction was launched on the simulated mine. The illumination laboratory became fully operational within six weeks and has received verbal approval from MESA Technical Support Group.

The laboratory was constructed in our Research and Development Center at Andover, Virginia, only a few miles from Eastern Operations headquarters at Big Stone Gap, Virginia. A steel frame was assembled on which corrugated steel sheets were attached to provide an irregular surface, and then sprayed with concrete sealant approximately 1/2 inch thick to exclude exterior light. The final step was to apply low reflectance black paint such as is used in photographic processing dark rooms.

The final result was a reflectance value of 2 1/2 percent, half the reflectance recommended by MESA for such testing installations. The purpose of this "dark room" is to simulate a coal mine entry by means of various grid systems, thus providing capability of simulating wide ranges of coal mine seam heights and entry widths.

The size of the "dark room" - 44 feet long, 30 feet wide, and 16 feet high - enables us to tram any machine into the lab for testing. Provisions have been made at the research center to truck machines to the site for unloading, with equipment connected to the electric power supply and moved into the laboratory.

Design of Westmoreland's illumination laboratory represents a unique approach in that it differs from the method utilized by MESA's Technical Support Group at its lighting laboratory in Beckley, West Virginia. MESA uses movable ceiling and sides to simulate seam height and seam width. The publication of standards in the Federal Register also specifies that "the ceilings and sides of the simulated working place shall be adjusted to their maximum height and width, or to the height and width specified by the applicant."

After considerable research, we decided on a simpler, more effective approach to the problem instead of movable ceilings and walls. First, the horizontal and vertical surfaces of the lighting laboratory were divided into 2-foot-square grids. Vertically suspended link chains, similar to those used in mine bathhouses, were suspended at the intersection of the 2-foot-square grids to simulate entry width, with the ability to move these chains in 1 foot increments. These chains are marked with luminous paint in 2 foot increments to simulate height. Seam height is simulated by horizontally suspended chains, movable in 1 foot increments.

For actual testing of lighting systems, the horizontal and vertical chains are adjusted to the minimum seam height in which the machine operates and then moved in 2 foot increments to determine the maximum height and width at which the lighting is effective and compliance can be obtained. Light readings obtained at these height and width variations determine the parameters at which the lighting system is approved. Light level readings at the various measuring points are taken by a Tektronic digital photometer that provides instant, accurate indication of the amount of light.

Fluorescent lighting fixtures have been temporarily installed on this continuous mining machine to check positioning of lights and to find if lumen output is sufficient to meet MESA standards.

Testing Procedure

The procedure for designing an illumination system would begin by using the actual machine. By using the full size machine instead of a mock-up, the exact number and location of luminaires or lighting fixtures can be determined, as well as wiring and associated components. Although most of the mining machines used by our company are basically the same, it is planned to bring in one of each model of continuous miner, roof bolter, scoop-tractor, loader, cutter, and face drill and design two or three lighting systems to fit various seam conditions and height restrictions. All illumination systems are designed for the worst case condition with safety the main consideration.

A prime factor that must be taken into consideration is the effect of the lighting on a person's vision. Through placement of the lights and use of diffusers, we will greatly reduce the problem of momentary blindness caused by looking into the light and then away into the darkness of the mine. The problem of glare is the most difficult design parameter we have to overcome.

After determination of the proper lighting system, detailed drawings will be prepared of the individual machine layout showing luminaire placement, orientation and associated wiring. Since we operate our own laboratory, we must submit additional data showing light measurements at approximately 500 different locations on the grid system around the machine. These readings require four to six hours for two men.

After all data has been submitted and MESA has granted the Statement of Test and Evaluation, the installation drawings will be sent to each of the company's four divisions for actual installation. The VPEO engineering staff will help coordinate installation of the lighting systems on individual equipment in each division mine.

Illumination Cost

The herculean task of installing systems on all face machines at Westmoreland's mines is emphasized by the estimates for illumination equipment only. The initial installation cost has been estimated as outlined on Table 1.

The total cost has been estimated at $1,635,030.00, of which $1,345,400.00 will be spent for hardware or lighting fixtures and $289,680.00 for installation labor. Installation labor estimates probably fall below actual costs incurred in mine installations, but that for estimating purposes a certain percentage of hardware cost was adopted as a criteria for labor cost. The costs listed above do not include research costs, MESA fees, equipment maintenance, replacement part cost, power cost increase, or miscellaneous other factors.

For most machines an average of six to eight lighting fixtures will probably include mercury vapor or sodium vapor headlights to illuminate the face, with very high output fluorescent machine lights to illuminate roof, floor and rib.

Required illumination for shuttle cars is a relatively simple matter since the same amount of illumination for machines operated in the face continuously is not required for these haulage vehicles. Compliance achievement for shuttle cars may require installation of only two incandescent headlights in addition to the two headlights furnished as standard equipment on the machine. This would place two headlights at each end of the shuttle car, if needed.

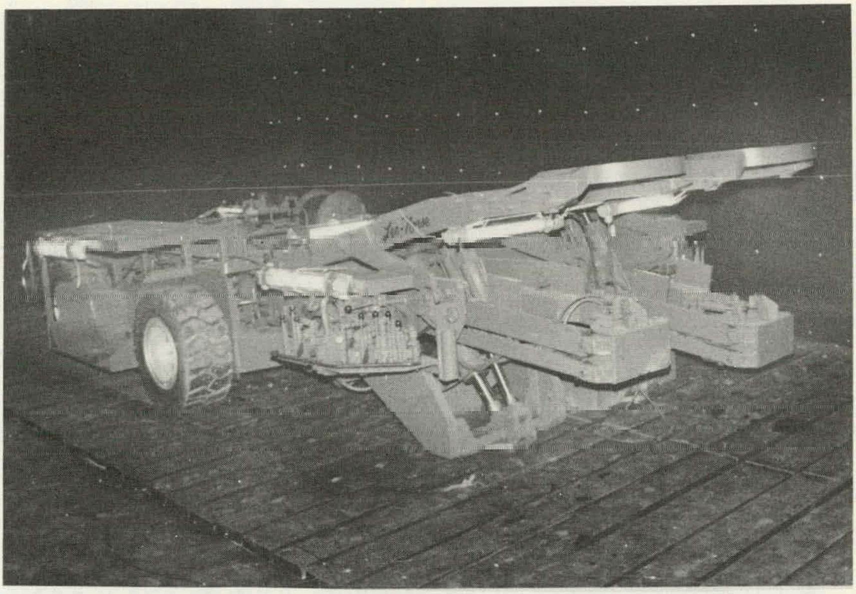
TABLE 1

Equipment (number and type)	Hardware Cost Each Machine	Installation Labor Each Machine
80 Continuous Miners	$4,200.00	$ 960.00
140 Roof Bolters	3,500.00	640.00
52 Scoop-Tractors	3,200.00	640.00
40 Loading Machines	4,000.00	800.00
30 Cutting Machines	3,800.00	800.00
5 Face Drills	3,800.00	800.00
200 Shuttle Cars	300.00	150.00

Testing Results

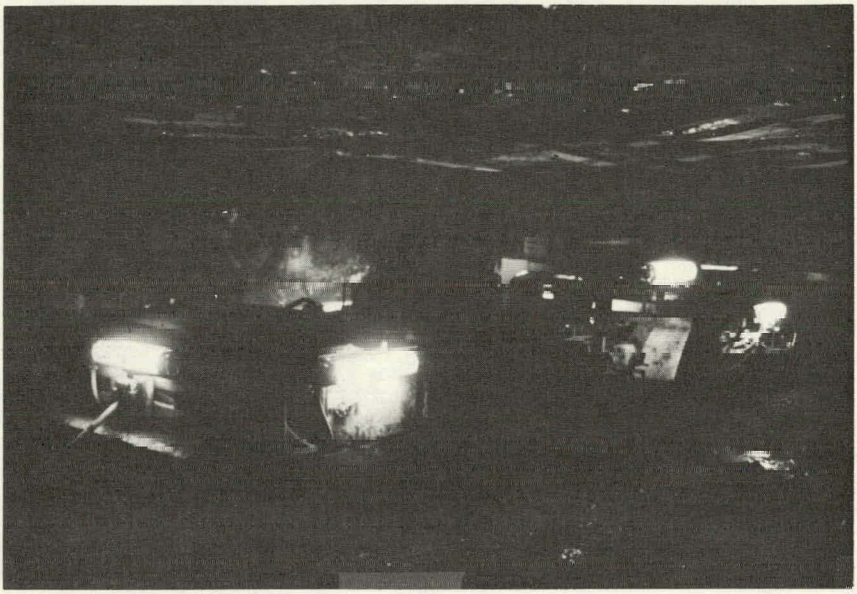
The first Statement of Test and Evaluation letter, based on data and tests submitted by our company, was issued October 15, 1976, on a Lee-Norse TD-2-34 doublearm roof bolting machine. The letter specified its use within the parameters of 8 foot height and 22 foot entry width. The all-fluorescent system utilizes ten very high output fluorescent fixtures and clears the way for its application on 27 such bolters purchased by our company. This system was also the first one we installed underground.

At present, we have over ten "STE" letters and are regularly making applications for others. We now have four continuous miners and one roof bolter illuminated underground. These installations are in seams of five to eighteen feet high. The correlation between our lab and underground installations have confirmed the simulated surface approach. With this approach, all systems installed have more than met compliance.



Lee Norse TD-34 Roof Drill in laboratory during testing.

Lee Norse TD-34 Roof Drill in laboratory during testing.



Lee Norse TD-34 Roof Drill in operation underground.

Lee Norse TD-34 Roof Drill in operation underground.

Installation and Maintenance

We have installed lighting systems on three continuous miners in our shop and lighting systems on a continuous miner and a roof bolter underground. The systems installed underground require 2 to 3 times more labor than those done in our shop. Those systems installed in our shop were far superior to those installed underground in terms of protecting wiring, power supply and luminaires.

During three months of operation on a Lee Norse Model 455 continuous miner, we have encountered four (4) lamp failures and one (1) ballast failure due to vibration and improper wiring. In that same section we have had three (3) premature lamp failures on a Lee Norse Model TD-2 Twin Head roof bolter.

On a Lee Norse Model 265 continuous miner we have had to replace three (3) headlights and two (2) fluorescent luminaires in only five weeks of operation. These fixtures were destroyed by roof falls and rib rolls. The lighting system on this machine consists of two mercury vapor headlights and six fluorescent luminaires.

Based on these preliminary results, an operator's replacement and maintenance costs could be 100% of the installation costs yearly. Operating costs such as these could become staggering.

Conclusions

It is readily evident to any person with practical knowledge of modern-day coal mining that the task of providing the required levels of illumination is not an easy one. Retrofitting many of the existing machines with permissible lighting will require custom installations. Special illumination systems will be required for certain types of mining.

It is a physical impossibility to have all Westmoreland's machines on all 100 working sections equipped with lighting systems by the required date of April 1, 1978, but we feel we will be able to get all of the lighting systems designed and approved within the required time.

The task of achieving these goals requires a concerted effort, but utilization of our illumination laboratory, we feel, gives us a better chance of meeting compliance.

ILLUMINATION EXPERIENCES AT PENNSYLVANIA MINES CORPORATION

Joseph Kreutzberger
Vice President-Safety & Training

Pennsylvania Mines Corporation
Ebensburg, Pennsylvania

At the inception of the proposed illumination regulations, an Illuminating Committee consisting of personnel from the Safety, Maintenance, and Electrical Departments was formed to decide on a standard system of illumination compatible with all of the Corporation mines.

The results of this Illuminating Committee's indepth study indicated that at this time there are two ways of illuminating a working section and complying with Section 317(e) of the Coal Mine Health and Safety Act of 1969 and 75.1719 of the regulations:

1. Area Lighting.
2. Machine-Mounted Lighting.

Area Lighting

Area lighting consists primarily of stationary type of luminaires positioned in such a manner that light would be directed to the surfaces required by law. These luminaires would then be advanced as the mining equipment went through its cycle. This type of light would appear to be more appealing in extremely low coal mining, such as machines of the augering type and longwall mining. It does not appear to be appealing in continuous mining sections where the equipment is mobile and moves repeatedly. It would require additional cables in the face area, and also additional personnel would be required to position the lights and maintain them in their proper position.

Machine-Mounted Lighting

Machine-mounted lighting involves mounting the luminaires on the machine itself. This would then make the lighting as mobile as the machine. No additional personnel would be required to maintain the luminaires to fulfill the requirements of the law regarding luminaire positions.

A decision therefore had to be made between area lighting or machine-mounted systems. It was decided to approach the problem by following the concept of machine-mounted lighting because of the following reasons:

1. Area lighting systems are more cumbersome.
2. Additional stationary equipment.
3. Additional cabling in the face area.
4. Restricts the movement of mobile face equipment.

The machine-mounted lighting developed for use at the time of the study was:

1. Complete fluorescent.
2. Complete mercury vapor.
3. Complete sodium vapor.
4. A combination of any two.

In order to obtain experience with the different types of hardware and fixtures available, it was decided to acquire a system of each type for test and evaluation. Therefore, testing was instituted on the following:

1. Continuous miner with mercury vapor.

2. Continuous miner with fluorescent, mercury headlight combination.
3. Continuous miner with fluorescent, standard headlight combination.
4. Roof bolter with mercury vapor.
5. Roof bolter with fluorescent, mercury vapor combination.

The purpose of these installations were to evaluate the type of system (mercury vapor or fluorescent) that would be most suitable for the seam height and mining procedures applicable to the various Corporation mines. The outcome of this portion of the evaluation was that more attention would be directed to a system consisting primarily fluorescent with as little mercury vapor as possible. The reasoning was the potential glare problem; and although mercury vapor luminaires have an exceptionally high output, it was felt that the quantity of light is not required in the less than four-foot to five-foot seam heights encountered in the Corporation mines.

The Illuminating Committee was of the opinion that the design criteria must meet the following standards:

1. Provide a certifiable system that would be acceptable to the machine operator and those personnel in close proximity of the machine, such as the helper and shuttle car operator.

2. The position of the fixtures must not be susceptible to abuse from the adverse conditions involved in coal mining.

3. The fixtures must not interfere with the outline of the machine, which meant that the fixtures did not increase the overall machine height or width to a point that these additional dimensions would hinder safety or production.

The various systems would need testing to evaluate the following:

1. Durability (Shock and vibration).

Tamp failure due to vibration and shock have been minimal although problems have been encountered with respect to blasting of coal and rock in longwall applications.

2. Light Output (Capable of providing enough lumens to bring the machines into compliance).

It is felt with proper placement, most luminaires would be capable of providing enough lumens to bring machines into compliance.

3. Cage Construction (Impact withstandability).

Care must be taken in cage selection. It should be able to withstand abusive conditions and show imagination in design to let light out. A rub rail was extended two inches on both sides of the continuous miner. This was done with the intention of protecting the luminaires by the physical layout of the machine. The theory was the rub rail extended beyond the luminaires, thus providing the primary protection and the protective cage providing the secondary protection. The decision for this alteration was based on past experience of protruding parts on mining machines.

4. Ease of Installation (Wiring).

In the initial installation and maintenance of the lighting system, it was felt the system should have as few packing glands as possible and incorporate the quick disconnect principle.

In January 1975, it was decided to attempt to illuminate a longwall in a thin seam with a Cooperative Agreement with the Bureau of Mines.

At the time of the first attempt to illuminate a longwall face, there were two types of chocks in service, the four-leg and six-leg chock. There was a restricted area between the legs of the four-leg chocks; therefore, it was necessary to mount the luminaires in some position in front of the legs, preferably as close to the conveyor panline as possible. The average height in the travelway was approximately 34 inches to 36 inches and this influenced the final decision to mount the luminaires on the armored face conveyor. The clearance area over the conveyor panline for the shearing machine must not be interfered with, that is, the area in which the shearing machine travels. There are occasions when the shearing machine clears the underside of the canopies by inches. The mounting of the luminaires to the conveyor panline had to be of a design that would flex and not be mounted in a rigid position. Little concern at the first installation was given to the position of the luminaires. The only concern at this time was to mount the luminaires to the conveyor panline in such a manner that they would not hinder the passage of light to the coal face and walkway. However, the design of the brackets had to be canted 23 degrees for operator's comfort. The Industrial Engineer's Report of June 9, 1975 stated complaints had been registered by several of the shearer operators. They felt the directional intensity of the lights should be cast almost entirely towards the face. The justification for their complaints is as follows: In the normal progression of his job duties, the shearer operator moves along the panline operating the shearing machine. When he moves, he looks down, crawls several feet and then looks up to the roof to see where his bits are cutting. During this process of looking up and then down again, the operator's eyes pass through a partial intensity of the light each time. This causes a semi-blind condition which interferes with the proper operation of the shearing machine.

The approach to the problem of illuminating the headgate and tailgate was accomplished by the use of luminaires mounted on pogo sticks. In our view, this method presented several undesirable conditions; such as, extra cabling in the tailgate and headgate entries; on the occasion of blasting, they would have to be removed; and rough usage caused premature failure of the springs in the pogo sticks. In the headgate and tailgate, the pogo sticks with luminaires were removed and luminaires were mounted in a fixed position on the stageloader and on the tail piece of the face conveyor.

The first longwall installation produced the following results:

1. The first attempt of mounting luminaires by the use of flexible springs was initially acceptable, but failed after they were stressed beyond their elastic limits.

2. The outer covering of the luminaire appears to be of sufficient strength. But on the occasion of blasting and even if the luminaires are removed from their brackets and placed on the floor in a secure position, damage does occur to the internal parts.

3. Alignment of the face is improved. You can visually look down the luminaires and readily observe face alignment.

4. If one light goes out, do we shut down the longwall section? Do we include overkill? A practical answer to this dilemma would be a definite time period such as 24 or 48 production hours to replace luminaires.

5. More emphasis should be placed on the lighting of the coal face. The .06 lamphents should only be required on the coal face. At this time, little safety value appears to be derived from illumination of the walkway.

6. It is felt that one additional maintenance man would be required to maintain the lighting system.

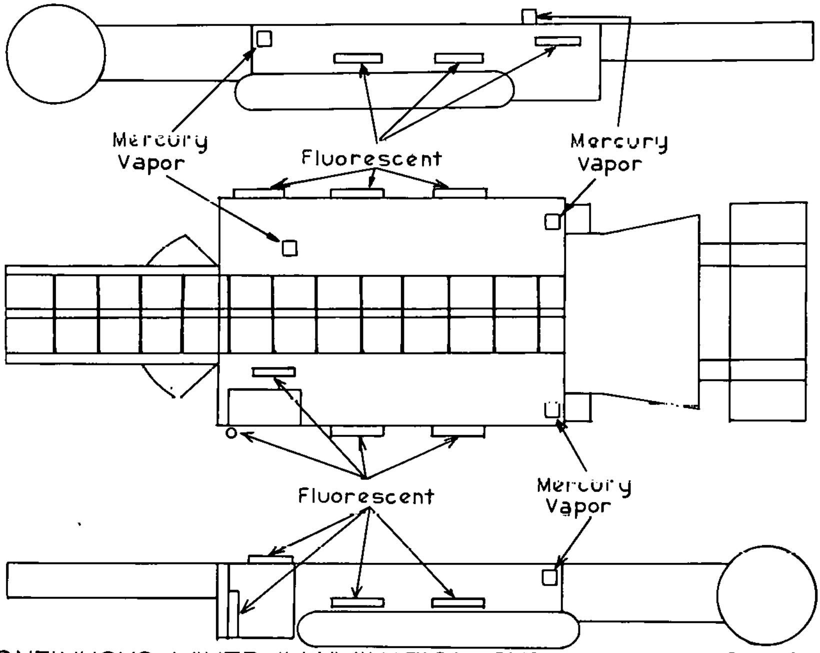
<u>Second Longwall Illumination (Six-leg Chock)</u>

From the experience gained in the first attempt to illuminate a longwall in a thin seam with primarily four-leg chocks, certain changes would need to be implemented to illuminate a longwall face utilizing the six-leg chock.

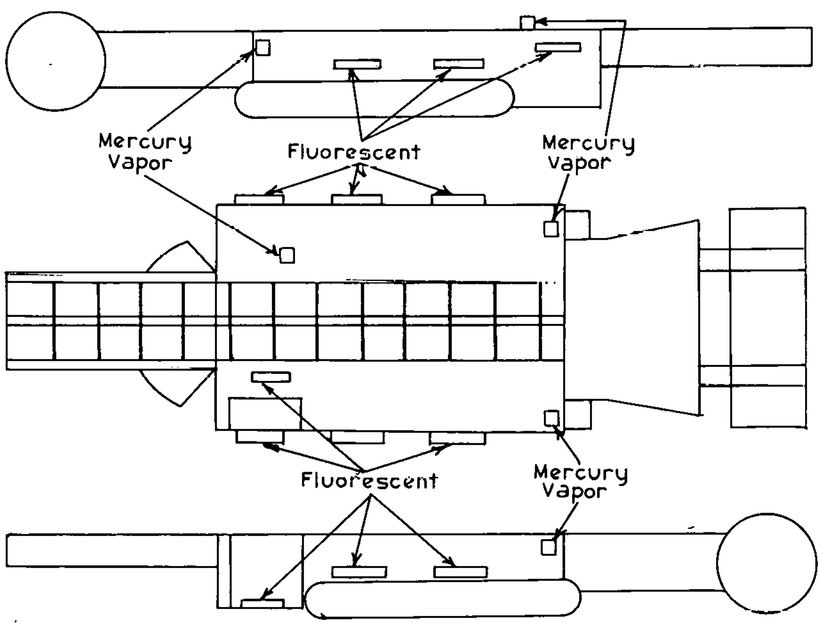
Since the walkway was between the legs, this area must not be utilized for mounting any fixtures or cables. The luminaires should be mounted in some position between the forward legs of the chock and the coal face. A system of preventing the power lines from being stressed beyond their limits by the physical movement of the chocks and armored face conveyor should be developed. The luminaires should be mounted in such a manner that the employees should not have to look through the luminaires to perform their job functions. Maintenance should be reduced; and where possible, the wiring should be simplified. With the above criteria in mind, the following changes were made in the illumination of the second longwall face. The luminaires were mounted under the canopy forward of the front legs. This location should illuminate the work area and reduce the operator's glare. All necessary lighting components were mounted in positions in which the employees do not normally come in contact. A 1-1/4 inch staple-lock hose, double steel, braided in 30-foot lengths, is used as a protective covering or conduit to provide primary protection for the 1-1/8 inch, 6/3 GGC cable which provides 110 volt, three phase, AC power. It has been determined by experimentation that 36 feet of 1-1/4 inch staple-lock hose and 36 feet of 1-1/8 inch trunk power line provides a lineal distance of 24 feet. This appears to provide sufficient slack for chock movement. Initially, the 110 volt, three phase, AC power was derived from a source approximately 300 feet outby the longwall face. This, therefore, made the working on the illumination system cumbersome. This three phase, 110 volt power was moved to a source near the stageloader where an employee is on duty at all times with telephone communication and can facilitate the necessary repairs of the illumination system.

A Cooperative Agreement was received from the Bureau of Mines to illuminate a continuous mining machine and roof bolting machine. The lighting systems for these machines were designed to meet MESA standards of .06 foot lamberts. The luminaires were positioned in such a fashion that the light would be directed to the required surfaces as indicated by Section 75.1719 of the regulations. Attempts were made at this stage to invision the operator's functions and correct as necessary. The machines were then put in service and monitored. Operators' comments and recommendations were considered and every attempt was made to fulfill their recommendations. If luminaires were bothersome, attempts were made to correct the situation through relocation or guarding. In most situations, the required foot candle level would drop in those areas where guarding or relocation was necessary. If selected areas are causing operator discomfort and cannot be illuminated to the required levels without objections, is the light in these areas increasing or decreasing the accident probability. A luminaire was placed near the operator's compartment to bring the floor area into compliance. The operators and helpers complained that this luminaire interferes with their viewing of the trailing cable. This luminaire was repositioned. The mercury headlights on the continuous miners were canted upward to offset the operator's complaints that the immediate face did not have enough illumination. The machine operators prefer to have the face area flooded with light. To ease maintenance and provide access to the various panels and compartments, the luminaires were mounted with a hinge mechanism so they could be flipped up and the necessary work completed in the panels.

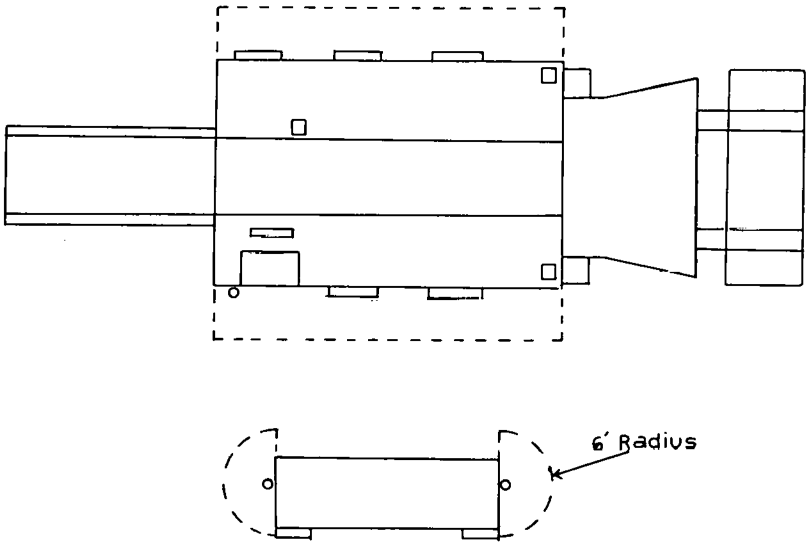
In conclusion, the passage of any regulation should be preceded by exhaustive experimentation in both low coal and high coal applications. The operating coal companies should not be forced into a position where in an attempt to meet the deadlines of new regulations begin a hit or miss program and not know if their attempts will meet standards set up by the regulatory agencies. What will MESA instructions be to their inspectors on illumination standards? When does he write a violation? Will the operating companies have to install the illumination systems and then hope MESA does not add requirements that will require extensive field changes.



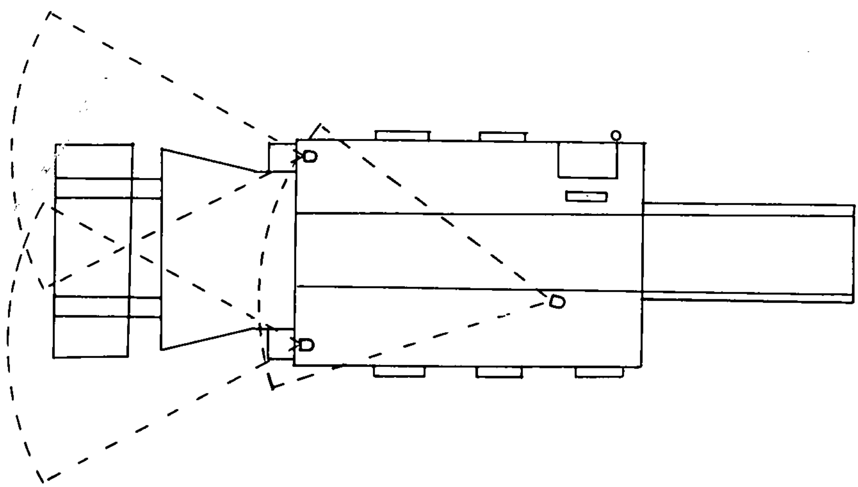
CONTINUOUS MINER ILLUMINATION SYSTEM WITH CANOPY



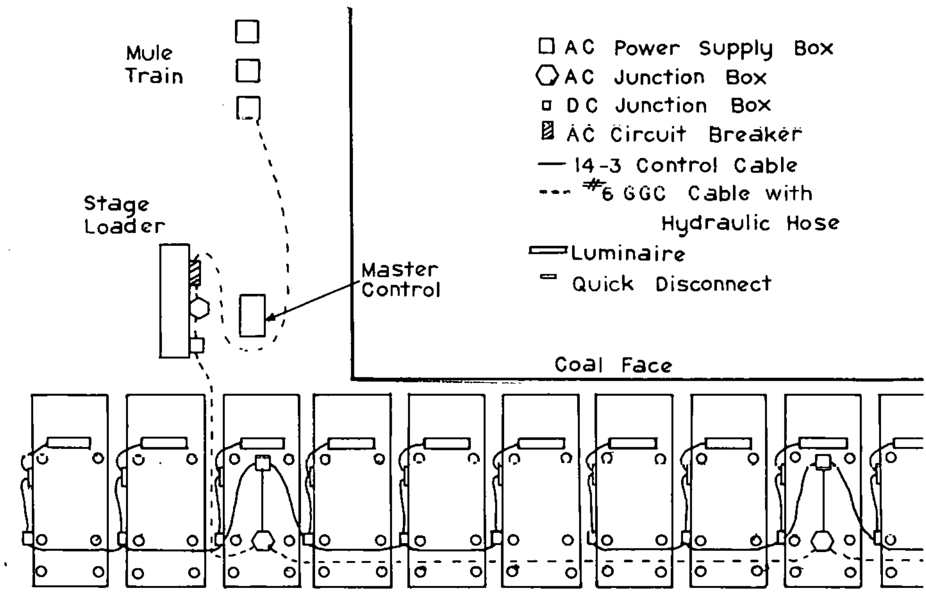
CONTINUOUS MINER ILLUMINATION SYSTEM WITHOUT CANOPY



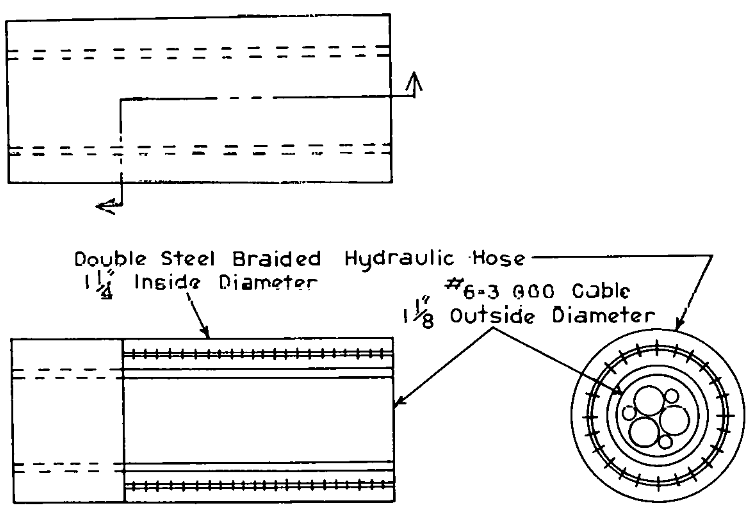
AREAS ILLUMINATED BY FLUORESCENT LUMINAIRES



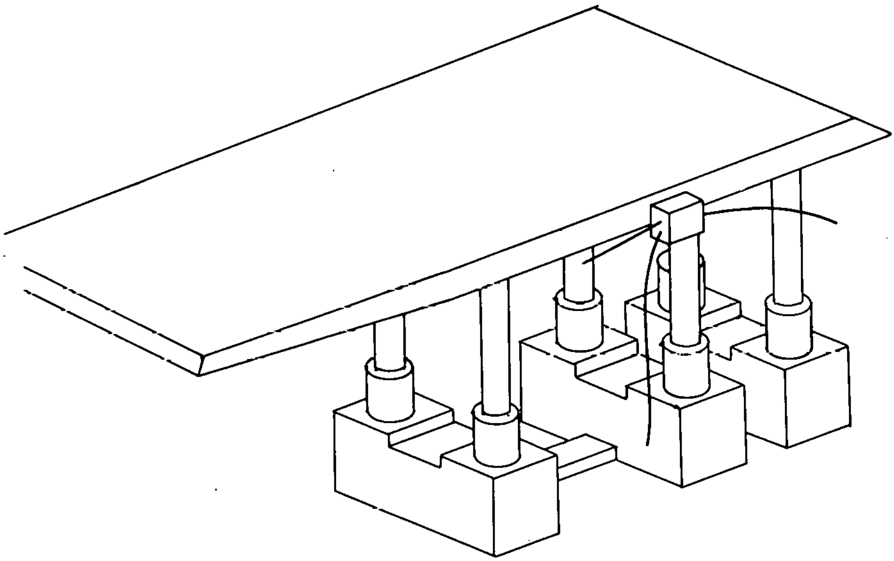
AREAS ILLUMINATED BY MERCURY VAPOR HEADLIGHTS



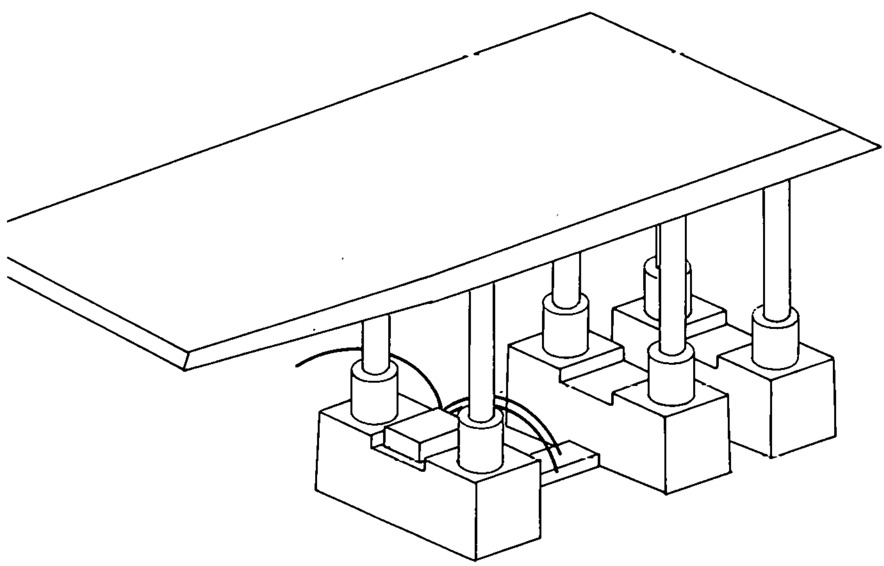
WIRING SEQUENCE FOR LONGWALL ILLUMINATION SYSTEM



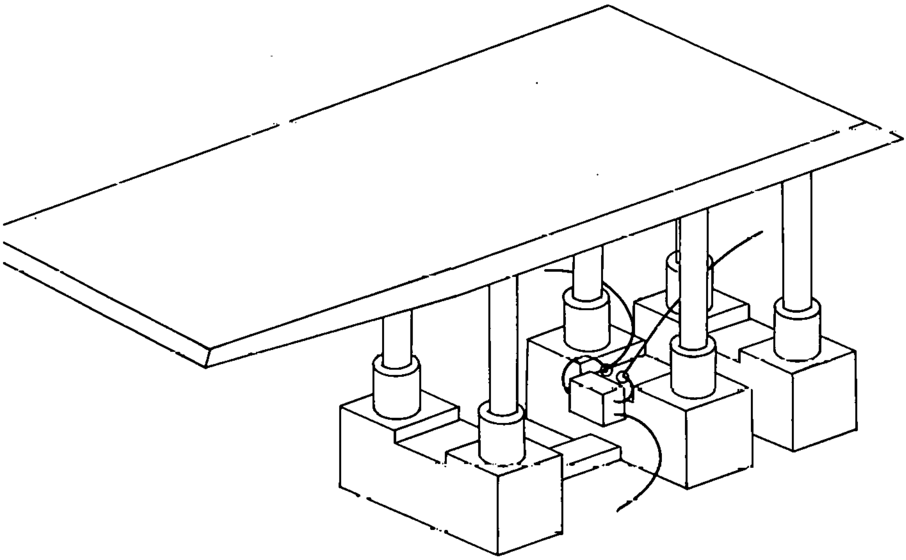
PROTECTIVE HOUSING FOR #6-3 GGC CABLE



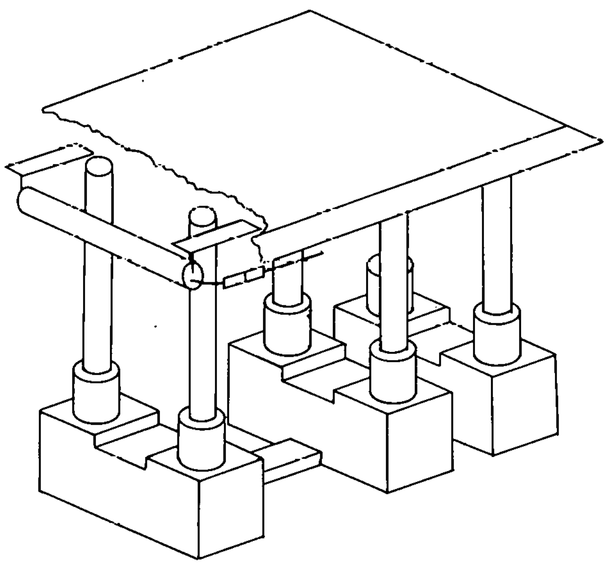
LOCATION OF D.C. JUNCTION BOX
Approx. Size 4"x4"x3"



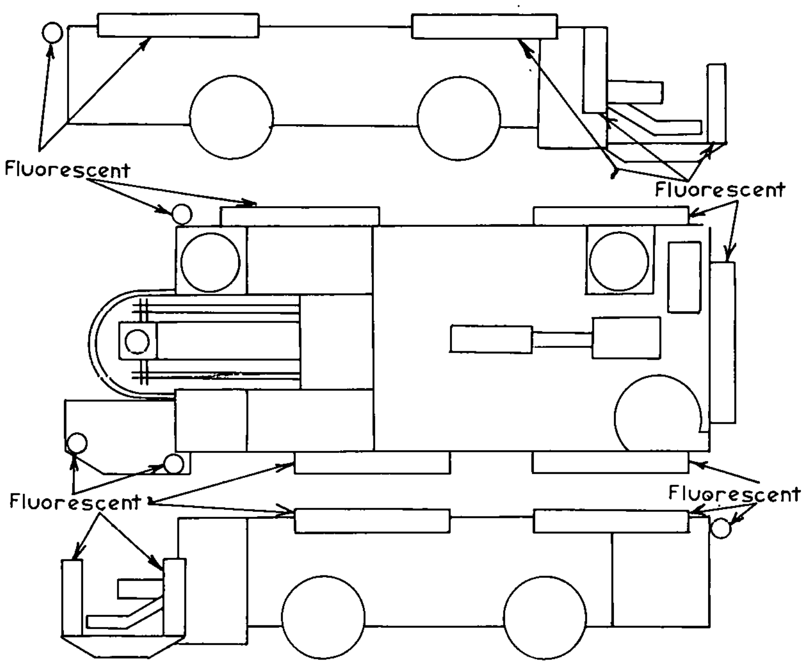
LOCATION OF A.C. POWER SUPPLY BOX
Approx. Size 11"x7"x7"



LOCATION OF A.C. JUNCTION BOX
Approx. Size 7"x3"x5"



LOCATION OF LIGHT ASSEMBLY
Approx. Size 3"Dia. x 24"



ROOF BOLTER ILLUMINATION SYSTEM WITH CANOPY

RESULTS OF CONSOL'S ONGOING PROGRAM ON MINE ILLUMINATION

L. D. Patts
Environmental & Metallurgical Engineer

Lee Engineering Division,
Consolidation Coal Company
200 Hidden Valley Road
McMurray, PA 15317

Consolidation Coal Company, in co-operation with the United States Bureau of Mines, began experimentation with machine mounted underground face illumination systems in early 1972. Although illumination requirements, hardware specifications and light measuring techniques were not yet established, we felt that expertise and experience in this field would become increasingly important to our Company.

Initial Efforts Aimed at Practicality

Our initial efforts in coal mine illumination were aimed at investigating practical lighting systems for face equipment. We were concerned with installing unobtrusive lights which provided sufficient face illumination for safety, but at the same time were readily maintainable, electrically reliable and physically sheltered from damage. We believe that our initial lighting systems provided sufficient face illumination for safety, but because only prototype lighting components were available the resultant poor system reliability and maintainability necessitated drastic improvement before face lighting could become practical.

When MESA's final illumination standards were announced and light measuring techniques were defined, Consol checked their lighting systems underground and determined that their initial lighting systems were not in compliance with the proposed illumination standards. After determining that our face lighting systems were not in compliance, we began adding additional lighting hardware to meet compliance. This additional lighting hardware, with its increased vulnerability and decreased reliability, rendered the lighting systems impractical, if not impossible to maintain.

Extensive Field Testing

Since 1972 Consol has installed every available variety of machine mounted mine illumination hardware. All hardware installed has been field tested under actual mining conditions to evaluate its practicality for use underground.

During the past 6 years Consol has installed face illumination systems on over 40 pieces of underground face equipment. The following charts offer a brief description of our illumination hardware field testing program.

REGION	MINE	TEST MACHINE	12 VDC Fluorescent	120 VAC Standard Output Fluorescent	120 VAC High Output Fluorescent	HgV Headlights	HgV Machine Mount	NaV Machine Mount	Machine Mount Lucalox
Eastern	Mathies	Joy 10CM Miner				X			
		Coal Transfer Vehicle		X		X			
	Montour #4	Galis Bolter					X		
	Ireland	Joy 11CM Miner							X
		Joy 11CM Miner	X			X			
		Joy 14BU-10 Loader	X						
	Renton	Marietta Drum Miner			X				
		Galis Bolter			X	X			
Midwestern	Franklin	Joy 11CM Miner							X
		Joy 11CM Miner		X		X			
	Rose Valley	Jeffrey 120M Heliminer		X		X			
		Fletcher Roof Drill		X		X			
		Jeffrey 120M Heliminer		X		X			
	Oak Park	Jeffrey 120M Heliminer			X	X			
Northern West Virginia	Williams	Joy 2BT-2H Borer	X			X			
		Joy 14BU-10 Loader	X			X			
		DC Fletcher Bolter	X			X			
		Joy 2BT-2H Borer		X		X			
		Joy 14BU-10 Loader		X		X			
		DC Fletcher Bolter		X					
		Joy 2BT-2H Borer				X	X		
	Blacksville	Fletcher Bolter						X	
	Humphrey #7	Joy 1CM Miner	X			X			
		Goodman Loader	X						
		DC Galis Bolter				X	X		
		Joy 1CM Miner		X		X			
		Joy 1CM Miner			X	X			

REGION	MINE	TEST MACHINE	12 VDC Fluorescent	120 VAC Standard Output Fluorescent	120 VAC High Output Fluorescent	HgV Headlights	HgV Machine Mount	NaV Machine Mount	Machine Mount Lucalox
Southern Appalachia	Rowland #4	Acme Bolter					X		
	Beechfork	Jeffrey 120M Heliminer	X			X			
		Galis Bolter	X						
	Lynco	Joy Cutting Machine		X		X	X		
		Joy Loader		X			X		
		Joy Face Drill				X	X		
	Maitland	Joy 11CM Miner	X			X			
	Bishop 36	Jeffrey 120L Heliminer			X			X	
		Galis Roof Bolter			X				

Experience Gained by Field Testing

We believe that we have gained considerable insight into the light distribution, ruggedness, maintainability and operator reaction of all major brands of mine illumination hardware.

There are presently three major types of illumination hardware approved for underground use: Mercury Vapor, Fluorescent and Sodium Vapor. Our field testing program has included the installation of hardware from each of the major lighting manufacturers. We have drawn the following conclusions for each type of lighting hardware from our field tests:

Mercury Vapor (HgV) -
There are presently two different methods of packaging Mercury Vapor lamps for underground mining use; the HgV headlight and the HgV machine mounted light. Mercury Vapor headlights have proven far superior to existing incandescent headlights with regard to light output and durability; however, they possess a long hot restrike time and sometimes reflect light from fine water mists and coal dust. The light reflections raise severe objections from machine operators. Mercury Vapor headlights have found limited application as machine lights because of their intense and directional light blinding anyone who may be inby this light source.

Mercury Vapor machine mounted lights provide excellent light output but also provide a source of glare for mining personnel if they are not shielded completely from the miner's field of vision. Some Mercury Vapor machine lamps have been frosted to reduce glare; however, this drastically reduces light output. As with HgV headlights, HgV machine mounted luminaries possess an unacceptably long hot restrike time. Our field testing has demonstarted that HgV lamps are most likely employed in high coal mines where all sources of light can be completely shielded from an operator's sight or on a continuous miner as headlights where minimal dust and water conditions exist.

Sodium Vapor (NaV) -
High pressure Sodium Vapor lights have recently been introduced into the underground coal mining industry as machine mounted luminaries and have been installed on several pieces of Consol face equipment for field evaluation. Our experience to date indicates that although the hot restrike time for Sodium Vapor lights is more favorable than that of HgV lamps (30 sec. for NaV), NaV lamps are an intense source of light and must be completely shielded from the miner's field of vision. Sodium Vapor lights have found acceptance when mounted on mining machines as headlights because of their ability to penetrate water mists and coal dust better than HgV or incandescent headlights.

Standard Output Fluorescent -
Standard output fluorescent lights have been used as face illumination sources in coal mines for several years. The soft white light characteristic of standard output fluorescent lights renders them acceptable to most underground employees. Unless the light source is directly in the visual path of a machine operator, few complaints concerning glare are registered. The unfortunate fact is that the output of standard fluorescent lights is relatively low and it is practically impossible to mount enough standard output fluorescent fixtures on a mining machine to achieve compliance with MESA lighting regulations. Consol has installed as many as 13 standard output fluorescent luminaries and 2 HgV headlights on a continuous miner, but has been unable to achieve compliance with MESA lighting regulations.

High Output Fluorescent -
High output fluorescent lighting is now offered by most manufacturers of underground mine lighting hardware. High output fluorescent lights offer considerably more light output per length of luminarie than do standard output fluorescent

lights. Because a higher light output is achieved per length of fixture, glare problems are often encountered with the use of high output fluorescent luminaries unless the lights are shielded from the operator's visual path. Miner operators have commented that they experience difficulty when trying to view objects or persons if they must look over high output fluorescent lights. Our experience indicates that face personnel overwhelmingly prefer standard output fluorescent lights to high output fluorescent lights.

Problems Still Remain

Consol's field experimentation with face illumination systems has demonstrated that the major problem in meeting compliance with MESA illumination regulations is not one of mounting enough lighting fixtures on a given piece of face equipment to generate the necessary light output. The problem is to install lighting hardware on equipment in a practical and maintainable configuration which will enhance the safe and efficient performance of the miner's tasks in the working place.

It has been stated by MESA that the intent of the illumination regulations is to require an illumination system which will:

1. Provide adequate levels of illumination so that the miner can perform the tasks involved in underground mining operations in a safe manner.

2. Allow the miner to recognize incipient hazards within his field of vision.

3. Not introduce objectionable glare sources into the miner's working environment.

4. Require minimal maintenance so that additional personnel are not required in the high risk areas for the maintenance of lighting fixtures and associated circuitry.

Our field testing since early 1972 has demonstrated that the above intentions of the required illumination systems have not been met by the majority of face illumination systems.

Illumination Systems Must Be Practical

If practical illumination systems are to be installed in our underground mines, many problems concerning illumination practicality and safety must be resolved. It is naive to assume that underground lighting systems can be proven acceptable for use solely on the basis of laboratory testing and installation on mockups of mining machines. It may be possible to achieve compliance with illumination regulations by showing the location of luminaries on machine blueprints and mounting this hardware on wooden machine mockups. It is an entirely different undertaking to install this same lighting hardware on actual mining machines in an operating coal mine. Illumination systems require installation and field testing on actual mining machines under actual mining conditions. Very few mining machines which have been in service for any length of time remain unmodified; therefore, many lighting installations will require custom fitting of lighting hardware.

The most favorable time to install lighting hardware is when the face equipment is being fabricated or when it is undergoing major rebuild in an outside machine shop. If mounting is done in the shop environment, lights can be properly protected by modifying the machine cover plates or structure and lighting cables can be properly guarded. It is unfortunate that the majority of lighting installations will of necessity be done in-mine which is the worst circumstance in which to do custom installation.

Even though the existing deadline for compliance with underground face illumination regulations is less than 6 months away, many coal companies possess limited experience with face illumination systems and their lighting programs, if established, are moving slowly. This past summer MESA Approval and Certification suspended, revoked or rescinded approvals on mine illumination hardware for underground use. Although no one can take issue with such action, many coal companies' lighting programs have suffered a severe setback and it is very unlikely that all face equipment can be illuminated by the April 1, 1978 compliance deadline.

Summary

Consol's underground field testing of face illumination hardware has demonstrated that the required 0.06 foot-lambert (fL) reflected light level is a convenient guideline as a starting point for face illumination. This same field testing has shown that 0.06 fL surface brightness need not be maintained for every 4 sq. ft. area of coal surface in the face area haphazardly defined as the "normal visual field of the miner". To illuminate every 4 sq. ft. area to the 0.06 fL level is often impossible without locating luminaries in positions totally unacceptable to machine operators or too vulnerable to damage. Severe light overkill is also realized in the majority of instances when attempting to achieve 0.06 fL on every 4 sq. ft. area of the face. It is our belief that the 0.06 fL level need not be maintained on every 4 sq. ft. grid of the face area for personnel to work safely. We have determined through practical example that the following areas need not be illuminated to 0.06 fL.

A. The area of roof directly above the machine operator's canopy.

B. The area of rib and bottom opposite the operator's side on continuous miners alongside and inby the gathering head.

C. The area of bottom and rib on the operator's side of the continuous miner alongside and inby the gathering head.

If the above mentioned coal surface areas were not required to be illuminated to the 0.06 fL level, face illumination could be achieved with less possibility of overkill and glare.

Consol has gained considerable experinece in the installation and field maintenance of face illumination systems during the past several years. Our experience indicates that more development and field work is necessary on mine illumination systems, and that MESA illumination regulations will have to be modified before it will be practical for coal operators to install and satisfactorily maintain safe illumination systems.

If implemented properly, face illumination could be an aid to conducting mining tasks in a safer manner; it will however be a detriment to safety if compliance with present illumination standards is the only goal. Mine workers should not be subjected to the obvious safety hazards associated with glare, and they need not be if reasonable modifications are made to existing illumination regulations. The application of face illumination must be done in a safe, workable manner which will not detract from an operator's ability to work safely.

NORTH AMERICAN COAL CORPORATION'S MINE ILLUMINATION PROGRAM

James R. Elliott
Chief Electrical Engineer

North American Coal Corporation
Powhatan Point, Ohio

The manuscript of this paper was not received in time to permit its inclusion in the book of preprints. Copies of the paper will be available at the Conference or can be secured by writing directly to the author at the following address:

James R. Elliott
Chief Electrical Engineer
North American Coal Corporation
Powhatan Point, Ohio 43942

BCR PROGRAM IN MINE ILLUMINATION

Felix du Breuil
Assistant Manager, Mining Research

Bituminous Coal Research, Inc.
Monroeville, Pennsylvania

Purpose of Program

In December 1976 the mining research staff of Bituminous Coal Research, Inc., was asked by its Board of Directors to determine the present status of the federally legislated face machinery illumination program. With the deadline for compliance set for April 1, 1978, many BCR member companies were looking for guidance in planning the installation of lighting equipment in their mines. Most of them had no experience at all in this area. Suppliers were ready to sell hardware, but many prospective customers felt they did not have the knowledge required to make an intelligent choice.

Regulations had been set by MESA, but few in the field knew precisely what they were. Many experiments had been conducted by the Bureau of Mines, but again the results were not widely known. Also, there were rumors about potential or imaginary harmful effects resulting from illuminating face machinery. Thus, the BCR directors felt that they needed a first-hand, up-to-date report on the true state of mine illumination and on current installation activities. With this in mind, BCR started establishing contacts with all groups concerned with mine illumination.

The first to be contacted were the manufacturers of mine lighting equipment--to find out what kind of hardware was available, and what kind of installation service was provided to the customer. Each manufacturer was also questioned about the problems encountered and about the difficulties still to be solved in satisfying the federal regulations on illumination, in obtaining MESA approval of his equipment, and in applying his equipment to various types of mining machines and seam conditions.

Contact was also made with the two groups in MESA which deal directly with the lighting program, the Illumination Group in Beckley and the Approval and Certification Center in Pittsburgh, in order to obtain background information on the procedures for approval and to discuss technical aid available to coal operators. MESA does organize seminars on mine illumination at the Academy in Beckley, and several members of BCR's staff attended sessions last May.

Comments were obtained from the major mining-equipment manufacturers on the problems of integrating illumination in the design of machines. Of interest, too, was what their policy would be regarding the retrofitting of machines already in the field, and their time schedule for delivery of machines with integrated lighting.

The final task (and probably the most important) was to investigate the experience of coal operators who had installed, or were in the process of installing, lighting systems in some of their sections. By June 1977, the BCR staff had visited 14 mines in Pennsylvania, Ohio, Virginia, and West Virginia, and observed 23 machines with functioning lighting systems. The companies visited accounted for approximately 25 additional machines lighted with similar systems. Four manufacturers were responsible for illuminating these 50 machines, which were either continuous miners or roof bolters.

Rather than describe individually the various installations visited, this paper will present some general remarks summarizing the experience of those interviewed during the course of our program.

Hardware

1. Type of Lamp

Because of their high efficiency and long life, two types of lamps are in common use for machine lights (as opposed to headlights)--the high-intensity discharge (HID) and the fluorescent. These two types are basically very different light sources, the former being almost a point source, very bright, while the latter is a low-brightness, large-area light source. At first it may appear odd that either type could be chosen for the same purpose on a face machine, except that neither one can satisfy all contradictory requirements of a lamp to be used underground. Coal has a very low reflectance, hence a large lumen output is needed. In addition, space is at a premium on any machine. Thus, HID is a good choice to provide a large amount of light out of a relatively compact luminaire. On the other hand, since lamps are likely to be in the field of view at some time, a lamp with low surface brightness, such as a fluorescent, is desirable. So one is faced with having to choose between few luminaires, but very bright; or low brightness luminaires, but in larger number and/or of relatively larger size.

Satisfactory installations have been achieved with each type of lamp, and it would be a mistake to simply dismiss the mercury or sodium HID as "too bright," or the fluorescent as "too low in output."

2. Mechanical Properties of Luminaires

A luminaire has to remain functional and retain its explosion-proof characteristics under "normal" operating stresses. The problem faced by the designer is to estimate what is "normal" abuse. The stronger the luminaire, the heavier, bigger, and the more expensive it is. Also, the addition of steel for mechanical strength usually means fewer or smaller lenses, thus reducing light output and, consequently, more luminaires have to be used.

The presently available luminaires are, perhaps, not strong enough in all cases, but sufficient for most applications.

The critical factor in luminaire survival is its location on the machine. If it is somewhat "recessed" between existing frame members or other heavy parts of the machine, it may not even need the "cage" provided by many manufacturers. On the other hand, a luminaire mounted at the edge of a continuous miner requires a heavy guard rail in addition to the cage if experience indicates that the machine commonly hits the rib. Although such additional protective devices may be suggested by the luminaire manufacturer, these custom features are better provided by the mine operators.

Another factor is local conditions and methods of operation. For example, deck-mounted luminaires can easily be damaged against the roof in an undulating or irregular seam, while they may never even be scratched in a regular seam. Operator procedures and habits are also factors which affect luminaire survival.

3. Maintenance of Lighting Systems

Ease of lighting system maintenance varies greatly with the design of the luminaire, the construction of the power supply, and the wiring arrangement utilized. Machine maintenance is affected, also, because it is often necessary to remove luminaires to gain access to other parts of the machine. For these reasons the mine maintenance personnel should be given a large say in the choice and installation of luminaires. They know how to judge equipment from the maintenance standpoint and they will install the lighting systems so as to retain sufficient accessibility to the parts of their machines which require frequent maintenance.

Illumination Requirements

1. Minimum Light Level

The main legal requirement (and the only one defined by a number) is to achieve a minimum brightness of 0.06 fL on specified surfaces surrounding the machine. Generally speaking, this is not a difficult level to achieve. However, there exist around almost every machine some small areas which are very difficult to illuminate to that level without placing luminaires in locations unacceptable to the operators or too vulnerable to damage. Attempting to raise the illumination level in those small areas usually results in large light overkill in all other areas. It is questionable whether these limited, somewhat darker, areas do, in fact, present a safety hazard. The question merits study if only because an excessive overall light output compounds the next problem, that of glare.

2. Glare

If obtaining the required light level is usually not a problem, obtaining it without glare is. "Glare" is a difficult problem because it is very subjective; it does not affect all individuals to the same extent, and so far it has not been amenable to a simple measurement.

The regulations state only that "lighting fixtures shall be designed and installed to minimize discomfort glare" [75, 1719-2(g)]. Without a definite numerical value to achieve, early installations perhaps tended to somewhat ignore glare, concentrating only on achieving a 0.06 fL level. Later it was recognized that avoiding glare was at least as important as achieving the proper light level.

There is also an acclimation period which comes into play, in that one trains oneself to limit the effect of glare by not looking directly into a light source. This comes after a period of unconscious adaptation to the machine and its arrangement of luminaires.

Whether using HID or fluorescent lamps, the most effective way to eliminate or reduce the undesirable effects of glare is to locate the luminaires out of the field of view of the men working with the machine. In addition, solid or translucent shields can be placed between luminaire and operator, or tapes used over the lenses to reduce the brightness in the offensive direction.

Lamps of low surface brightness, such as fluorescent, obviously cause less glare problems than HID. However, even low-brightness, standard fluorescent lamps are objectionable when located too near a task to be done. An illustration of this is the luminaires mounted on top of roof bolters which shine in the face of the operator assembling bolts on the deck of the machines. Even when using fluorescent luminaires the positioning is very important, and shields should be used if necessary. This can be more difficult than with HID lamps because of the large size and/or number of luminaires.

The literature distinguishes between disability glare and discomfort glare. Disability glare is defined as that which causes actual damage to the eye, and it does not appear that this is the problem here. However, even in discomfort glare, such as when approaching a car with high beams on, there is an element of disability in that our ability to see in the darkness is temporarily impaired. The same applies when an HID source is in the field of view. The effect is much less evident with fluorescent lamps; however, it is still there. More research is needed in the area of eye response to the light level and light contrast prevalent at the face, in order to determine the conditions most conducive to workers' awareness of hazards.

Reaction of Personnel

Reaction of the men to illumination was found to vary from total rejection to wholehearted acceptance. Beyond this general statement it is very difficult to evaluate the reaction of personnel in any particular case. Some installations were found which were poorly designed or maintained, and rejection would be expected. However, it is also possible to have identical machines with identical luminaires with similar conditions and get opposite reactions from the operators.

As a help in judging the men's comments, several observations can be made:

1. Anything new, whether machine or procedure or organization, requires some adaptation period. Everyone needs some training with a new tool before he can make efficient use of it. Illumination at the face is something new, and as such, requires a training period.

2. The normal human response is to resist any change. Efforts should be made to build up a receptive attitude well in advance of introduction of illumination. Most personnel appreciate being "in" on projects affecting their tasks. Particularly when confronted with such a subjective matter as the effect of glare, which is not measurable, it is important to consider the reactions and suggestions of those directly affected.

3. The reason for a negative reaction should be carefully examined. It was observed several times that an operator's objection was not directed at the idea of illuminating the working place, but was to a specific luminaire particularly annoying to him.

4. Acceptance of illumination has to be tested both ways, i.e., what is the reaction of a crew which has worked with an illuminated machine for a period of time, and then has to return to a non-illuminated one, as well as vice-versa. Often the "old days" look good only until one has to return to them.

Conclusion

During these past few months we have been in contact with coal operators, lighting equipment suppliers, and government agencies, each group concerned with a different aspect of mine illumination. We have found many people among these groups who are very knowledgeable, have good engineering sense, are open to new viewpoints, and willing to listen to and learn from each other in spite of their different affiliations and sometimes conflicting interests.

From what we have seen we believe that increased illumination at the face is beneficial and possible. However, the technology is presently at the critical stage of expanding from a few demonstration units to widespread use of illumination; problems are bound to surface, whether in interpretation of the law, or reliability of the materials and designs used, or even in new hazards unwittingly introduced. We believe these problems can be solved if studied carefully and cooperatively by all concerned and no one adopts an "I know best" intransigent attitude. Our experience indicates there are capable people in all phases of the field of mine illumination who can reasonably resolve these problems if permitted to work together protected from outside, unqualified, self-serving pressures. For the sake of safety, let us all cooperate in that effort with an open mind.

Mine Illumination

Kenneth Klouse
Electrical Engineer

Mining Enforcement and Safety Administration
Electrical Testing Project, Office of Technical Support
Beckley, West Virginia

In the Federal Coal Mine Health and Safety Act of 1969, Congress empowered the Secretary of the Interior to develop standards that would require illumination of all working places in a mine. Only now, is there enough data, experimentation, and equipment available to promulgate a practical and enforceable regulation.

Mine illumination did not start just six years ago. Since 1938, numerous papers have been presented on illumination in underground coal mines in the United States. In a 1950 meeting of the Coal Mine Lighting Clinic, which included representatives of industry, the UMWA, and State and Federal agencies, one of the conclusions reached was that improving seeing conditions at the face was the most crucial problem. The Mining Development Committee of the Bituminous Coal Research, Inc., has been conducting tests since 1953 in underground illumination. Technical data is also available from some European countries where underground lighting has been compulsory since the 1940's. From these countries, the effect of lighting on safety has been demonstrated.

In a paper published in 1954 by A. Roberts, a lecturer in mining at the University of Nottingham, the correlation between accidents, visibility, and lighting is discussed. The accident investigation covered a 12-year span during which lighting became mandatory in certain areas of underground mines midway through the period. The death rate and serious injury rate significantly declined in the second half of the sample period in areas requiring lighting. In other words, the second six-year period showed a 27 percent lower death rate than the first six-year period and correspondingly, an 11 percent lower serious injury rate. Serious accidents because of haulage runaways showed a decrease of 40 percent between the two six-year periods. These numbers show a definite relationship between underground safety and underground lighting. Mr. A. Roberts stated that "while it is not possible to deduce any simple relationship between such accidents (mining accidents) and the lighting environment, yet there is evidence that in certain types of accidents, where visibility is an obviously important factor, the effects of improved lighting are reflected in falling accident trends. . ."

More difficult is the correlation between production and illumination. Installations in the United States tend to show an increase in production in longwall mining; however, the increase is not statistically significant. When more illumination systems are introduced underground and operational time is extended, definite trends in the industry will become apparent.

Mining produces unique problems that make production comparisons almost impossible. Even between adjacent sections of the same mine, conditions usually vary. The only way to develop statistics, when the variables cannot be held constant, is to acquire a large data base. The number of illumination systems underground do not have sufficient working time to supply us with this data base. However, in other industries good lighting has been shown to increase both safety and production. This should be applicable in a limited respect to the mining industry.

Compliance with the illumination regulations will not be easy to achieve. It will require a concentrated effort by the coal industry, manufacturers, and MESA. The Technical Support division of MESA is offering services to the coal industry so that compliance can be achieved. These services can be broken down into three areas.

The first and most important area is technical assistance. Technical assistance is available to coal mine operators and equipment manufacturers to enable them to comply with the illumination regulations. In the area of illumination, the Beckley Electrical Testing Project (BETP) will be available to assist in laying out illumination systems. This assistance will be for machines that cannot be handled by the normal test and evaluation procedure. This service is free but does not result in a Statement of Test and Evaluation.

The Statement of Test and Evaluation is the second area that service is offered to the coal industry. The statement is an evaluation of an illumination system that states that the system will be in compliance with the Federal regulations when used within certain operational parameters. It is not mandatory to obtain a Statement of Test and Evaluation; however, this service has many advantages:

1. Testing is accomplished in surface facilities which will not interfere with mining operations.

2. Operators can purchase illumination systems for equipment and be assured that the system will be in compliance when installed and operated according to the Statement of Test and Evaluation.

3. Systems can be installed at maintenance or rebuild shops which will assure a better installation in a shorter time.

4. Part of the Statement of Test and Evaluation is a one-line diagram approved by Pittsburgh Approval and Certification Center. This one-line diagram can be used to supply much of the information required for the field change. This will save a lot of paperwork by coal mine operators.

The Statement of Test and Evaluation is issued only through BETP; however, there are presently available ten laboratories where the actual work can be performed. These laboratories have been certified by BETP as capable of supplying data comparable to Technical Support's facilities.

There are five lighting equipment manufacturers' laboratories:

1. General Energy Development Corporation

2. Ocenco, Incorporated

3. McJunkin Corporation

4. National Mine Service

5. Bacharach Instrument Company

Three mining equipment manufacturers' laboratories:

1. Lee-Norse Company

2. Joy Manufacturing Company

3. FMC Corporation

Two coal company laboratories:

1. North American Coal Corporation

2. Westmoreland Coal Company

This is quite a list of laboratories available to service the industry in underground illumination and provide a means of achieving compliance. Already more than two hundred Statements of Test and Evaluation have been issued on a wide range of mining machines. These Statements can be used by anyone in the mining industry as long as the operational parameters are observed. Some of these parameters are:

1. Machine model, type, and optional equipment

2. Height and width constraints

3. Equipment function

A catalog of these Statements of Test and Evaluation is being assembled by Bituminous Coal Research, Incorporated (BCR) for the coal industry. This will be a center where a coal operator can contact BCR and obtain a listing of illumination systems that have been tested for his machines. All an operator will have to do is tell BCR the machine type or model and the maximum height and width the machine will be working. BCR can then send the coal operator a copy of all Statements of Test and Evaluation that cover his mining conditions.

The third area is in support functions. MESA is able to calibrate meters for coal mine health and safety inspections and the coal industry. This will assure that all meters used in illumination measurements underground give reliable results and are traceable to the National Bureau of Standards. Test equipment is also available to analyze individual luminaires and/or systems.

As I said earlier, compliance with the regulations will not be an easy task to achieve; but with the support services available to the coal industry from MESA, compliance is a realistic goal.

Coal mine operators are not required to use the services offered by MESA Technical Support for a Statement of Test and Evaluation. Operators can illuminate working places using the Illumination regulations, 30 CFR, Section 75.1719 through 75.1719-4, and compliance will be determined by MESA inspectors. The underground illumination regulations can be broken down into four main areas. They are: (1) areas to be illuminated; (2) power requirements to luminaires; (3) methods of determining compliance; and (4) procedures in taking light measurements.

Areas to be Illuminated

Illumination shall be provided in each working place of underground coal mines while self-propelled mining equipment is being operated in the working place. This illumination shall be in addition to that provided by personal cap lamps and will require all surfaces in the miner's normal field of vision to have a surface brightness of at least 0.06 foot-lamberts.

The surfaces in a miner's normal field of vision varies with the types of equipment being operated. When continuous miners and coal-loading equipment are operating, the face, ribs, roof, floor, and machine surfaces from the conveying equipment to the face, are required to be illuminated. For self-loading haulage equipment and cutting and drilling equipment, all exposed coal and machine surfaces from five feet outby the equipment to the face are required to be illuminated. Shortwall and longwall mining equipment require illumination from the gob side of the travelway to, and including, the block of coal being extracted for the entire length of the self-advancing roof support system. In addition, illumination shall be provided for the headpiece, tailpieces, and control stations and roof and floor, for a horizontal distance of five feet from these areas. Whenever roof-bolting equipment is operated, all surfaces to the sides and front of the machine must be illuminated when they fall within a horizontal distance, from the machine, equal to the mining height or five feet, whichever is greater. In addition, for roof-bolting equipment, all surfaces from the machine to a point five feet outby the machine must be illuminated. All other self-propelled equipment must provide sufficient illumination to illuminate a coal surface equal to the height and width of the machine and located 10 feet to the front and rear of the machine.

Power Requirements to Illuminaires

Luminaires used to illuminate the surfaces mentioned above must be permissibl whether they are mounted on the machine or stationary fixtures. Stationary lighting fixtures cannot be energized by conductors at alternating-current voltages greater than 70 volts to ground or direct-current voltages in excess of a nominal voltage of 300 volts. The cables containing these conductors for stationary lighting fixtures (other than intrinsically safe fixtures) are considered trailing cables and must meet the requirements of Subpart G of the Federal Coal Mine Health and Safety Act of 1969. They must be protected against overloads and short circuits. In addition, resistance grounded lighting circuits must be protected against ground faults. Alternating-current circuits, energized at 100 volts or more and used to supply power to stationary lighting fixtures, must originate at a transformer having a center or neutral tap grounded to earth through a proper resistor, designed to limit fault current to not more than 5 amperes. A grounding conductor from the grounded terminal of the grounding resistor shall extend along with the power conductors and serve as the grounding conductor for all equipment receiving power from the circuit. Grounding conductors are also required to electrically ground machine-mounted lighting fixtures.

Methods of Determining Compliance

MESA inspectors will determine if the surface brightness level of 0.06 footlamberts is present by using a Go/No Go 27^{o} photometer. The photometer will be held approximately five feet from the surface being measured. If the brightness of that surface is greater than 0.06 footlamberts, a green light will glow, and if the brightness is less than 0.06 footlamberts, a red light will glow. The surface area being measured will be approximately four square feet when the photometer is at a distance of five feet. When clearances do not permit a five-foot distance from a surface, one reading will be taken in each corner of a square, two feet by two feet. Each reading will not exceed 100 square inches, and if any one of the readings create a green indication on the photometer, that area will be considered in compliance. If the MESA inspector selects the darkest surface area, and that area is in compliance, then all of the brighter surface areas are in compliance.

Procedures in Taking Light Measurements

Readings will be taken with continuous miners, coal-loading equipment, cuttin equipment, drilling equipment, and self-loading haulage equipment, located in the center of the working place, not more than three feet from the face. The equipment will be idle while being measured. Measurements can be made on longwall and short-wall equipment while it is being operated, except when measurements are made in the vicinity of shearers, plows, or continuous miners. Roof-bolting equipment will be positioned so that face or rib surfaces are at a distance equal to the mining height or five feet, whichever is greater, from the front and one side of the machine. After measurements are taken, the equipment will be repositioned so the face and other rib surface is at a distance equal to the mining height, or five feet, whichever is greater. If the roof-bolting equipment can be positioned in the center of the working place, and the ribs fall within the mining height, or five feet from the sides of the machine, then rib measurements can be made without repositioning. The roof-bolting machine will be idle while being measured. All other equipment will be positioned at a distance not less than nine feet from a coal surface and will be idle while measurements are being taken.

Inspectors will not take measurements where shadows are cast by roof-control posts or ventilation equipment. When machine-mounted light fixtures are used, except on self-advancing roof-support systems, measurements will not be made within one foot of machine surfaces. The illumination regulations also specify that paint used on exterior surfaces of mining machines have a minimum reflectance of 30 percent, that reflectors be used on machines when stationary lighting systems are being used, and that hard hats have a minimum of six square inches of reflecting tape or paint on each side and back.

Summary

The Federal Coal Mine Health and Safety Act of 1969 reads, ". . . the Secretary shall propose the standards under which all working places in a mine shall be illuminated by permissible lighting, within 18 months after the promulgation of such standards, while persons are working in such places." The Act provides an 18-month period for compliance which ends on April 1, 1978.

THE BUREAU OF MINES RESEARCH PROGRAM ON MINE ILLUMINATION

George R. Bockosh
Electrical Engineer

U.S. Department of the Interior
Bureau of Mines
Pittsburgh Mining and Safety Research Center
Industrial Hazards and Communications
Pittsburgh, Pennsylvania

Abstract

The Federal Coal Mine Health and Safety Act of 1969 provided for the formulation of illumination standards for underground coal mines; such standards were promulgated on October 1, 1976. The Bureau's program of research on mine illumination, which has evolved supportive data for standards and the technology necessary for their implementation, is presently directed toward demonstrating cost effective and reliable means to provide underground illumination, providing the technological means by which these standards can be expeditiously realized, and insuring that the industry is aware of existing hardware and also possesses the expertise needed to apply it efficiently. The realization of a thoroughly informed industry and the existence of readily available, mineworthy hardware will enable the implementation of the standards to progress in a reasonable manner. The current work of the Bureau of Mines on mine illumination is herein described.

Introduction

Federal illumination standards for underground coal mines were promulgated on October 1, 1976, as parts 75.1719 and 75.1719-1 through 75.1719-4, Title 30, Code of Federal Regulations. The Bureau of Mines research efforts have recently been directed toward providing appropriate hardware and insuring that the industry is aware of the existence of the hardware and possesses the expertise needed to efficiently apply it. These objectives are manifest in continuing projects designed to demonstrate the feasibility and benefits of supplying additional illumination in even the most demanding mining environments. The means by which this is to be accomplished will be described through a discussion of the Bureau's ongoing activities in this area.

In-Mine Demonstrations of Illumination Hardware

In demonstrating the feasibility and benefits of providing additional illumination in the working places of underground coal mines, the Bureau of Mines has illuminated 38 machines and 4 longwall faces in a total of 22 mines. These installations provided guidelines by which most of the machines utilized by the industry could be illuminated. Representative demonstrations will be briefly discussed.

Two recent installations are in use at the Greenwich Collieries mining complex, near Cookport, Pa. The mines are operating in the lower Freeport seam, which is approximately 48 inches thick in this area. Both continuous and longwall mining produces approximately 1.7 million tons of coal per year.

Initial efforts illuminated a longwall face which consisted of 72 Joy* RB 4-legged chock supports and 35 Gullick Dobson 6-legged chock supports.

* Use of brand names is for identification purposes only and does not imply endorsement by the Bureau of Mines.

The 6-legged supports were located in the center of the face, providing improved roof control in this area. An Eickhoff EW170L shearer is utilized for coal cutting.

The illumination system includes one model 15/3 (21-watt) fluorescent luminaire mounted on each pan section, nineteen 14-amp power supplies, and one 120-vac power-distribution box manufactured by Ocenco Inc. The general location of the luminaires is shown in Figure 1. The power supply and all cabling were also located on the panline. This arrangement provided good protection for all components and eliminated relative movement complications associated with support-mounted hardware.

Analysis of performance over a 3-month period showed that the component reliability of the illumination system was excellent. However, tight clearances between the 6-legged supports and the panline demanded the relocation of the illumination hardware in this area. In addition, severe roof control problems forced Greenwich to replace the complement of 4-legged supports with the Gullick Dobson 6-legged supports.

In view of the past clearance difficulties experienced on the section of the face supported by the 6-legged chocks and the better equipment protection afforded by these units, it was decided to redesign the illumination system. The final design places all illumination hardware on the supports. Figure 2 illustrates the new mounting location of the luminaires. System installation will be completed during March 1977 and its performance will be monitored.

A continuous miner section consisting of a Lee-Norse 265HH miner and a Fletcher LTDO roof bolter was also illuminated at the Greenwich Collieries. This installation requires special consideration because of the low clearance generated by seam conditions and the use of rails for roof support. In addition, the rails were slid along the top of the miner during roof support installation. These restrictions required that the top of the miner remain as unobstructed as possible.

The machine was illuminated to meet the requirements of the Federal Coal Mine Illumination regulations, utilizing seven VHO, 55-watt Mine Safety Appliances fluorescent luminaires and three 100-watt Control Products mercury vapor headlights. As shown in Figure 3, the only hardware protruding above the machine main frame is one fluorescent luminaire, located beside the operator, and one headlight, located opposite the operator. All hardware is well protected from external forces.

The performance of this illumination system was monitored for 3 months, during which time there was no hardware failure.

The Fletcher LTDO roof bolter utilized in conjunction with the 265HH miner was also equipped with a similar illumination system. The layout of this system is illustrated in Figure 4. A noteworthy innovation incorporated in this design is the hinged mounting of the fluorescent luminaire located over the boom of the machine. This allows the unit to remain in a proper position for all seam heights encountered in this mine.

Continuing and future efforts in this area will be directed toward the more difficult mine illumination problems. Current work includes the designing of systems for use on diesel and battery-powered LHD's, slope and shaft sinking operations, and 600-vdc machines. In addition, newly developed illumination systems will be evaluated with both reliability and efficiency being noted.

Factory Integration of Illumination Hardware Into Mining Machinery

The continuing evaluation of mine illumination systems has shown that the major cause of illumination system downtime is physical damage due to machine collisions with coal surfaces or other machines. The protection of this additional hardware requires the good utilization of machine architecture, as shown in Figure 5, for only the massive machine main frames can withstand these extraordinarily large contact forces. Although existing machines have been successfully modified on the section or during a major overhaul, the most efficient time to build in an illumination system is during the early design of the machine, at the factory.

Present Bureau efforts have resulted in the integration of illumination systems into a Galis 300 roof bolter and a Long-Airdox coal drill. These machines have been redesigned to accept an illumination system and the manufacturers plan to offer the illumination system as an option in the near future. Work has also begun to modify a Marietta drum miner.

The availability of new, illuminated mining machines will provide the industry with excellently protected illumination systems, which will drastically reduce the maintenance costs associated with this additional hardware.

Information Dissemination and Training

The required introduction of additional illumination in the coal mines has found much of the industry lacking the expertise necessary to implement effective programs, thus forcing the operators to rely entirely on externally designed illumination systems that do not always fulfill the specific needs of their mines.

This endeavor, which has been continuing since the conception of the mine illumination program, has taken on increased importance and activity because of renewed interest within the industry sparked by the recent promulgation of regulations. In reaction to the industry's need, the Bureau has initiated a two-part program designed to inform the industry.

A program has begun to formulate a one-day seminar which will present the history of mine illumination, an explanation of the new regulations and photometry, design procedures for laying out an illumination system to meet the requirements of the regulations, and other pertinent data. The program is to be taken into the mine fields and presented throughout the country during approximately 40 seminars. The seminars are directed toward the people at the mine level who will be "living" with the lights and, as such, are designed to minimize the time such personnel would need to spend away from the job. A second meeting or workshop will be considered if interest warrants.

In addition, the Mining Enforcement and Safety Administration (MESA) has initiated a means by which mine operators and equipment manufacturers can obtain a Statement of Test and Evaluation (STE) for any given mine illumination system. The STE is issued by MESA and enables the operator to forego any inmine measurement of luminous intensities by local MESA inspectors while the illumination system is in adequate repair and the machine is being operated within the specifications stated in the STE. Although anyone can apply for an STE, the majority of applications are presently being submitted by manufacturers of mine lighting equipment and each manufacturer is presently completing a set of STE's. These files are not cross-referenced, making it very difficult for a mine operator to find existing STE's that apply to his particular mining conditions. The time expended in these endeavors could seriously affect the speed by which the mining sections will be illuminated. The Bureau has initiated the formation of an STE "library" through which an interested party could obtain a list of all tested illumination systems for a given machine.

The realization of this program will enable the industry to quickly determine what systems are available and to efficiently evaluate their applicability to specific conditions.

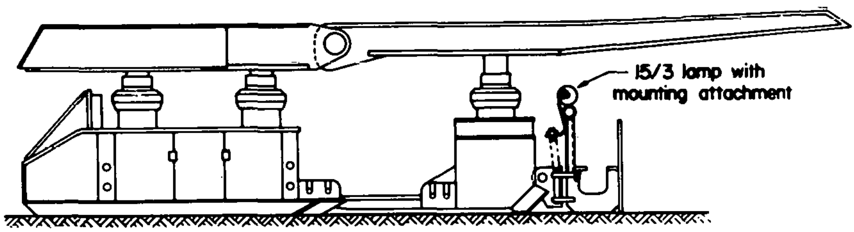
Computer Evaluation of Mine Illumination Systems

The method of test and evaluation of illumination systems that is presently being used by MESA's Technical Support Electrical Testing Project located at Beckley, West Virginia, utilizes a simulated mine entry and requires the construction or alteration of a "mock-up" mining machine for each type of equipment to be evaluated. Due to the multitude of machine and luminaire types and combinations, this present technique becomes a time-consuming process affecting industry as well as Government.

The Bureau of Mines is developing a computerized system by which mining machines and luminaires can be mathematically modeled and displayed on a TV-type monitor. A typical display is shown in Figure 6. The computer evaluation of illumination systems would enable anyone with access to a computer terminal and telephone to quickly, inexpensively, and accurately determine the luminous intensity levels associated with any hardware configuration. Thus, in addition to reducing evaluation costs, the computer simulation will drastically reduce system design time.

Conclusions

The Federal Coal Mine Health and Safety Act of 1969 provided for the formulation of illumination standards for underground coal mines. The Bureau's program of research on mine illumination, which has evolved data for standards and the technology necessary for their implementation, is presently pursuing an overall major objective of demonstrating cost effective and reliable means to provide the illumination required by the newly promulgated underground standards and to provide technological means by which these standards can be expeditiously implemented.



6 post chock luminaire arrangement

Figure 1. A cross section of the longwall system in use at the Greenwich Collieries, showing a 6-legged support and the general location of the luminaire.

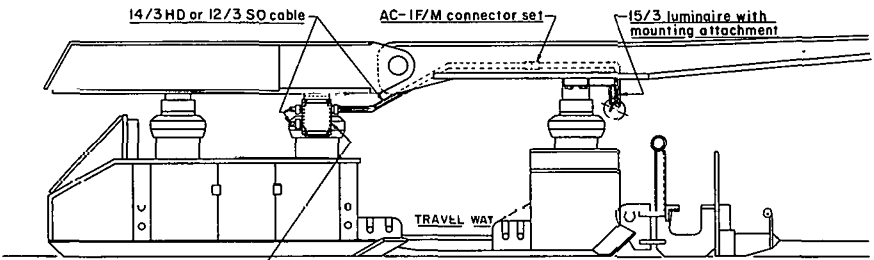


Figure 2. A cross section of the Greenwich Collieries' longwall illustrating the location of luminaire in the revised illumination system design.

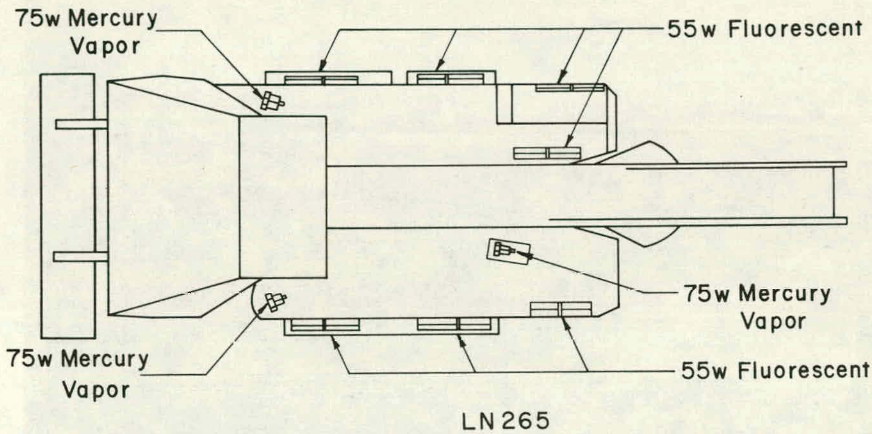


Figure 3. Layout drawing of a Greenwich Collieries 265HH miner, equipped with an illumination system which is well adapted to the low coal seam in which it is operated.

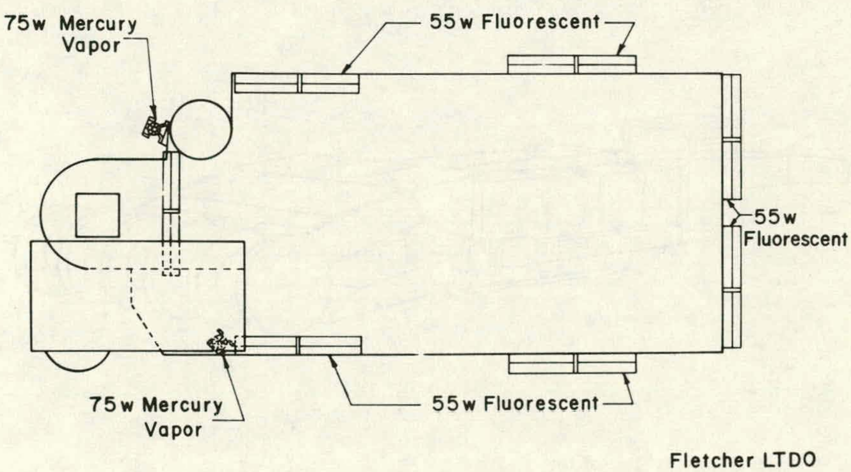


Figure 4. General layout of an illumination system for a Fletcher LTDO roof bolter.

Figure 5. A well protected luminaire, mounted on a Long-Airdox coal drill. Machine architecture is utilized here to protect the unit from external forces.

Figure 6. A computer generated line drawing of a continuous miner.

FACTORY INTEGRATION OF AN ILLUMINATION SYSTEM INTO A CONTINUOUS MINING MACHINE

Stephen W. Corbitt
Project Engineer - Drum Miners

National Mine Service Company

Introduction

With the approaching deadline date on lighting requirements for underground mining by the federal government, there has been some activity and concern by the industry, as to obtaining the lighting goal on their equipment. National Mine Service Company has been involved in an extensive test and design program to comply with the conditions of U. S. Bureau of Mines Contract Number H0366066 "Factory Integration of an Illumination System into a Continuous Miner." This symposium is one method to share the progress and problems encountered in this project.

The contract, as the title implies, is basically aimed at light systems installed at the original equipment manufacturer of new machines, notat the add-on systems to equipment in the field. However, much of the information to be presented could be of use to those attempting to add lights to existing equipment.

Requirement

The federal requirements are known to most people involved in underground mining; however, as a refresher and in case someone may not be acquainted with the required amount of light, areas to be illuminated, and methods of measuring the light, we will briefly mention these pertinent items.

The proposed standards state in part that the minimum luminous intensity shall be 0.06 ft-L required within the visual field of miners on rib, floor exposed equipment, and work areas where self-propelled equipment is employed during cutting, loading, or mining of coal up to and including the outby end of such machines.

There are two basic techniques by which one can measure illumination—either by incident light or by reflected light. For incident light, the unit of measure is the foot-candle (ft-c); for reflected light, the foot-lambert (ft-L). The relationship between the two is:

Reflected light (ft-L) = incident (ft-c) x reflectivity (%)

Although either method is usable for the problem under consideration, the Bureau finds that the reflected light measurements are preferable for three reasons: They can be conducted remotely from the specific surface that is being measured; they take into consideration the actual reflectivity of the surfaces; and from a visual viewpoint, one sees by reflected light, not by incident light.

However, instruments to measure reflected light are not common and tricky to use. National Mine Service Company elected to use an incident light meter, the Gossen Panlux Electronic Foot-candle Meter, and conducted our test as per the criteria on page 14110 of the 1 April 1976 <u>Federal Register</u>, which reads as follows:

"CRITERIA FOR TESTING AND EVALUATION OF ILLUMINATION SYSTEMS IN A SIMULATED WORKING PLACE AND IN WORKING PLACES OF UNDERGROUND COAL MINES USING INCIDENT PHOTOMETERS

A. *General.* A simulated working place shall be constructed in such a manner so as to exclude all exterior light, and the surface of the interior ceiling, walls, and floor of a simulated working place shall have reflectance value of not greater than five percent. The ceiling and sides may be constructed so that they are adjustable.

B. *Testing and Evaluation of Machine-Mounted or Stationary Illumination Systems using Incident Light Measurements.*

1. The illumination system shall be installed in the simulated working place or in a working place of an underground coal mine in accordance with the drawings and specifications submitted in the application for testing and evaluation.

2. The illumination system shall be energized and allowed a warm-up period.

3. Incident light for each 4-square-foot field shall be considered as the average of four uniformly spaced measurements taken at the corners of the field.

4. Measurements will be taken for all areas to be illuminated, using an incident photometer. Foot-candle averages for each field shall be adjusted by the following formula:

Average foot-candle measurement for a field
x Light loss factor (0.77) x Reflectance of underground
surface (0.02) 0.00 f-L

5. The luminous intensity of all surfaces required to be illuminated by 30 CFR 75.1719-1 shall be at least 0.06 foot-lamberts when adjusted by the formula in B.4 of these criteria."

Using this formula, the minimum average foot-candle measurement is 3.89 foot-candles. One requirement as yet not mentioned is the date of compliance, which is 1 April 1978.

In addition to these regulations, the following requirements were included as conditions in the USBM Contract H0366066:

1. It must comply with all applicable MESA illumination and permissibility regulations.

2. All illumination hardware shall be incorporated into the machine mainframe such that it cannot be damaged by usually expected external forces caused by machine collisions and minor roof falls.

3. Any necessary machine dimensional increase shall be minimized with possibly no dimension increase.

4. The illumination hardware shall be reasonably accessible for maintenance activities.

Light Tests

The single most important data required in order to engineer an effective illumination system into any machine is the light envelope (foot-candles vs. distance chart) of the light hardware to be used. Of the four light suppliers contacted, only National Mine Service had any light envelopes available and these were only completed on the 38-watt and 65-watt fluorescent lights. It is surprising that such basic information is so lacking in the mining industry.

Since this information is so essential, we ran a series of tests on all of our lights including the 150-watt incandescent headlight, the 75-watt mercury-vapor headlight, the 65-watt fluorescent, and the 70-watt HID sodium-vapor lights. Scaled isointensity diagrams of all these lights are presented on the following pages. Each of these diagrams shows the 4 foot-candle envelope produced by the various lights.

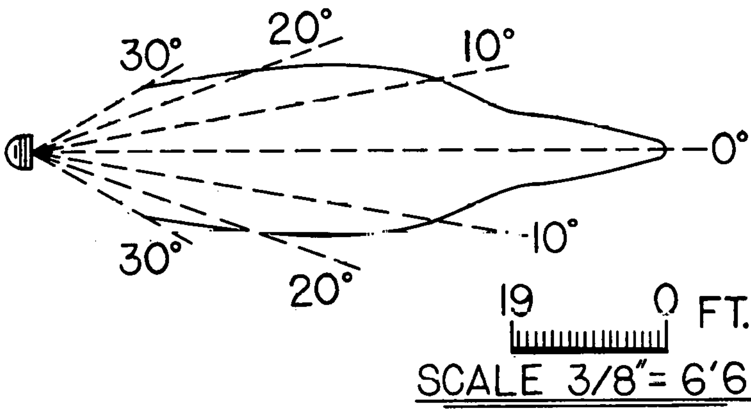
These light envelopes were very useful in trying to develop a light package on paper. However, the most interesting side effect of these tests was the rejection of the mercury-vapor headlight from consideration on our light system.

Originally, on the lighting plan, mercury-vapor headlights were to be used due to the assumed low light level of incandescent headlights, which seems to be their reputation throughout the industry. However, our test showed that the 150-watt incandescent headlight produced very good light. In fact, after the light test, it was decided to replace all planned mercury-vapor headlights with the incandescent. Not only did the incandescent produce satisfactory light, but it also reduced the space requirements in the ballast box since the mercury-vapor requires a ballast almost as big as the light itself. The incandescent headlight requires no ballast and therefore not only requires less space but also is the cheaper of the two lights. One more reason to abandon the mercury vapor light is its re-arc time, if the light is turned off or drops out due to low voltage. A mercury-vapor light that is hot will require three to five minutes to re-start whereas an incandescent re-arcs immediately and will not drop out due to low voltage.

In order to be able to perform light tests not only on individual light, but also on mock-up of entire machine lighting systems, NMSC constructed a darkroom at their Greenup, Kentucky plant in compliance with the criteria as published in the 1 April 76 Federal Register. The darkroom is 12 feet high, 20 feet wide and 36 feet long with adjustable walls and roof. The interior is painted flat black with a reflectivity of less than 5%. Also, the interior is divided into areas of 2 feet x 2 feet with a mark on the inside surfaces to assist in taking the required measurements. The room was expensive, costing almost $6,000 to construct, but it has proved to be an absolute necessity in order to engineer lighting systems on equipment mock-up.

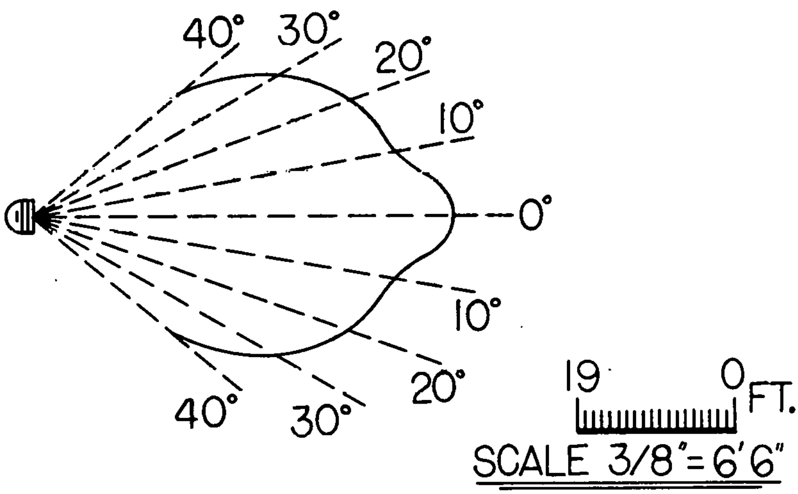
Another necessary item required to complete the light test is either the actual equipment being lighted, or as in this case, a mock-up of the machine. Again, the wooden mock-up was expensive, but valuable in testing. The NMSC mock-up was adjustable to match any configuration of miner manufactured by the Company to include at least one miner that is only in the early design stages. This mock-up provided the platform on which to mount the actual light in position, so that the overall system could be checked for light output.

4 FOOT-CANDLES

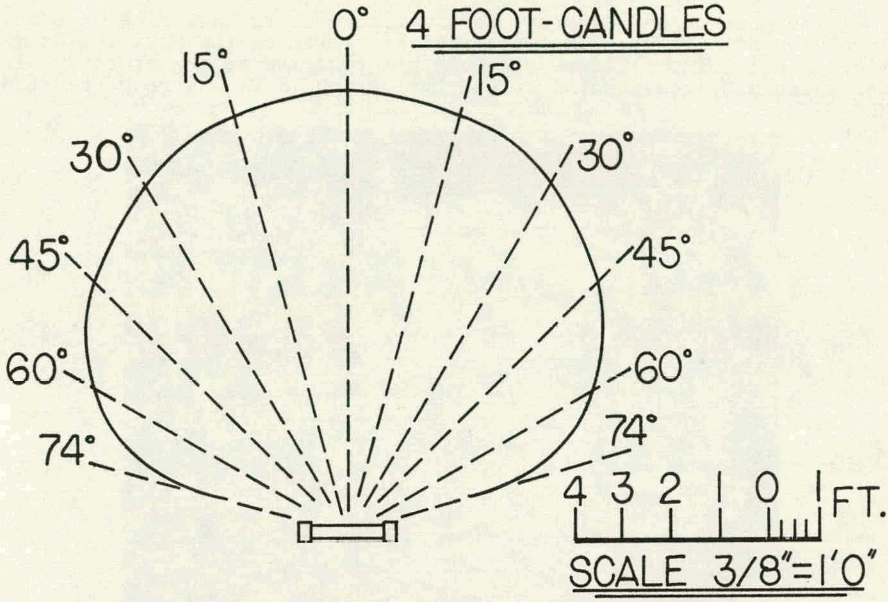


ISOINTENSITY DIAGRAM FOR 150 WATT INCANDESCENT HEADLIGHT

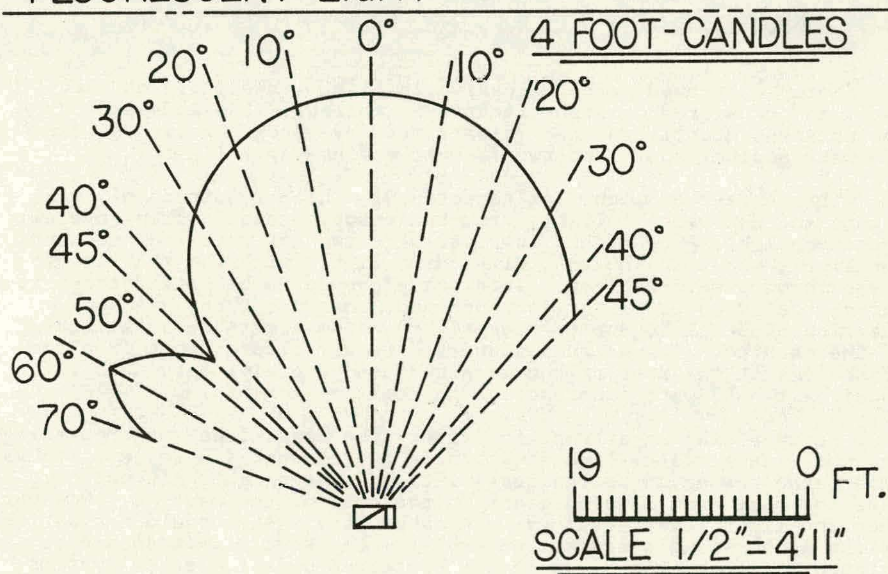
4 FOOT-CANDLES



ISOINTENSITY DIAGRAM FOR 75 WATT MERCURY-VAPOR HEADLIGHT

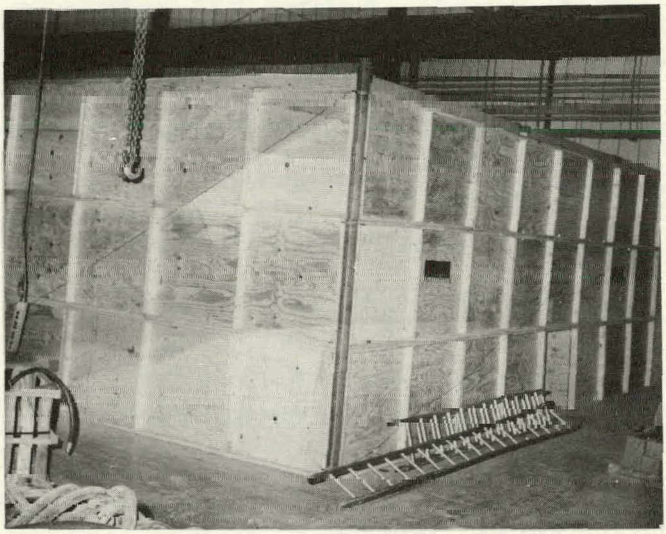


ISOINTENSITY DIAGRAM FOR 65 WATT
FLUORESCENT LIGHT



ISOINTENSITY DIAGRAM FOR 100 WATT
H.I.D. SODIUM-VAPOR LIGHT

The mock-up and darkroom were utilized to run a series of tests involving six different setups of light system on the Marietta Drum Miner Model 5012S. The results of this test led to the prototype 1 for which a Statement of Test and Evaluation (STE) was requested from MESA.

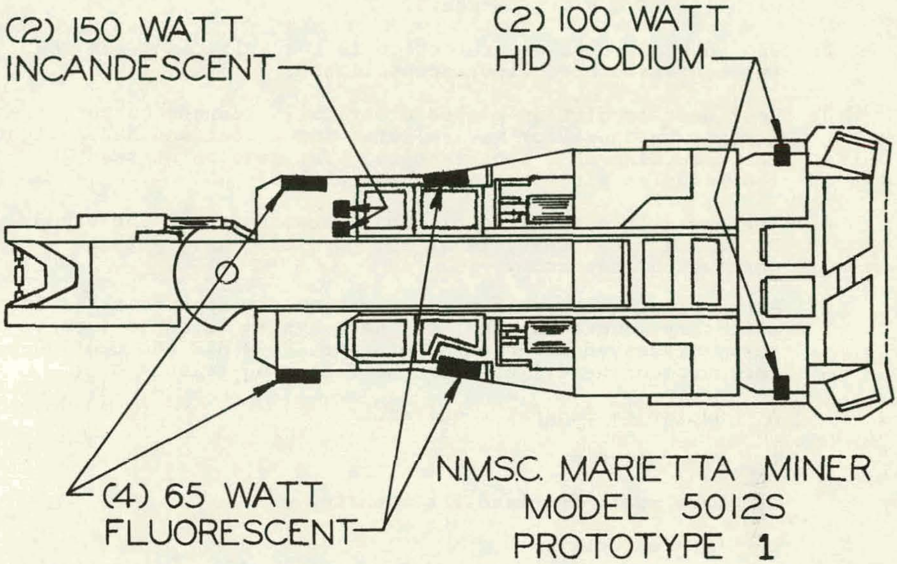


NMSC DARK ROOM GREENUP, KENTUCKY

Setup #1 proved to be excessive in light, especially on the face, and consisted of eight lights, as follows: two 150-watt incandescent headlights, two 75-watt mercury-vapor headlights, two 100-watt sodium-vapor, and two 70-watt sodium-vapor lights.

This excess of light led to setup #2, which required only six lights and removed all lights from the canopy, thus lowering overall machine height. The lights involved were two 150-watt incandescent headlights and four 100-watt HID sodium lights with the rear lights moved forward several feet. This setup proved to be satisfactory as far as the amount of light is concerned; however, there was some question about glare due to the bright sodium lights near the rear of the machine. It was then suggested to put fluorescents in place of sodiums at the rear of the machine, and generally agreed that it requires two 65-watt fluorescents to replace each sodium.

Setup #3 was an attempt to satisfy the conditions with replacing the two (rear) sodium lights with only two 65-watt fluorescent lights This setup seemed to be the best—with adequate light, minimum hardware and cost, and reduced glare on the rear of the machine. However, when the light level readings were taken, the system could not satisfy the 4-foot-candle average on every 2-foot square. It should be pointed out that this setup #3 was preferred over the other systems due to the reduced number of lights and the fact that the light output was satisfactory to the observer. Actually, the 4-foot-candle average was satisfied in more than 92% of the readings with the remaining, mostly rear top, falling into the 3- to 4-foot-candle range.



Setup #4 was a suggestion of the Pittsburgh Production Shop, the light manufacturer, which involved two 100-watt sodium, and four 65-watt fluorescents. This system did satisfy the light condition; however, the sodium position was extremely open to damage due to its 2 1/2 inch overhang. At the approximate point, the machine tends to rib when making a 90° turn.

Setup #5 was an attempt to use the smaller 38-watt fluorescent instead of the 65-watt fluorescent to light the machine. These smaller lights would not provide the proper amount of light at the required distance in order to be utilized on this machine.

Setup #6, which is prototype 1, is the system that has been sent to MESA for approval. It utilizes eight lights—two 150-watt incandescent headlights, four 65-watt fluorescents, and two 100-watt HID sodium.

Prototype 1

On 31 May 1977 a Marietta Drum Miner S/N 7683 (Prototype 1) a machine with the approximate proposed lighting system was delivered to a customer. This machine was not intended to be the contract machine and, in fact, we were not really ready to manufacture an integrated light system, but due to customer insistance and our own sales department, a system similar to a factory integrated system was installed.

This machine was inspected in the mine, after two weeks of operation. This inspection resulted in the following facts and recommendations about this lighting system:

1. The lights should be attached with at least 1/2" capscrew. The necessity of damper or "soft mount" has not yet been determined; therefore, we will bolt these lights up solid with 1/2-13 H.H.C.S. Grade 5.

2. The rockfall type of protection is not sufficient for the two center-mounted fluorescent lights.

3. Coal dust or dirt on globes drastically reduces output, i.e., a point on the floor was selected for a test and had a light meter reading of 6 foot-candles. The same point read 26 foot-candles after the globe was cleaned.

4. The operator complained of the light mounted on the canopy. It is suggested that, if at all possible, no lights be mounted on the canopy.

5. After viewing the system in operation, it appears that, for a factory-mounted system, the best system would be a previously-discussed arrangement of four 100-W HID and two 150-W incandescent headlights for the following reasons:

   A. Satisfies federal requirement

   B. Fewer lights

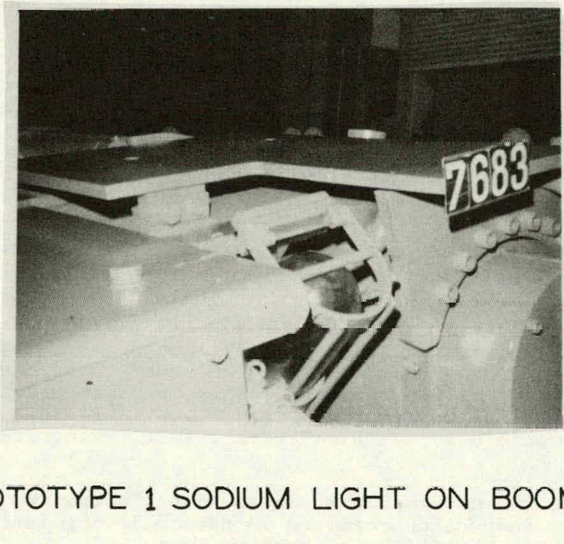
   C. Less space required for mounting

   D. Less expensive

   E. Less than 40% globe surface area to keep clean

   F. Glare did not appear excessive on test machine

   G. Remove objectionable light from canopy



PROTOTYPE 1 SODIUM LIGHT ON BOOM

A second trip was taken to investigate complaints about the light system installed on the 5012S miner, S/N 7683, as mentioned in the previous report. This trip uncovered some interesting aspects of underground lighting and some drawback or falacies of the proposed regulation, as published in the 1 April 1976 Federal Register.

The requirment for .06 lumens reflected light or the MESA acceptance of 4 foot-candles incandescent light on all surfaces (roof, face, ribs and floor) around the machine is not the best method to judge the effectiveness of a lighting system. The only factor that this proposed regulation has in its favor is that it is simple to understand and evidently easy to write, once the level of light was selected. But it does not really consider the actual use of the system by the operators, and the fact that light levels coming from a point source are inversely proportional to the square of the distance from that source, therefore requiring extra or overly bright light in order to comply with the regulation.

As an example, refer to the light system known as setup #3 in this report. There are 455 areas of 4 square feet each, that by the regulation must average 4 foot-candles each. The light measurements show that some of these areas did not average 4 foot-candles, therefore this system would be unacceptable. However, there are only 32 areas or 7% unacceptable and they averaged over 3 foot-candles. Because of this 7% with a 25% low light level or actually an overall low light level of less than 2%, two more lights have to be added to the system.

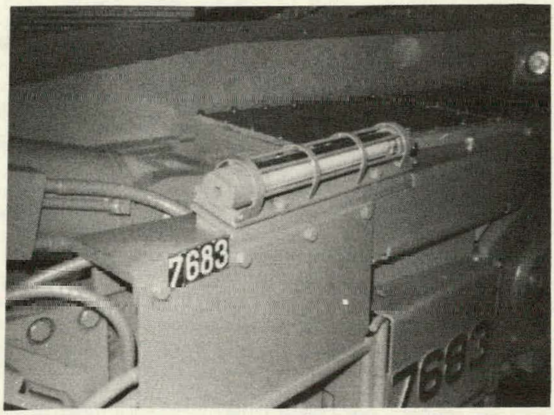
Another interesting lighting problem was discovered when the miner operator complained about not being able to see the face on a machine that we had tested and know to be in compliance with the four foot-candle regulation. The problem was that although the face may show 4 foot-candles, the area 3 feet back from the face was reading 40 to 50 foot-candles, which is evidently too much contrast. This was compounded by dust and fog-type water sprays which effectively concealed the face behind a brightly-lit cloud.

The corrective measures taken to satisfy the customer on prototype 1 were to (1) adjust reflector on the 65-watt fluorescent on the canopy to keep light out of operator's eyes, and (2) move the 100-W sodium vapor back and upon the machine and add reflectors to put the light on the face.

The second change made the system an add-on type, as opposed to a factory integrated system. Any existing machine can be fitted with an "add-on" system to satisfy the lighting requirement, but their main problem is usually the susceptability to damage due to being placed on top or sides of the machine in such a manner that they cannot be protected.

These examples point out some of the problems with the current requirements. The miner wants light on the face to see what he is cutting and he is not concerned about light in other areas, and in fact is usually opposed to lights on the rear of the machine (i.e., canopy) that may irritate him or his shuttle car operator. Why should the rear roof be required to have the same light level as the face? Why should extra lights, increasing system cost by 20%-30%, be added for a 2% overall low light level, as in "setup B?"

Considering these factors, I feel that a much improved system on the MESA requirements would be as shown on the figure titled "Suggested Light Pattern." This type of pattern would put the light where the miner wants it and give the manufacturer some option in light mounting on the machine's rear.



PROTOTYPE 1 FLUORESCENT 65 WATT

PROTOTYPE 1 FLUORESCENT & INCANDESCENT

PROTOTYPE 1 SODIUM LIGHT ON BOOM

At a minimum, there should be some provision for a percentage of low level light to allow more flexibility in the light location. A percentage of 5% to 10% does not seem unreasonable for a lighting system. This would not only solve the problem as stated in "setup 3" but also make dark spots or shadings caused by heavier guards acceptable.

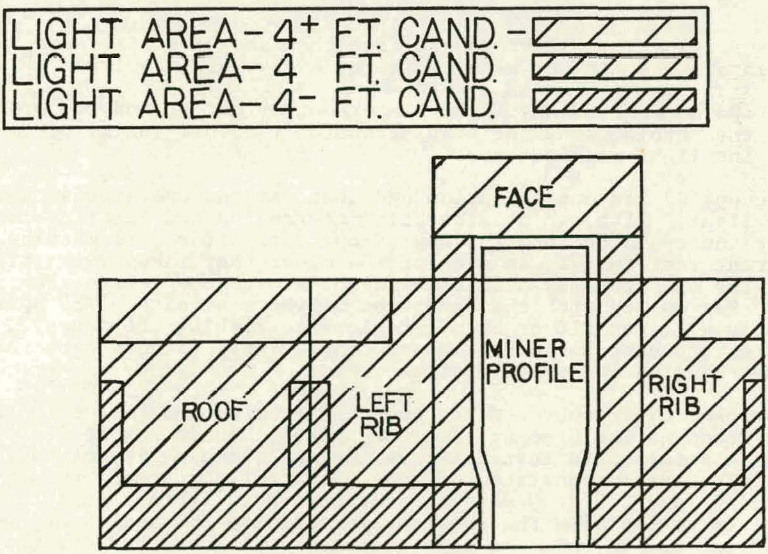
All of the NMSC light work up to this point has been aimed at our 5012S Drum Miner for which we have requested a Statement of Test and Evaluation from MESA. However, all of our 5012S Drum Miner orders are either going to South Africa, or being put into service as rock machines.

In April of '77, we introduced and delivered our first 5012N Drum Miner for which we now have a much greater sales order volume than for the 5012S. (The 5012N, or narrow, has an 8'-6" fixed cutter head, while the 5012S, or standard, has a 10' to 11' retractable cutter head). Due to this shift in customer orders, the 5012N is the logical choice for the Continuous Miner to be equipped with a factory integrated illumination system.

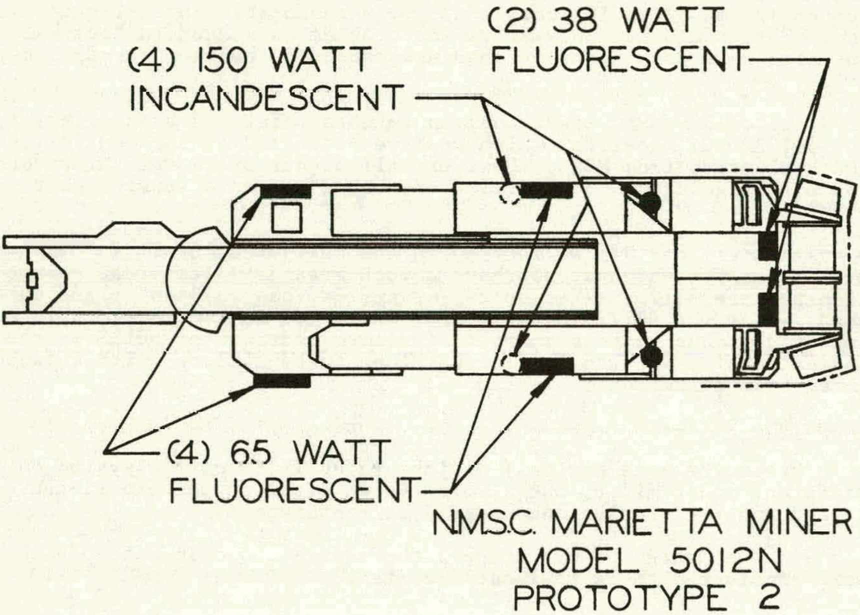
Prototype 2

Prototype 2 is not only an integrated illumination system on a different model miner, but also a system based on the experience gained and the problem encountered on Prototype 1.

In the design of Prototype 2 there were two main concepts that were considered to be the most important:



SUGGESTED LIGHT PATTERN



1. Provide more light to the face than anywhere else, but maintain the mininum required light everywhere required.

2. Do not extend any light fixture up or out beyond the profile of the machine, and use heavier guards than originally provided by the light manufacturer.

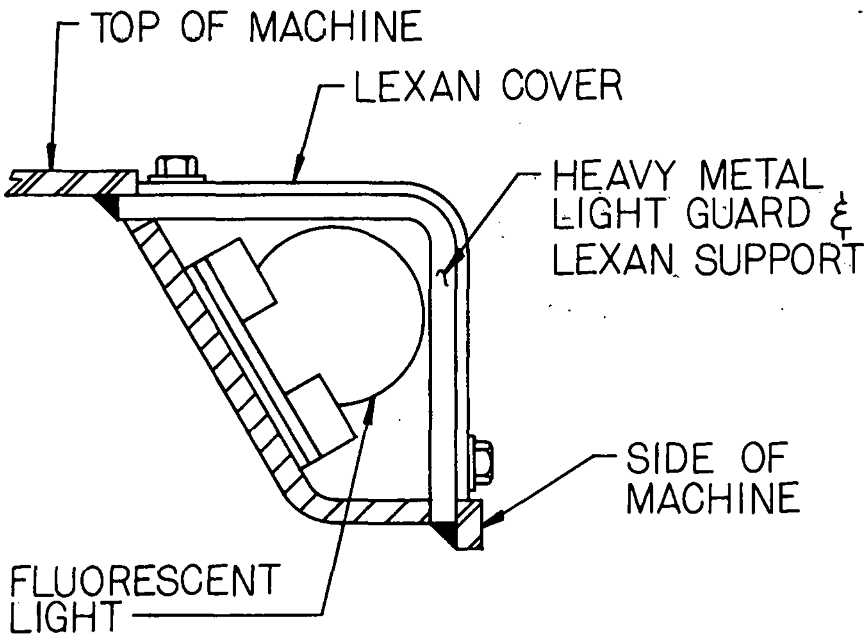
Concept #2 had one exception and that was the operator's canopy-mounted light. Although considerable integration and light protection are incorporated into the new canopy design for this machine, it is apparent that this is an area of the miner that takes very little abuse. The operator may handle the machine with complete disregard for any area on it, with the exception of where he sits. The operator simply will not rib or even get close to ribbing the canopy; therefore, the canopy can be widened considerably without concern about overhanging the profile of the miner.

A mockup of Prototype 2 has been constructed and tested in the NMSC darkroom at the Greenup, Kentucky plant. Again, as in Prototype 1, several setups were tested before the one shown as Prototype 2 was chosen. The system consisted of four 65-watt fluorescent, two 38-watt fluorescent, and four 150-watt incandescent headlights. An attempt was made to comply with the previously mentioned "Suggested Light Pattern" and due to this the face averaged 8.72 foot-candles, the ribs averaged 6.52 foot-candles and the roof averaged 4.87 foot-candles. This system should be more satisfactory to the miner operators than one that averages 4 foot-candles overall, since more light is on the face where he wants it, and the remaining areas comply with federal requirements.

Another attractive area of this machine is that all light fixtures, with the exception of the canopy, are mounted within the profile of the machine. Experience shows that light fixtures are crushed by falling rocks and sheared off by rocks sliding down the boom or off of the conveyor. The intent of the design for the light fixture, as shown in the figure titled "Typical Cross Section - Fluorescent Lamp" is an attempt to remove these lamps from the crushing or shearing problems that would be common in the add-on system. By being inside the miner profile and covered with a layer of tough Lexan supported by heavy metal guards, the fixtures are completely protected from the rock hazards. Also, this type of setup allows easy cleaning of the outside layer of Lexan with no outside cover to remove. The problem of dirty globes with the associated light loss was mentioned earlier in this paper. Another idea which may be tried in Prototype 2 is the possibility of providing a spray from the dust suppression system to be directed on the outside Lexan to keep it clean. It seems that the only clean areas on the miners are the areas around the water sprays which have continuous-running water on them.

Prototype 2 has not yet been manufactured or field tested. Only the mockup work and darkroom light test have been completed. It is possible that some of the field testing will be completed by the oral presentation of this report.

One other item that is important to everyone is cost. A light system for any machine will not be cheap. Adding eight or ten lights, ballasts and ballasts box, cable, transformers, switches and accessories, all of which are explosion-proof, cost money. Some of the initial estimates are in the \$3,000-\$3,500 range. Of course, this is only about 1% of the cost of a continuous miner, and in that respect the system does not seem overly expensive.



CONCLUSIONS

The experience gained so far through NMSC work on illumination, for the most part, centers around light levels both required and desired. All light systems will have to comply to the MESA requirements, but for better customer acceptance, more light should be concentrated on the face or, in the case of other underground mining equipment, the area being worked. Also, the overall illumination requirement could be reviewed as to the possibility of reducing the required light level on the roof and rear ribs. In this same area, an allowable percentage of low level areas could considerably reduce design and cost of lighting systems.

Second, only in importance to light level, is light fixture pro tection. The guards provided by light manufacturers are in general not heavy enough for continuous miner applications and basically any fixture that extends beyond the profile of the machine is in danger of either being crushed or sheared-off by falling or sliding rocks.

Another conclusion was the advantages of incandescent headlights over the mercury-vapor type to the point of dropping the mercury-vapor from consideration on our lighting system. This is because of the extra cost and space requirements of the ballast needed for the mercury-vapors, and the surprisingly satisfactory light envelope of the incandescent.

One final conclusion is the fact that the lights will have to be kept clean in order to stay in compliance with the requirements.

USBM-SPONSORED MINE ILLUMINATION PROJECT AT BITUMINOUS COAL RESEARCH, INC.

Kenneth L. Whitehead
Supervising Engineer

Bituminous Coal Research, Inc.
Monroeville, Pennsylvania

The illumination program at Bituminous Coal Research, Inc., has included a BCR-sponsored project, discussed in an earlier session by Dr. Felix du Breuil, and second one funded by the U.S. Bureau of Mines. The Bureau project is designed to ovide a means by which industry personnel may obtain information to assist them specifying and maintaining illumination systems. This project includes three tasks: (1) the consolidation, maintenance, and distribution of MESA-evaluated mine illumination specifications; (2) the development and presentation of illumination training sessions; and (3) the determination of installation costs associated with mine lighting systems.

The initial task involves the establishment of a file of Statements of Test and Evaluation (STE) issued by MESA to lighting system manufacturers, mining equipment manufacturers, and coal mine operators. These STE's are obtained by having lighting systems tested under actual or simulated mine conditions on a specific type of machine. If the light levels produced are in compliance with the 0.06 footlambert requirements of the regulations, an STE is issued to approve use of the system on the type of machine and under the conditions specified in the STE without requiring an inspector to take underground light measurements. Copies of all STE's are being supplied to BCR by the manufacturers and operators or by MESA, with approval by the applicant.

This file has been established; and the individual entries normally include the STE number, the machine specifications, the dimensions of the working place, a listing of the lighting hardware used, a diagram showing luminaire location on the machine, and a line diagram of the lighting system power circuit. Mine operators can request copies of this information by writing to BITUMINOUS COAL RESEARCH, INC., Attention: STE File, 350 Hochberg Road, Monroeville, Pennsylvania 15146, or telephoning 412/327-1600. Requests should include information on the type of machine to be equipped and description of the working place. It is intended that this file will provide the operators with easy access to information on all approved lighting systems available for specific machines to facilitate the selection of an appropriate lighting system.

The second task involves the preparation and presentation of 40 to 50 one-day illumination seminars in all coal-producing regions. These seminars will cover:

1. A brief history of the development of mine illumination hardware and results of in-mine experience to date.

2. An explanation of the requirements of the Federal Mine Illumination gulations.

3. An explanation of relevant photometric terms and practices.

4. Explanation of the procedure by which isointensity diagrams may be utilized to design an illumination system which will meet the requirements of the law.

5. A summary and comparison of available mine illumination hardware.

Each attendee at the seminars will receive a manual that will include more detailed information on the material covered by the oral presentation.

In addition to the formal presentation, the attendees will be encouraged to participate in an informal discussion of their illumination experience to bring out problems they have encountered and solutions that have been tried. Any pertinent information developed during these discussions will be incorporated into the seminars.

These seminars will provide operators with background information on the fundamentals of illumination and on the type of lighting systems that are available. This information should provide a basis for operators to make a more judicious choice of lighting systems for their equipment.

The final phase requires BCR personnel to collect lighting system installation cost data. This is being done in two ways; first, by consulting the records of mines where the installation procedure was well documented, and, second, by actually monitoring typical new installations from start to finish. This should provide more reliable information on the economic aspect of illumination than is available at present.

It is hoped that this program will help industry install lighting systems which not only comply with the federal regulations, but improve working conditions with a minimum of maintenance problems.

PRINCIPAL AUTHOR MAILING ADDRESSES

ALDRIDGE, DR. M. DAYNE
Professor, Department of Electrical Engineering, West Virginia University, Morgantown, West Virginia 26506

BAILES, C. MAX
Vice President, Clinchfield Coal Company, Division of the Pittston Coal Group, Lebanon, Virginia 24266

BARTHOLOMAE, ROY C.
Electrical Engineer, Industrial Hazards and Communications, Pittsburgh Mining and Safety Research Center, U.S. Bureau of Mines, 4800 Forbes Avenue, Pittsburgh, Pennsylvania 15213

BLACK, SIGMUND
Corporate Vice President of Engineering, Fairchild Inc., P. O. Box 1184, Beckley, West Virginia 25801

BOCKOSH, GEORGE R.
Electrical Engineer, Industrial Hazards and Communications, Pittsburgh Mining and Safety Research Center, U.S. Bureau of Mines, 4800 Forbes Avenue, Pittsburgh, Pennsylvania 15213

BOYD, DOUGLAS C.
Operations Research Analyst, Denver Mining Research Center, U.S. Bureau of Mines, Building 20, Denver Federal Center, Denver, Colorado 80225

CARSTENS, DR. M. R.
Professor, School of Civil Engineering, Georgia Institute of Technology, Atlanta, Georgia 30332

CORBITT, STEPHEN W.
Project Engineer - Drum Miners, National Mine Service Company, P. O. Box 1447, Ashland, Kentucky 41101

CURRY, JESSE J., JR.
Superintendent of Underground Maintenance, Virginia Pocahontas Division, Island Creek Coal Company, P. O. Box 444, Keen Mountain, Virginia 24624

CURTH, ERNEST A.
Mining Engineer, Roof Support Group, Pittsburgh Mining and Safety Research Center, U.S. Bureau of Mines, 4800 Forbes Avenue, Pittsburgh, Pennsylvania 15213

DEUL, MAURICE
Research Supervisor, Methane Control and Ventilation, Pittsburgh Mining and Safety Research Center, U.S. Bureau of Mines, 4800 Forbes Avenue, Pittsburgh, Pennsylvania 15213

BREUIL, DR. FELIX
Assistant Manager, Mining Research, Bituminous Coal Research, Inc., 350 Hochberg Road, Monroeville, Pennsylvania 15146

ELLIOTT, JAMES R.
Chief Electrical Engineer, North American Coal Corporation, Powhatan Point, Ohio 43942

ELLISON, DR. RICHARD D.
Executive Vice President, D'Appolonia Consulting Engineers, Inc., 10 Duff Road, Pittsburgh, Pennsylvania 15235

FORNEY, ROSS
Director-Coal Services Division, Ebasco Services Inc., P. O. Box 986, Golden, Colorado 80401

FRIANT, JAMES E.
Manager - Government Projects, The Robbins Company, 650 South Orcas Street, Seattle, Washington 98108

GREEN, LARRY E.
Industrial Engineer, Roof Support Group, Pittsburgh Mining and Safety Research Center, U.S. Bureau of Mines, 4800 Forbes Avenue, Pittsburgh, Pennsylvania 15213

HARGRAVES, DR. ALAN J.
Superintendent, Rock Mechanics and Strata Control, The Broken Hill Proprietary Company Ltd., Australian Iron & Steel Pty. Ltd., P. O. Box 1239, Wollongong, N.S.W. 2500, AUSTRALIA

HERRON, R. D.
Corporate Vice-President of Maintenance, Island Creek Coal Company, P. O. Box 11430, Lexington, Kentucky 40511

JOHNSTONE, JAMES
Managing Director, Buchanan Borehole Collieries Pty. Ltd., P. O. Box R221, Royal Exchange, Sydney, N.S.W. 2000, AUSTRALIA

KARLOVSKY, JERRY, JR.
Manager - Development Engineering, National Mine Service Company, Mine Road, Nashville, Illinois 62263

KLOUSE, KENNETH P.
Chief, Electrical Power Systems Branch, Mining Enforcement and Safety Administration, 4800 Forbes Avenue, Pittsburgh, Pennsylvania 15213

KREUTZBERGER, JOSEPH
Vice President, Safety & Training, Pennsylvania Mines Corporation, P. O. Box 367, Ebensburg, Pennsylvania 15931

McNICOL, JACK B.
Director, Mining Operations, Allied Chemical Corporation, Semet-Solvay Division, P. O. Box 1835, Beckley, West Virginia 25801

MORGAN, BENNIE E.
Superintendent-Maintenance Services, U.S. Steel Corporation, Resource Development Group, Cumberland Coal Operations, P. O. Box 711, Waynesburg, Pennsylvania 15370

OLAF, DR. RER. NAT. JÖRN
Bergbau-Forschung GmbH, 4300 Essen 13, Frillendorfer Straße 351, GERMANY

PATTS, LARRY D.
Metallurgical & Environmental Engineer, Lee Engineering Division, Consolidation Coal Company, 200 Hidden Valley Road, McMurray, Pennsylvania 15317

RUPERT, DR. GERALD B.
Chairman, Department of Geology & Geophysics, University of Missouri-Rolla, Rock Mechanics and Explosives Research Center, Buehler Building, Rolla, Missouri 65401

SHARKEY, ANTHONY
Staff Engineer, Division of Mining Research--Resources, U.S. Bureau of Mines, 2401 E Street, N.W., Washington, DC 20241

SLONE, RANDOLPH E.
Electrical Engineer, Vice President--Eastern Operations, Westmoreland Coal Company, Drawer A and B, Big Stone Gap, Virginia 24219

THOMAS, V. M.
Deputy Director (Electrical Engineering), National Coal Board, Mining Research and Development Establishment, Ashby Road, Stanhope Bretby, Burton-on-Trent, Staffs DE15 OQD, UNITED KINGDOM

WHITEHEAD, KENNETH L.
Supervising Engineer, Bituminous Coal Research, Inc., 350 Hochberg Road, Monroeville, Pennsylvania 15146

WILSON, FRED T.
Manager, Central Rebuild Center, Clinchfield Coal Company, Division of the Pittston Coal Group, Dante, Virginia 24237