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THE DEVELOPMENT AND TEST OF AN UNDERGROUND AUGER PANEL EXTRACTION SYSTEM

Final Report
Contractor—FMC Corporation

March 1981

Contract No. U.S.D.O.E. AC01-75ET12485
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U. S. Department of Energy
Assistant Secretary for Fossil Energy
Office of Coal Mining

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THE DEVELOPMENT AND TEST OF AN UNDERGROUND AUGER PANEL EXTRACTION SYSTEM

Final Report

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U. S. Department of Energy
Assistant Secretary for Energy Technology
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This report represents work on a program that was originated by the Interior Department's Bureau of Mines and was transferred to the Department of Energy on October 1, 1977.

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FOREWORD

This report outlines the results of a program to develop and test two separate prototypes of an auger panel extraction system in an underground coal mine The system consisted initially of a modified commercially available Coalbadger auger miner which was used to drill large-diameter (42 inch) boreholes at distances up to 100 feet through specially prepared panels of coal in the Pittsburgh Number 8 coal seam. The major modifications consisted of the addition of a two-stage coal conveyor and a unique through-the-stem borehole ventilation methane monitoring system.

After extensive underground testing, the system was rebuilt on an FMC-designed frame, and several improvements in drill steel handling and tramming were incorporated. This second prototype was returned underground for additional testing. Operational and production aspects of these two systems are discussed in detail in the report.

Since these two machines differ widely in form and function and since their construction and testing were displaced temporally, this report treats them separately.

Part 1 covers the First Prototype and contains that information initially documented as an interim final report while the project was under the cognizance of the U.S. Department of the Interior, Bureau of Mines.

Part 2 covers the Second Prototype and contains information previously unreported since the project sponsorship changed to the U.S. Department of Energy.

There are separate tables of contents for both parts, although figures and appendixes are designated sequentially to avoid the confusion of duplicity.

FMC Corporation takes this opportunity to express appreciation to the Consolidation Coal Company, Ireland Mine, for its splendid cooperation during both demonstration tests. We are particularly grateful to present and past employees Walt Mueller, Hugh Lucas, Mike Janc, Norbert Becker, Don Yoders, Charles Hagedorn, John Snyder, John Baker, Steve Harrison, Rick Geimer, Hestle Riggle, Jim Glendenning, Matt Preston, Kit Phares, and Dave Ryan, whose continued interest and constructive suggestions contributed greatly to the program's success.

ABSTRACT

From June 1975 to March 1981, the Engineered Systems Division of FMC Corporation engaged in the development and testing of two prototype underground augering machines, first under contract to the U.S. Department of the Interior, Bureau of Mines and later to the U.S. Department of Energy. Actual underground testing of the first machine (a modified Coalbadger manufactured at the time by Badger Manufacturing Company) was in 1976. Shift production averaged 39 tons.

The second machine, highly modified from the Coalbadger and built at FMC, was in service in late 1978 and early 1979. Average shift production with this machine increased to 88 tons.

Boreholes up to 110 feet deep were drilled with these machines, a hole was successfully backreamed from its original 42-inch diameter to 48 inches, and several experiments were performed to determine drilling horsepower. Consolidation Coal Company's Ireland Mine near Moundsville, West Virginia, provided the demonstration site for both machines. This mine is in the Pittsburgh Number 8 coal seam.

It was concluded that underground augering of coal is a viable means of extracting coal from certain difficult areas of many mines. Future machines must have improvements which will result in less downtime, and average production must increase to 125 tons per shift or more to be attractively profitable.

EXECUTIVE SUMMARY

Underground augers, which bore long horizontal holes in coal pillars, have been the subject of periodic experimentation for several decades. It is recognized that augers can enhance recovery in certain circumstances, such as areas of bad roof and areas where surface subsidence cannot be tolerated, and yet some secondary pillar recovery is desired.

In 1974 the Badger Manufacturing Company of West Virginia developed an auger machine they called the Coalbadger. In 1975 FMC reviewed the Coalbadger and determined that, with certain modifications, it could be very productive.

In 1975 FMC entered into a contract with the U.S. Department of the Interior, Bureau of Mines to purchase, modify, and demonstrate a Coalbadger machine. Two significant additions were made to the Coalbadger:

- A series of two conveyors were added to the basic machine to carry the coal away from the boreholes.
- New, hollow-stem auger sections were fabricated with a ventilation system so that the boreholes could be ventilated while being drilled (in conformance with Federal laws).

The modified auger system was tested underground in a Consolidation Coal Company mine in Northern West Virginia during 1976. It was determined that holes as deep as 110 feet could be drilled, but problems in guiding the boreholes and keeping the machine operating had limited the average shift production to 39 tons. This was unacceptably low to the mine's management, but FMC determined that productivity could be increased by severely modifying the machine in a way which would make it more reliable and allow a greatly reduced cycle time for drilling.

The original Coalbadger was scrapped, and FMC fabricated a new underground auger on an FMC-designed chassis.

The machine was returned to the same mine in late 1978 and had completed its demonstration by early 1979. The combination of mechanical and operational improvements allowed the average shift production to rise to 88 tons with a record shift of 182 tons.

To be economically attractive to mining companies, future augers must be improved to the point where they can produce at least 125 tons per shift with a crew no larger than five persons.

Much basic knowledge about borehole drilling was gained, and it seems certain that auger drilling can, in certain situations, offer a method of achieving good recovery and safety in hard-to-mine areas.

1.0 INTRODUCTION (Part 1)

USBM Contract HO252076 -- The Development of an Underground Auger Panel Extraction System -- was awarded to FMC Corporation on June 23, 1975. The objective of the program was to develop an underground augering machine and ancillary components and to demonstrate the system underground in a production situation.

This report documents the activities, accomplishments, and problems encountered during the engineering design, fabrication, and in-mine testing phases of the program for the first prototype.

The portion of the program concluding with the demonstration of the first prototype equipment underground continued as a U.S. Bureau of Mines project until September 1976. The program was then modified and resumed when it was transferred to the U.S. Department of Energy. The effort associated with the development of the second prototype is discussed in Part 2 of this report.

2.0 SUMMARY (Part 1)

2.1 HARDWARE DEVELOPMENT

Since at least the 1940s, it has been recognized that underground coal production in certain "difficult" mining areas or situations could be enhanced by utilizing a horizontal drilling or augering machine to bore into the virgin seam from a stablilized entry and recover a high percentage of the coal. Such an operation would also offer a measurable increase in safety, as all the excavation would be done in an unmanned area, and the machinery itself would be mounted in a well-bolted or timbered area. Several sporatic attempts were made to build and operate such "auger miners," and while most of them were able to drill coal, none reached truly profitable production.

The type of "auger miner" which we refer to here is the true augering machine which produces long, relatively large-diameter horizontal boreholes through the seam. There are other continuous mining machines also called auger miners which use horizontally mounted auger cutterheads with cutterbits mounted all along the periphery of the scrolls. These auger-head continuous miners cut coal and produce an entry by oscillating their frames laterally in a wide swath as they move forward. They are generally used in low coal seams, operate well, and are very productive.

Experimentation on boring-type auger miners dwindled in light of the introduction of and advancements being made in mechanized continuous miners. There are many reasons why early auger miners failed. Some of the more important reasons are as follow:

- Lack of sufficient power to drill a large-diameter hole sufficiently deep to be efficient in recovery.
- Lack of a method of steering the cutterhead and auger string to avoid cutting roof and bottom.
- Lack of an effective method of loading, unloading, and storing the auger flights while drilling. Hand loading of individual auger flights was practiced exclusively and, after completing a hole, each flight was laboriously removed from the hole and stored in the entry while the drill base was moved to the site of the next hole.
- Lack of a coal haulage system to remove the cut coal from the throat of the borehole and keep the machine from becoming muckbound.

In 1972, Badger Manufacturing Company revived the idea of underground augering with a machine they called the Coalbadger. Aware of the failings

of earlier auger machines, Badger improved upon the prior art in a series of prototype machines which ultimately resulted in the following features:

- A 150-horsepower main drive calculated to drill a 36-inch-diameter hole to a depth of at least 100 feet

- A system of hydraulic jacks which are capable of leveling the machine on uneven ground so that auger trajectory can be better controlled

- A unique sliding frame arrangement which allowed the drill frame to move from borehole to borehole storing and retrieving drill sections from the adjacent hole, thereby eliminating most of the tedious hand loading of augers.

In 1974 Badger Manufacturing Company previewed their Coalbadger auger miner to the mining industry. As part of new product development activities, FMC visited Badger and reviewed the machine specifications and operating scenario with Badger personnel. It was immediately apparent to us that, while Badger had solved many of the problems which had plagued prior augers, certain other features had been overlooked which would tend to diminish the machine's utility. Specifically,

- There was still no coal haulage system to carry the coal away from the borehole.

- There was no provision for ventilating the borehole as it was being excavated as would be required to meet the intent of the 1969 Coal Mine Health and Safety Act.

- The controls and operator's station lacked proper human factors' attention.

- Badger had not successfully drilled larger than a 36-inch-diameter borehole, which limited the number of seams in which the machine could provide acceptable recovery.

Encouraged, however, by the advancements in the baseline Coalbadger machine, FMC proposed a development program to the United States Bureau of Mines whereby FMC would undertake to modify the Coalbadger, correcting the apparent shortcomings, and add such ancillary equipment as was necessary to emplace a complete auger miner system for demonstration in a producing mine. Such a program was approved and work began in June 1975.

The baseline machine, designated Coalbadger IV (fourth prototype), was ordered from Badger in July 1975 and delivered in November. Simultaneously, FMC designed and built several modification "kits" for the Coalbadger:

- Face Conveyor

  To solve the problem of hauling coal away from the collar of the borehole, a two-stage conveyor was provided to carry the coal from the borehole to the center of the entry. The first stage was fixed -- bolted to the Coalbadger, and the second stage was mobile -- wheel-mounted and positionable in the entry. This conveyor system was designed to deposit the coal in conventional face haulage vehicles -- shuttle cars or scoops.

- Airbox Chuck

  To solve the problem of ventilating the borehole as it was being drilled, a hollow, rotating chuck with a stationary (nonrotating) air collection box was designed through which an exhausting fan could be connected via flexible tubing. This arrangement was necessary to carry both dust and methane from the borehole to the mine's return airways. The airbox was also fitted with two remote-reading methanometers.

- Hollow-Stem Augers

  To accomplish in-hole ventilation and dust collection, FMC designed a unique set of hollow-stem augers. In operation, air was drawn back through the auger stems, through the airbox, and on through the fan and into the returns. The stems were designed with tang and box torque connections and spring-loaded shear pins to react withdrawal force.

- Fan and Rockdusting System

  Calculations were performed which allowed the selection of a ventilation fan which would adequately aspirate the borehole. A minirockduster was purchased and fitted with a set of rigid tubes with specially constructed quick-acting connectors so that the interior of the boreholes could be rockdusted.

Several other minor machine modifications were made; these, along with the major modifications listed, are discussed in more detail in the following sections of this report.

2.2 DEMONSTRATION SITE AND MINE PLAN

It was determined early in the program that to best demonstrate this prototype system, certain criteria for the demonstration site should be set forth as requirements:

- 56- to 72-inch seam height
- Dry bottom
- Reasonably competent roof
- Minimum 18-foot-wide entries
- Freedom from major rock inclusions in the coal.

These criteria are tantamount to a "textbook" mine. Nonetheless, after holding conversations with numerous mines, the Ireland Mine of Consolidation Coal Company was selected as the test site as it did indeed possess all the desired attributes. The Ireland Mine is in Moundsville, West Virginia, and operates in the Pittsburgh Number 8 coal seam.

Management, engineering, and operating personnel worked with FMC project personnel to develop a mine plan which was then submitted to and accepted by

MESA[1] and the West Virginia Department of Mines. Three test panels were mined to FMC specifications in preparation for the underground test.

2.3 PRODUCTION TEST

All the system components were massed at Ireland Mine's main shop. Here all the final joining of component parts and other minor modifications were completed. Members of the Mine's UMWA Safety Committee observed the machine on the surface and were instructed in its operation. After all systems were checked out on the surface, the miner and the other components were disassembled and moved underground. Reassembly was accomplished underground and the equipment was set up on the first prepared panel.

Details on the demonstration are covered in the latter sections of this report, but in summary:

- The equipment operated from May 6 to August 19, 1976, experiencing only two major breakdowns -- the loss of the main hydraulic motor and damage to one cutterhead.

- During this period the section operated 51 of a possible 58 shifts. Seven shifts were lost (i.e., crew not dispatched to the section) due to equipment breakdown. Note: Numerous shifts were lost due to the 1976 miners' vacations and to periodic strikes, and these lost days are not tabulated for the purpose of evaluating system performance.

- Reasonably straight holes up to 100-foot depths were drilled at which point the machine's practical ultimate horsepower limit was reached. For reference, a 42-inch-diameter, 100-foot-deep borehole yields 40 tons of coal which we show to be the tonnage per setup with the existing machine.

- It was demonstrated that 90-degree intersecting holes through blocks of coal could be drilled with the machine. This demonstration was done as proof of the utility of the auger as a ventilation control system used in conjunction with a continuous miner, i.e., using the right-angle boreholes to carry face air and dust into the returns.

- During the test period the section mined 2,000 tons ROM coal. Shift production varied from 0 (on days when the equipment was moved) to 130 tons.

- On the 58-shift base, shift production averaged 34.5 tons per shift. On the 51-shift base, shift production averaged 39.2 tons per shift. The difference, 4.7 tons per shift, illustrates the damaging effect of the major machine malfunctions.

1. The Mining Enforcement and Safety Administration (MESA) was part of the U.S. Department of Interior until March 9, 1978 when reorganization placed it under the U.S. Department of Labor, retitled Mine Safety and Health Administration (MSHA). References to MESA and MSHA throughout this report indicate the agency in authority at the time.

- On the 31-shift base, productivity averaged 7.8 tons per man-shift for the auger section; and while this is less than the average tonnage of a continuous miner section in Ireland Mine, it is very respectable performance for a prototype machine. It should be noted, however, that the machine's performance fell short of the calculated potential. This was due in large part to difficulties in exchanging auger flights from hole to hole, to breakdowns, and to an excess of idle time spent waiting for mobile face haulage. Instantaneous production rates compared very favorably with initial predictions.

Several design and construction defects contributed to the machine downtime, but these were corrected in the second prototype.

Production and productivity can be expected to increase substantially if certain design, construction, and operational changes are implemented in future models. These are discussed in the following sections of the report.

3.0 TECHNICAL DISCUSSION (Part 1)

3.1 HISTORY

Highwall augers which drill coal from prepared benches on the surface have been utilized for several decades. However, due primarily to increasing environmental constraints, modern highwall augers, such as that shown in Figure 1, now account for only a few million tons of surface production per year.

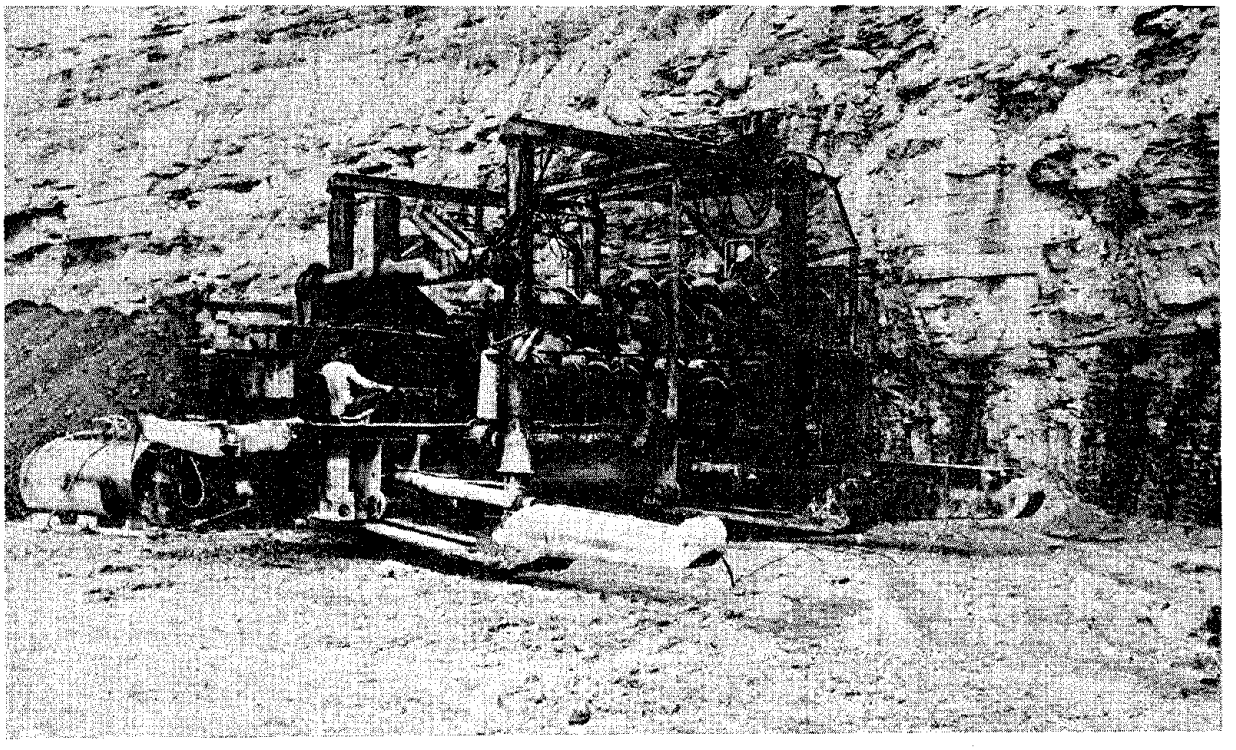


Figure 1 TYPICAL HIGHWALL AUGER OPERATED ON SURFACE

The idea of mining coal underground by auger has been attempted experimentally since the 1930s. Several attempts, both in the U.S. and in England and France, have resulted in workable machines. Their major drawback was that they were not as productive as conventional drill and blast mining and in the 1940s major industry emphasis was placed on the mechanized continuous miner. That emphasis persists even today.

Although almost all highwall auger manufacturers and some continuous miner manufacturers have built experimental underground augers, none have proved sufficiently productive to merit introduction of a salable machine. Spurious attempts by independent inventors and small-mine owners at building one-of-a-kind underground augers have been recorded, but again, none offered much hope as a commercial product.

The reasons that early underground augers did not perform better are easily understood when one pictures the previous operating scenario:

- The auger platform was skidded, jacked, or wheeled into position against the coal face. Cribs or props were set around the machine and timbers, or jacks were placed between the frame and the rib to the rear of the frame to react thrust.

- One by one, the cutterhead and following scrolled flights were rolled and lifted (with pry bars and cables, but mostly by hand) into position on the platform, matched to the drive chuck, and drilled into the coal.

- As the cut coal exited the borehole, it was shoveled by hand into the face conveyance or shoveled out into the clear so that a mechanized loader could gather it. Often the auger would have to be stopped two or three times per 6-foot advance for shoveling or the machine would become muckbound.

- When the hole had been drilled to its deepest penetration, all auger sections would be laboriously removed by hand and stored in the entry. Then the auger platform could be moved one spacing away and the entire process would be repeated. A good crew of four or five hardworking miners might make 24 tons of coal a shift (80 feet of 36-inch-diameter borehole).

These early experimental auger mines were characteristically:

- Labor intensive

- Dangerous (because of all the hand work)

- Marginally productive

- Less interesting to mine operators as a mining system than the emerging improvements in continuous miners.

During the 1960s and early 1970s, little experimentation was done on underground augers in the U. S. Two events, however, occurred which caused renewed interest in underground augering. The 1969 Coal Mine Health and Safety Act was passed and began to be enforced, which caused productivity in the mining industry to drop dramatically in both continuous and conventional operations. The "Energy Crisis," as it has come to be called, of 1973-1974, caused renewed interest in coal production and utilization, and in particular in the conservation of our national coal reserves -- in this case conservation meaning the efficient recovery of the coal resources already being mined.

A great quantity of coal was heretofore (and still is) left in the ground simply because productive methods of extracting it had not been explored. This "unrecoverable" coal is coal which:

- underlies "bad roof" areas, or
- is left in barrier pillars and against outcrops for surface ground protection or to protect more immediately recoverable coal in the seam, or
- is left in old mines or workings because no efficient secondary recovery methods existed when the development work was done.

This resurgence of interest in mining "difficult" coal caused Badger Manufacturing Company of Summersville, West Virginia, in 1972, to reinstitute the development of an underground auger which the company's principals had conceived a decade before. The machine which emerged was an improved underground auger which they designated the Coalbadger. Their patent outlines the features of the machine which constituted improvements over earlier attempts:

- Roof and floor jacks which acted to wedge the entire machine in place in the entry to react drilling and withdrawal thrust.
- A unique sliding frame arrangement which allowed the base frame of the machine to remain fixed in the entry while the frame to which the torque motor was attached shifted transversely between the hole being drilled and the previous hole. This feature allowed storage of the drill steels in the previously drilled hole and retrieval in sequence as they were needed in drilling the next hole. This eliminated the problems of tedious hand loading of auger sections and the unloading and storage of the sections at the end of each completed hole. The sliding frame also allowed maintenance of more exact spacing between boreholes. The entire sliding frame/moving sequence is described in Section 3.3.

Three prototype Coalbadger machines were produced by Badger Manufacturing Company between 1972 and 1974. The first, built and tested in 1972, proved that the shifting frame concept of drilling and storing augers was successful, but the machine was structurally deficient and its frame and other components required redesign before it could be offered commercially. The second prototype was greatly improved and tested for a short time in the summer of 1973 in the Alyce Mine near Morgantown, West Virginia. Here the machine successfully bored several 60-foot-deep holes and a number of 84-foot-deep holes using a 36-inch-diameter auger. It was removed from the Alyce Mine and taken to another site for additional testing, but before testing could begin a massive cave-in totally buried the machine and a good portion of the mine. Machine 3 was an updated version of the second machine but with an extended auger trough. It was specially reconfigured for long-term lease to the Shannon Branch Mine of Semet-Solvay Division of Allied Chemicals where it was used to bore airways only and was not used as part of the production cycle. In 1974 this Coalbadger showed that it could drill relatively long (54-foot) straight holes in the very soft coal of the Pocahontas Number 3 seam.

In 1974 Badger Manufacturing Company announced the Coalbadger auger miner as being available for limited production, and interest was high from within the mining industry. Along with many other companies, FMC inspected the Coalbadger; and while we recognized the potential utility of the machine, its shortcomings were also readily visible.

- There was still no conveyance device to carry the coal away from the borehole.

- Although a methane detector and shutdown system had been added, there was no provision for diluting the methane liberated within the borehole nor for controlling the float dust created while cutting. This we felt made the machine noncompliant with the intent of the 1969 Coal Mine Health and Safety Act.

- There were obvious improvements to be made in the layout of controls and in the operators' stations themselves.

Recognizing that these major deficiencies could be overcome by relatively straightforward design changes, FMC approached the U. S. Bureau of Mines offering to modify a production Coalbadger and demonstrate the improved system underground in a producing mine. In June 1975, USBM Contract HO252076 was awarded to FMC Corporation for the development and test of an underground auger panel extraction system.

3.2 SYSTEM DESCRIPTION

The underground auger panel extraction system is shown in an artist's concept drawing in Figure 2. The major system components are called out in this figure and are:

- Coalbadger auger miner with protective canopy

- Sub-auger conveyor

- Entry conveyor

- Hollow-stem augers

- Flexible ventilation tubing

- Auxiliary fan (not shown)

- Rock duster and in-hole rockdusting tubes.

All the components were either designed or specified by FMC as we conceived the panel extraction system. The design criteria and/or performance specifications set forth for all system components are outlined in the following subsections.

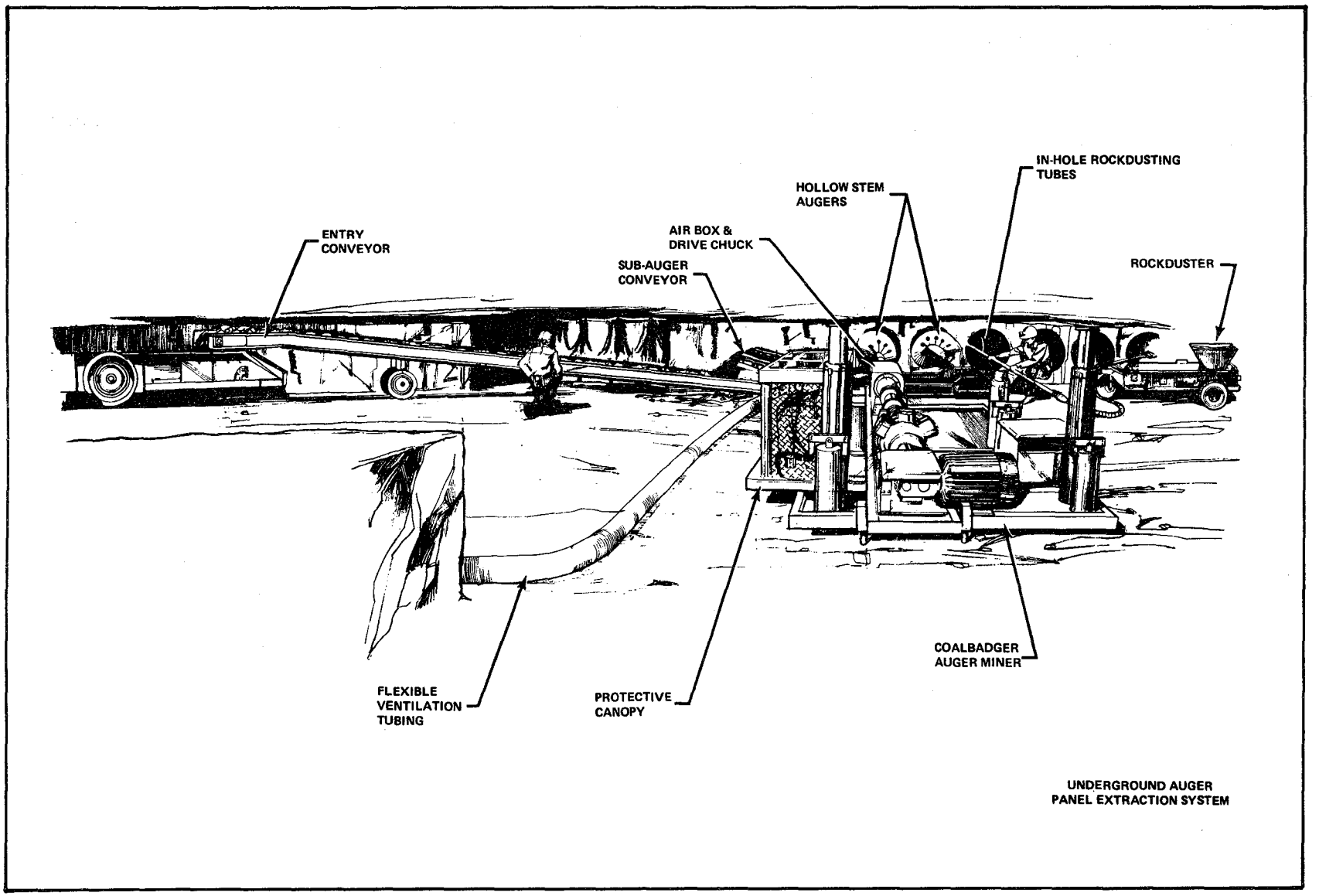


Figure 2 UNDERGROUND AUGER PANEL EXTRACTION SYSTEM

3.2.1 Coalbadger Auger Miner

As previously mentioned, when the underground auger panel extraction system was proposed to the USBM, the Coalbadger was selected as the auger miner on which to base the system. FMC purchased the fourth prototype, designated by the manufacturer as the Coalbadger IV. A great number of design improvements had been made by Badger Corporation and several modifications were suggested or specified by FMC.

Figures 3, 4, and 5 show several views of the Coalbadger IV in its "stock" condition. Figure 4 shows the protective canopy and operator's station, and Figure 5 shows the rear control station for the actuation of the floor and roof jacks.

Table 1 outlines the operating and performance specifications of the Coalbadger as advertised by the manufacturer. The major modifications to the "stock" Coalbadger as purchased by FMC included the addition of a more powerful auxiliary hydraulic system and elimination of the mechanically troublesome powered steerable wheel set. Conversations with several mine operators indicated that towing was their preferred method of transit for short moves and that the machine need only tram for major moves. Also because the wheel set is removed before the machine is put into operation, short moves are made by dragging the machine on its base frame rather than reassembling the wheel set.

The second major modification was the substitution of a 30-horsepower auxiliary hydraulic power unit for the 20-horsepower unit normally supplied by Badger. This was done because early in the program it was decided to power the belt conveyors hydraulically by tapping into the Coalbadger hydraulic system. A power-demand table was compiled and it was found that a 30-horsepower unit with a three-section pump was required to suit our particular system. The Coalbadger IV was delivered to FMC in November 1976 and underwent minor structural modifications for the mounting of the conveyors and additional controls at FMC's Morgantown, West Virginia, facility.

3.2.2 Sub-Auger Conveyor

The sub-auger conveyor is a low-lying pan-type belt conveyor mounted beneath the auger trough along the front frame of the Coalbadger as shown in Figure 1. The conveyor alone is shown in Figure 6 as it underwent testing in the FMC shops.

The conveyor is unique in several respects. It has a very low overall height on the flat (5-3/4 inches) and, at its lowest position, the top strand of belt is only 1-3/4 inches from the ground. The return strand is fully shrouded for freedom from clogging from accumulated debris. Because the horizontal portion of this conveyor is so low, it was necessary to angle the tailboom upward at a 15-degree angle. This required a special set of rollers to guide the belt upward at a sharp angle while retaining it flat against the pans.

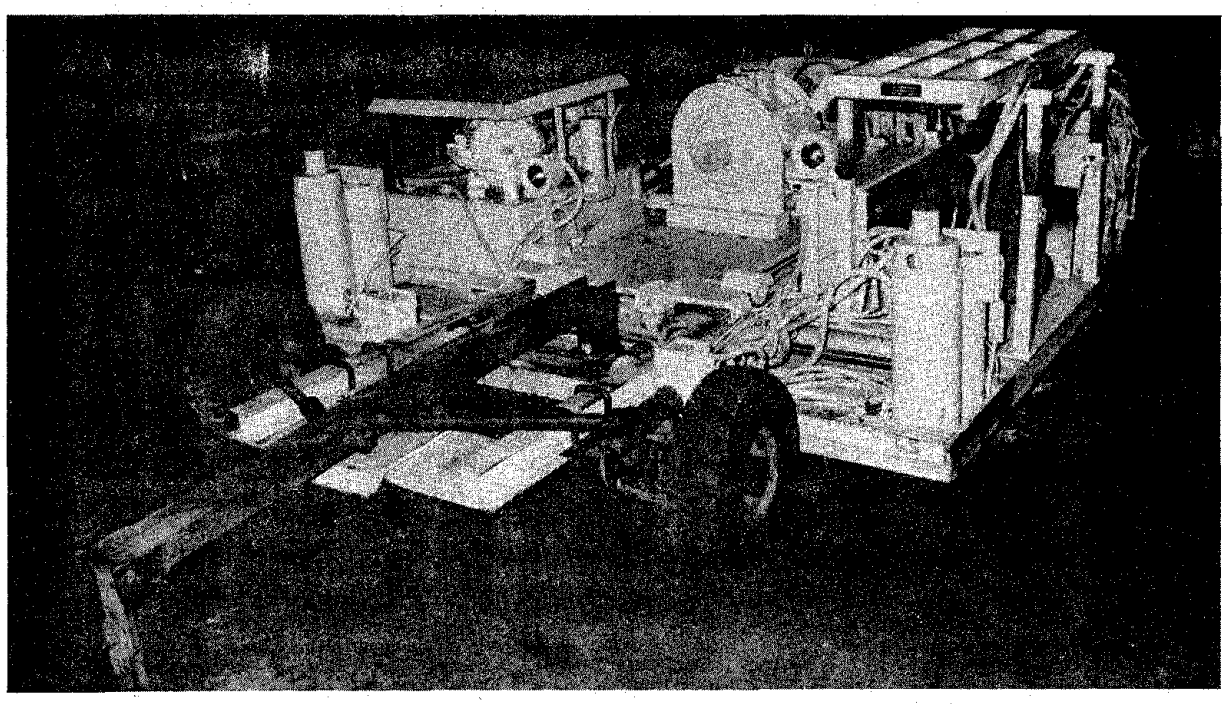


Figure 3 FRONT VIEW OF COALBADGER AUGER MINER

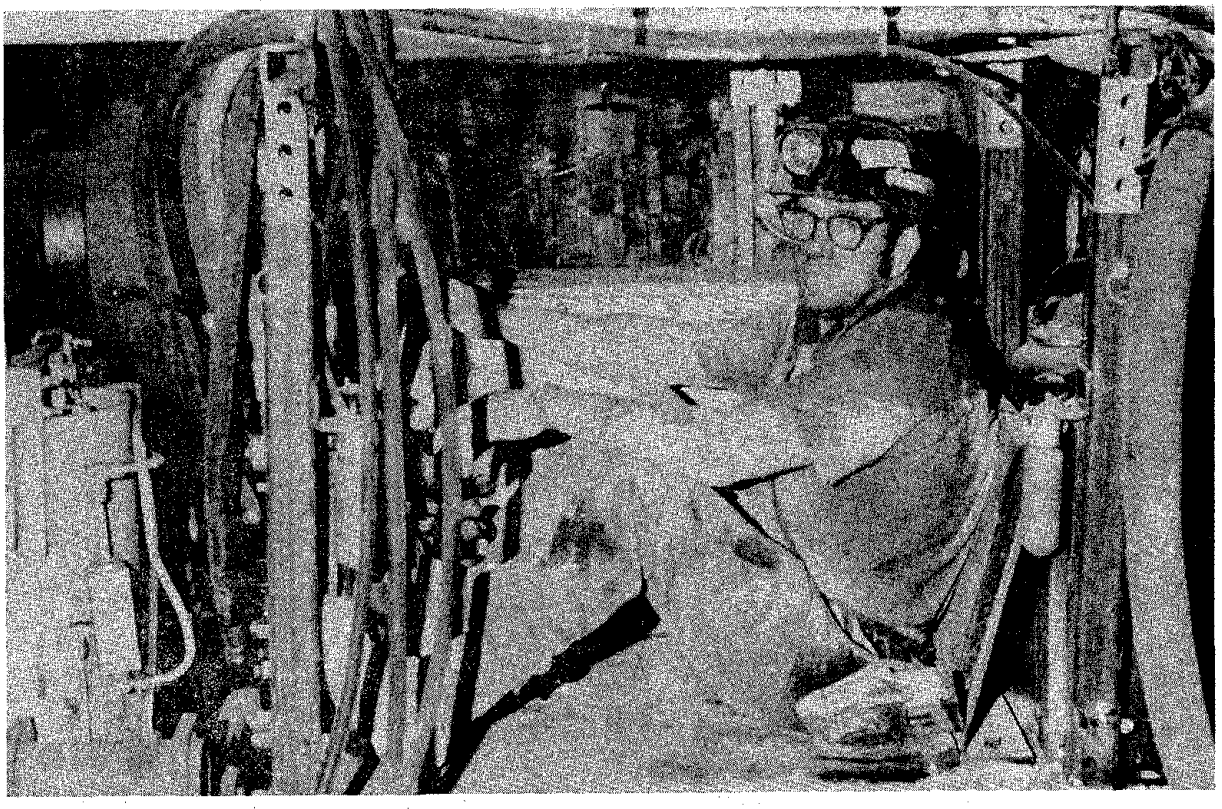


Figure 4 PROTECTIVE CANOPY AND OPERATOR'S CONTROL STATION ON COALBADGER AUGER MINER

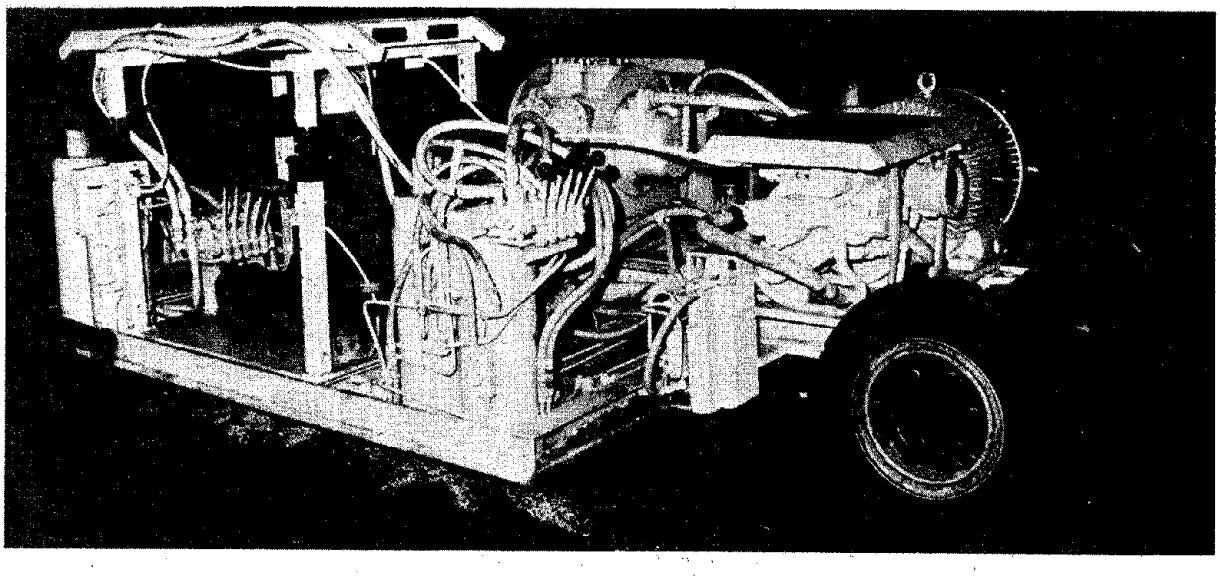


Figure 5 REAR VIEW OF COALBADGER AUGER MINER SHOWING REAR CONTROL STATION

Table 1 OPERATING AND PERFORMANCE SPECIFICATIONS FOR COALBADGER AUGER MINER

Power	Main auger drive	150 horsepower
	Auxiliary hydraulic power	30 horsepower
Dimensions	Overall length	15 feet, 2-1/2 inches
	Overall width	11 feet, 8 inches
	Tramming height (minimum)	47 inches
	Working height (minimum)	43 inches
	Sump stroke (maximum)	82 inches
	Traverse stroke (maximum)	57 inches
	Weight (approximate)	25,000 pounds
Utilities	Input power	440 volts AC, 30-60 Hertz, 200 KVA
	Water	10 gpm at 30 psi minimum
Features	Fire suppression system	Dry chemical, dual cylinder, manual discharge
	Lighting	3 incandescent floodlights
	Auger capacity	36- to 48-inch diameter
	Methane detection system	Preiser-Mineco
Performance	Sump speed (maximum)	8.4 feet per minute
	Auger drive speed	Infinitely variable to 62 rpm reversible
	Auger torque (maximum)	14,800 foot-pounds
	Auger thrust (maximum)	50,000 pounds
	Production with 36-inch auger	2.5 tons per minute maximum
	Production with 42-inch auger	3.5 tons per minute maximum

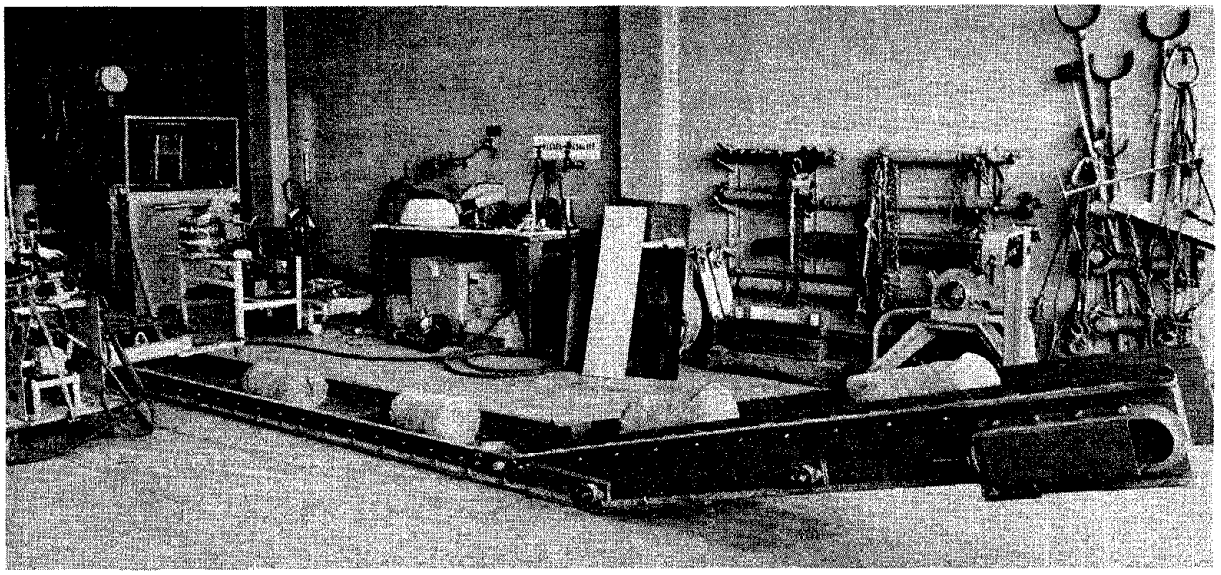


Figure 6 SUB-AUGER CONVEYOR BEING TESTED WITH SIMULATED LOAD (500 Pounds Bagged Gravel)

The sub-auger conveyor was designed to handle maximum anticipated production of the Coalbadger plus a generous margin and to allow a minimum of lost recovery due to offset of the auger from the floor. The hydraulically driven unit was powered from the Coalbadger auxiliary hydraulic system. All hydraulic lines feature quick-disconnect fittings and the entire conveyor was reversible by 180 degrees for discharge either to the right or to the left of the auger miner. General specifications of the sub-auger conveyor are given in Table 2.

Table 2 SUB-AUGER CONVEYOR SPECIFICATIONS

Dimensions	Height under auger	5-3/4 inches
	Overall height at discharge boom	36 inches
	Overall width	32 inches
	Overall length	23 feet, 7 inches
Drive	Type	Hydraulic motor
	Flow requirements	7 gpm at 1,950 psi maximum
	Power	4.3 horsepower continuous at 1,300 psi; 5.7 horsepower peak at 1,950 psi
Belt	Type	Long-Airdox Poly-Vinylad single-ply woven PVC-impregnated polyester yarn with PVC cover one side
	Belt speed	360 feet per minute
	Belt width	20 inches
Capacity	Tonnage loaded level full	5 tons per minute maximum
Weight	Unloaded weight	2,250 pounds

3.2.3 Entry Conveyor

Because the sub-auger conveyor discharges very close to the rib, it was not possible to position a shuttle car or scoop directly under the boom. Therefore, an entry conveyor was designed and built to carry the coal from the center of the entry to the face haulage vehicles. The entry conveyor (Figure 7) is a wheeled pan-type belt conveyor designed to be towed into position.

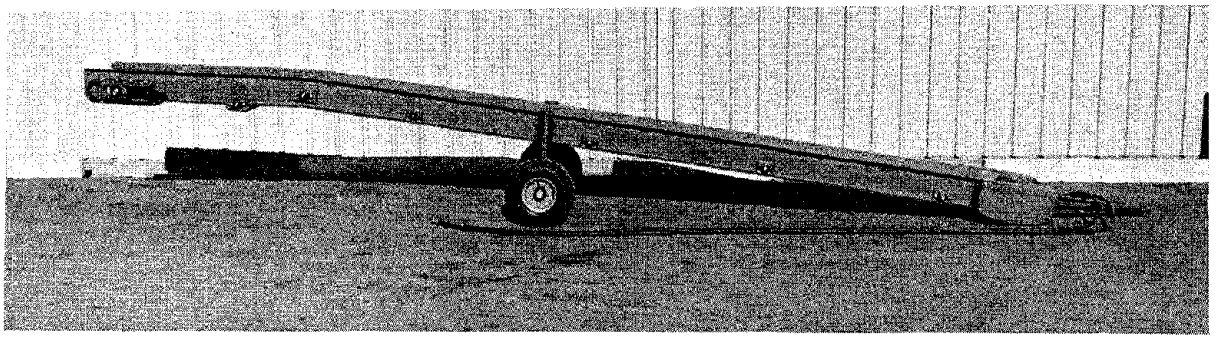


Figure 7 ENTRY CONVEYOR

This conveyor was also hydraulically powered from the Coalbadger via umbilical hoses. During place changes, the hoses were broken at self-sealing quick-disconnect fittings which minimizes the preparation time for moving. The general specifications of the entry conveyor are given in Table 3.

Table 3 ENTRY CONVEYOR SPECIFICATIONS

Dimensions	Overall length with towbar	30 feet, 5 inches
	Length with towbar removed	27 feet, 3 inches
	Width over siderails	30-1/2 inches
	Width over wheels	54-1/2 inches
	Height at discharge	36 inches minimum, 55-1/2 inches maximum (adjustable)
Drive	Type	Hydraulic motor
	Flow requirements	13 gpm at 1,950 psi maximum
	Power	7.9 horsepower continuous at 1,300 psi, 11.5 horsepower peak at 1,950 psi
Belt	Type	Long-Airdox Poly-Vinylad single-ply woven PVC-impregnated polyester yarn with PVC cover one side
	Belt speed	400 feet per minute
	Belt width	24 inches
Capacity	Tonnage loaded level full	6-1/2 tons per minute maximum
Weight	Unloaded weight	2,950 pounds
	Tongue weight	360 pounds

3.2.4 Hollow-Stem Augers

From the beginning of the program, it was recognized that ventilation through the entire borehole was necessary for the prevention of methane buildup and the possibility of explosion. Two ventilation methods were considered. One was a pressure system in which air from a fan or blower is directed down the borehole, through the stems of the augers, to the back of the borehole. It then travels a helical path back out to the collar of the borehole and exhausts into the entry. The second method, the one selected, was a vacuum system in which the intake side of a fan or blower is connected to the stems of the augers. Air is then drawn into the borehole, around the helical path to the back of the hole, and into the auger string. Then by means of suitable flexible and rigid tubing, it is directed into the mine's return air course. The two systems, represented diagramatically in Figure 8, offer nearly identical control of methane in the borehole, but the vacuum system offers superior control of dust. It does, however, introduce some danger of coal transport through the hollow stems which could result in clogging.

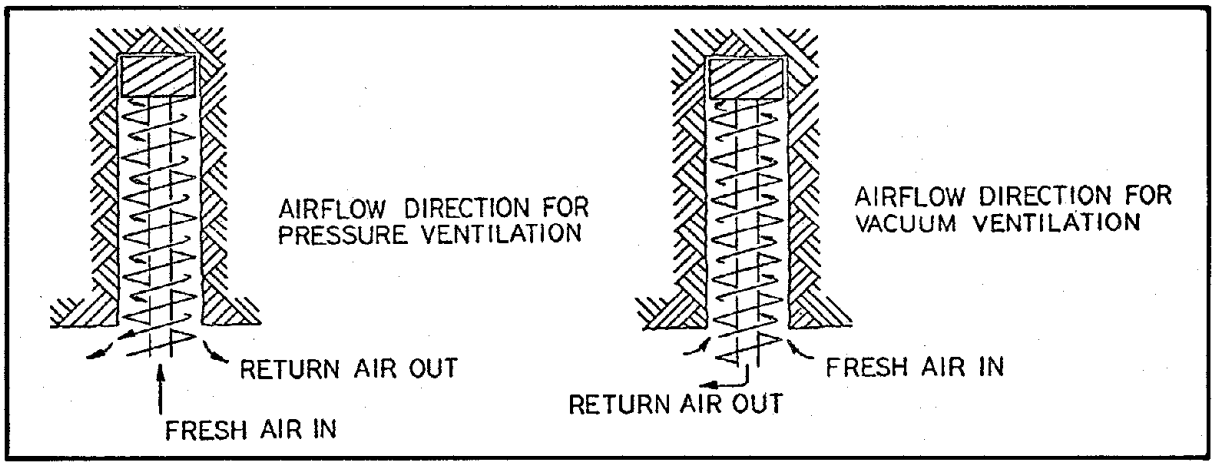


Figure 8 ALTERNATE AUGER VENTILATION SYSTEMS

To balance the requirements of airway area and mechanical strength for the augers themselves, 12-inch Schedule 30 pipe was selected for the auger stems. The outside diameter of the stems is 12.75 inches and the wall thickness is 0.33 inch. A tang and box style of connector with a spring-loaded latch pin which is fast and foolproof to operate is utilized in making auger connections. The individual stems are 5 feet, 8 inches long, with a coupled length of 5 feet each. The completed stems and the male and female portions of the connectors are shown in Figures 9 and 10.

A graph of theoretical air velocities for the unrestricted portions of the auger string versus airflow quantities is shown in Figure 11. It is of interest to note that an air velocity of 90 feet per second in a closed conduit is sufficient to cause the horizontal transport of coal lumps up to 3 inches in size, and high velocities such as these are to be avoided in sizing the ventilation system fan.

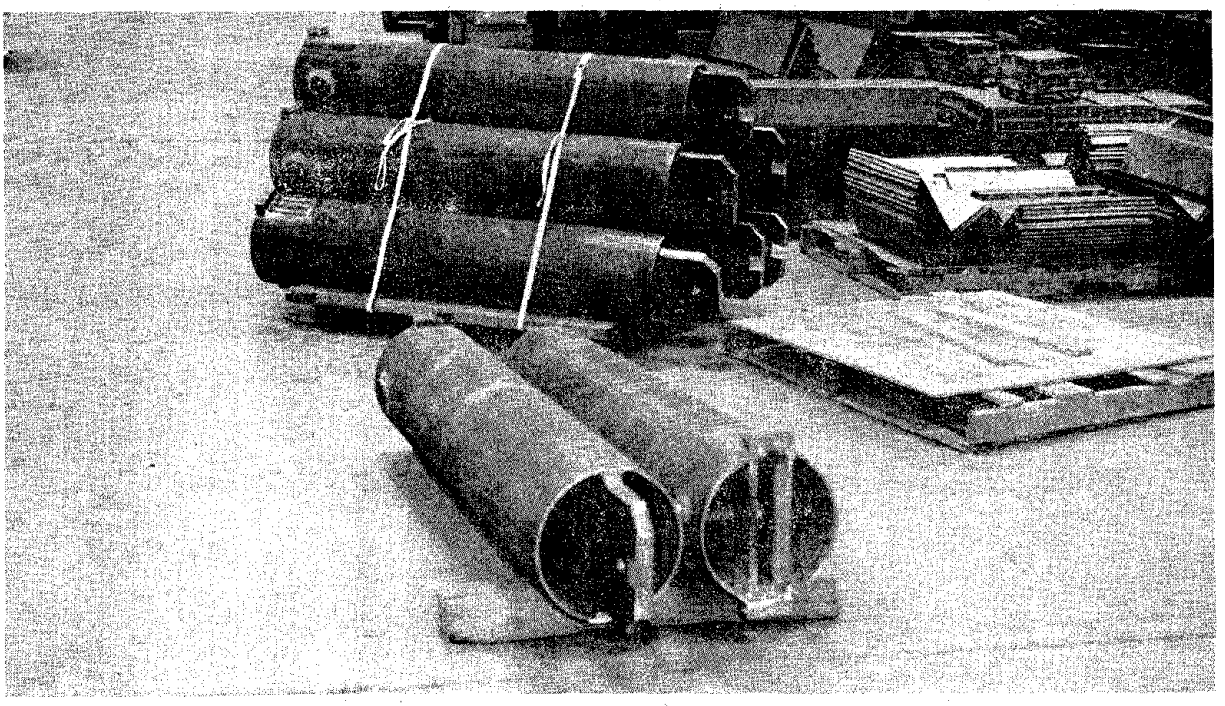


Figure 9 AUGER STEMS PRIOR TO APPLICATION OF SPIRAL

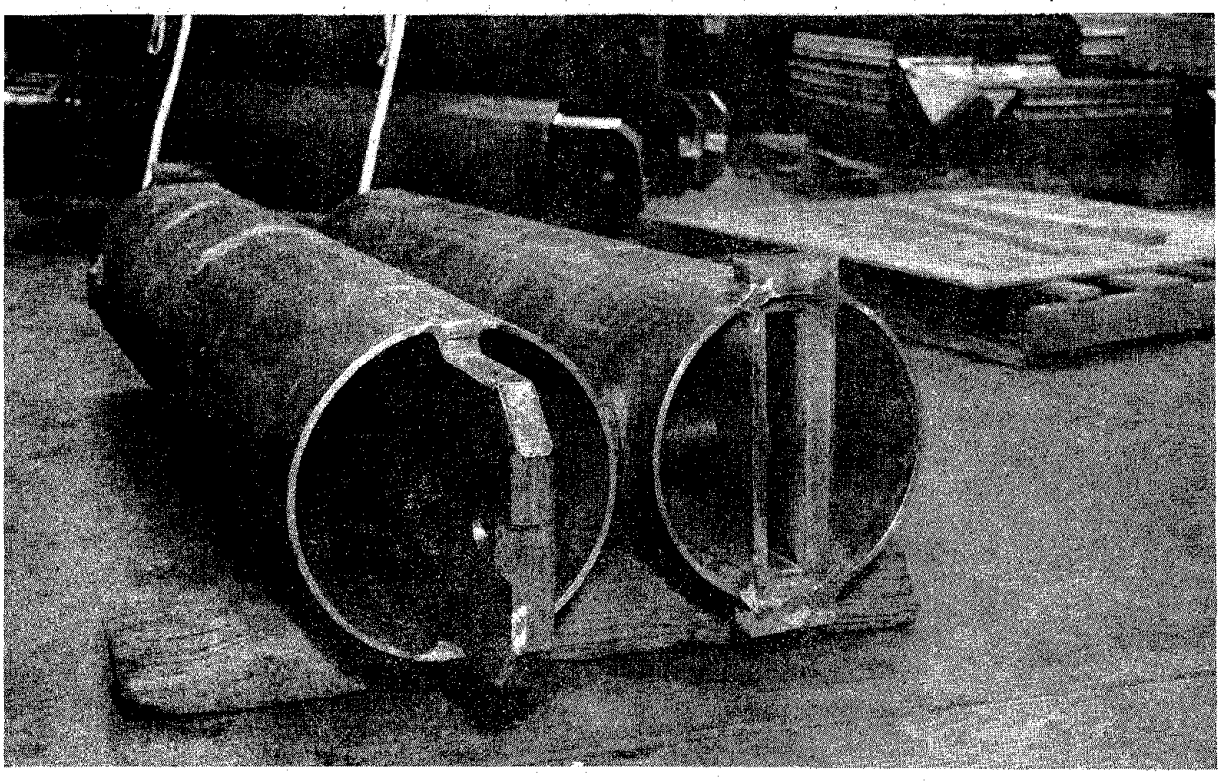


Figure 10 CLOSEUP OF TANG AND BOX ENDS OF AUGER STEM CONNECTORS

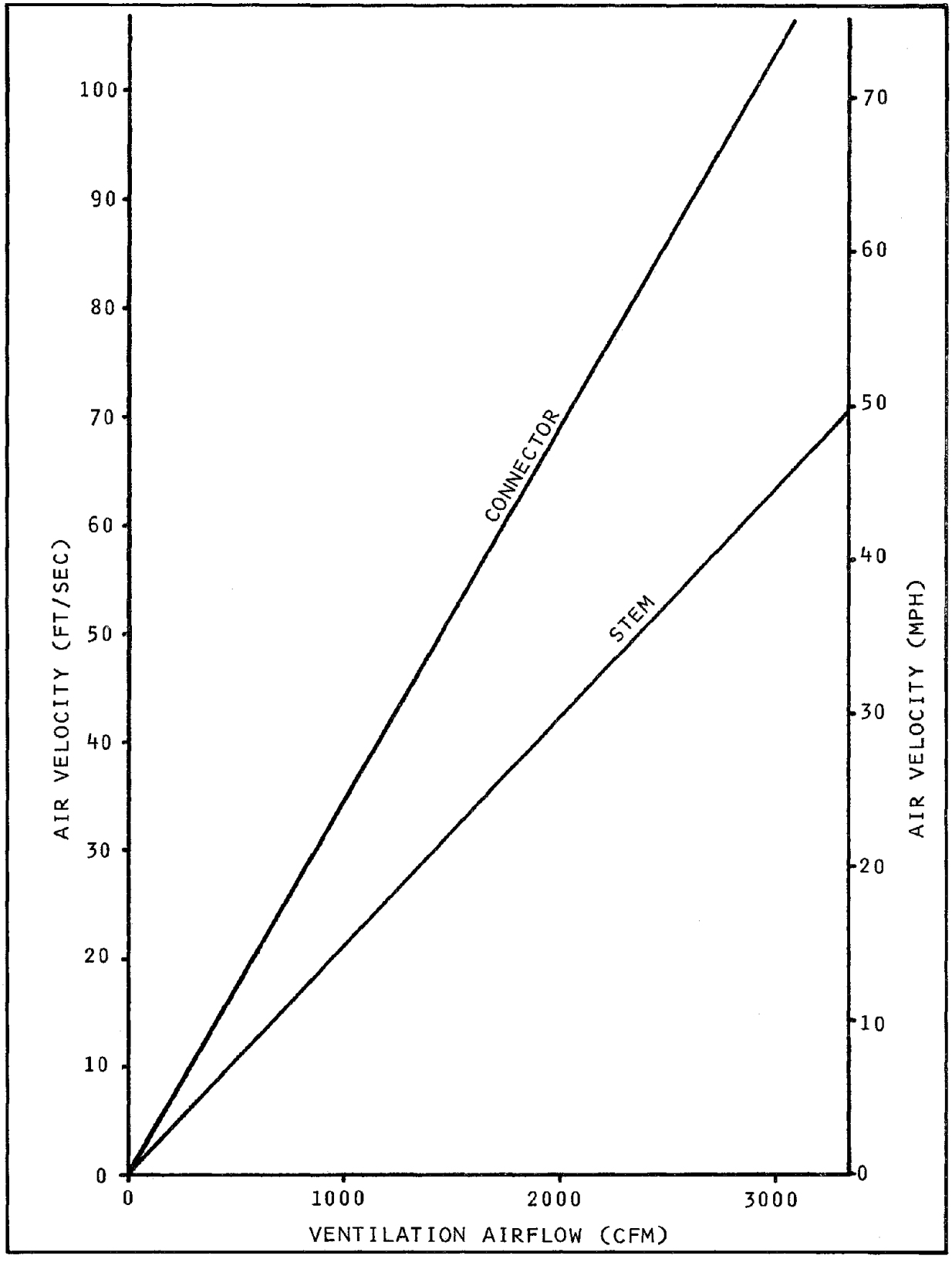


Figure 11 AIR VELOCITIES IN AUGER STEMS

These special stems were fabricated by FMC and provided to W&S Welding of Neal, West Virginia, for fabrication of the cutterheads and spiral flights. W&S has pioneered a drum style of cutterhead which has proven very sucessful both in cutting coal and in maintaining guidance through the virgin coal. The configuration of the cutterhead is shown diagrammatically in Figure 12. Figure 13 illustrates a completed cutterhead.

The drum portion of the cutterhead contains the bit blocks and spiral stabilizing lacings called "duster" lacings as shown and is connected to the stem by webs which continue to the rear of the head as the beginning of the spiral. The circular breaker bar which is also fitted with bits is set slightly behind with the leading edge of the drum and is attached to the end of the stem. The cutting action resembles that of the familiar hole saw.

As the hole is bored, a cylinder of coal is first exposed as shown in Figure 14A. These annular cuts allow relief so that the coal can be broken into pieces by the flutes on the breaker bar. The annular spacing between the main cutter drum and the inner breaker bar and the size and shape of the breaker bar flutes determine the sizing of the lumps.

As the cutterhead advances into the coal (Figure 14B), the entire drum portion finally is embedded in the coal. The duşter lacings act as stabilizers to guide the head much as the bourrelet and driving bands do on an artillery projectile. As the hole is advanced farther, the head continues to guide on its original heading with the spiral flights trailing behind. Highwall augers using this style of head have drilled relatively straight holes to depths up to 200 feet. A listing of specifications of the auger heads and flights is given in Table 4.

All auger sections have blocks welded to the box couplings as shown in Figure 10.

A failure mode study of the auger string predicted that the most likely failure of the string would be the torsional failure of a drive tang or the shear failure of a latch pin. In either event, at the suggestion of the mine operators, an auger retriever has been designed and fabricated which, when attached to the remaining undamaged auger sections, can be used to fish out the broken string. The auger retriever utilizes two hooks which engage the blocks on the box couplings.

It should be noted that it is not theoretically possible for the Coalbadger to exert enough torque or thrust to fail the auger string. It is possible, however, that a roof squeeze or fall could sever the auger.

3.2.5 Airbox and Drive Chuck

To match the drive shaft of the standard Coalbadger (which is made for augers with no provisions for ventilation air passages) with the new hollow-stem augers, it was necessary to fabricate an adapter-type drive chuck. It was also necessary to provide an airtight air coupling which would allow the auger string to rotate while the vacuum is applied through fixed tubing. A special airbox was designed to provide this feature. The assembly is shown in Figure 15 and a diagrammatic sketch of its operation is shown in Figure 16.

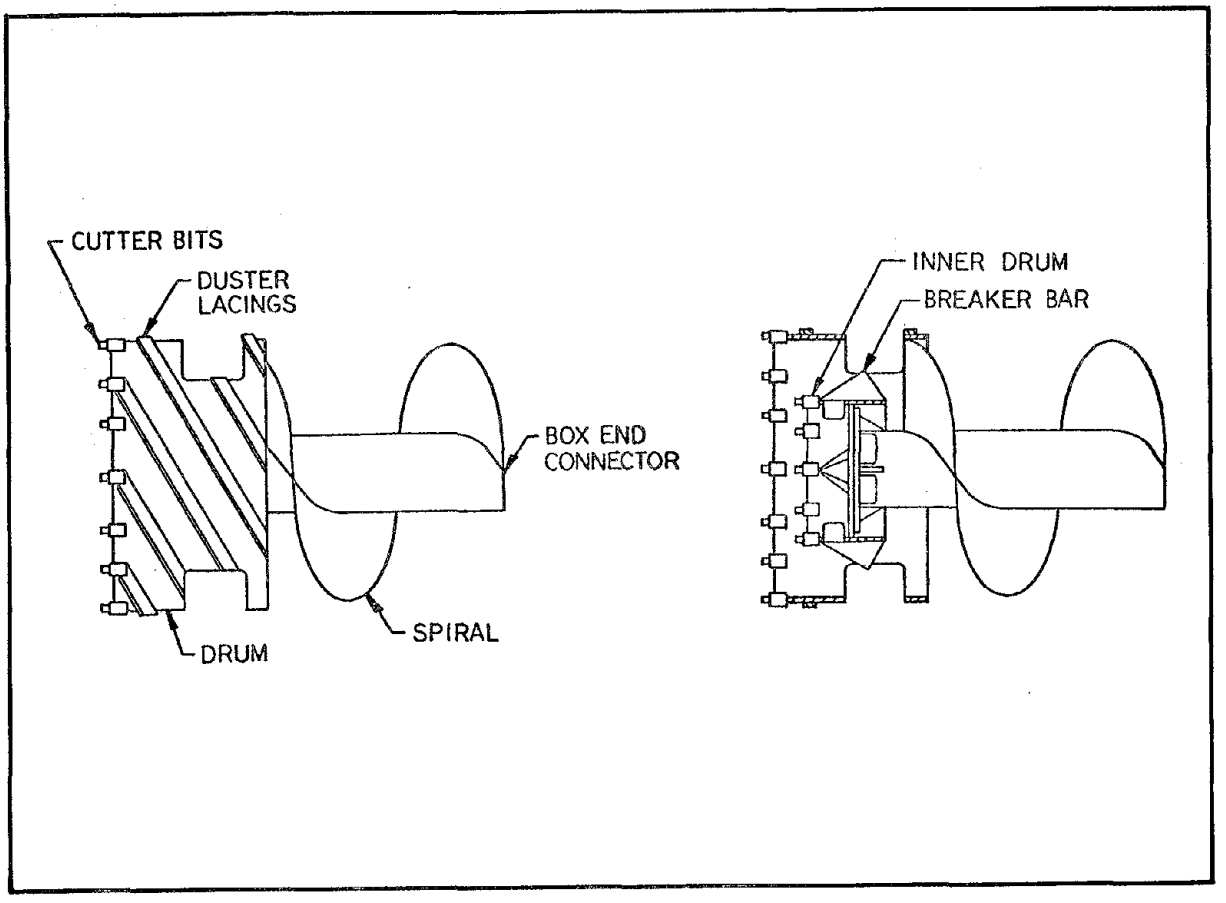
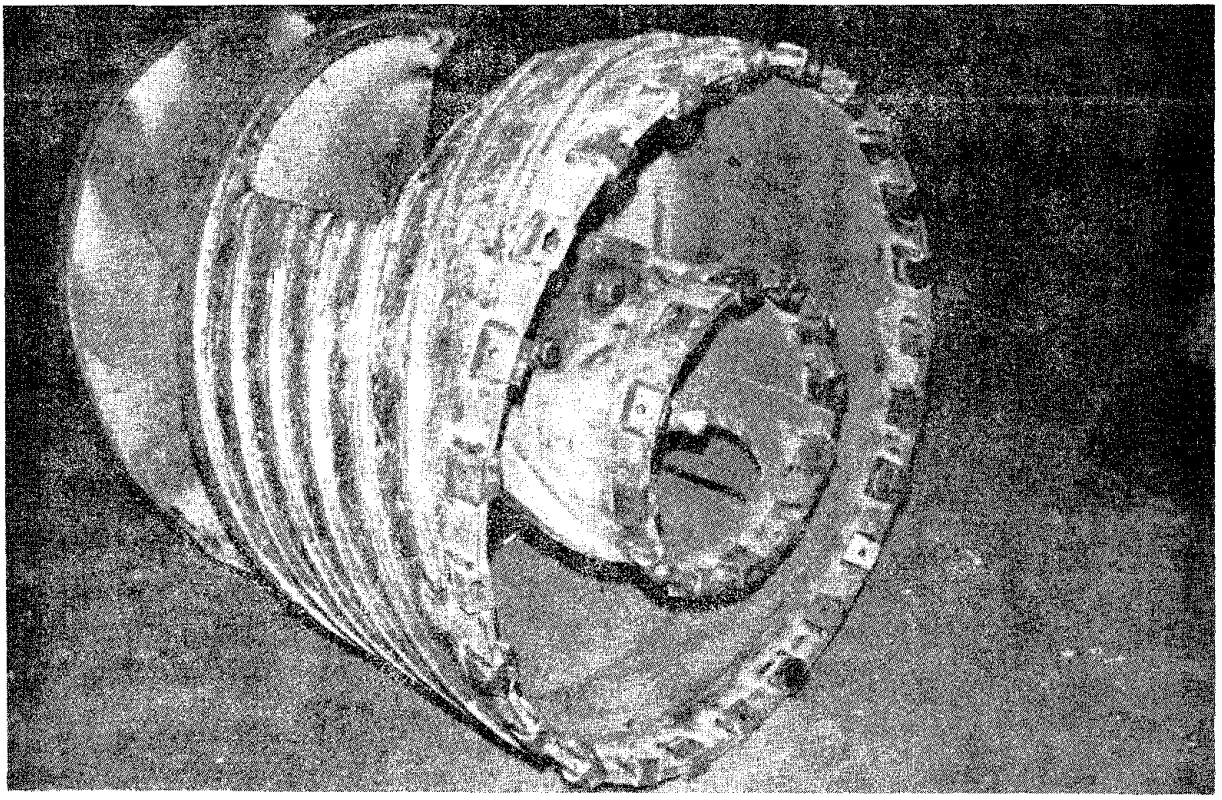


Figure 12 AUGER CUTTERHEAD

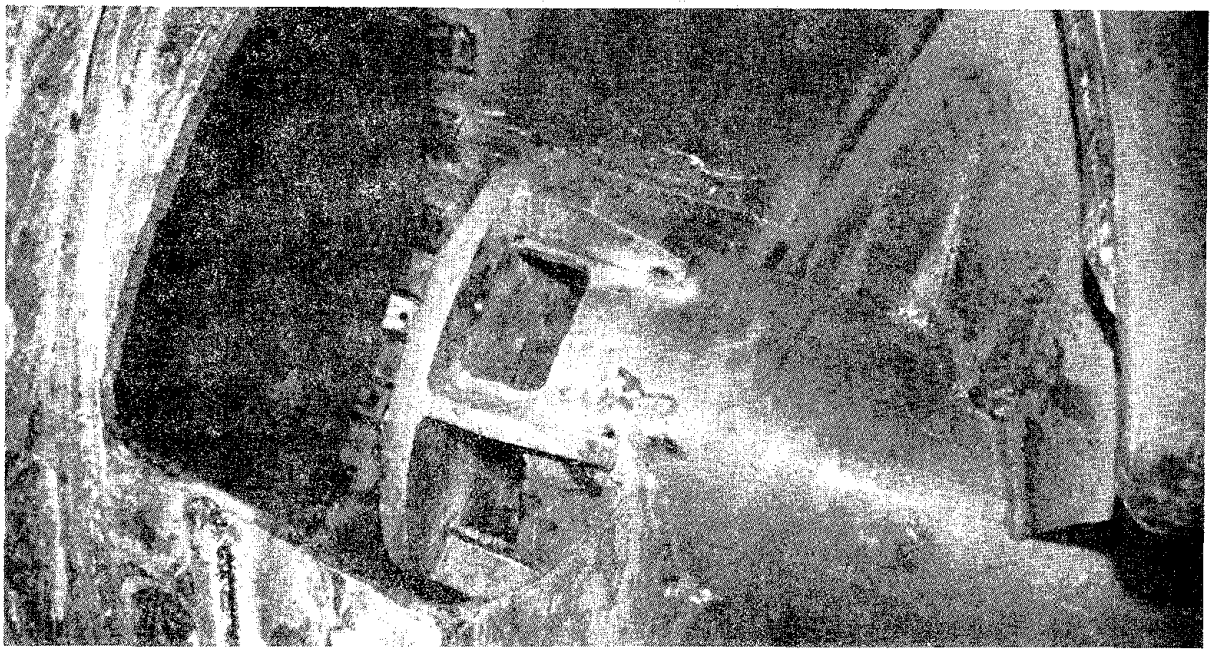
In addition to providing a nonrotating air conduit, the airbox is used as a sampling point for the methane monitoring systems on the Coalbadger as shown diagrammatically in Figure 17.

Two pipe taps are placed in the airbox at locations where there is a calculated pressure differential of approximately 1/2 inch of water. These taps are connected by hose to a small methane sampling box in which is mounted the detector head of the methane monitor/shutdown system on the Coalbadger. A sample of the air entering the airbox is naturally aspirated through the sampling box and the methane content of the exhaust air is continuously monitored.

To satisfy the requirement for 20-minute methane checks, a third tap is made in the airbox and a sample is drawn back to a hand-held methanometer in the operator's compartment. Because the air sample is being drawn from the back of the borehole, the provisions of the law are met.



A. Completed Cutterhead With Breaker Bar Attached



B. View Through Cutout in Drum With Breaker Bar Removed, Showing Ventilation Air Passages

Figure 13 AUGER CUTTERHEAD AS FABRICATED FOR FMC BY W&S WELDING, INC.

In the event that the methane content of the exhaust air exceeds 1 percent, the methane monitor will alarm. Air containing 1 percent or more methane cannot be passed through an auxiliary fan; therefore, the dilution port on the air box (Figures 14 and 15) must be opened sufficiently to allow enough fresh entry air to dilute the air coming from the borehole. When the methane level drops, the dilution air port can be closed and augering can resume.

3.2.6 Flexible Ventilation Tubing

As can also be seen in Figure 14, the connection from the airbox to the permanent tubing (if used) or directly to the auxiliary fan is by means of spiral reinforced flexible fabric tubing. Because of the high vacuum pressure differentials expected, a special configuration of flexible tubing was provided by the Flexaust Company to FMC's specifications. The material is Flexaust's Type SD which conforms to ASTM E-162, flame spread index less than 25, and thereby met USBM approval. The system included two 25-foot lengths of this flexible exhaust tubing with cuffed ends and quick-release clamps.

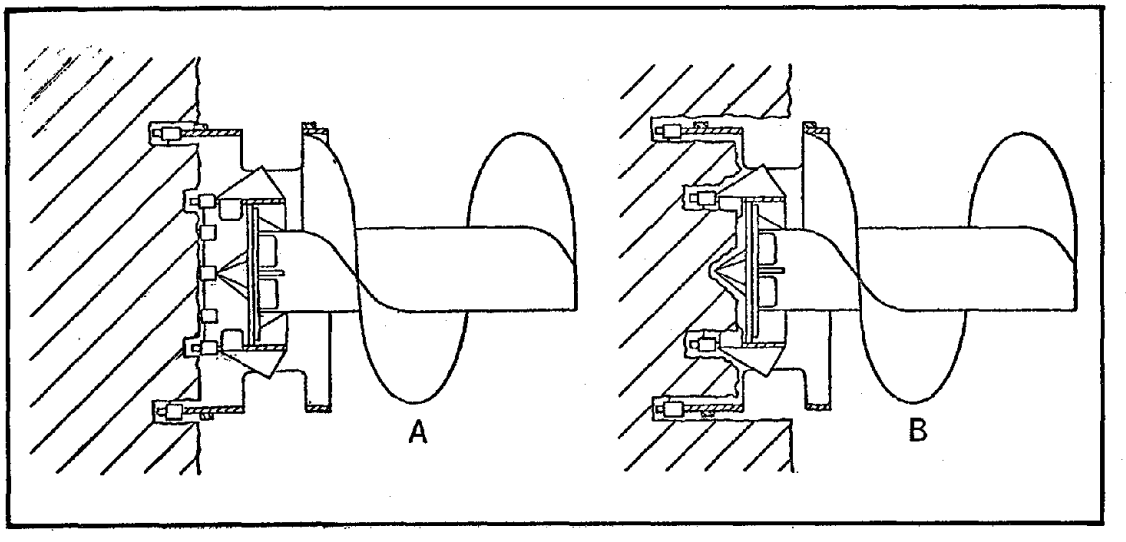


Figure 14 CUTTING AND BREAKING ACTION OF AUGER CUTTERHEAD

3.2.7 Auxiliary Fan

Matching an auxiliary fan to the auger miner is a matter of matching the system air flow curves to the fan curves for various fans. The points of intersection will be the operating points for the system. If the air flow of these operating points is adequate and not excessive, the fan is properly matched.

Determining what air flow is adequate is largely a question of interpreting Federal law and conforming to the requirements imposed by the local MESA district in which the auger miner is operating. This, in turn, is largely a question of determining the hazard to be expected in terms of methane content of the coal, presence of inclusions in the coal, type and speed of cutters, use of water sprays, etc.

Table 4 AUGER SPECIFICATIONS

Cutterhead	Diameter	42 inches
	Length (overall)	60 inches
	Cutterbits	Rectangular shank, side cutting, carbide tipped
	Bits on drum	24
	Bits on breaker bar	12
	Weight	1,000 pounds (approximately)
Spiral Flights	Diameter	40 inches
	Length (overall)	68 inches
	Length (coupled)	60 inches
	Spiral	5/16 plate
	Pitch	30 inches
	Weight	600 pounds (approximately)

This has been a topic of much discussion and many calculations for the duration of the program. Several of these calculations which sought to predict or characterize methane hazard at the demonstration site are presented in Appendix A. The most conservative of these studies indicated that an air flow of 775 cfm would be adequate to keep the exhaust air methane content below 1 percent by volume.

Figure 18 is a curve which shows auger system air flow curves for auger penetrations of 0, 10, 60, and 100 feet respectively. Plotted to the same scale is the performance curve for the special high-pressure, low-volume centrifugal fan which completes the borehole ventilation system. This fan was custom assembled by CSE Company to FMC specifications.

Points of intersection are the aforementioned operating points. As can be seen from the curve, even with full 100-foot auger penetration, this fan will provide approximately 900 cfm which is in excess of the 775 cfm believed to be required for safety. Air flows for shallower penetrations are, of course, correspondingly greater.

Based on these calculations, MESA[1] and the West Virginia Department of Mines approved this ventilation plan subject to experimental verification.

3.2.8 Rockduster and In-Hole Rockdusting Tubes

Because in some mine plans, especially with blind auger holes, it was necessary to rockdust the inside surface of the boreholes, a method of introducing the rockdust into these deep holes was required. The surface area of the holes is small compared to the mine entries; therefore, a small mobile

1. The Mining Enforcement and Safety Administration (MESA) was part of the U.S. Department of Interior until March 9, 1978 when reorganization placed it under the U.S. Department of Labor, retitled Mine Safety and Health Administration (MSHA). References to MESA and MSHA throughout this report indicate the agency in authority at the time.

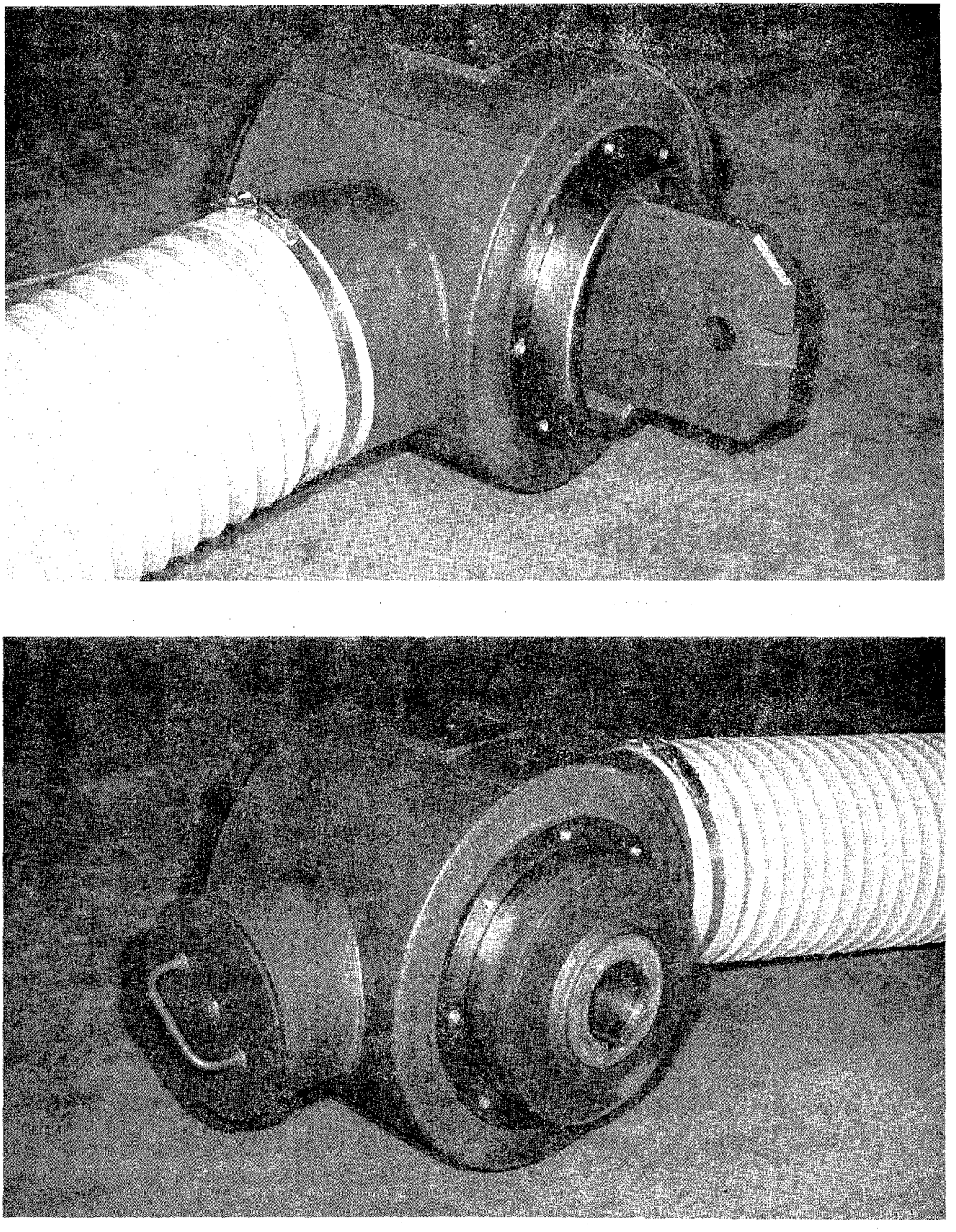


Figure 15 AUGER DRIVE CHUCK WITH AIRBOX IN PLACE

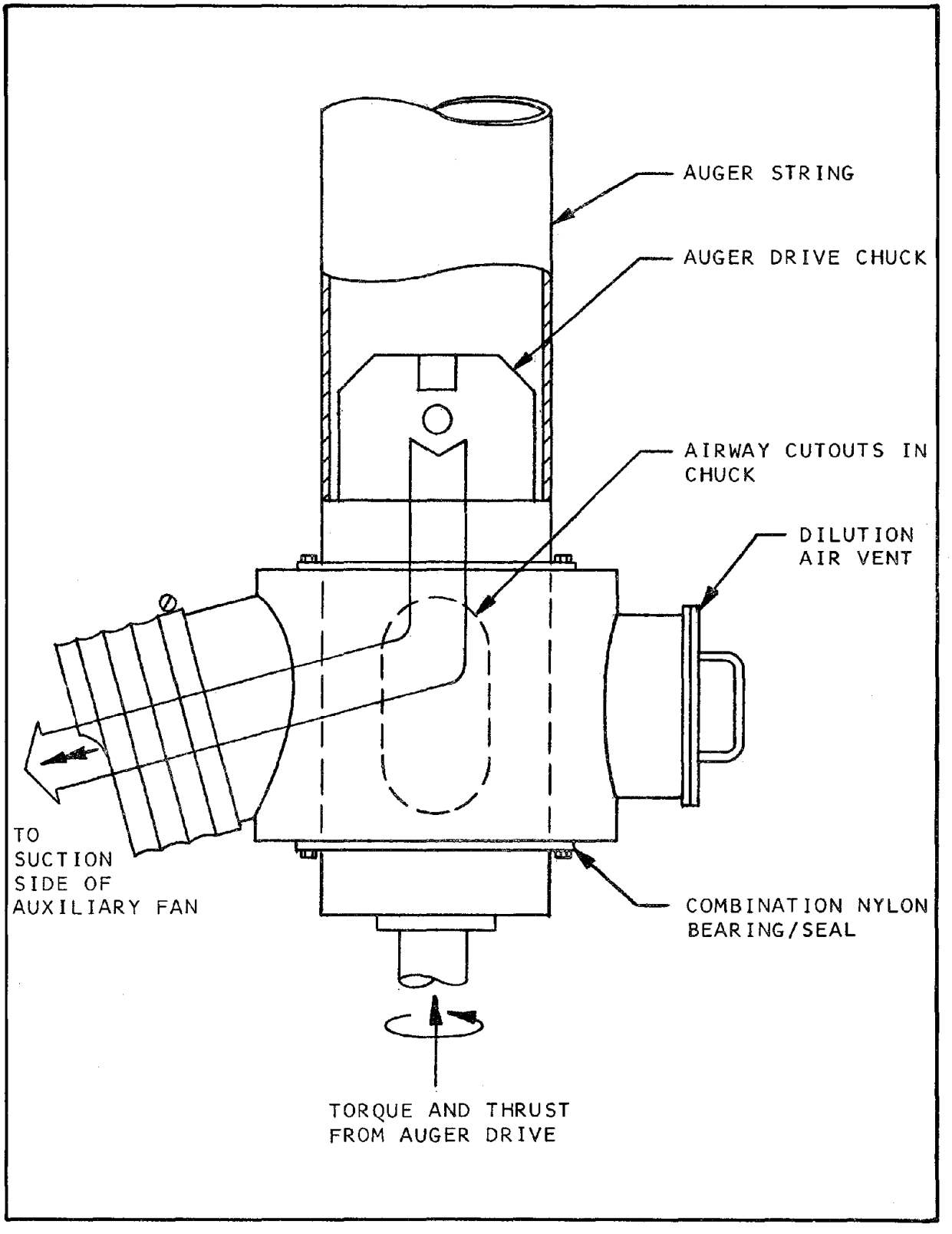


Figure 16 AIR FLOW THROUGH AUGER CHUCK AND AIRBOX TO EXTERNAL VENTILATION TUBING

3.3 SYSTEM OPERATION

3.3.1 Equipment Setup

In general, large production equipment such as miners, loaders, shuttle cars, etc., are moved underground partially disassembled. The same was true of the auger miner system. For moving, the sub-auger conveyor had to be removed from the front frame. Depending on clearance in the haulageway, the canopy and the main hydraulic drive motor may be lowered or removed. If brought in by rail or a low flatcar, the wheel set may be removed for additional vertical clearance. The same option is true for the entry conveyor because its wheel set was also removable. All other system components, including the individual auger cutterheads and flights, were small enough to be handled on standard supply cars.

Once individual system components were offloaded, partial reassembly was accomplished. The wheeled components were towed to the intersection of the entry in which they were to be operated. The exact order of installation in the entry depended on the configuration of the panels or pillars. In general, the flights and cutterheads were brought to the back of the entry and stored in the last crosscut. The best vehicle for moving augers was a scoop/tractor, but the augers could be rolled over fairly level dry bottom by two people.

Next the auger miner was brought in. The wheel set had to be removed at the last intersection because wheel travel direction was at right angles to the sidesliding frames of the machine. The Coalbadger with all jacks retracted could be skidded on its main frame by a scoop/tractor. As an option, power could be applied to the machine and it could be "walked" on its jacks into the entry (this walking operation is discussed in Section 3.3.2). The positioning of the frame of the auger miner at the starting point was critical and was best accomplished by using a string line to obtain a reference along the rib to be drilled as shown in Figure 19.

Because rib surfaces were always irregular and pillar lines are not always straight, some judgment had to be exercised in setting the limits of the string line. In general, the line would not allow the gap between the auger trough and the rib to exceed 1 foot. Minor rib projections could be removed by pick and shovel.

The string line when completed was used to set the machine's frame square to the rib and at the standoff distance. The sub-auger conveyor could then be attached and the hydraulics connected. The entry conveyor was wheeled into position under the discharge end of the sub-auger conveyor and angled away from the rib out to the center of the entry, and the hydraulic lines were connected. One by one the auger sections were loaded onto the platform as the first hole was drilled. A hydraulic jib crane accessory attached to a forward roof jack was used to lift the augers. The rockduster could be placed wherever it was convenient and the flexible hose could be used to reach the auger hole. In some mine plans, depending on air flow, the holes may be rockdusted from the far side of the panel. The auger miner system in a typical setup for operation is shown diagrammatically in plan view in Figure 20.

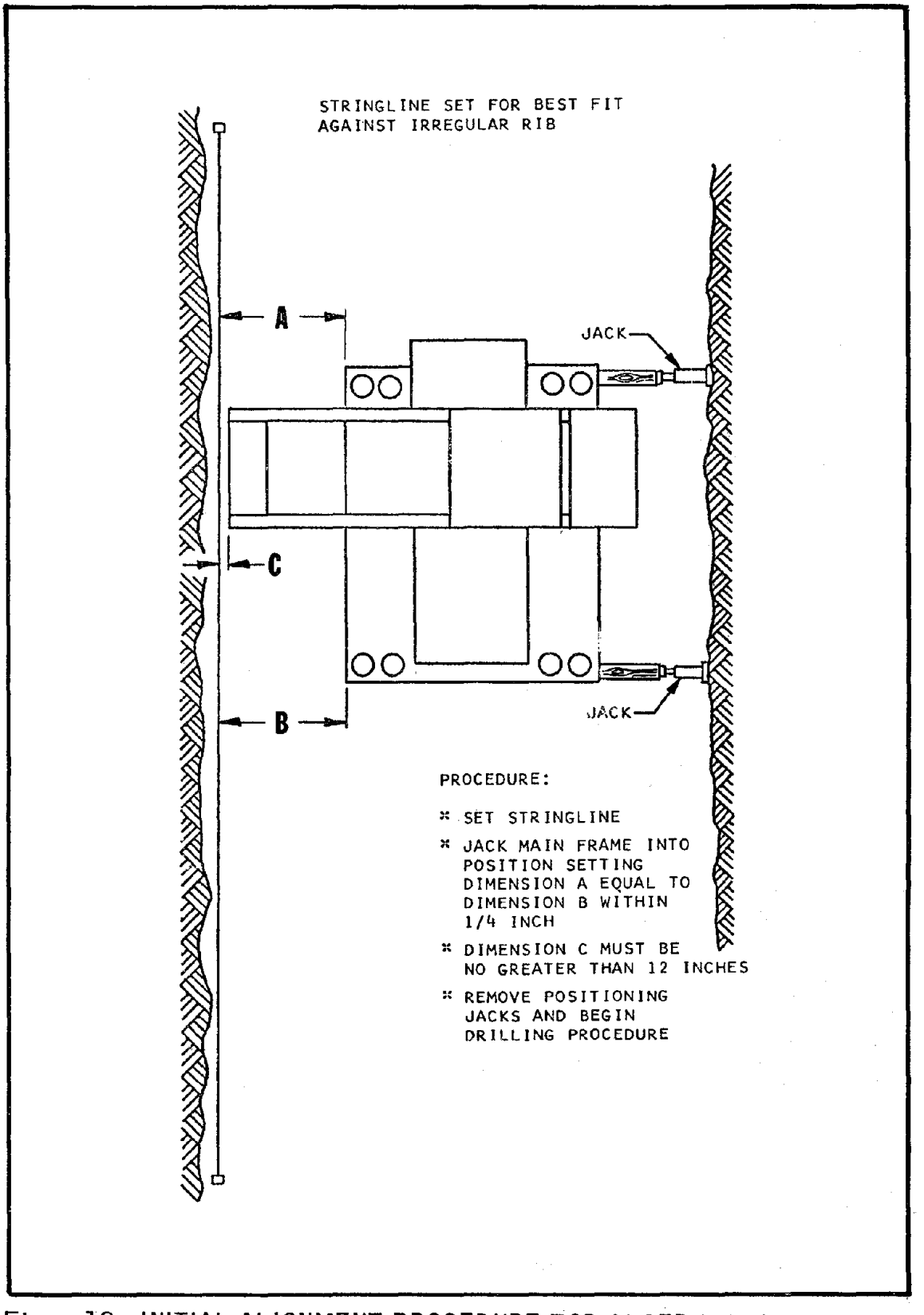


Figure 19 INITIAL ALIGNMENT PROCEDURE FOR AUGER MINER

3.3.2 Borehole Drilling

In operation, the auger miner could drill two parallel holes from one setup.

It drilled the first hole, loaded a second cutterhead, and drilled the second hole by retrieving auger sections from the first hole. For the third and subsequent holes, it was necessary to shift the main frame of the auger. This procedure is shown in Figure 21 and the following explanatory steps are keyed to this figure. Note that parallelism of borehole axes is always guaranteed because the main frame and sump frame always move parallel to the rib line along their guide rails, and the sump carriage always moves perpendicular to the sump frame.

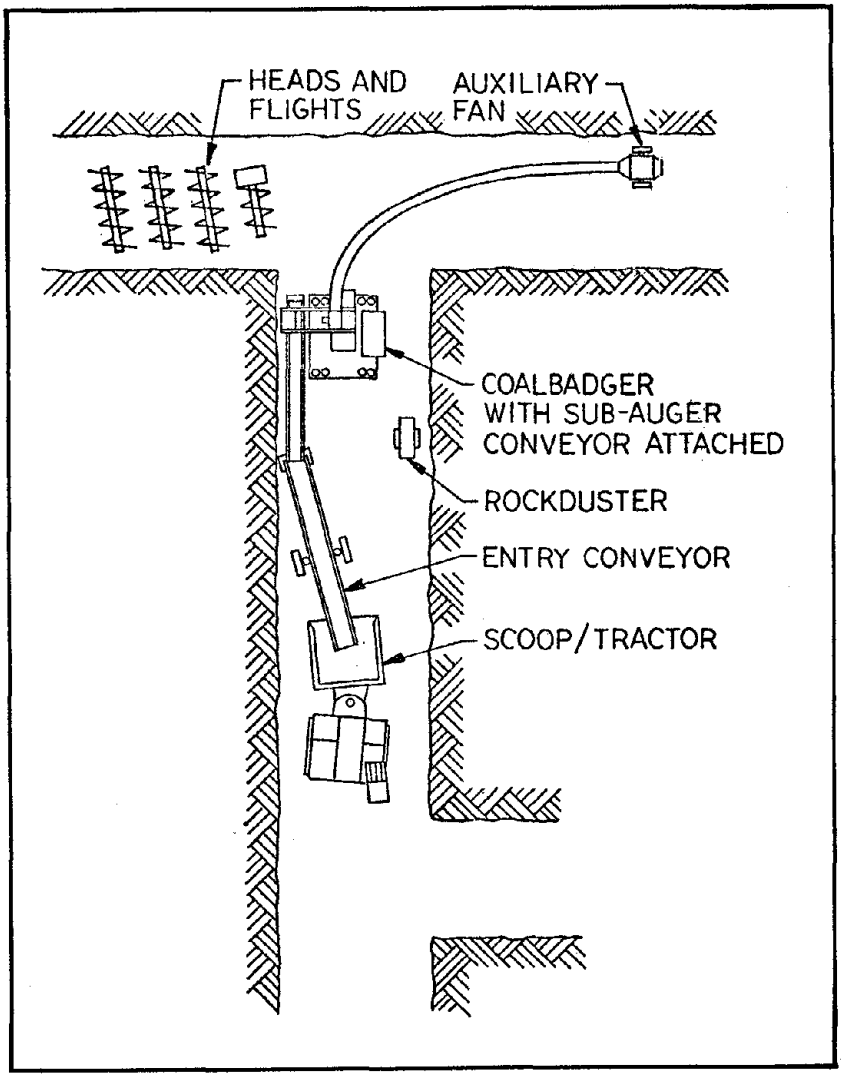


Figure 20 AUGER MINER SYSTEM IN TYPICAL POSITION TO BEGIN OPERATION

Referring to Figure 21A, the machine was moved into position and the floor jacks were adjusted to put the machine at the correct attitude. Roof jacks were then extended to lock the machine between roof and floor. Carriage traverse was then used to drill the first two holes.

To shift the machine main frame, the bump jacks were lowered until all had contacted the floor (Figure 21B). The roof jacks were then lowered and the floor jacks were raised (Figure 21C). The machine was now totally supported on the bump jacks. The traverse cylinders were operated to shift the machine main frame to the side (Figure 21D). After the roof jacks were raised, the floor jacks were lowered (Figure 21E). The bump jacks could now be raised and drilling could continue as shown in Figure 21F. The machine

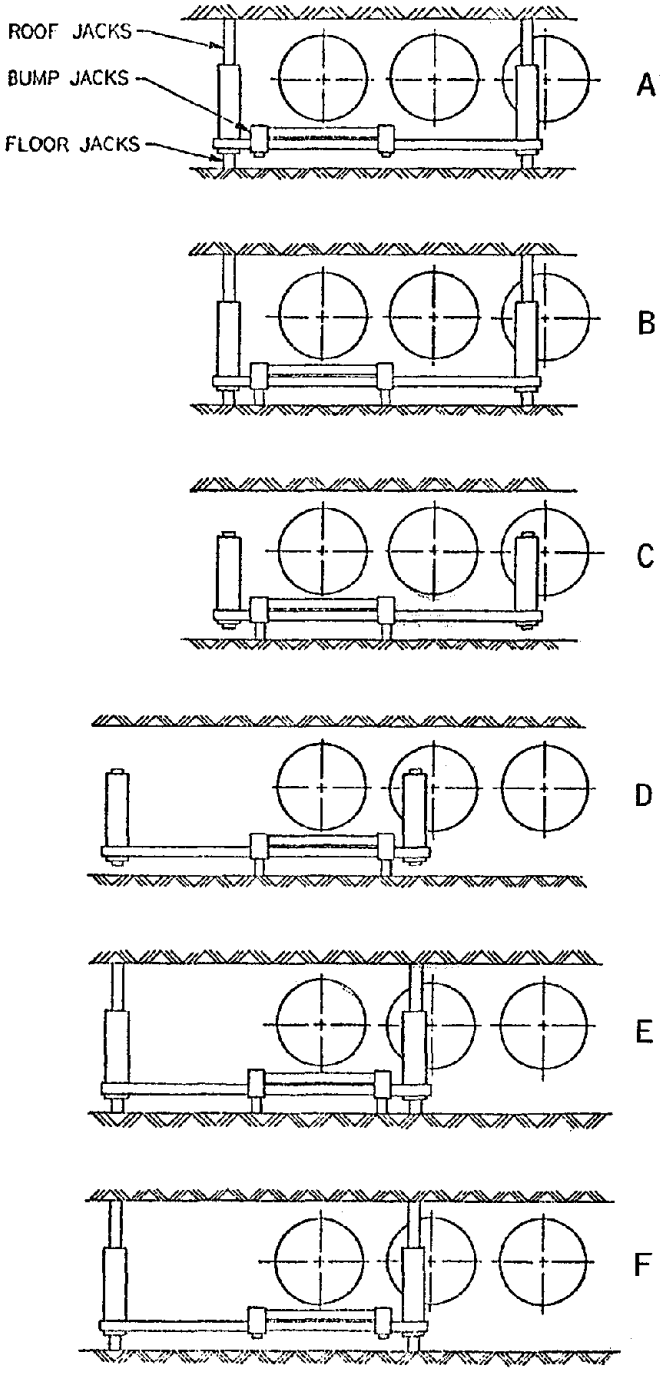


Figure 21 SEQUENCE FOR SHIFTING AUGER MINER LATERALLY FOR BOREHOLE SPACING

then moved one hole spacing and remained not only parallel to the previous holes but also horizon (i.e., horizontal centerline) to the previous holes.

A synopsis of the side moving sequence, which must be followed exactly to maintain alignment, is as follows:

Initially:

1. Adjust attitude of frame with individual floor jacks.
2. Raise roof jacks.
3. Operate traverse and sump to drill first two holes.

To move:

1. Lower bump jacks.
2. Lower roof jacks.
3. Raise floor jacks.
4. Shift frame to side.
5. Raise roof jacks.
6. Lower floor jacks.
7. Raise bump jacks.
8. Return to traverse/drill sequence.

When a panel was completed, the system was disassembled in reverse order of assembly and moved to the next location by towing and/or skidding depending on the distance of the move.

3.3.3 Auger Guidance

While drilling, the auger operator relied on the rigidity of the auger string and the self-guidance capability of the cutterhead to control the straightness of the borehole in the horizontal direction. There was no external control on left and right auger deflections. In the vertical direction, gravity tended to drive the cutterhead down until it contacted the bottom material. More often than not, the bottom was approximately the cutting consistency of the coal, while the roof material was generally harder. By a combination of "feel" and observation of the product being removed from the borehole, a skilled operator could tell approximately where the cutterhead was.

If the head dipped into the bottom, it could sometimes be brought out again by slowing the sump, increasing the auger rotational speed, and allowing the cutterhead to "climb" up the sidewall of the hole. This procedure is successfully used in highwall augering.

Occasionally, the bottom was hard or locally pushed up and caused the head to rise into the hard roof. This was usually noticed as the auger began to slow down or stall. To drop the head, the operator had to slow or even stop the sump so that the side-cutting action of the head could elongate the borehole downward. After a sufficient pause, the forward sump could resume.

Unfortunately, when a cutterhead did wander into harder material, the additional resistance acting on the rim of the cutterhead tended to produce a torque which rotated the head axially and redirected it into -- not away from -- the harder material. Thus, once a guidance misdirection was encountered, it was self-perpetuating and could end in an aborted borehole.

3.4 SUPPLYING THE AUGER MINER SYSTEM

The auger miner system was supplied with the following utilities and consumables:

• Electric power	440 volts AC, three phase, 250 KVA
• Water	10 gpm at 30 psi minimum
• Rockdust	400 to 500 pounds per 100-foot hole
• Cutterbits	Average two changes per shift (36 bits per change)
• Hydraulic and lubricating oil and grease	as required.

Note that there were normally no roof support supplies required because the system operated from previously bolted entries established during panel development. Supplies would generally be carried to the machine on the same scoop or shuttle car which provided the face haulage of the coal.

3.5 DEMONSTRATION SITE SELECTION

From the beginning of the program, FMC endeavored to find a demonstration site for the auger miner system. Some criteria set forth for the mine itself were:

- 56- to 72-inch seam height
- Dry bottom
- Reasonably competent roof
- Minimum 18-foot-wide entries
- Freedom from major rock inclusions in the coal.

The qualities were observable and measurable. A less tangible criterion was the cooperation and interest shown by mine management -- equally as important for a successful demonstration.

Numerous mines were contacted in Kentucky, West Virginia, Tennessee, Illinois, and Utah. Many mines were found unsuitable for various reasons and were eliminated from consideration. The management of a number of suitable mines declined to participate for various reasons such as:

- Too busy with production work
- Did not care for this style of mining
- Did not wish to become involved in a government-sponsored program
- Mine interested but lease agreement provided penalty for low recovery.

Several false leads were followed only to have discussions ended by the normal uncertainties caused by the fluctuating and often depressed coal market.

Ultimately two coal companies emerged as viable demonstration partners. FMC personnel visited their various properties, made physical measurements, talked to both the mine management and the UMWA labor force, and made an honest appraisal of the overall attributes of both companies.

We recommended that the auger miner system be demonstrated at the Ireland Mine of Consolidation Coal Company near Moundsville, West Virginia. Meetings were held at Ireland Mine between FMC and USBM project management and representatives of Ireland Mine Management, local MESA district representatives, local West Virginia State Department of Mines personnel, and members of the UMWA Safety Committee at Ireland Mine. At these meetings the general configuration of the system elements was shown, and the operating procedures were discussed. The general tone of these meetings was very positive with many of the attendees making suggestions or constructive criticism. All comments were noted and a number of suggestions were acted on. At the conclusion of these meetings, FMC petitioned the USBM to approve Ireland Mine as the demonstration site for the Phase II portion of the contract. This was done and a demonstration agreement was drafted between FMC and Ireland Mine.

3.5.1 Test Area

Ireland Mine management identified several areas in the mine where the auger miner system could be set up and demonstrated with a minimum of disruption to the balance of the mine. Personal visits, telephone conversations, and correspondence between FMC and Ireland Mine personnel resulted in the final selection of a test area and the adoption of a test plan for submittal to MESA and the West Virginia Department of Mines.

Figure 22 shows the test area in which the auger miner demonstration was held. The 1 Right, 8 North, and 2 Right, 8 North, sections were approximately 1-1/2 miles from the Ireland Mine's Rosby Rocks Portal which is a shaft entry east of the main slope entry. Three panels were developed in 1 Right, 8 North, as shown in in Figure 23. The dimensions of these panels (feet) were 50 by 200, 75 by by 200, and 100 by 200. It was intended that once the first three experimental panels were mined out, additional panels of optimum depth

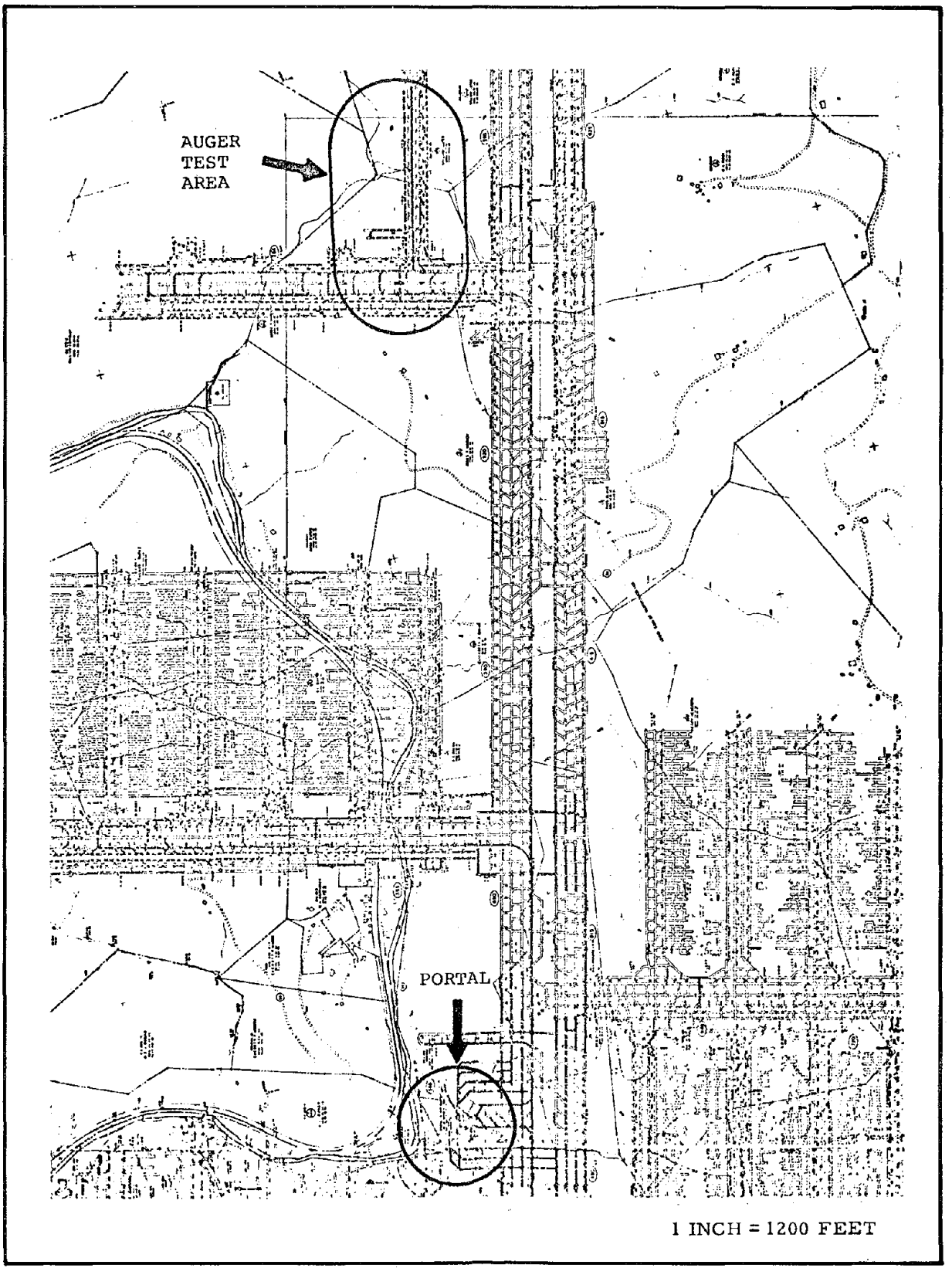


Figure 22 IRELAND MINE 1 RIGHT, 8 NORTH, TEST AREA

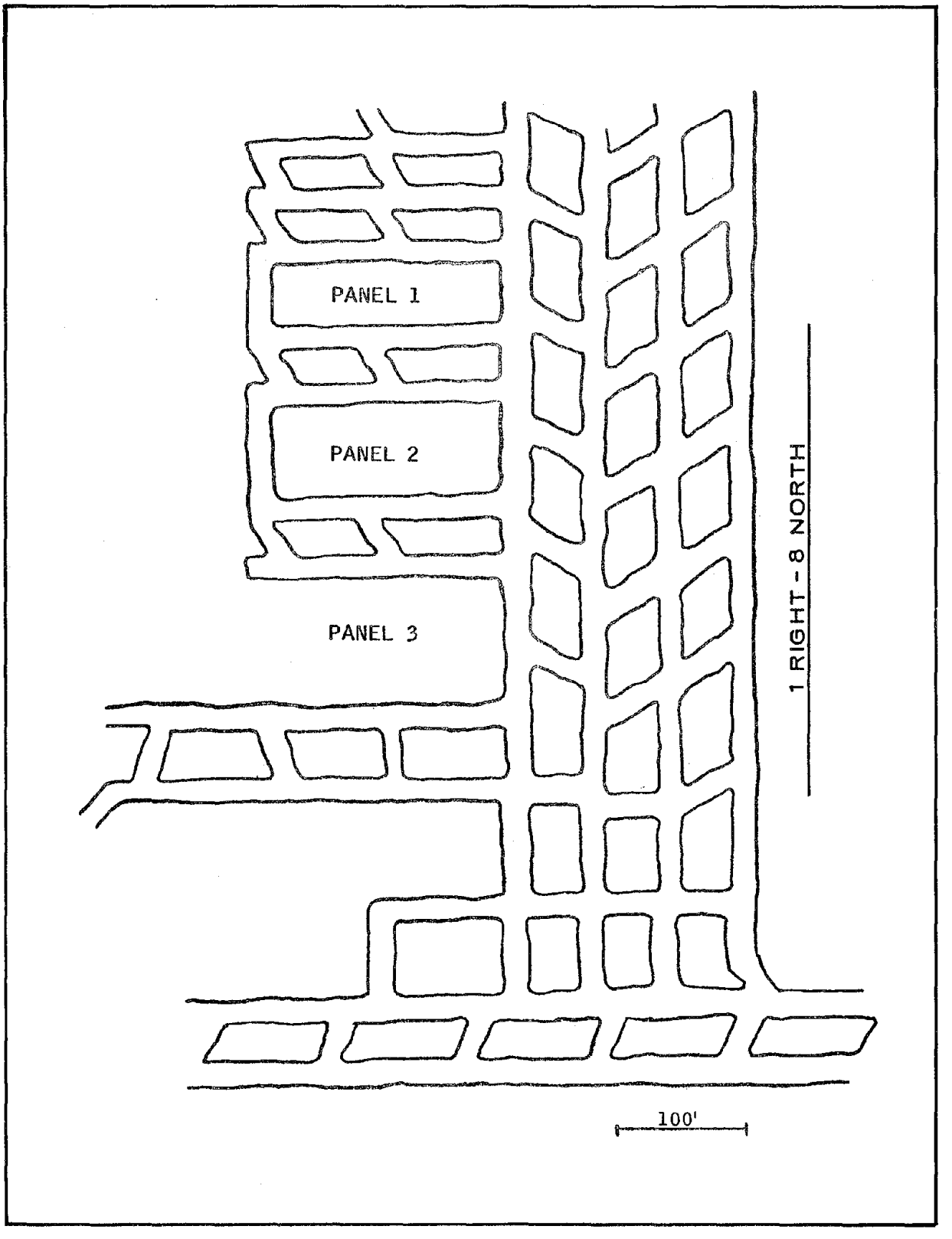


Figure 23 FIRST THREE PANELS PREPARED FOR AUGER MINER IN 1 RIGHT, 8 NORTH, AREA

would be prepared in 2 Right, 8 North. However, by the end of the demonstration, only the 1 Right, 8 North, panels had been mined. Besides preparing these special panels, Ireland Mine widened the entries around the panels to 18-1/2 feet minimum to allow clearance for the machinery.

In retreat mining such as this, it is Ireland's practice to drive entries and crosscuts 15-1/2 feet in width with a two-bolt pattern. This allows good roof support during the retreat operation with caving being typical after the area has been abandoned.

Because this demonstration was to be too lengthy for this normal roof support, all entries surrounding the panels were center bolted, the three row bolts passing through a 12-foot-long 2-by-12 timber beam. Rib bolts were also added between normal rows approximately 2 feet from either rib.

As previously mentioned, Ireland Mine extracts coal from the Pittsburgh Number 8 seam, a seam of wide expanse occurring in Ohio, West Virginia, Pennsylvania, and Maryland. The seam ranges in thickness from about 2 to 10 feet; the average thickness is 5 to 6 feet. The coal is typically inter-bedded with bands of soft shale and harder veins of clay and shale. Often thin rider seams of coal are found above the main seam. There are also inclusions of pyritic sulphur. Pittsburgh Number 8 coal tends to be firm and somewhat blocky with a Hardgrove Grind Index of 56 to 57. It is low in moisture (2 to 3 percent), moderately low in sulphur (3 to 5 percent), and therefore is used mainly as steam coal.

That portion of Ireland's 1 Right, 8 North, demonstration site location is characteristic of the above description. The local seam height is about 5 feet with a soft shale band 2 to 12 inches thick above the main seam and a 2- to-12-inch-thick rider coal seam above the main seam. In practice, both the main seam, shale band, and rider seam are taken by the continuous miners making the roof height in this area about 6 to 7 feet average. The 1 Right, 8 North, area lies under about 650 feet of cover, making the local pillar pressure somewhat high and caving after retreat mining very common. In addition, both the coal and shale roof are moderately degraded by exposure to moisture in the ventilation air. There is essentially no ground water seepage present, though, leaving the shale/clay bottom dry and suitable for tramming of the auger miner.

3.5.2 Mine Plan

FMC conceived four general mine plans for the auger miner system which included equipment positioning and a general ventilation circuit. These four plans are shown in Figures 24 through 27; descriptions follow.

Figure 24 is an equipment layout and ventilation air flow diagram for coal discharge to the right, outby the auger miner with the auxiliary fan inby.

Figure 25 is an equipment layout and ventilation air flow diagram for coal discharge to the right, outby the auger miner with the auxiliary fan outby.

Figure 26 is an equipment layout and ventilation air flow diagram for coal discharge to the left, outby the auger miner with the auxiliary fan inby.

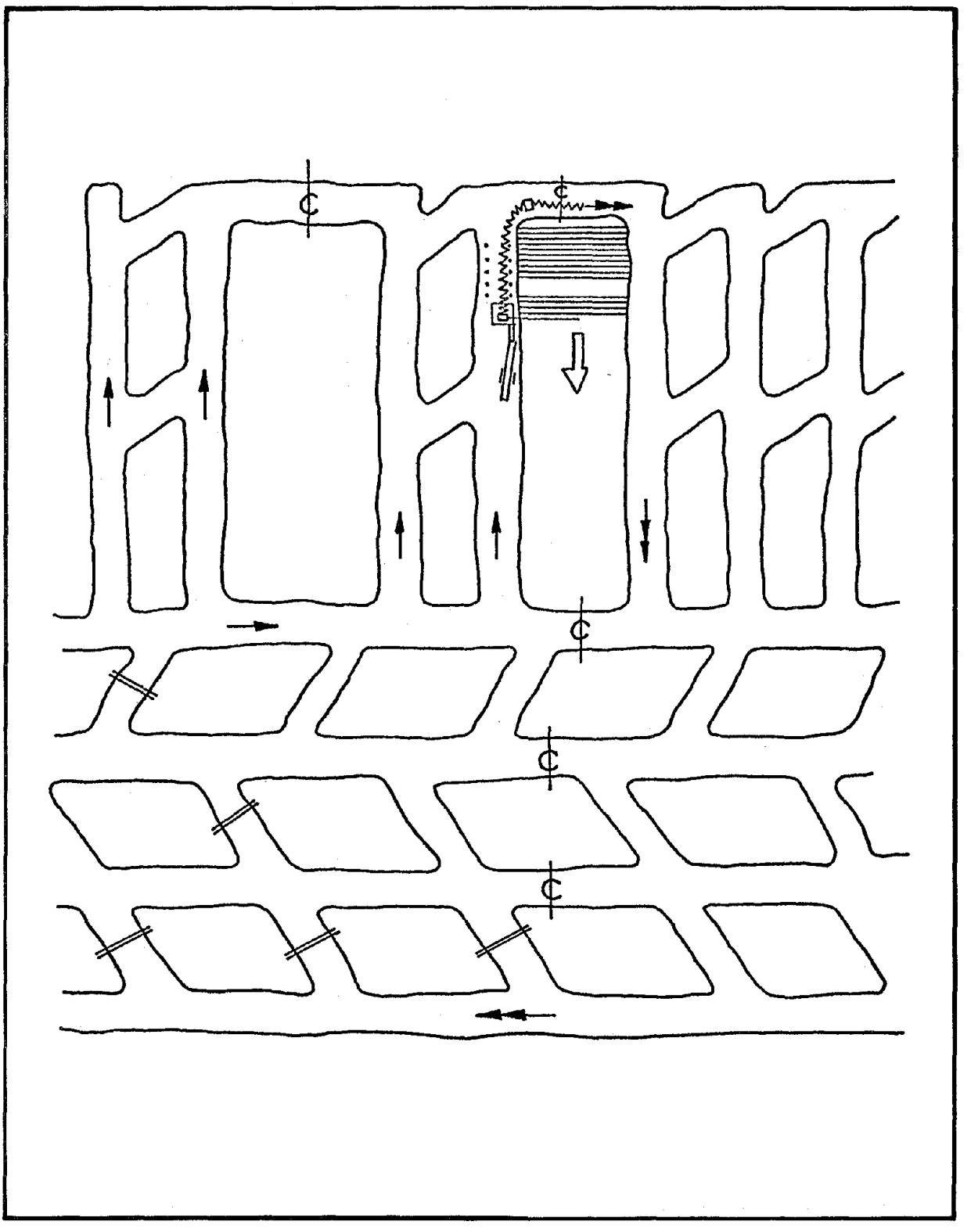


Figure 24 VENTILATION AIR FLOW DIAGRAM FOR COAL DISCHARGE TO THE RIGHT OUTBY THE AUGER MINER WITH THE VENTILATION FAN INBY

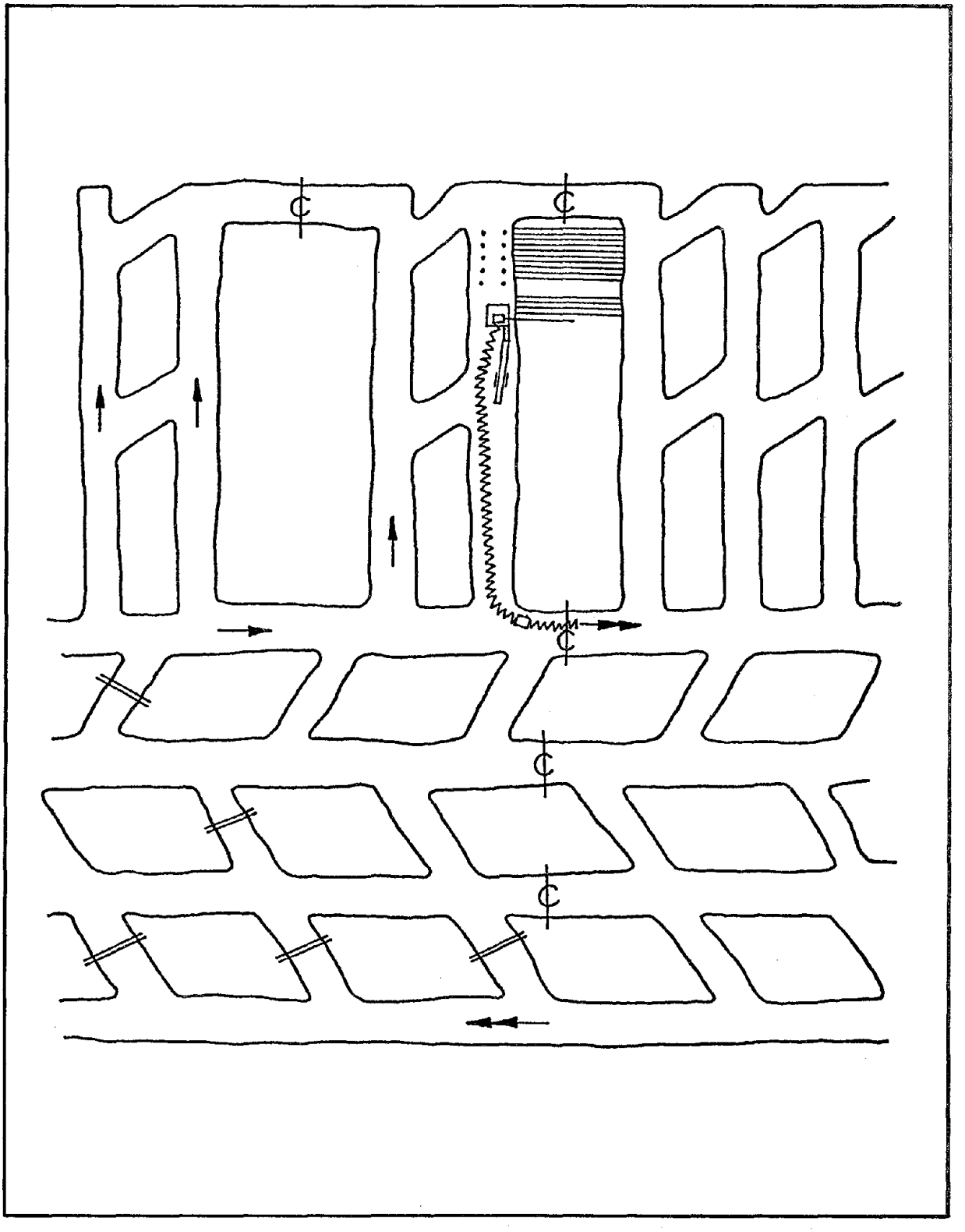


Figure 25 VENTILATION AIR FLOW DIAGRAM FOR COAL DISCHARGE TO THE RIGHT OUTBY THE AUGER MINER WITH THE VENTILATION FAN OUTBY

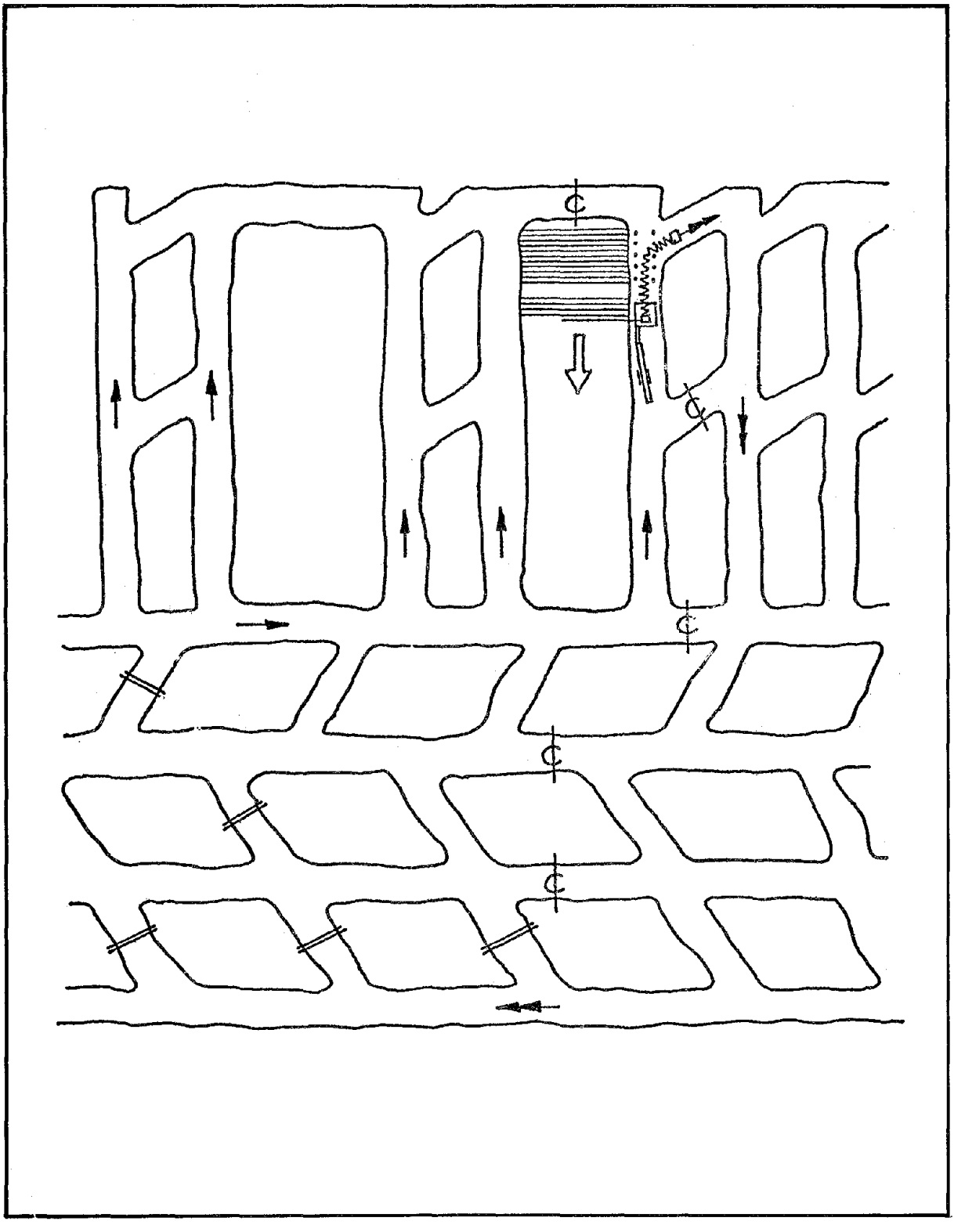


Figure 26 VENTILATION AIR FLOW DIAGRAM FOR COAL DISCHARGE TO THE LEFT OUTBY THE AUGER MINER WITH THE VENTILATION FAN INBY

Figure 27 is an equipment layout and ventilation air flow diagram for coal discharge to the left, outby the auger miner with the auxiliary fan outby.

In discusssions with the local MESA district and West Virginia Department of Mines personnel, it was their opinion that, as the panels were bored, the inby areas should be posted as a danger area and this area should not normally be entered. It was agreed that the postline could lag the auger miner by about 12 to 15 feet so that the auger steels could be positioned, loaded, stored, etc. This precluded placing the ventilation fan in by the miner.

It was Ireland Mine's preference to operate the equipment with the conveyor discharge to the right, outby as shown in Figure 25, and the 50-foot-deep panel was mined according to this plan.

Before the auger demonstration began, a massive roof fall occurred in one entry adjacent to the 75-foot-deep panel as shown in Figure 28. It was originally planned to abandon this panel, but mine management convinced the local MESA and West Virginia Department of Mines inspectors that enough of the fall would be removed by the auger cutterhead so that a ventilation path could be maintained. It was decided to mine the 75- and 100-foot panels using the discharge to the left, outby configuration as shown in Figure 27.

FMC planned a drilling pattern based on using a 42-inch-diameter auger cutterhead with the boreholes spaced on 57-inch centers (i.e., leaving a 15-inch web). From the recovery curve which appears as Appendix B, it could be calculated that the recovery, assuming only the 60-inch main seam is considered, was 41 percent.

If a 40-hole pattern is begun 6 feet from the end of the panel and ends about 5 feet from the opposite end, the net recovery is 38.5 percent. Roof control specialists from Consolidation's parent company, the Continental Oil Company, formulated a drilling plan which was thought to be conservative enough to preclude any catastrophic roof or pillar failure. A discussion and drawing of this drilling pattern appear in Appendix C. Unfortunately, the extra coal left solid in the panel by drilling on this pattern reduced net panel recovery to only 28.9 percent. As discussed in Section IV, this pattern proved to be competent in that no total pillar collapse occurred; however, there were severe squeezes in the first (50-foot) panel. It was hoped that in subsequent testing other drilling patterns could be tried which would yield higher recovery.

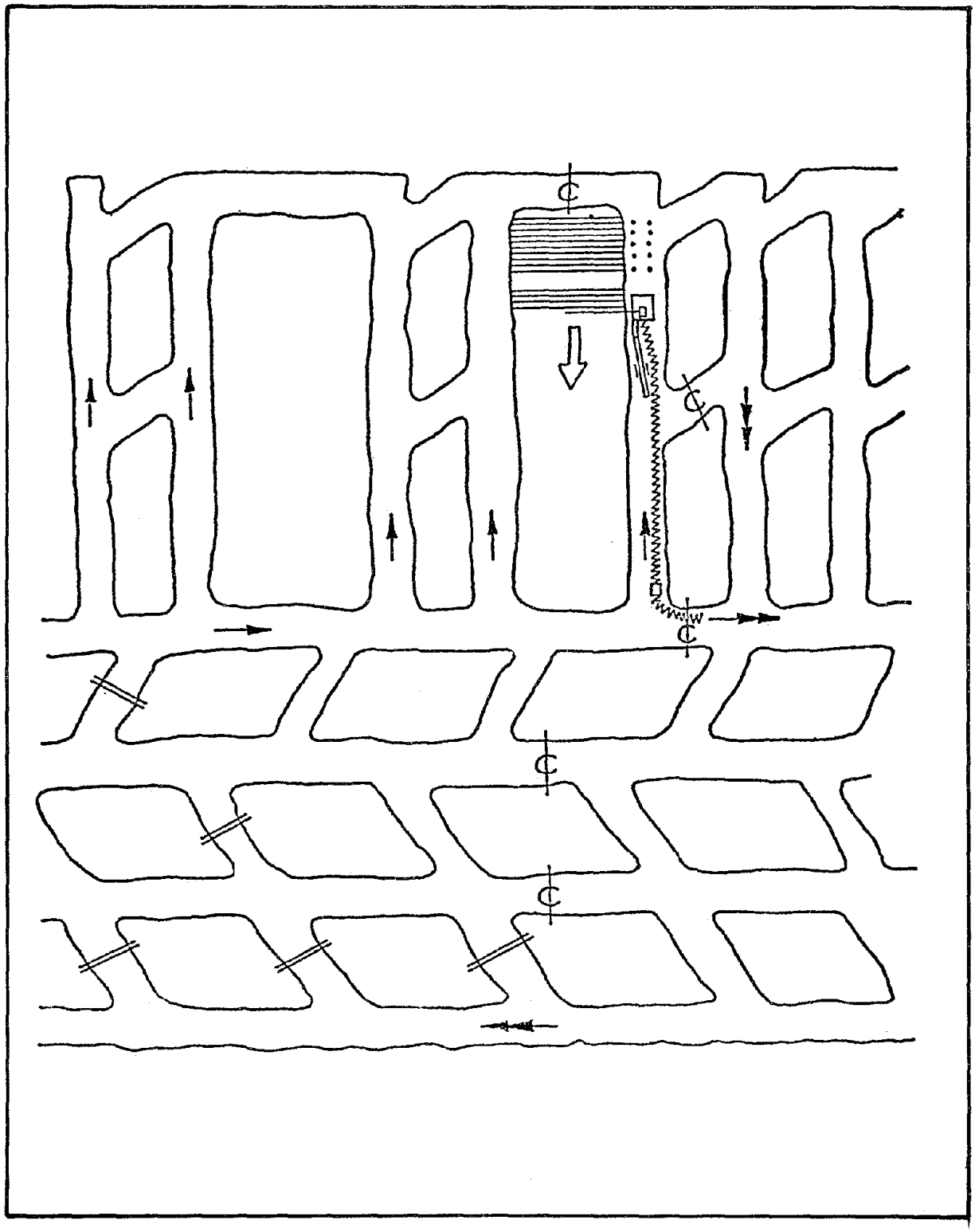


Figure 27 VENTILATION AIR FLOW DIAGRAM FOR COAL DISCHARGE TO THE LEFT OUTBY THE AUGER MINER WITH THE VENTILATION FAN OUTBY

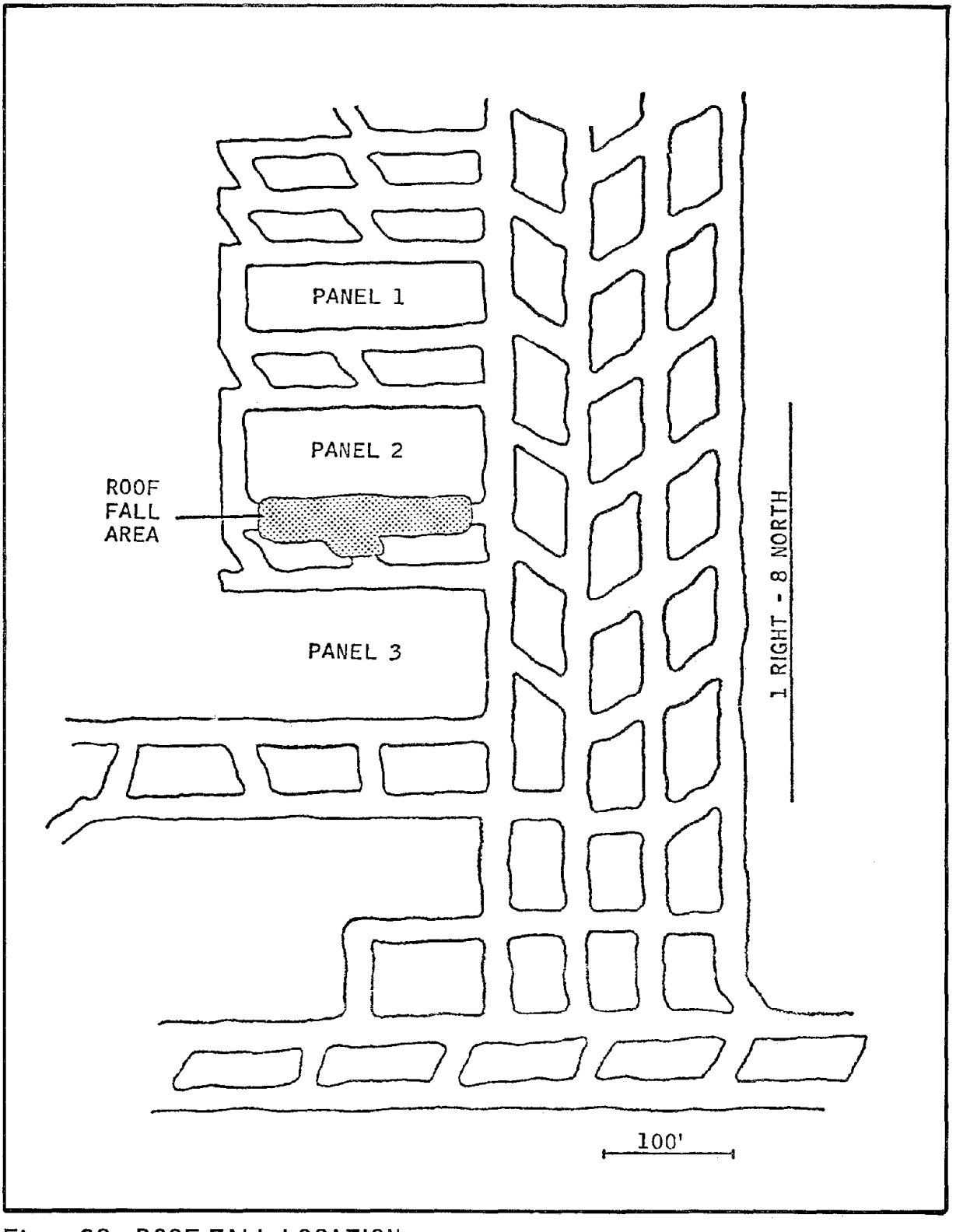


Figure 28 ROOF FALL LOCATION

4.0 DEMONSTRATION RESULTS (Part 1)

4.1 PREPARATION FOR OPERATION

After final modification work had been accomplished in the Ireland Mine main shop and after some above-ground inspection of the equipment by representatives of MESA,[1] the West Virginia Department of Mines, the UMWA Safety Committee, various members of Ireland Mine management, and some UMWA miners who were curious about bidding on the crew, the equipment was loaded on flatcars and taken into the mine.

The offloading and reassembly of the conveyors and auger steels proceeded fairly smoothly as expected. The reassembly of the auger miner itself was very time consuming because the frames had been separated for the move underground, and it was necessary to refit and weld them underground. For the future, knowing that the auger miner must be knocked down and reassembled, certain changes were recommended so that this operation would require only the fitting of bolted assemblies (no welding). Also, many mines could take the auger miner into the mine fully assembled with only the canopy lowered. At Ireland Mine, all equipment enters the mine through a slope entry which happened to be 6-1/2 miles from the 1 Right, 8 North, test area, and the route passed an area of restricted clearance.

The system was fully set up against the 50-foot panel in late April 1976. After inspections by MESA, the West Virginia Department of Mines, the UMWA local, and Ireland Mine management, approvals were given to begin mining. There was some controversy about the procedures for selecting a crew -- whether selection would be by a normal bidding procedure or by assignment. It was encouraging to us that several miners expressed interest in bidding onto the crew. The crew was finally selected by assigning miners on a temporary basis, mostly from volunteers. The first production shift was May 5, 1976, on the 50-foot panel. The last production shift was August 19, 1976.

A minimum of one FMC representative was on hand for the duration of a setup and production work. Logs were maintained by the shift foreman and additional data was collected by the FMC representatives.

Figures 29, 30, and 31 show the auger miner set against the 50-foot panel just prior to beginning production.

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1. The Mining Enforcement and Safety Administration (MESA) was part of the U.S. Department of Interior until March 9, 1978 when reorganization placed it under the U.S. Department of Labor, retitled Mine Safety and Health Administration (MSHA). References to MESA and MSHA throughout this report indicate the agency in authority at the time.

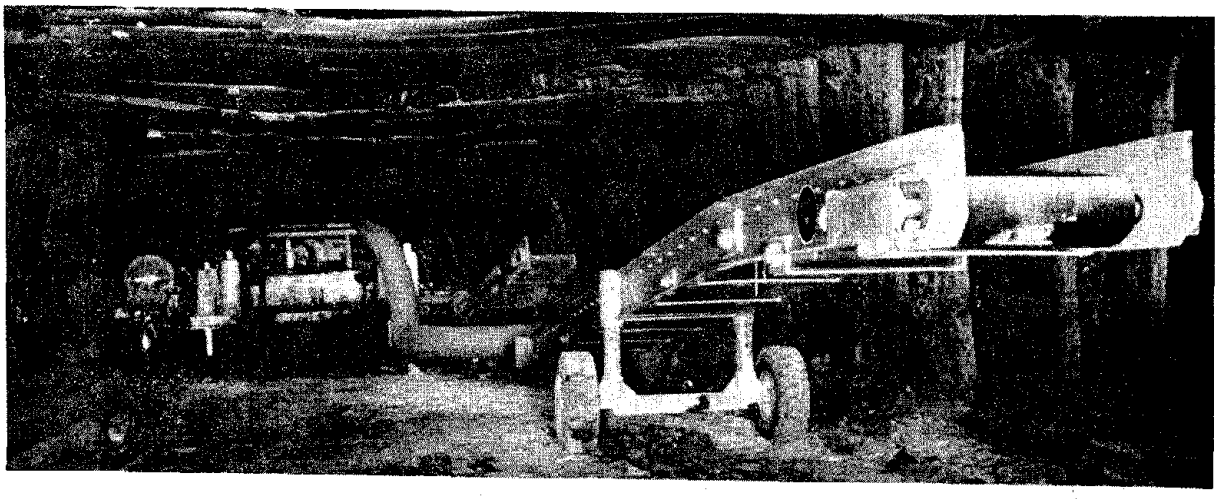


Figure 29 VIEW OF AUGER MINER FROM THE DISCHARGE POINT OF THE ENTRY CONVEYOR

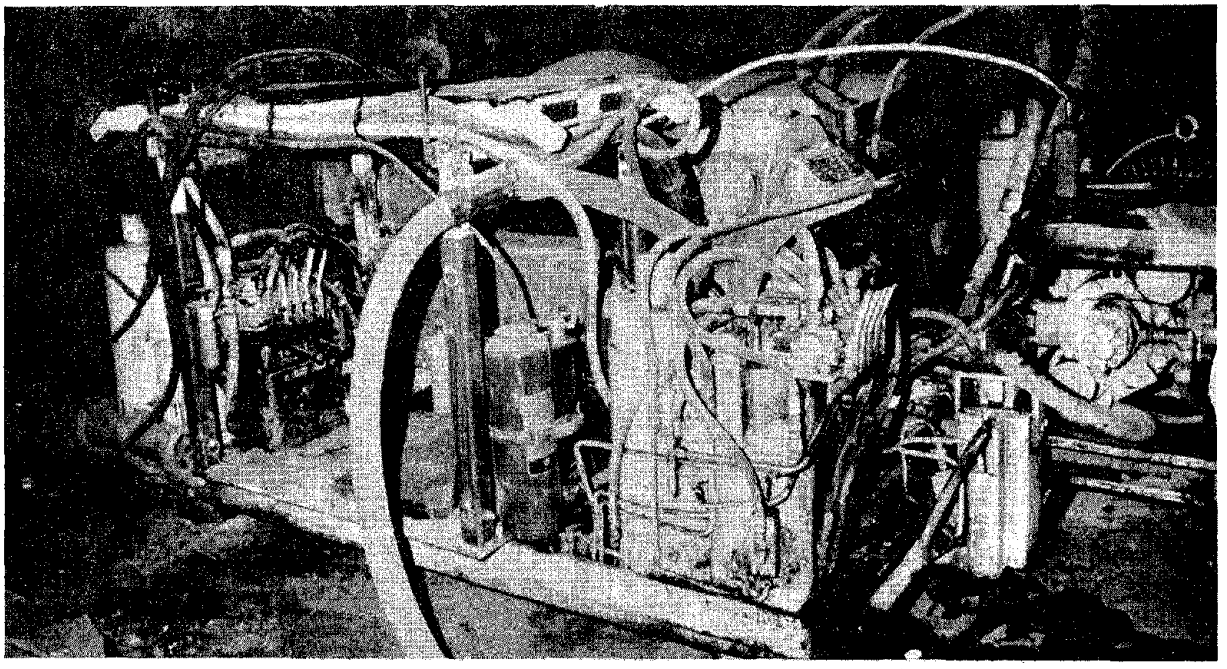


Figure 30 VIEW OF LEFT SIDE OF AUGER MINER SHOWING OPERATOR'S CANOPY AND REAR JACK CONTROL STATION

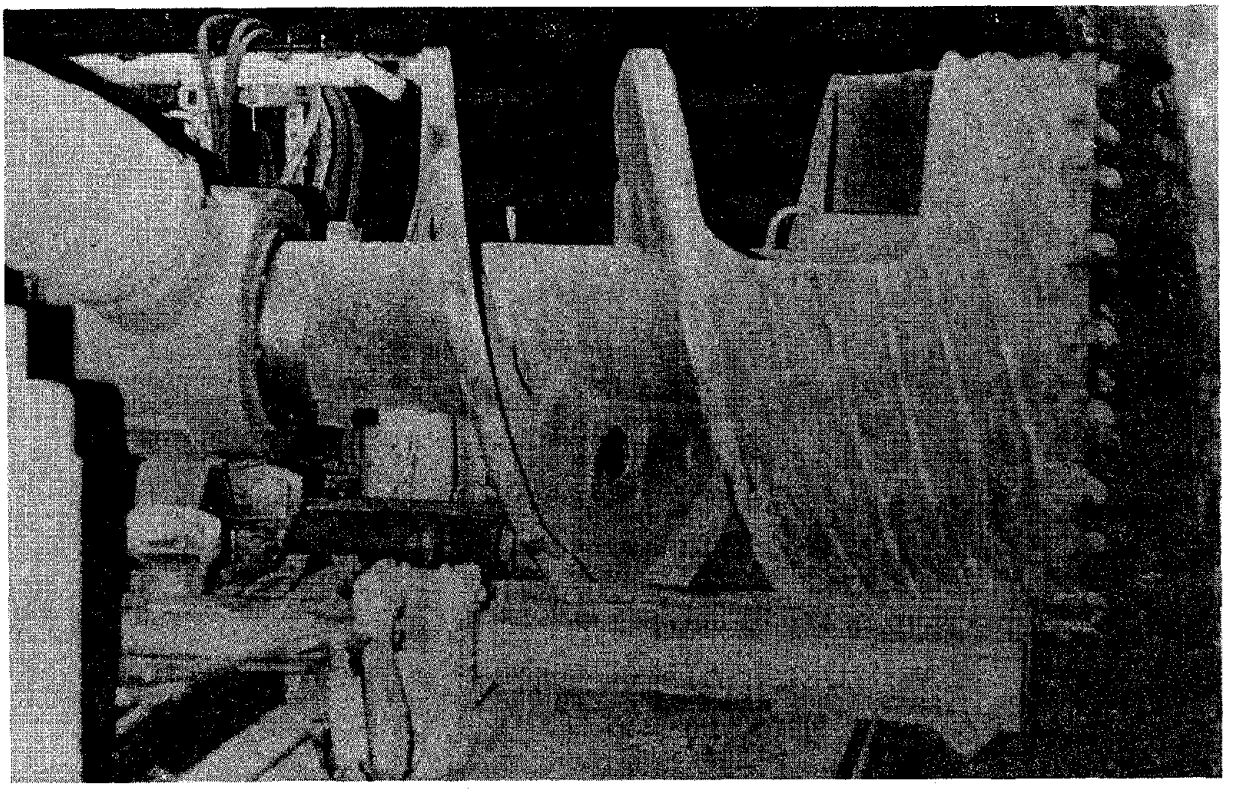


Figure 31 CLOSEUP OF AUGER CUTTERHEAD RESTING ON GUIDE RAILS

4.2 DEMONSTRATION RESULTS

4.2.1 Demonstration Criteria

To judge the performance of the auger miner system, it was necessary to propose some subjective levels of success, bearing in mind that this was a prototype system. The following subsections outline generally these criteria, with a discussion of the rated performance of the system.

4.2.1.1 Criterion 1 -- The system should possess structural and mechanical integrity for reasonable periods of time with reasonable preventive maintenance and repairs.

The system as a whole performed well for the duration of the demonstration. There were no major structural failures of any system components. One auger cutterhead was damaged in the hole. This cutterhead had penetrated roof rock, and a combination of dull bits and overthrusting of the string by the operator caused the drum portion of the cutterhead to collapse rearward on the stem. As elaborated on later in this report, this failure could have been prevented by a better auger alignment procedure (or instrument) and by more operator training.

The main hydraulic motor failed after approximately 3 months of operation. The problem was found to be a scored crankshaft, most likely due to inadequate lubrication. The practice of starting the main pump with the rotation control advanced allowed the torque motor to rotate before sufficient charge pressure could be built up within the motor crankcase. This practice should be avoided in the future.

Numerous small mechanical failures contributed to the high percentage of lost time experienced during the demonstration. Some examples were:

- Failed subplate mount on conveyor motor
- Numerous pinched hydraulic lines
- Stuck pins on augers
- Sloppy control cable on hydraulic motor speed control
- Failed auger guide rails
- Jammed conveyor belting
- Leaky hydraulic fittings.

While shortcomings of the above type are annoying and counterproductive, all were felt to be easily correctable by minor redesign on the next prototype.

4.2.1.2 Criterion 2 -- The system should be operable by mine personnel within the skill levels of the various classes of available labor.

FMC had analyzed the operation of the system and recommended a crew of five, consisting of:

- Operator
- Helper
- Shuttle car driver
- Beltman
- Foreman.

The crew Ireland Mine assigned to the section followed our recommendation except that a full-time mechanic replaced the beltman. This was a wise decision because the expected startup problems could be more quickly resolved with a mechanic on the crew.

The operation of the equipment does not exceed the abilities of available classifications of UMWA labor. It is not necessary for a miner to have previously operated a continuous miner to do well operating this machine. A better qualification would be reasonable mechanical aptitude and a good working knowledge of the seam, floor, and roof of the mine because augering

is largely a matter of "feel" for what is going on in the borehole. Future demonstration crews should maintain the listed personnel with the addition of an extra shuttle car driver for additional haulage capability.

4.2.1.3 Criterion 3 -- The auger miner should be able to drill at least a 75-foot-deep hole sufficiently straight so as to prevent breakthroughs when using a 42-inch-diameter auger on 57-inch centers.

This criterion was only partially met. The machine did in fact succeed in drilling a number of 100-foot holes through the third block of coal; actual measurements were almost 110 feet. Power measurements taken at the hydraulic motor, which appear as Appendix D, indicate that the electric motor was operating at overload in this range and that about 80 feet is a practical maximum depth which could be drilled in continuous duty with this machine. To go deeper than 80 feet, a higher-horsepower electrical motor should be substituted for the 150-horsepower motor now standard on the Coalbadger.

The guidance of auger holes through the coal seam was a major problem. It was not possible to instrument a cutterhead; therefore, the exact cause of hole misguidance cannot be pinpointed, but external observations lead us to believe the problem is twofold:

- Errors in estimating the initial trajectory of the hole cause the major guidance problem, i.e., initial "aiming" of the cutterhead when beginning a new hole is poor using the available tools and procedures, which consist of adjusting the attitude of the Coalbadger frame while setting a carpenter's level atop the cutterhead before starting a hole.

- The nonhomogeneous nature of the coal and certain unpredictable dynamic responses of the auger string cause variances from expected trajectories in both the vertical and the transverse directions.

Poor operator "feel" for the drilling operation might have also contributed to the errant trajectories.

Statistically, of the 68 holes which were augered in the Phase II demonstration, only 36 stayed entirely within the coal panel the full length. The various trajectory errors are shown in Table 5.

Table 5 DISTRIBUTION OF BOREHOLE TRAJECTORIES

Description	Number of Holes
Hole stayed entirely in coal through to breakout	36
Hole strayed into roof rock but holed through	12
Hole strayed into bottom but holed through	4
Hole strayed into floor or roof rock and could not be continued; these holes were left blind and were sealed	10
Hole intersected the previous hole	6

Improvements to the setup and drilling operation which would likely result in straighter and more predictable holes are as follow:

- An automatic leveling system which would adjust the base of the auger miner to a predetermined and repeatable position
- A more rigid cutterhead-to-flight and flight-to-flight connection at least for some distance behind the head
- A sturdier trough for the head and flights to rest on before entering the hole so their entry can be more precisely guided
- A hydraulic or electric servo system to control cutterhead RPM and direction more smoothly.

4.2.1.4 Criterion 4 -- The system should provide sufficient reliability to provide at least 85-percent machine availability on the average.

As detailed in tabular form in Appendixes E and F, mechanical and electrical downtime accounted for 30.5 percent of the total available shift time. When travel, lunch, firebossing, and general mine delay time is considered, mechanical and electrical downtime accounted for 38.4 percent of the available work time at the face. This figure is somewhat misleading because it includes seven full shifts (3,360 minutes) of downtime during the failure of the main hydraulic drive motor, the loss of the auger cutterhead, and the one-day loss of the auxiliary fan -- shifts when no crew was dispatched to the section. If this time is subtracted from both the recorded downtime and the recorded time at the face, the result is a more representative downtime ratio of 29.7 percent or a system availability of about 70 percent.

While not up to the desired level of 85 percent, it must be remembered that this was a prototype machine with a novice crew. It should also be noted that 70-percent availability is very respectable when compared to many, if not most, continuous miner sections running today. It was encouraging that so very many of the contributory factors to the system downtime appeared to be easily correctable in the next prototype. These items are identified and discussed in the section on Criterion 7 -- Productivity.

4.2.1.5 Criterion 5 -- The operator and crew should be satisfied with the layout, operation of controls, visibility, comfort, etc., afforded by the equipment.

The auger miner system layout and operation was in general very well received by the miners. The major elements of the machinery involving human factors are discussed below:

- Visibility

  The position of the rear control station offered good visibility of the entire operation. The view from the canopy operator's station was good but partially obstructed by the drive motor and chuck itself. This can be improved by raising the canopy and seat height where roof conditions permit.

- Illumination

The face area and the rear of the machine were adequately lit for safety. Inside the canopy, the operator's cap lamp was able to provide good illumination of the controls. The lighting on the first prototype machine would not, however, meet the latest laws. To meet these new standards, a totally new lighting arrangement would have to be designed.

- Controls

All controls were identified by nameplate and by individual legends on the knobs and switches, etc. Controls were laid out in a logical order and the operators were able to learn the controls in a very short time. One troublesome control, though, was the auger rotation/direction control lever. It was noted that this flexible cable control should be replaced by a ball-bearing-type flexible cable or, preferably, a changeover should be made to an electrical or hydraulic servo system on future machines.

- Comfort

The size and layout of the canopy and controls made it necessary for the operator to sit cross-legged which became uncomfortable. Having an adjustable seat height would be better because in most 5- to 6-foot seams headroom is not a problem. Entry to and egress from the operator's station was somewhat difficult when the traverse frame was shifted to the right because the operator had to climb over part of the base frame. In normal operation the canopy station traverses across the base frame, creating many pinch points. An on-the-spot modification was made consisting of adding a flap of conveyor belting material and a panel of sheet metal to the edges of the canopy to act as a guard while sliding by possible pinch points. This worked well, but it was still advisable for the operator to keep his hands and legs totally inside the canopy at all times and especially while traversing.

- Noise

No official instrumented sound level tests were made during the demonstration; however, Ireland Mine safety personnel made a spot check of the auger system using their own sound measuring instrument. The results on the A-weighted scale, slow response, were below 90 dbA, therefore allowing 8 hours exposure per day.

- Dust

In accordance with Ireland Mine's MESA-approved schedule, personnel dust samplers were worn by appropriate members of the crew and the cassettes were submitted for analysis. No notification of dust exposure in excess of legal limits was received. A radiation absorption-type dust sampler was used on the section by Ireland Mine safety personnel, and spot checks were made at various locations during various portions of the augering cycle, none showing a concentration of respirable

dust above the legal limit. The direction of borehole aspiration air and the use of water sprays at the throat of the hole and at the conveyor transfer points kept float dust to a very tolerable minimum.

4.2.1.6 Criterion 6 -- To be competitive in the Ireland Mine operation, the auger miner system should recover at least 40 percent of the coal. This is roughly the average recovery during Ireland's normal "rooming-out" retreat mining plan.

By calculation, Ireland Mine's usual retreat mining plan yields a recovery of 44.6 percent. By similar calculations based on the 75-foot by 100-foot panel, the retreat mining plan utilizing a continuous miner to create panels and the auger system to drill out the panels yields a recovery of 45.4 percent, a 1.8-percent increase. Normal mining variations would cause both values to fluctuate, so for all practical purposes the recovery is identical for both systems. Thus the system is successful in offering equivalent recovery potential to the mine.

4.2.1.7 Criterion 7 -- Productivity of the auger miner system should be equivalent to or greater than that of a continuous miner section in the same area of the mine.

Prior to the underground demonstration at Ireland Mine, we had arbitrarily set our productivity goal at 35 tons per man-shift. This corresponds to 175 tons per machine-shift and was felt to represent an average attainable tonnage based on a theoretical maximum shift tonnage of slightly more than 300 tons. These figures were in line with Ireland Mine's production goals and compare favorably with the average tons-per-man-shift output they currently achieve on their continuous miner sections.

Basically, the auger miner system fell considerably short of achieving our productivity goals. Daily activity logs which recorded shift production, face time, operating time, and various delay times were kept for the entire demonstration. This detailed data is presented in tabular and graphical form in Appendixes E, F, and G.

The major production data is presented in Table 6 and is a synopsis of the appendix data.

Table 6 PRODUCTION DATA

Total shifts considered in demonstration	58	shifts
Total shifts in which crew was dispatched to section	51	shifts
Total shifts system was down full shift due to breakdown (1 minus 2)	7	shifts
Total face time (production)	5,916	minutes
Total of all delays	21,594	minutes
Total tonnage for demonstration	2,000	tons
Average tonnage per machine (crew) shift	39.2	tons
Best shift tonnage	130	tons
Average tonnage per man shift (51 shifts base)	7.8	tons

The high ratio of delay to production time (3.65:1) indicates that the low production lies in the inability of the machine to stay on-line. A look at the average temporal distribution of the time elements of a work shift over the 51 machine/crew shifts is more revealing (see Table 7).

Table 7 AVERAGE SHIFT — ELEMENTAL TIME BREAKDOWN

Time Element	Minutes	% of Shift
Normal Production Activity	118	24.5
Delays:		
Machine Setup		
• Hole-to-Hole Move	42	8.8
• Panel-to-Panel Move	56	11.8
Mechanical/Electrical	100	20.7
Functional Delay	17	3.5
Travel, Lunch, Firebossing	115	24.0
General Mine Delay	32	6.7
Total Shift	480	100.0

For reference, the various delay elements are defined as follows:

- Machine Setup

  Machine setup entails moving the machine up 57 inches after each augered hole and moving 13 feet between hole patterns per the mining plan, or moving to a new panel.

- Moveup

  Each moveup requires checking and leveling, aligning with the rib, rehanging the ventilation duct, setting the rib jacks, moving the entry conveyor, and usually adding blocks above the roof jacks for spacers.

- Functional Delay (Auger Guidance and Conveyors)

  Functional delays represent time required to remove the auger flights from the hole when the holes intersect or when the hole is abandoned due to entering the roof or the floor. Functional delay also is the time lost when the conveyors become jammed due to insufficient clearance for passing mined material or when the belts are stuck after shutdown. Stalling is an additional loss caused by underpowered conveyors.

- Travel, Lunch, Firebossing

  This is a normal, recurring loss which decreases the number of working minutes per shift. It includes portal-to-portal travel time which varies depending on how soon the portal bus can depart at the beginning of the shift, and of course on the distance from portal to section.

- General Mine Delay

  General mine delay is an element of delay entirely independent of the auger miner system or crew. It includes loss of water pressure, loss of electric power, inoperative elevator, breakdown of portal bus, etc.

The last two elements, travel, lunch, firebossing and general mine delay, are beyond the control of the mining system -- they are inherent to any mining operation. They do diminish the working time at the face by a significant amount -- during the demonstration it averaged 2 hours, 27 minutes, per shift. Of the remaining 5 hours, 33 minutes, only 1 hour, 58 minutes, were spent running the system in its normal mining cycle. The balance of the shift -- 3 hours, 35 minutes -- was spent in various non-production delays. None of these delays can be eliminated entirely -- machine moves, for example, must still be done, and some electrical and mechanical failures will always occur, but Table 7 identifies those areas where large improvements are necessary if productivity is to increase to an acceptable level.

4.2.2 Delay Elements

4.2.2.1 Machine Setup

Moving from hole to hole by shifting the machine frames using the procedure described in Section III required an average of 42 minutes per shift. Moves occurred once per shift in 19 shifts, twice per shift in 14 shifts, and three times per shift in 2 shifts. In the remaining 16 shifts, no hole-to-hole move was made. In all, 53 hole-to-hole moves were made or about one per shift average, with an average time of 40 minutes. The theoretical time for a hole-to-hole move including resetting the machine attitude to the operator's and foreman's satisfaction was estimated at 15 minutes. The difficulty in resetting the alignment of the cutterhead with reference to the coal seam accounts for the additional 25 minutes of delay. This difficulty occurs because the base frame of the auger inevitably changes attitude during a move because of:

- Settlement in the soft bottom
- Limited stroke of the "bump" jacks (the floor jacks on the traverse frame)
- Local changes in the contour of the seam which require raising or lowering the machine for compensation.

The long delay associated with these moves is caused by the present practice of aligning the machine frame and cutterhead attitude by eye, using a carpenter's level held on top of the head. This procedure is inherently time consuming, difficult, and only moderately accurate. A significant improvement in this area can be made by modifying the floor jack hydraulic circuit to incorporate an automatic leveling device which would return the machine to a preselected attitude after a move. This is a simple change which will result in much more time being available for production work.

4.2.2.2 Panel-to-Panel Moves

Panel-to-panel moves include moves between drilling sites on the same panel per the approved drilling plan (Appendix C) and moves between panels. All moves of this type involve removing the entire auger string from the last borehole, moving it to the site of the next setup, and reloading the auger steels one by one into the machine for the first hole of the new series. This laborious handling of auger steels is precisely the element which has reduced underground auger production since the first experimental machines.

Site-to-site moving on the same panel will normally be practiced in the Ireland Mine because of roof conditions and the overburden depth in the 8 North area. Improvements which would reduce the time lost in these moves include the addition of a powered winch to move the auger sections as opposed to manually rolling them into position, and an improved jib crane arrangement to lift the auger sections on and off the auger frame. For panel-to-panel moves, the addition of a powered tramming undercarriage, either wheels or tracks, would save time now spent in the laborious "walking" of the machine. For long moves, auger steels should still be transported by scoop/tractor.

4.2.2.3 Mechanical/Electrical Delays

An average 100 minutes per shift were lost due to mechanical and electrical malfunctions which had to be corrected before normal production could resume. These delays include replacement of burst hoses, damaged fittings, etc.; replacement of fuses, floodlight bulbs, motor starter contacts, etc; unjamming the frames when coal lumps would fall into the interior of the machine; and numerous other minor but annoying and contraproductive delays.

Because delays in this category accounted for 20 percent of the full shift and 30 percent of the available face time, it is obvious that improvements in this area will have one of the largest beneficial effects on system productivity. Some of the troublesome electrical and mechanical delays are attributable to the fact that the auger miner system was a prototype system with many developmental problems remaining to be resolved. For example, Badger Manufacturing Company elected to use a combination of rigid hydraulic tubing within the frame of the Coalbadger, terminating in flexible hoses running to valves, motors, etc. The mixture of hard line and hose and of Swagelock fittings and JIC fittings made maintenance and repair of the hydraulic system difficult. Also, many hoses and hard lines were routed and positioned in such a way that they were vulnerable to damage from flying coal or from normal operation of the miner. It was recognized that in future miners, simple improvements should be incorporated to reduce the number and severity of mechanical and electrical malfunctions. Better operator training and maintenance procedures will also contribute to fewer hours downtime for these failures.

4.2.2.4 Functional Delays

Although contributing only 17 minutes of delay per shift, functional delays such as the extra time taken to abandon a misguided auger hole and restart can be reduced as a result of improvements to the methods of machine setup.

Adjustments to the conveyors and replacement of the conveyor pans with stainless steel pans would reduce or eliminate these functional delays. Again, improvements such as these are part of the normal evolution of a product through prototype stages.

4.2.2.5 Normal Production

Many improvements in the normal production cycle are possible. The 118 minutes average shift production time produced only about 39 tons of coal or about 1/3 ton per minute -- far less than the approximate 1 ton per minute which is the system's calculated theoretical output. One major cause of lost production is the absence of haulage. At the start of the demonstration, one 2-ton scoop was assigned as the haulage vehicle and it was thought that the round-trip haulage time would be about 2-1/2 minutes. However, the round-trip haulage time proved to be between 4 and 6 minutes, and a higher-capacity (6-ton) shuttle car soon replaced the scoop. This was an improvement, but the miner still sat idle for a short time after every third flight (each flight produced 2 tons) waiting for the haulage to return. Ultimately, both the shuttle car and the scoop were used to eliminate haulage delays. FMC recommended that in the future, the scoop-shuttle car combination or two shuttle cars should be used. With this improvement in haulage, production improvement of as much as 30 percent can be expected.

Other portions of the mining cycle took longer than expected; for example, extra coordination between the operator and the utility man was required to stop the main drive at a position where the latch pin could be reached. Often this latch pin stuck and had to be forced out with a pry bar, making a 5-second job last a minute or more. Routine cleanup also delayed the cutting cycle, as there was quite a bit of coal spillage between the conveyor and the throat of the borehole. These inherent delays were reduced by simple redesign of some machine components in the second prototype.

It was realized that many improvements could be simply incorporated into the design, construction, and operating procedures of the auger miner system which promised to increase significantly production. It is interesting to note that in the latter days of the demonstration, through the 100-foot-deep panel, shift production averaged about 71 tons -- almost twice the average of the entire demonstration. This points out that as the crew became more skilled in the operation of the system and as the "infant-mortality" failures became less prevalent, the production increased as predicted.

With the improvements as outlined in this section and as further described in Section V, it was estimated that the average shift production of the auger miner system would increase to a level of about 140 tons average with peak shifts of 200 tons occurring about 10 percent of the time.

4.3 PERFORMANCE OF MAJOR SUBSYSTEMS

In the design and execution of the auger miner system, several new or novel techniques were tried, many with very interesting results. A brief synopsis of the performance of each of the major subsystems is provided in the following paragraphs.

4.3.1 Auger Miner

The basic auger miner, the Coalbadger IV, irrespective of inherent design shortcomings, performed much as the manufacturer had specified. The machine demonstrated the ability to drill 100-foot boreholes using the 42-inch-diameter cutterhead. A power-demand curve was plotted versus borehole depth (see Appendix D) which indicated that at the 100-foot depth, the shaft horsepower at the hydraulic motor was about 170. Thus, the electric motor was operating at about 40 percent overload. Considering the drilling duty cycle, this amount of overload is probably acceptable, but as other machine improvements increase the drilling duty cycle, it would be advisable to convert to a 200-horsepower electric motor or limit the drilling depth to about 80 feet.

An examination of the power-demand curve in Appendix D yields some interesting information. It is known that the horsepower delivered to the auger string goes into three separate functions:

- Cutting of the virgin coal
- Conveying of the broken coal
- Rotating and advancing the auger string (i.e., overcoming inertia and friction).

The general shape of the power-demand curve indicates that the cutting of the virgin coal consumes about 90 horsepower; the balance is consumed by conveying the broken coal and rotating the string at a combined rate of about 0.85 horsepower per foot of borehole depth. A linear approximation, then, of boring horsepower with the Coalbadger IV and a 42-inch auger is:

$$\text{Shaft hp} = 90 + 0.85\,d \qquad (d = \text{depth of borehole}).$$

As can also be seen in Appendix D, the auger miner peak efficiency in terms of energy consumption and cutting rate occurs at a borehole depth of about 35 feet. Beyond about 70 feet, the machine's efficiency drops off significantly. This reinforces the observation that about 80 feet of 42-inch borehole is the practical limit of the present machine.

The cutting rate of the Coalbadger IV was predicated on advancing the 42-inch-diameter borehole 5 feet (one flight) in about 36 seconds, corresponding to an instantaneous production rate of about 3.5 tons per minute. In practice this rate was easily attained through virgin coal and even exceeded in several cases by increasing the flow to the sump cylinders.

The "leapfrog" walking/sliding feature of the Coalbadger, which is one of the major claims in the patent for the machine, worked well; but it was learned that for situations where roof height is ample, the strokes of the floor jacks and "bump" jacks should be increased. The theory of being able to hold a horizon while traversing to a new borehole site often failed in practice because of the machine's tendency to settle into the typically soft bottom. An independent method of attitude control was needed.

4.3.2 Auger Cutterheads and Flights

The auger cutterheads and flights as fabricated for FMC by W&S Welding on FMC-fabricated stems performed very well while cutting coal. When the cutterhead entered roof rock or a clay vein in the coal, it was necessary to "brute force" the auger through or abandon the hole. During one of these "brute force" attempts the scrolls and part of the stem on one cutterhead yielded, allowing the drum portion of the head to shift rearward about 6 inches. This put the head out of round and out of balance, necessitating rebuild.

It was decided to increase the cutterhead stem wall thickness from 0.33 inch to 0.50 inch because it was not possible to guarantee that the heads would never penetrate rock. The scroll was also rebuilt from heavier material, and the modified cutterhead performed well for the balance of the demonstration. One other cutterhead was modified by attaching a 6-inch-diameter pilot drill 8 inches ahead of the main drum. It was thought that this pilot would help the cutterhead to guide through the coal in a straighter trajectory, but such was not the case. The tang and box connections between flights suffered no failures either torsionally or by pullout. On several occasions, however, a latch pin assembly slipped out of its housing when the retaining pins bent or sheared off. Twice this resulted in a parted string in the borehole. Both times, the remaining sections in the borehole were easily retrieved using the special auger retriever accessory built for this purpose. An auger retriever tool is a necessity for use in the case of a parted string because it is not allowable for a person to enter the borehole.

During the demonstration, three models of cutterbits were tried -- the Kennametal U-15, which was recommended by the cutterhead manufacturer, and the Carboloy CCH 66 and CC 19. The Carboloy bits exhibited superior life and provided faster cutting and overall better production. The probable reason is that both the Carboloy bits have a shank shoulder-to-tip dimension approximately 1/4 inch greater than the Kennametal U-15. Hence, they cut a borehole with a diameter approximately 1/4 inch larger than the U-15s.

It is known from instrumented tests that the boreholes in the 1 Right, 8 North, test area squeeze as much as approximately 1/2 inch in the first few hours after neighboring holes are drilled (no conclusive convergence data is included here because of unauthorized removal of the gages before all data could be taken). If part of this squeeze occurs as the holes are being drilled, then the additional clearance provided by the Carboloy bits likely reduced the drag forces on the head and string and resulted in improved bit loading and more efficient cutting. The rake angle is also steeper on the Carboloy bits allowing a better bite into the coal. Under test conditions in this area of Ireland Mine, the Carboloy bits were superior in performance -- in other locations, this may not be the case. It should be noted that when rock or clay was encountered, all bits tested dulled rapidly, even to the point where the carbide inserts were torn off and the bits and the remaining bit bodies were deformed into mushroomed nubbins.

Approximately 600 bits were used in the demonstration producing 2,000 tons of coal. At a unit cost of about $1.70 per bit, the bit cost per ton was $0.51. This cost was very economical; in many continuous miner sections the bit

cost per ton was estimated at about $1.00. Also, many bits were prematurely worn by contact with rock, and when auger guidance improves, bit costs should diminish even further.

4.3.3 Borehole Ventilation and Methane Detection System

As described in Section III, the borehole ventilation system consisted of an exhausting blower connected by means of flexible and rigid tubing to a non-rotating airbox enshrouding the rotating auger drive chuck. The hollow-drive chuck and hollow-stem augers continued the conduit to within 2 feet of the back of the borehole. The path of ventilation air was from the entry into the throat of the borehole, around the helical path formed by the auger flights, into the labyrinth formed by the mount of the cutterhead breaker bar (designed to keep lumps larger than 2 inches from entering the stems), back down the auger stems, through the airbox and tubing, to the blower. Discharge from the blower was directed into the return airway.

When operating at a minimum air flow of 900 cfm at a 100-foot penetration depth, the borehole air is changed approximately once a minute and air from the back of the borehole reaches the airbox in about 6 seconds. Two taps on the airbox leading to an air sampling box provided an air sample to the methane monitor detector head. The differential air pressure between the two taps was estimated at 0.5 inch w.g. and was calculated to aspirate a sufficient sample of air to provide a more or less instantaneous indication of the methane concentration of the air passing through the airbox. In practice the differential pressure at the taps was only 0.1 inch w.g., but by modifying the taps with elbows and tubes to form a pitot-tube arrangement, the differential pressure was increased to about 0.2 inch w.g. This allowed a few seconds of delay in making a reading; nonetheless, the methane monitor gave continuous indication of air quality at the back of the borehole in about 10 seconds.

MESA had established a procedure in the approved test plan so that when a hole had remained unventilated over the lunch break or during a period of downtime or between shifts, the dilution port on the airbox would be opened, the fan started, and the dilution port slowly closed, never allowing the methane concentration passing through the airbox to exceed 1 percent. This practice assures that the law is met limiting ventilation air across an auxiliary fan to 1 percent methane. In routine operation, the dilution port must remain closed, and when resuming operation after short unventilated periods, the methane monitor should be closely watched. If the methane concentration reaches 1 percent, auger rotation must be stopped and the dilution port must be quickly opened.

In practice these procedures worked well. Typically, after a short unventilated period such as when changing flights, when the string was reconnected, the methane reading would rise and then fall as the hole was cleared. Under normal drilling conditions, the methane content of the borehole ventilation air ran typically from 0.1 to 0.2 percent methane; occasionally it ran as high as 0.5 percent for brief periods.

The required 20-minute-interval methane checks were taken with a portable battery-operated methanometer connected by rubber tubing to the airbox. Readings between the portable methanometer and the onboard methane detector were always in close agreement.

In all, the borehole ventilation system on the auger miner was an outstanding success. It proved that, at least in the Pittsburgh coalbed, deep-hole augering can be safely done using a through-the-stem air path and moderate airflows. The calculations on which airflow predictions were based are given in Appendix A. They proved conservative in that the methane concentration never reached 1 percent. Undoubtedly, the fact that the test panels had stood for several months prior to drilling allowed some of the methane in the panels to drain.

4.3.4 Conveyors

One of the major modifications to the Coalbadger was the addition of the sub-auger conveyor and the second-stage entry conveyor. The sub-auger conveyor was unique in that it was low lying -- its top strand was only 1-3/4 inches from the ground when the conveyor frame rested on the ground. The return strand was fully enclosed as were the drive and idler rollers. The tailboom of this conveyor angled upward at 15 degrees and the belt was held down, forming this bend by a special set of rollers which gripped the edges of the belt. The conveyor was originally powered at 7 gpm at 1950 psi from the Coalbadger auxiliary power supply. At this power level, it slipped and stalled frequently enough to affect adversely production.

The hydraulic circuit was changed to provide 13 gpm at 2,100 psi which afforded a great improvement in performance. Slipping occasionally occurred when the belt was very wet. Stalling still occurred when a piece of coal or rock would jam the belt. Jams could usually be quickly removed by reversing the belt and sometimes by manually removing the jamming lumps. It was noted that future sub-auger conveyors should be provided with still more power, mostly for reserve for surge conditions and for breaking jams.

The entry conveyor was also umbilically connected to the Coalbadger. Originally, it was operated at 13 gpm at 1,950 psi and performed well. When the modification to the sub-auger conveyor was made, it necessitated reducing the flow to the entry conveyor to 11 gpm at 1,950 psi. It was run at this level for the balance of the demonstration, but it stalled occasionally and had no reserve power. FMC estimated that future entry conveyors should be provided with about 15 gpm at 1,950 psi for proper operation.

Efficiently conveying the coal away from the auger miner was one of the major advancements accomplished during this demonstration. Lack of good coal removal was one of the major factors which had contributed to the failures of earlier underground auger experiments.

4.3.5 Borehole Rockdusting

In the development of the auger miner system, it was realized that the boreholes would have to be rockdusted for safety, and a small rockduster with both flexible and rigid segmented, quick-connecting tubes was provided for

this purpose. It was planned that for through holes and especially for blind holes the rigid tubes would be inserted back into the boreholes to allow dusting of the entire inside surface. In practice it was found that by inserting the flexible hose 10 to 15 feet into the borehole and allowing the normal ventilation circulation to carry the injected dust through the hole, the boreholes could be adequately dusted in a much shorter time. This practice should be followed in the future. Blind holes were rockdusted in this manner without benefit of circulation air and then sealed. Two types of seals were tried; one was a round sheet-metal plug inserted and wedged into the borehole and then plastered over, the other was the more conventional built-up concrete block and plaster style. Both were fabricated with ease and proved durable. Local costs of materials and labor would dictate which type was more cost effective.

4.4 SPECIAL TESTS AND EXPERIMENTS

In addition to the routine operation of the system, several special tests were performed to gather information on various aspects of auger mining. These tests and their results are summarized in the following paragraphs.

4.4.1 Drilling and Withdrawal Forces

To better understand the forces at work in augering coal so that future miners could be designed more efficiently, tests were made to determine the thrusts required both to drill the coal and to withdraw the auger string. These measurements were made by attaching gages on each leg of the sump cylinders. The sump and withdrawal forces were then:

$$F_{sump} = \left[P_{high}\left(A_{piston}\right)\right] - \left[P_{low}\left(A_{piston} - A_{rod}\right)\right]$$

$$= \left[\left(P_{high} - P_{low}\right)\left(A_{piston}\right)\right] + \left[\left(P_{low}\right)\left(A_{rod}\right)\right]$$

$$F_{withdrawal} = \left[P_{high}\left(A_{piston} - A_{rod}\right)\right] - \left[P_{low}\left(A_{rod}\right)\right]$$

$$= \left[\left(P_{high} - P_{low}\right)\left(A_{piston}\right)\right] - \left[\left(P_{high}\right)\left(A_{rod}\right)\right]$$

The sump force was monitored in this manner during the drilling of an entire 100-foot-deep borehole. It varied from 16,000 to 20,000 pounds -- the maximum capability of the machine being approximately 50,000 pounds at a relief valve setting of 2,000 psi. The variance in sump force is attributable to several independent factors:

- Borehole depth (i.e., string length)
- Amount of coal in transit
- Hardness of medium (coal, coal/rock, clay, etc.)
- Operator technique

- Bit sharpness
- Squeeze on borehole
- Sump and rotational speeds
- Frictions in mechanism and string
- Lead angle on auger flights.

It would be difficult or impossible to predict or detect the contribution of each of the above factors to sump force.

Withdrawal force was measured with the entire string in the borehole, loaded with coal and rotating clockwise (as if cutting) at about 60 rpm. The withdrawal force under these conditions was measured at 26,000 pounds.

Withdrawal force is also a function of various independent factors:

- Borehole depth (i.e., string length)
- Operator technique
- Withdrawal speed
- Rotational speed and direction
- Frictions in mechanism and string
- Amount of coal in transit
- Lead angle on auger flights.

Again, it is difficult to predict the contribution of each factor to withdrawal force. Some interesting conclusions were drawn, however.

It had been theorized that the lead angle of the auger flights contributed significantly to the sump force at the cutterhead when the string is rotating, and, hence, would oppose the withdrawal of the string under similar conditions. This theory was reinforced by the fact that the withdrawal force was higher than the highest recorded sumping force, even though more work was being done while cutting. There is an apparent and complex interaction whereby auger torque is partially converted to thrust. Instrumentation was insufficient to explore fully this interaction, but an interesting conclusion can be drawn if two simplified force balances are postulated for drilling and extracting while rotating:

Drilling:

$$S_d + L = \Sigma F + R$$

Withdrawing:

$$S_w = L + \Sigma F$$

where

S_d = Sump cylinder force while drilling

S_w = Sump cylinder force while withdrawing

L = Lead angle force (assumed to exist)

ΣF = All combined inertial and frictional forces

R = Cutting resistance

Combining: $S_w - L = S_d + L - R$

or $$L = \left(\frac{R}{2}\right) + \frac{(S_w - S_d)}{2}$$

using observed values of S_w = 26,000 pounds

S_d = 20,000 pounds

$L = \frac{R}{2}$ + 3,000 pounds.

Because R is real and positive (even though it may in fact be small), L is real and positive and hence acts in the direction assumed in the force balance. It also, by this example, has a minimum value of 3,000 pounds which, as theorized, is a significant contribution to the total sump force at the cutterhead.

Note that this calculation did not necessarily prove the presence or magnitude of a lead-angle force; it only reinforced the argument for its existence. Future tests on the second prototype discussed in Part 2 confirmed its existence and further advanced our understanding of the coal drilling process.

4.4.2 Power Absorption Tests

It was desired to determine how much power was being absorbed by the torque motor in performing the complex function of cutting the coal and conveying the cut coal out of the borehole. The torque motor was instrumented by placing a pressure gage in each supply port. During the drilling of a borehole, the differential pressure was recorded and the rotational speed was measured by stopwatch.

Then, using the torque motor manufacturer's specifications for output torque versus port pressure, the theoretical power output at the motor shaft could be calculated. These results are plotted in Appendix D. As discussed earlier in this section, the conclusion to be drawn from this curve is that, estimating electric motor horsepower at 25 percent above theoretical horsepower (i.e., 80 percent conversion efficiency), the present Coalbadger miner with a 150-horsepower (minimum) electric motor when operated on a

duty cycle which produces 150 tons per shift should be limited to cutting boreholes at most 80 feet deep. Continuously cutting deeper holes could lead to motor failure.

4.4.3 Roof Control Studies

The method of perforating the large coal panels gave rise to much concern about roof failure in two areas:

- Would the entries remain open and stable?
- Would the boreholes squeeze significantly and would the squeeze, if it did occur, hamper the boring of holes or the retrieving of flights from a previous hole?

As previously mentioned and as shown in Appendix G, Consolidation's parent organization, the Continental Oil Company, devised a drilling pattern which they predicted would allow safe drilling. For all intents, this pattern provided for safe operations. No entries were lost although some posted areas took considerable load as evidenced by bowed wooden posts and noticeable roof sag.

Convergence gages were placed on the vertical axis about 5 feet back from the break through area in certain test boreholes, and readings were taken as adjacent boreholes were drilled. Test holes were monitored in the 50-foot panel and the 75-foot panel, but insufficient tests were done to draw conclusive information. The data taken (see Figure 32 for a set of sample convergence curves) exhibits evidence that borehole squeeze takes place more or less independently of time but rather as a function of the creation of adjacent boreholes.

In the example in Figure 33, the drilling of Borehole 4 in a series of holes in the 50-foot panel significantly affected the convergence of Boreholes 1, 2, and 3. That the convergence registered in Borehole 2 was the most severe can be explained by the fact that it was the farthest from the solid coal and hence received the least "carry across" support of any of the holes.

For as long as the tests were carried out in any given series of holes, typically the convergence increased to 1/2 to 5/8 inch and apparently stabilized -- this conclusion was drawn from visual observations only because the gages were withdrawn at the end of a hole series for safety. The first hole set in the 50-foot panel crushed out catastrophically, leaving the holes lenticular in shape and only about 2 feet high. This crushing took place 16 days after the first hole was bored. No further massive failures occurred for the balance of the first demonstration.

4.4.4 Ninety-Degree Intersecting Boreholes

One of the proposed uses of the auger miner is to predrill ventilation holes into a face to be cut by a continuous miner. These holes would be at least 100 feet deep. A second hole from a point ahead of this face in the adjacent entry would then be drilled into the block to intersect the first hole at about 90 degrees. This passage, as shown in Figure 33, would then be used as a ventilation air passage to carry dust and methane out ahead of the miner

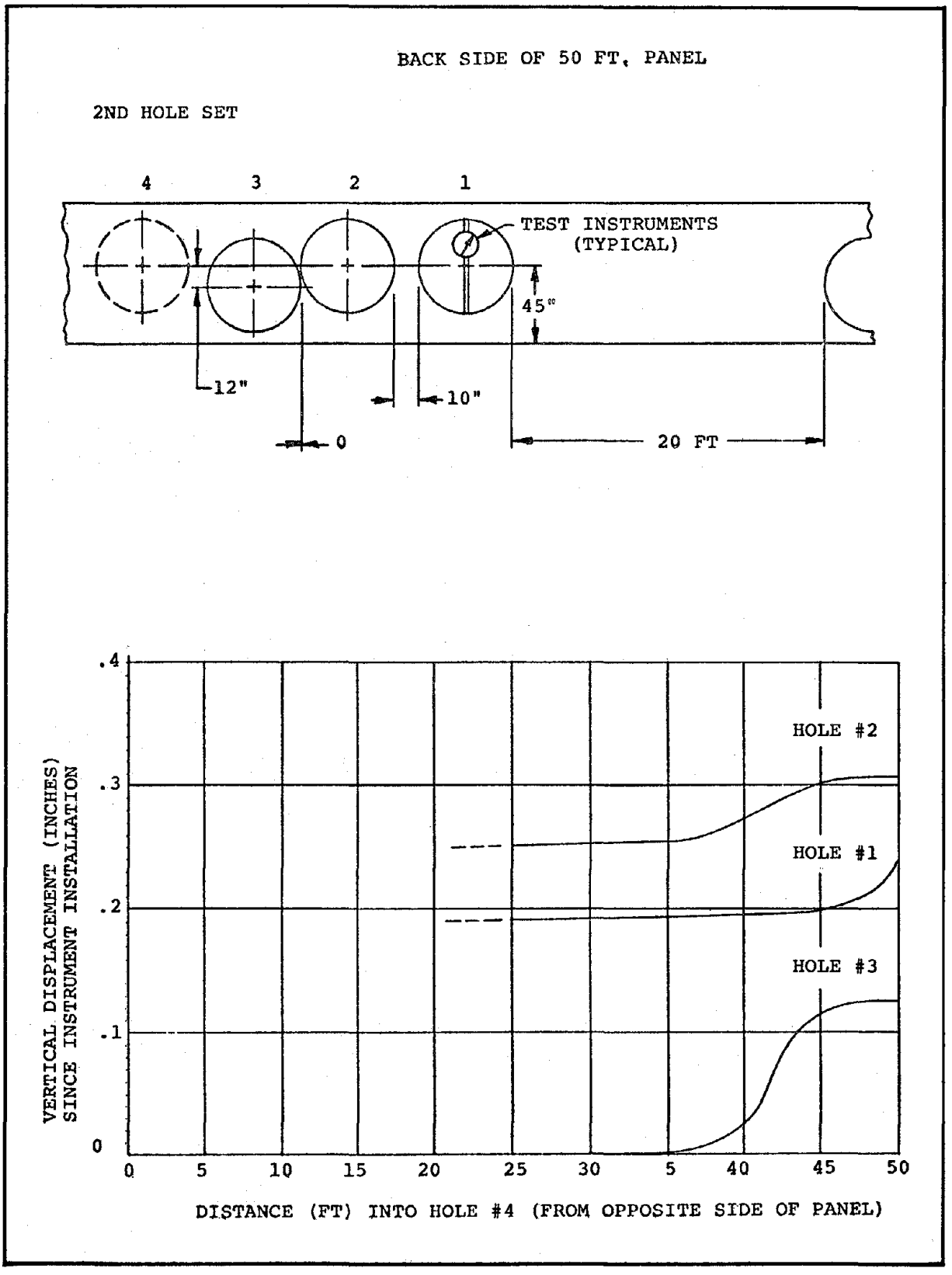


Figure 32 TYPICAL BOREHOLE CONVERGENCE CURVES

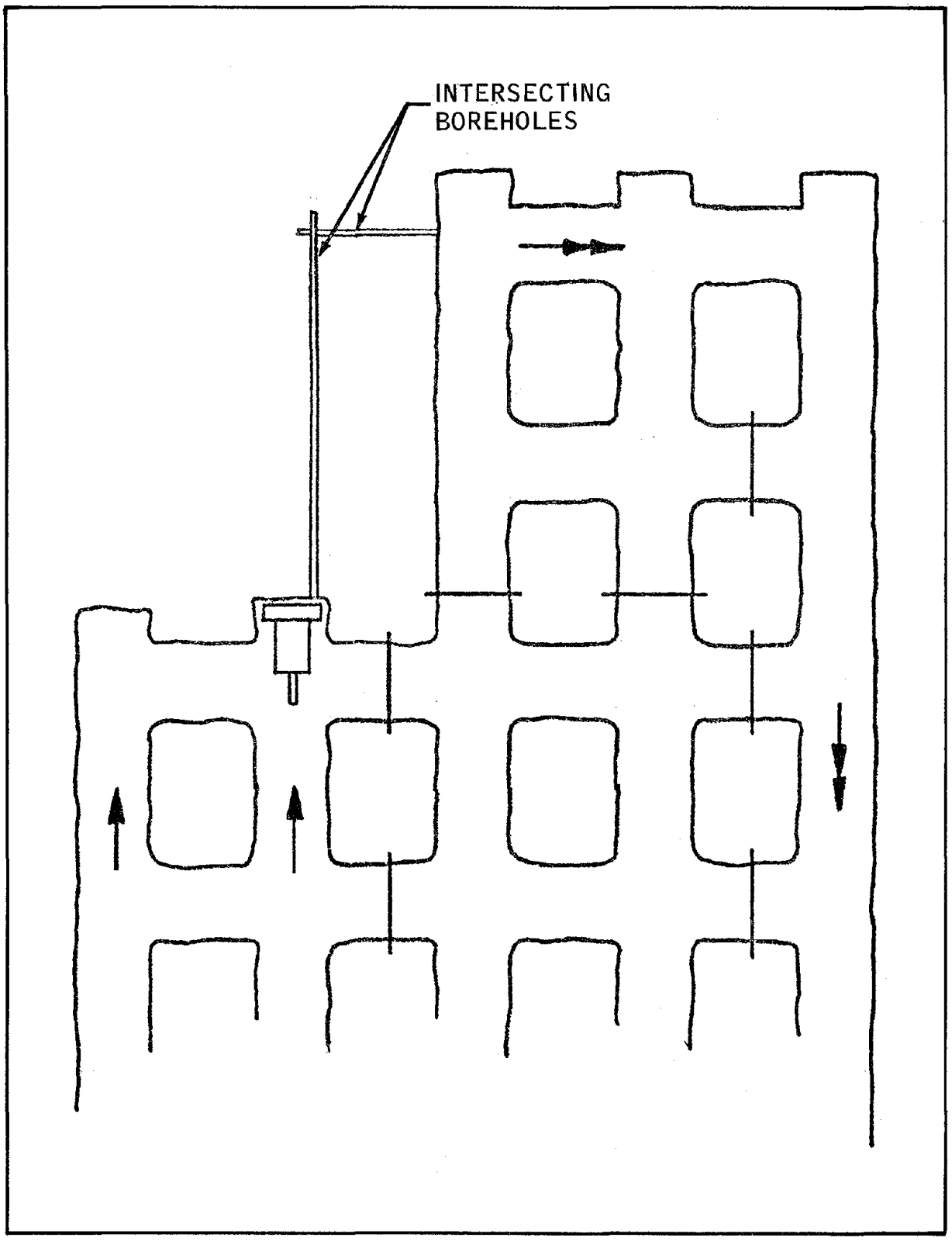


Figure 33 AUGERED BOREHOLES AS USED IN "AHEAD OF THE FACE" VENTILATION PLANS

and into the returns. A test intersection of this sort was drilled at the end of the 100-foot panel from a point about 40 feet around the corner of the pillar. The second borehole was approximately 30 feet deep, and the point of intersection of the two holes was offset about 1 foot vertically.

4.4.5 Safety

Although not specifically a test item, for the record no lost-time accident was sustained by any member of the auger miner crews during the first underground demonstration.

5.0 RECOMMENDATIONS (Part 1)

5.1 GENERAL

The auger miner system demonstrated the potential for acceptable productivity in the general mining conditions afforded by Ireland Mine which are typical of a good part of the Pittsburgh Number 8 seam. The system achieved a record shift production of 130 tons, which was 87 percent of the mine's stated goal of 150 tons.

Certain manufacturing, operational, and procedural shortcomings which lessened the system's availability and reduced average production were identified, and without exception all were correctable by redesign or by adopting more effective operating procedures.

The system was well received by the crew members and by mine management. It was relatively easy and safe to operate and the work environment (dust, noise, methane, lighting, work assignments, etc.) was equivalent to or better than the typical continuous miner section.

To the best of our knowledge, the Coalbadger/FMC auger miner system operated by Ireland Mine outlasted, outproduced, and outperformed any other experimental underground auger in the U. S. at that point in time.

The system satisfied all stipulations set down by MESA[1] and the West Virginia Department of Mines for operation and for the health and safety of the crew. This included area ventilation, borehole ventilation, methane detection, borehole stability, dust suppression, mining practices, roof control, hole abandonment procedures, etc.

5.2 RECOMMENDATIONS

As discussed in detail in the earlier sections of this report, we recognized that two tasks needed to be accomplished to bring the auger miner system to full potential:

- Haulage away from the machine must be improved by adding an additional haulage vehicle or a continuous haulage system.
- Redesign of certain portions of the system must be undertaken to reduce downtime and make more effective use of available production time.

1. The Mining Enforcement and Safety Administration (MESA) was part of the U.S. Department of Interior until March 9, 1978 when reorganization placed it under the U.S. Department of Labor, retitled Mine Safety and Health Administration (MSHA). References to MESA and MSHA throughout this report indicate the agency in authority at the time.

The former can be done simply by the mine assigning another shuttle car to the section. The latter included many specific design changes which are outlined in the following subsections, which became our list of recommendations for the second prototype.

5.2.1 Increase Sub-Auger and Entry Conveyor Power

More powerful hydraulic motors, compatible with mine duty, should be provided. The hydraulic motors driving the two conveyors have been underpowered to the point of stalling under heavy load conditions. Kits for increasing power have been added to the sub-auger conveyor, and the entry conveyor speed has been increased, improving performance of both conveyors. However, due to the high torque imposed on the sub-auger conveyor motor, the mounting screw holes have cracked through to the motor housing. Auxiliary pump capacity will have to be increased as required to power the system.

5.2.2 Power Stabilizing Jacks to the Rib

Suitable hydraulic jacks to stabilize the machine against the auger thrust should be provided. During heavier thrust while augering, the machine could be pushed out of alignment, causing difficulty and delay in operation. These jacks, one at each corner of the rear of the machine, could be operated from the operator's compartment, and would eliminate manual handling and erection of wooden props and jacks (which is done at present), decreasing the setup time.

5.2.3 Auger Rotation Control

A positive control for aligning the tang of the chuck with the mating head or flight socket should be provided. Excessive time is now required when stopping rotation of the auger drive to align with the flight. Excessive start and stop of the electric drive motor is now practiced which is detrimental to both the motor and electrical controls and the hydraulic torque motor.

5.2.4 Roof Jack Pressure-Replenishing System

An automatic valve system which will maintain a constant pressure on the roof jacks should be provided. The present manual system for maintaining pressure on the roof jacks is not adequate, allowing the pressure to bleed off and allowing the machine to move out of position. This creates an alignment problem and a potential safety hazard.

5.2.5 Crane Arm Control

The control station for the job crane should be relocated from the cab to the flight loading area. The control for the flight loading crane is now located in the operator's cab. This requires communication and coordination between the operator and the utility man. More efficient and safer operation will be realized by making this modification.

5.2.6 Relocate Controls to Canopy

Floor jack and roof jack controls should be relocated from the present position at the rear of the operator's cab to a location within the operator's

compartment. This will eliminate lost operator time caused by moving from one station to the other and will place all machine controls near the operator.

5.2.7 Roof Jack Cylinders

Roof jacks with adequate extended length to contact a wider range of roof heights should be provided. Present roof jacks are inadequate in stroke, requiring extensions and the use of cribbing to reach the roof. This practice is hazardous and time consuming.

5.2.8 Machine Alignment and Auger Head Positioning

Methods of positive leveling, alignment with the rib, and positioning of the auger head should be provided. Misalignment of the machine in the vertical plane or horizontal plane has allowed the auger to penetrate the roof line, the floor line, or the preceding augered hole. Any of these deviations causes excessive time to complete the hole or retrieve the flights. Methods of leveling and aligning the machine will eliminate this problem. A method of positioning the auger head before augering should be provided.

5.2.9 Improve Tramming Capability

An improved tramming system should be provided. The existing method of "walking" the machine in increments of 57 inches is time consuming and imprecise.

5.2.10 Conveyor Clearance, Sub-Auger

A modified suspension structure for the sub-auger conveyor to provide adequate clearance for passage of mined material should be provided. The support for the sub-auger conveyor is now restrictive in the size of particles that can be discharged between the support and the belt, which occasionally jams the system. An improved support will allow passage of this material and reduce machine delay.

5.2.11 Conveyor Pans

Suitable stainless steel or other acceptable pans under the loaded portion of both the sub-auger and the entry conveyor belts should be provided. Stainless steel is widely used in mining conveyors to reduce friction and corrosion. On idle days, the belt tends to stick to the existing carbon steel pans, causing difficulty in starting the conveyor on resumption of production.

5.2.12 Auger Vent-Hose Supports

A vent-hose support system which would be attached to the machine should be provided. The hose is now supported by the roof support timbers and the supports must be relocated each time the machine is moved up. This support method is inadequate as there are no consistent points of support. The two directions of travel of the airbox-chuck assembly to which the hose is attached cause the hose to droop and become damaged. This then causes restrictions in the hose and often the hose is torn. Consequently, air aspiration from the borehole is slowed and production time is lost while corrections are made.

6.0 INTRODUCTION (Part 2)

At the conclusion of the first demonstration at Ireland Mine, FMC and the Bureau of Mines discussed the possibility of rebuilding the auger miner. The rebuilt auger miner would incorporate the many improvements which FMC and the Ireland Mine had recommended. Ireland Mine supported the idea of a new style machine provided that it would produce more coal per shift and indicated in advance their willingness to continue the demonstration of such a machine. FMC proposed a redesign program and a contract modification was subsequently awarded to FMC. In October 1976, design work commenced, and in September 1978, the refurbished and improved equipment was shipped to Ireland Mine.

Between October 1978, and March 1979, the auger miner system, as shown in Figure 34, was operated underground in a normal production section. Data from 53 test shifts were recorded.

Ireland Mine and FMC agreed that the revised machinery should embody changes which would enhance productivity. In some instances, the modifications also improved safety, although the original Coalbadger had no serious safety defects. It was also agreed that nonproductive experiments would be held to a minimum during the underground demonstration since most of this type experimentation had been done during the previous test.

FMC had calculated that the improved machine could achieve 200 tons per shift, 10 percent of the time, with an average shift production of about 123 tons. Breaking the 200-ton-per-shift barrier, then, became a stated goal for the demonstration.

In analyzing the results of the earlier demonstration, it was apparent that the instantaneous production rate of the auger miner was adequate, and that productivity increases would result from changes to the equipment which would increase its availability. Thus, reliability became important as well as operational techniques which would increase the portion of the shift during which the machine could drill.

In brief, during the second demonstration the improved auger miner fell somewhat short of its productivity goals. The average shift production was 88 tons and the record shift of 182 tons was 18 tons short of the target "200-ton" shift. The demonstration was successful, though, in that continual improvements were made in operating efficiency and productivity as the demonstration progressed. The demonstration was prematurely ended when the main drive motor for the drill failed catastrophically. Due in part to the expected time and expense of repairs and in part to Ireland Mine's desire to finish mining in the test area so it could be sealed, the equipment was removed from the mine and placed in storage. Had the planned additional

50 to 60 shifts of demonstration been completed, there is reason to believe that both the shift average and record shift production tonnage figure would have been increased.

It should be pointed out that the second-generation auger miner outperformed the modified Coalbadger tested earlier by more than double on shift production, and by almost half again on record shift tonnage. Thus, the revised FMC auger miner became, insofar as we know, the new record-producing underground auger miner in the U.S. and probably the world.

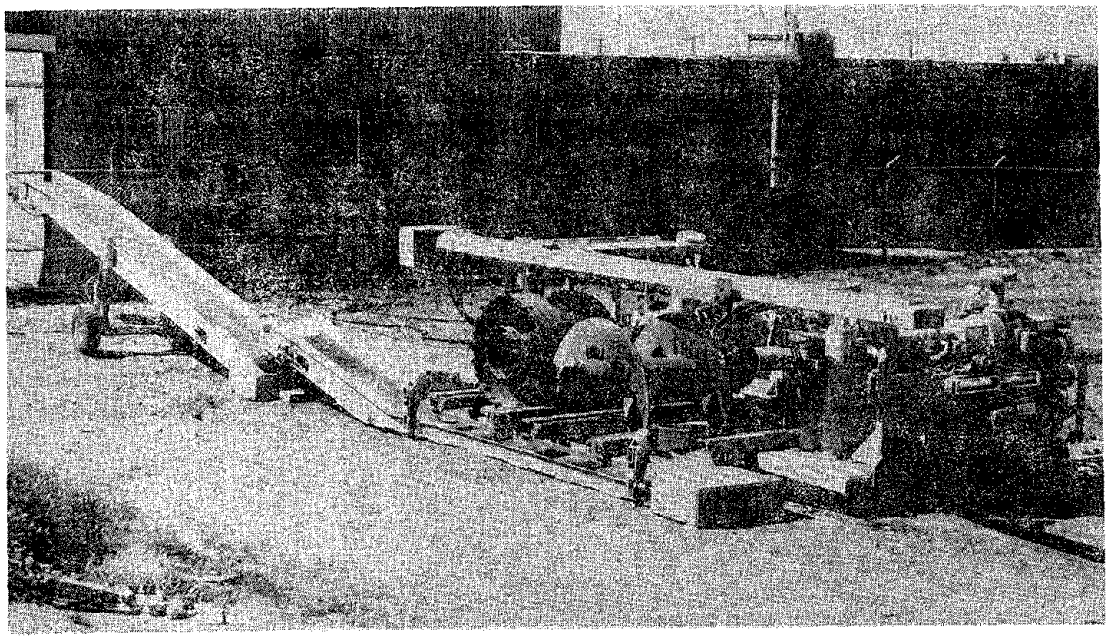


Figure 34 UNDERGROUND PANEL EXTRACTION SYSTEM - SECOND PROTOTYPE

7.0 SUMMARY (Part 2)

The auger miner system as delivered to Ireland Mine in September 1978, contained dozens of functional and operational improvements. The main frame was rebuilt and modified to incorporate an extractor device to withdraw drillsteels from the borehole. The sub-auger conveyor was mounted on a position-able mechanism. Both conveyors had their drive horsepower increased. The entry conveyor had a skirt attached to its intake side to reduce spillage at the transfer point. It also had a powered lift installed to allow better loading into the shuttle cars. The cutterheads, flights, fan, and rockdusting equipment were retained from the previous demonstration.

The mine plan was changed to allow an array of 12 to 13 holes per block in 84-foot square blocks, thus minimizing nonproductive moves.

In the aggregate, the improvements were expected to result in an average shift tonnage of about 123. In actuality average tonnage only reached 88.

Slight increases in system availability were achieved and the best shift tonnage of 182 tons nearly reached the 200-ton target. As in the previous demonstration, there were no lost-time accidents.

A failure of the main drive motor caused the mine to opt for the early termination of the demonstration. There is some evidence that average shift production was on the rise at the time of the termination of the demonstration, so it is unfortunate that 40 to 50 additional shifts could not have been run.

FMC attempted to find an alternate demonstration site but after several promising leads failed to lead to an agreement, the equipment was put into storage.

In the final analysis, the average shift production of 88 tons represented approximately the break-even point for Ireland Mine. Greater production would be needed for this system to be profitable.

The demonstration furthered our knowledge on the behavior of cutterstrings' trajectories through coal and the power required to auger coal. A borehole was successfully backreamed to a larger diameter. These facts will make it possible to design future generations of augers which can be more productive.

8.0 TECHNICAL DISCUSSION (Part 2)

8.1 MACHINE IMPROVEMENTS

At the conclusion of the first demonstration, 19 specific improvements to the original auger miner were recommended. As the design progressed, these improvements were implemented. Since they were important to the improved performance of the equipment and were, in fact, tasks of the modified contract, it is worthwhile to discuss them individually.

8.1.1 Increase Sub-Auger and Entry Conveyor Power

To improve the ability of the conveyors to remove coal, and prevent stalling and jams, both conveyors had new hydraulic motors installed as the conveyors were cleaned and rebuilt. The sub-auger conveyor reentered service with 17.6 nominal horsepower and a belt speed of 400 feet per minute. This represented a 309-percent power increase. Likewise the entry conveyor reentered service with 8.8 nominal horsepower and a belt speed of 400 feet per minute. This represented an 11-percent power boost.

8.1.2. Power Stabilizing Jacks to the Rib

Two double-acting hydraulic cylinders with adjustable thrust tube extensions were added to the frames of the new machine -- one at each rear corner. Each jack had a thrusting capability of about 14 tons and could react several times that force statically. This was determined to be adequate both to maneuver the machine into final drilling position and to react any drilling thrust which the sump carriage was capable of delivering.

8.1.3 Auger Rotation Control

Control of cutterhead rotation direction and speed on the original Coalbadger was by means of a flexible control cable running between the operator's compartment and a pilot valve lever on the Sundstrand Series 26 variable displacement pump. In the first demonstration, fine control of auger speed and return to the null position (zero rotation) was difficult. Sometimes after nulling, the cutterhead would abruptly begin slow rotation causing a possible hazard. It was felt that changeover to an electromechanical pilot valve would offer better control which would benefit both production and safety.

At the same time, the electrical control would allow logic interlock of the rotation and pump/start function which would disallow starting the pump in anything but the null position. It was felt that the miner operator's previous (unapproved) practice of leaving the flex cable control in the run position during pump startup allowed pump operation before charge pump

pressure could rise and cause the pump to run inadequately lubricated (momentarily). This practice also contributed to, or actually caused, the pump failure during the first demonstration.

The final system consisted of a commercially available, single axis "joystick" with a null position detent and a Sundstrand intrinsically safe electromechanical servovalve. Low voltage power from a protected, current-limited, DC power supply completed the system which was granted intrinsically safe designation from MSHA. The neutral start feature required that the joystick be in the detent (null) position before the motor driving the pump could be started. If an emergency rotation stop switch were activated, or a power interruption occurred, rotation restart could only be accomplished by moving the lever to null and depressing the pump start button. The detent feature was intended to prevent accidental rotation (for example by bumping the control lever), while cutterhead maintenance was taking place with the main motor and pump running.

A very useful feature of this control system was the "homing" feature, a button which when depressed caused the cutterhead drive motor to slow to about one-tenth speed and stop at a predetermined position such that the drive tang would align perfectly with the next auger section. When seeking the "home" position, the joystick was locked out, and a reset button needed to be intentionally depressed before joystick rotational control could be resumed. This feature virtually eliminated hunting for the proper tang engagement position, thereby shortening cycle time and increasing production.

Functionally the system performed flawlessly; however it was discovered during initial setup that the Sundstrand pump (reused from the original Coalbadger) required an additional mechanical adjustment to hold the null position, even when the electrical control was nulled or even disconnected. It was surmised that the original null problems were due to the pump itself and not the flexible cable controller as suspected.

While Sundstrand had an intrinsically safe servovalve (pump stroker) approved and in use in several other underground applications, FMC worked with Honeywell to gain MSHA approval for a similar unit of theirs in order to have a second source. Honeywell provided a loaned demonstration unit for the duration of the demonstration, but it was never used and was returned.

8.1.4 Roof Jack Pressure Replenishment System

A system was devised whereby the four roof jacks could be extended or retracted at a high rate and where, once extended against the roof, a sensing valve would, when engaged, bleed sufficient hydraulic oil into the cylinders to maintain a preselected (adjustable) pressure on the roof jacks. In operation, if the pressure were to fall, the valve automatically shifted to flow oil into the cylinders and restore the set pressure. In practice this set pressure was maintained at a low value, usually about 300 pounds per square inch, to minimize overloading the roof. When used in conjunction with the rib jacks, the roof jack system effectively locked the auger miner into position in the entry in preparation for drilling.

8.1.5 Crane Modification

It had been intended to remove the controls for the previous jib crane to a position near the flight loading operation. In actuality, the jib crane was replaced by a monorail hoist running the full width of the machine as shown in Figure 35. Dual controls for the hoist were relocated to the left side of the machine, one set near the cantilevered end of the monorail and one set a few feet inboard (see Appendix H).

The outboard set of controls were used when loading flights off the ground when starting a new borehole. The inboard set of controls were used when transferring flights between the extractor and drilling carriages.

The monorail hoist was required to replace the jib crane when it was discovered that no single jib crane position could reach all locations where flights needed to be moved. The monorail hoist allowed the operator to lift or lower flights and traverse them across the full width of the machine. In operation the hoist was easy to operate, fast and safe. It did, however, suffer from repetitive failure of a vendor-supplied splined driveshaft and ball runner as is discussed later in this report.

8.1.6 Relocate Roof and Floor Jack Controls to Canopy

In an effort to increase production by eliminating wasted operator motions, all machine controls except those for the monorail hoist and the extractor mechanism were moved into the operator's enclosure. The controls are shown in Figure 35 and are identified in Appendix H. During the demonstration, MSHA raised an objection to the hoist/extractor operator working his controls without benefit of a separate canopy. Their objection was complicated by the fact the auger miner worked from a fully bolted entry and the hoist/extractor operator worked in close proximity to a roof jack capable of providing many tons of roof support. MSHA's technical support group took the question under advisement, but the demonstration ended before a resolution could be rendered.

8.1.7 Improve Roof Jacks

The roof jacks on the Coalbadger had two shortcomings -- one of which was their inability to provide continuously some load against the roof as the machine settled or moved. This problem and our solution to it were previously discussed. The second shortcoming was that the jacks had a stroke of only 15 inches and often could not reach the local roof without using tubular steel extensions and/or crib blocks. Ireland Mine requested roof jacks which could extend to at least 8 feet but which would retract to a height to enable maneuvering the machine throughout the mine.

The new roof jacks provided by FMC had a stroke of 30 inches and would reach a height of 8 feet, 7 inches. Large, ball-jointed pads were added to the tops of the jacks to help minimize localized roof pressure. A typical roof jack can be seen in Figure 36.

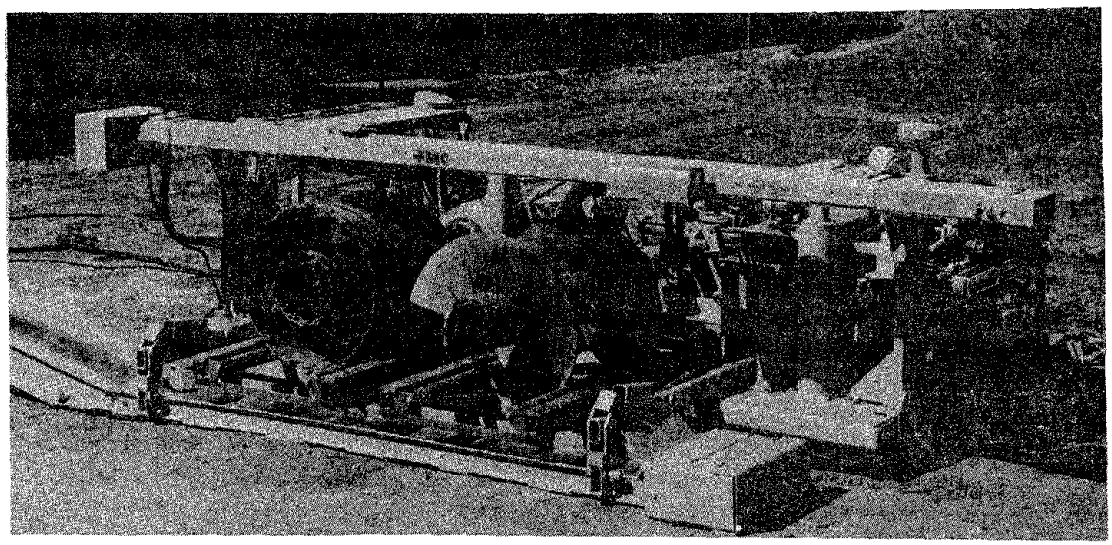


Figure 35 MONORAIL/HOIST

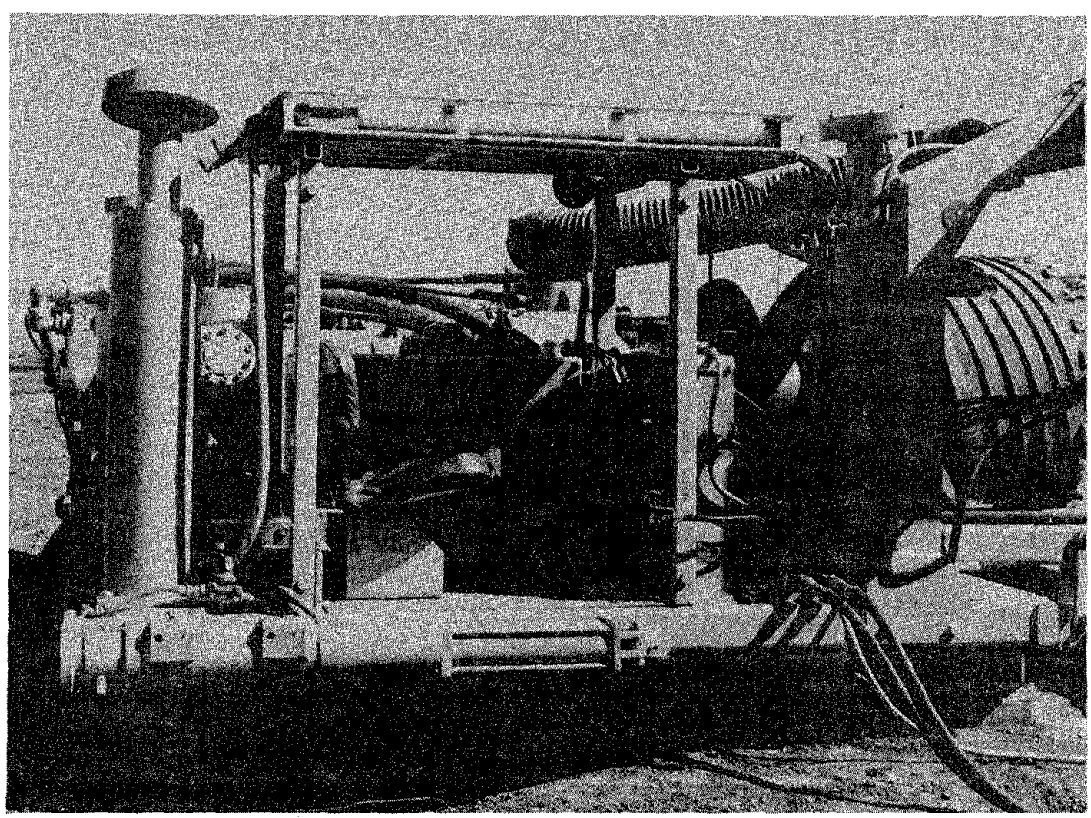


Figure 36 VIEW OF RIGHT REAR ROOF JACK AND RIGHT RIB JACK

8.1.8 Improved Tram Systems

The Coalbadger system of sliding frames and jacks, while innovative, was slow and imprecise. Several improved tram systems were conceived but no conventional crawler sets or powered wheelsets were found to be practical. Their drawbacks included unacceptable maneuverability and excessive frame height. Finally, a cylinder-powered walking shoe system was devised similarly to those used on surface highwall augers and on some European underground mining equipment. Although Ireland Mine personnel were skeptical, they conceded to accept this method of tram and in practice it worked well -- offering no more maintenance problems than crawlers.

The procedures for tramming the machine are shown in Appendix H which is the simplified instruction set given to the operator to aid him in learning and remembering the various maneuvers. A diagram depicting the sequence of "walking" the machine from hole-to-hole, in this case left-to-right, are shown in Figure 37. The sequence is as follows:

A) The machine frame is elevated using the floor jacks until the walking shoes are clear of the floor. The walking shoes are shifted to the right side of the frame.

B) The floor jacks are retracted, allowing the machine to rest on the walking shoes.

C) The walking shoes are shifted, causing the machine frame to move to the right.

D) The floor jacks are adjusted to drilling height and the walking shoes are moved to the right in preparation for the next step.

The feature of pivot steer was used to an advantage in aligning the drill carriages with the local rib and in negotiating intersections. A good operator could tram the machine about 10 feet per minute.

8.1.9 Machine Alignment and Auger Head Positioning

Control of initial drilling trajectory on the Coalbadger was provided by a spirit level placed on the cutterhead while the frame was rocked to some angle. This initial angle of attack was arrived at with difficulty and was erratic due to inconsistencies of the cutterhead and position from hole to hole -- it was always a different cutterhead. It was felt that rapid and repeatable alignment of the frame prior to drilling would prevent stray holes -- those which intersected roof or floor rock or wandered into a previously bored hole.

An electrohydraulic control system was designed for the auger miner which allowed the operator to adjust the front floor jacks to elevate the front of the auger frame to his preference. Engaging a pair of toggle switches activated two solenoid valves which routed hydraulic oil to extend or retract the rear floor jacks. Flow continued until interrupted by a level detecting switch -- a commercially available device manufactured by Honeywell. Regardless of whether the rear jack started out high or low, the appropriate solenoid on the valve brought the frame to the desired attitude.

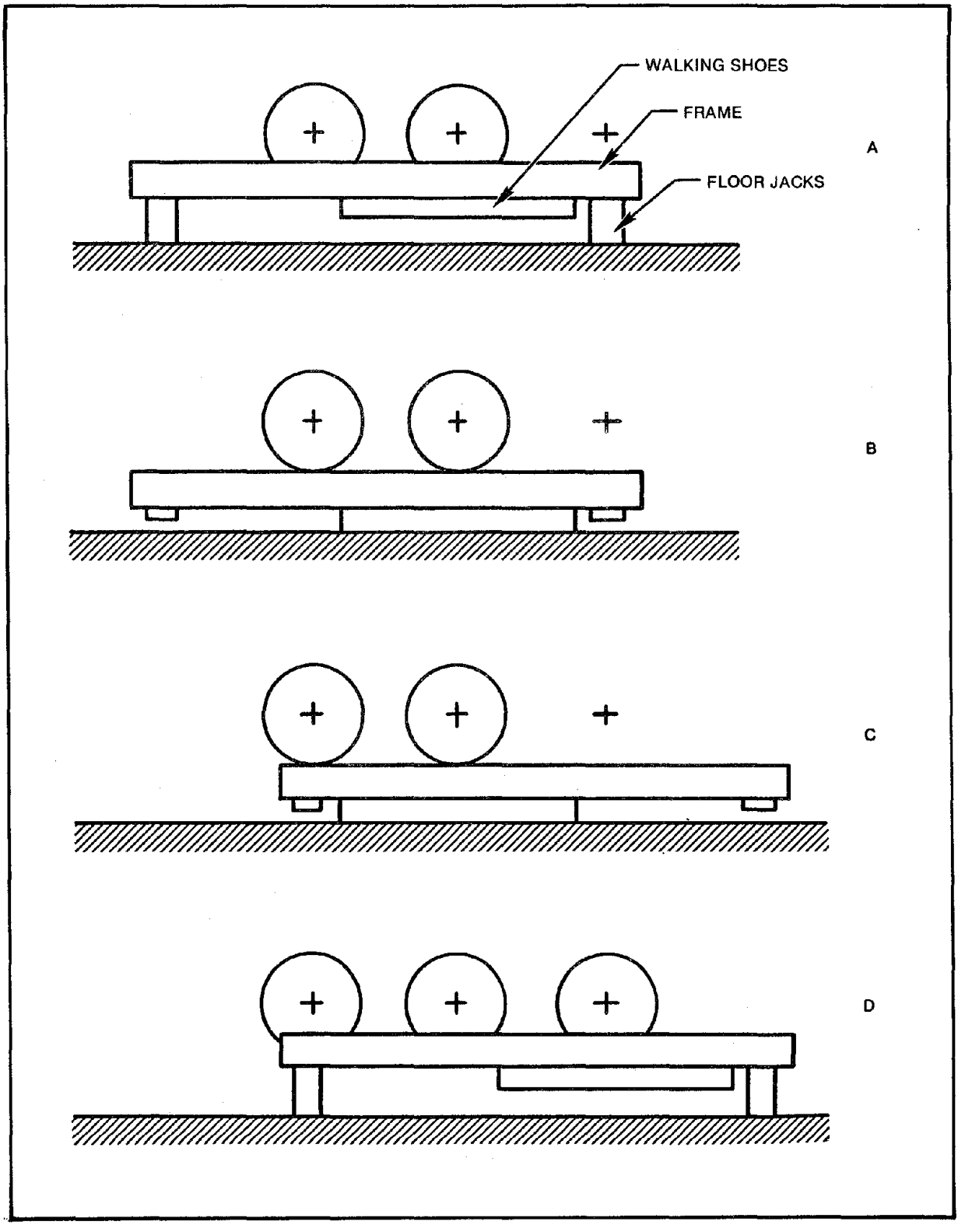


Figure 37 SEQUENCE FOR SHIFTING THE AUGER MINER LATERALLY FOR BOREHOLE SPACING

This control system utilized the same DC power supply as the cutterhead rotation control system, and it too was granted intrinsically safe designation from MSHA.

To indicate that the frame had reached the desired attitude, panel lights which glowed when an out-of-level condition existed extinguished themselves when the solenoid valves nulled.

During operation, if the frame tilted more than one-quarter degree, the indicator light(s) would glow to alert the operator that a releveling was necessary.

The automatic "leveling" system on the auger miner, as it came to be called (although usually the null position had the machine's frame rocked backwards slightly), worked very well mechanically. Unfortunately this added and repeatable control did not significantly improve our proportion of misdirected boreholes. As will be discussed later, internal forces within the coal block affected borehole trajectory far more than any external attempt at guidance by angle-of-attack setting.

Rotational alignment with the rib to bring the drilling carriage into perpendicularity with the local rib was accomplished by using the pivot steer feature of the tram system.

8.1.10 Sub-Auger Conveyor Clearance

The sub-auger conveyor was mounted on the Coalbadger on fixed horizontal tubular mounts. When spilled coal or other rubble prevented the conveyor from settling on the floor, drilling attitude could be affected. Also, tramming was compromised because the conveyor often scrubbed along the floor. A cylinder-powered, four-bar linkage mount on each end of the auger frame was devised for the second prototype. It allowed the operator to extend the conveyor forward and down to the floor for drilling, and then retract it rearward and up in an arc to a position which offered increased clearance for tramming. It was felt that some rubble could be "plowed" out of the way of the conveyor when positioning it for drilling and to a certain extent this was true, but some shoveling was occasionally necessary to prepare the corner so that the conveyor could be moved close to the rib. Figure 38 shows the linkage which suspended the sub-auger conveyor.

8.1.11 Conveyor Pans

Rust on the previous conveyor pans when left idle for a while caused adhesion between the belt and pans. Because the pans were not especially worn, an attempt was made to apply a plastic sheet wear surface to them. This process was abandoned when the material supplier was unable to deliver material without bonding defects. The carbon steel pans were removed from both conveyors during rebuild and were replaced with stainless steel pans. These pans resolved the belt adhesion problem during the second demonstration.

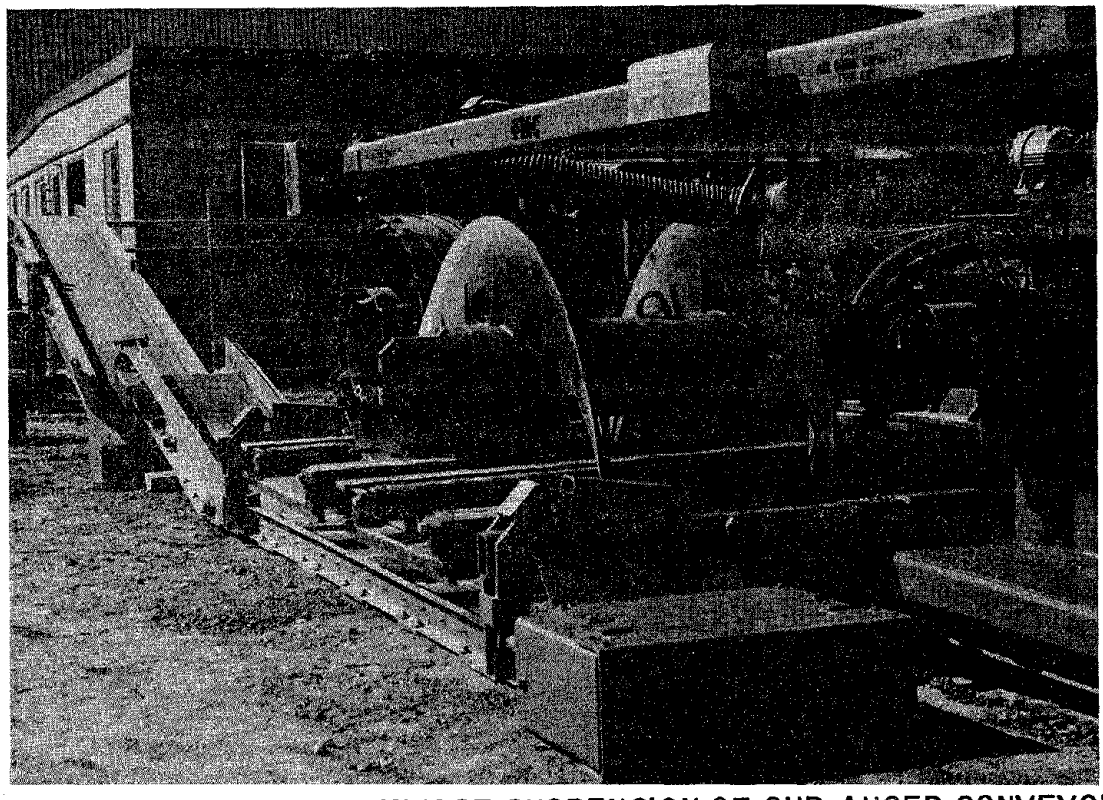


Figure 38 FOUR-BAR LINKAGE SUSPENSION OF SUB-AUGER CONVEYOR

8.1.12 Ventilation Hose Support

The wire-stiffened corrugated hose used on the Coalbadger was difficult to handle and continually became pinched or torn by movement of the drilling carriage or the shiftable frames.

The airbox from the first prototype was modified as follows (see Figure 39):

- The exit conduit was relocated to the top of the airbox
- The exit conduit was reshaped into a sharp right-angle elbow and reduced in diameter to 10 inches
- The elbow was detachable at its lower base and allowed to rotate freely about a vertical axle as the drill carriage pumped and retracted.

Smaller diameter (10-inch) corrugated tubing was installed on the airbox and run to a methane monitoring chamber mounted on the monorail. A tubular steel rub rail supported the tubing between the airbox and the chamber. Beyond the methane monitoring chamber, identical tubing ran from the machine to a sheet metal adapter mounted to the mine's standard rigid fiberglass ventilation tubing as shown in Figures 40 and 41.

In practice, a half-length of tubing was used as the last section near the miner when necessary to allow the corrugated tubing to always reach the rigid tubing. The half-length was inserted as required when moves were made.

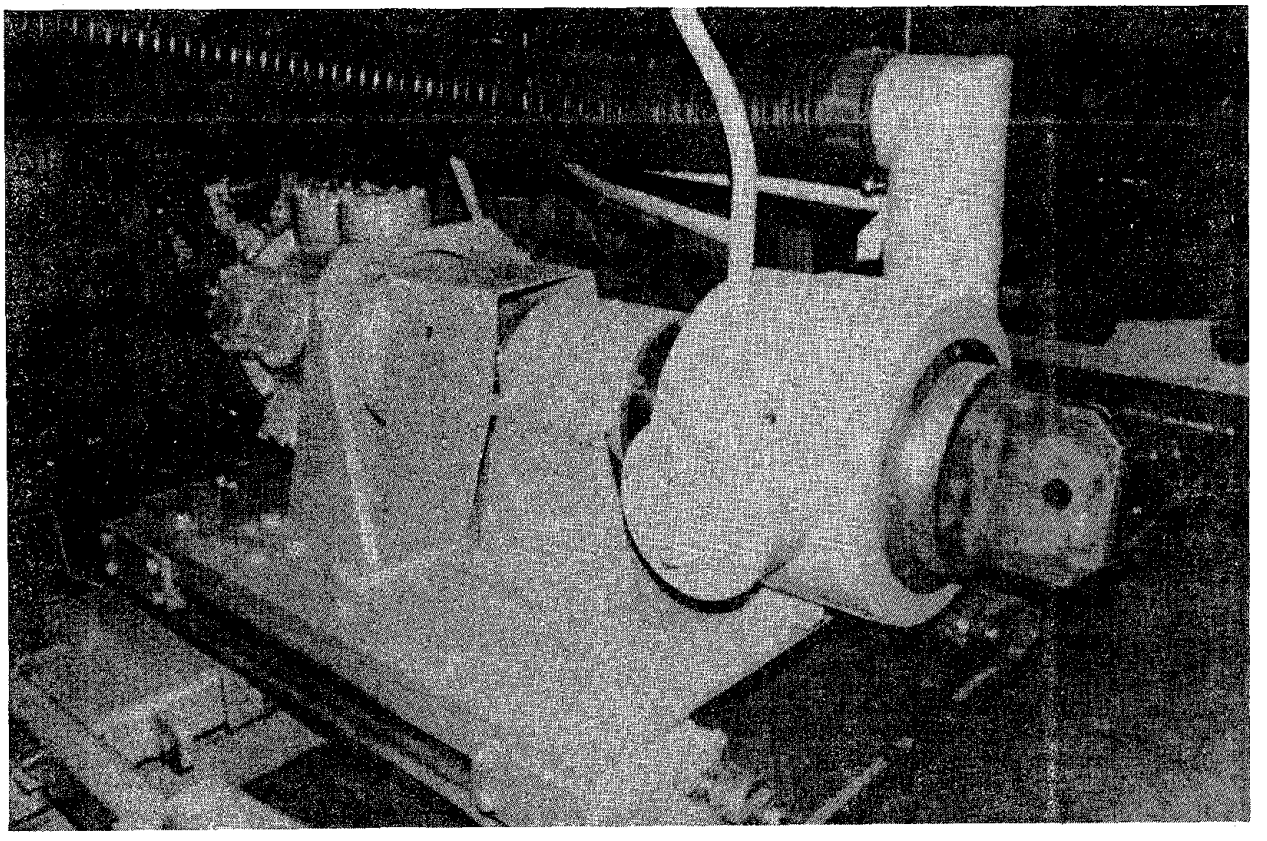


Figure 39 IMPROVED AIRBOX AND VENTILATION TUBING SUPPORT

Although MESA did not object to the previous methane monitoring system, FMC felt it would be preferable to place the detector directly in the air-stream. A diagram of the redesigned system is shown in Figure 42. Note that the hand-held methanometer -- required for the 20-minute methane checks -- sampled the air from the same chamber through a small diameter tube. The readings could be compared rapidly.

8.1.13 Auger Handling System

The mine had desired a hydraulic winch to pull cutterheads and flights along the ground during certain operations, such as setting up to drill the first hole after crossing an intersection. Such a winch was mounted at the left rear corner of the machine -- the same side as the cantilevered portion of the monorail hoist. The miners soon discovered that pulling the flights over the uneven ground and trying to guide them was more trouble than simply manhandling them and the winch saw little service.

8.1.14 Canopy

The original Coalbadger canopy and operator's station was crowded and required that the operator sit cross-legged or with his knees tucked up (refer to Figure 4 in Section III, Part 1 of this report). It also introduced a dangerous pinch point which FMC had corrected with a sheath.

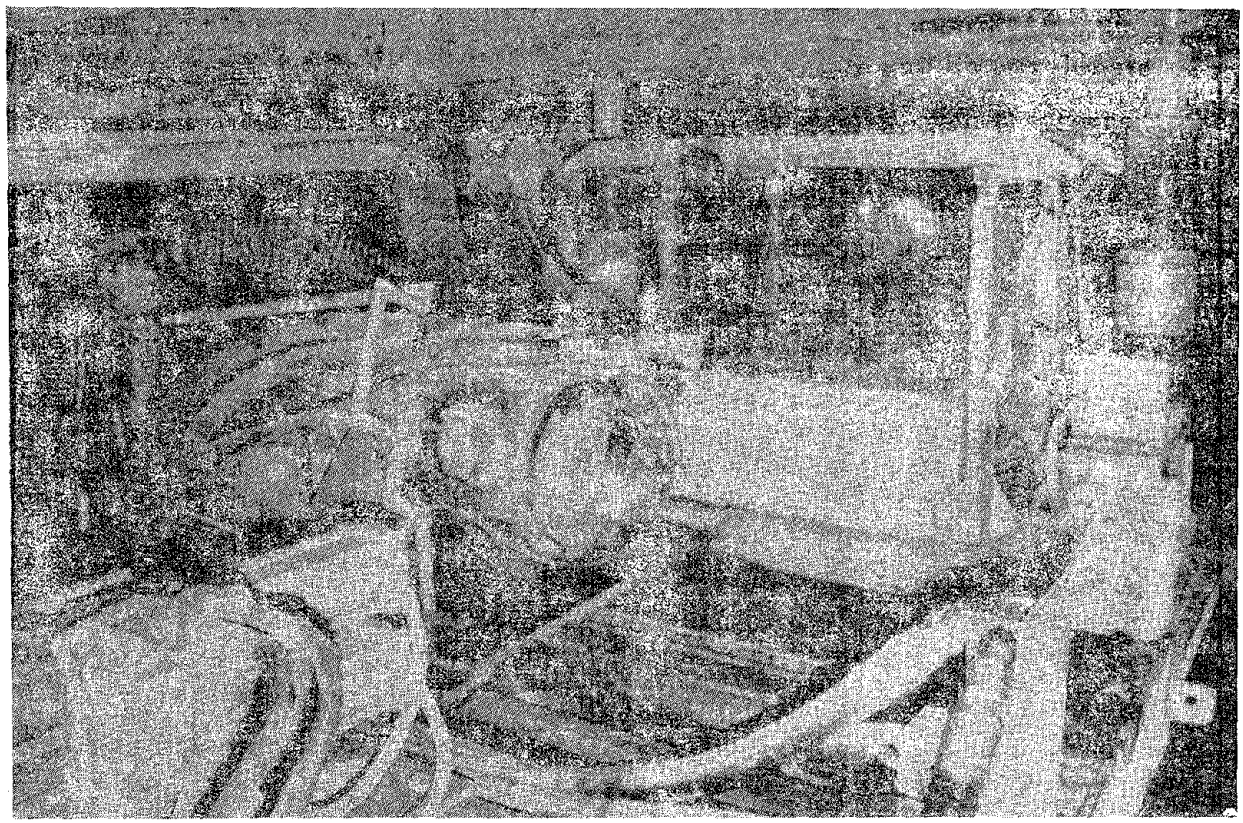


Figure 40 VENTILATION TUBING CONNECTIONS AND GUIDE RAIL

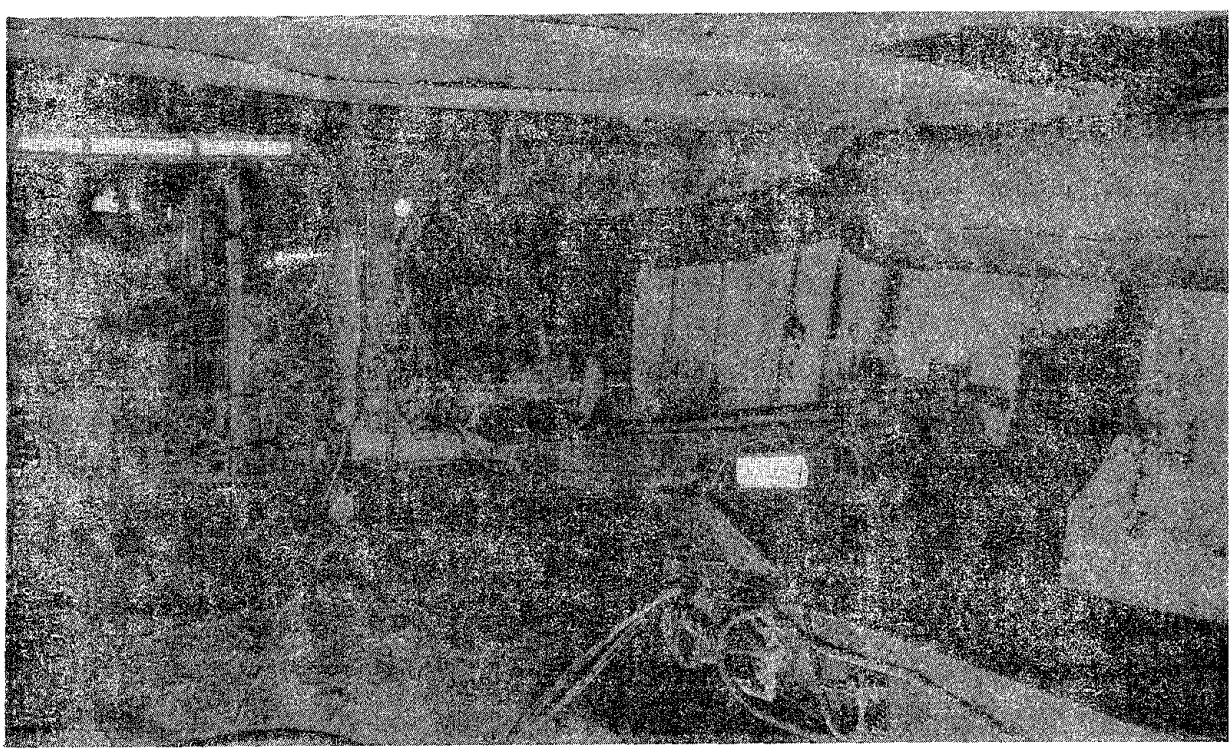


Figure 41 CONNECTION OF AUGER VENTILATION TUBING TO MINE'S RIGID TUBING

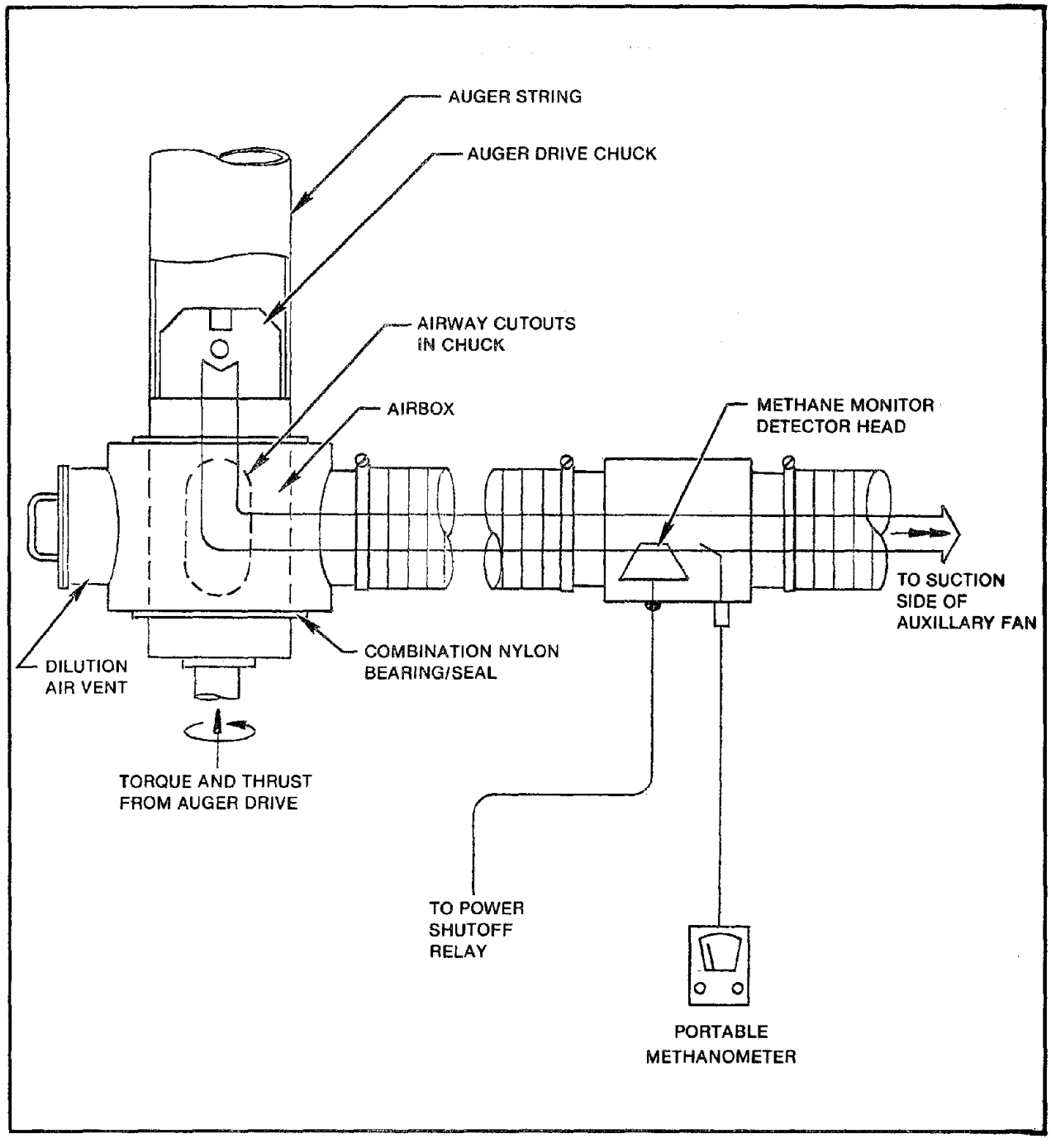


Figure 42 AIR FLOW THROUGH AUGER CHUCK AND AIR BOX AND METHANE SAMPLING DIAGRAM

The canopy/operator's station on the second prototype, as shown in Figures 36 and 43, featured an adjustable seat (up-down, forward-back, and inboard-outboard). The hydraulic controls were laid out in functional groups as discussed in Appendix H, and the seat incorporated an armrest and the rotation control joystick.

Since the operator sat near the high-pressure hoses leading to the drive motor, a lexan scattershield was fitted to the inboard side of the canopy. This layout afforded the operator a clear view of the drill and extractor carriages and the hoist operator stationed on the opposite side of the machine.

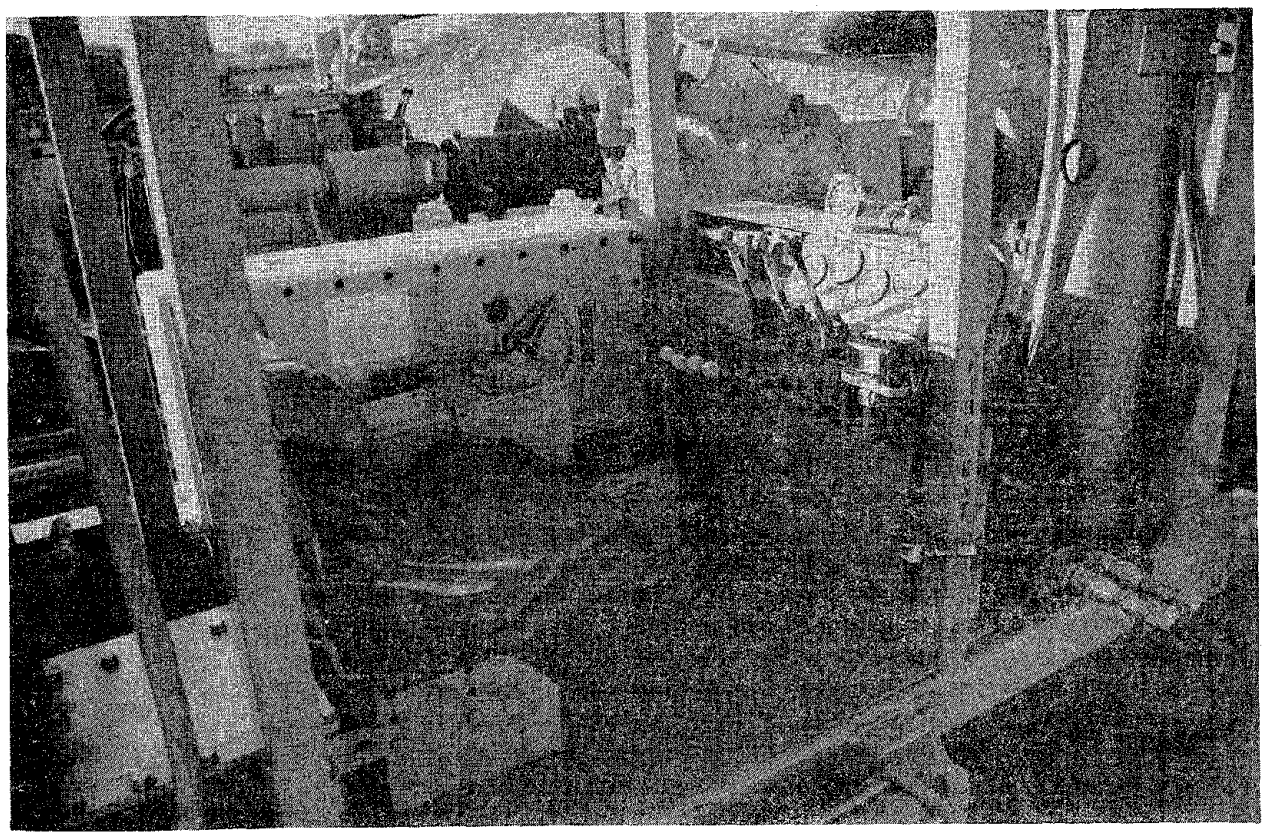


Figure 43 OPERATOR'S COMPARTMENT ON SECOND PROTOTYPE
(Note bracket holding the portable methanometer.)

8.1.15 Improved Coal Cleanup

Both FMC and Ireland Mine foresaw the need to reduce spillage. Not only does spilled coal present a hazard, coal not loaded out does not count as production. However the irregularities of Ireland Mine's pillar ribs did not lend themselves to a simple solution to the spillage problem. It was finally concluded that pivoting the sub-auger conveyor to allow it to seat as close to the rib-floor intersection would in itself minimize spillage. While in the final analysis it did help to reduce spillage, the

spillage was still greater than desired and a significant amount of shoveling was done. Control of spillage remains one of the unsolved technical challenges of auger mining.

8.1.16 Improved Entry Conveyor Handling

Ireland Mine had been unable to use the original rigid tow bar on the entry conveyor and instead had attached a dangling wire rope and hook to the underside of the tailboom which was attached through a hole in the tip of a scoop bucket. Obviously this approach was imprecise, and precise relocation of the entry conveyor was required. The basic idea, however, had merit because connection and disconnect times were shortened and no special hitches or mounts were necessary on the scoop. Rather than return to a rigid tow bar, an improved wire rope tow bridle was conceived for the underside of the tailboom with a three-point cable suspension terminating in the same type hook as before. The main improvement was in lateral control, with no sacrifice in hookup/unhook speed. The operation was about as good as could be expected from an unpowered (undercarriage), towed conveyor.

Subsequent discussions with Ireland Mine as the second prototype auger was being fabricated revealed their desire to revise the entry conveyor to have its discharge height adjustable. Their reasoning was logical, both from the standpoint of better filling of the shuttle cars, and the desirability of lowering the profile for towing from place to place.

The undercarriage of the entry conveyor had always been manually adjustable by pulling a pair of pins in the tubular steel supports and adjusting the tailboom height with the scoop bucket. Basically the same structure was retained, and two small hydraulic cylinders were added to power the conveyor structure up or down in relation to the wheeled undercarriage. For safety, a position-holding valve was used to keep the cylinders from creeping under load, and the safety pins through the tubular uprights were also retained. Figure 44 shows the elevating undercarriage. Note the high discharge height of the conveyor which allows a good engagement into the shuttle car.

The entry conveyor also had a large spill pan or "bib" attached to its intake end. This was required to prevent minor misalignment of the two conveyors from causing excessive spillage. The spill pan is shown in Figures 38 and 41.

8.1.17 Spare Auger Flights

Ireland Mine requested three spare auger flights, based on the continued presumption of drilling 100-foot blocks. This requirement became unnecessary when it was later decided that future drilling would be limited to 84-foot blocks.

8.1.18 Auger Guidance

It was well recognized by all participants in the first demonstration that improvements in drillstring guidance were necessary if long-hole augering were ever to become a viable underground extraction technique. It was

observed that the play between sections of the drillstring might be contributing to the wanderings of the cutterhead. As such it was felt that rigidizing at least the cutterhead and its following flight might enhance control of borehole trajectories.

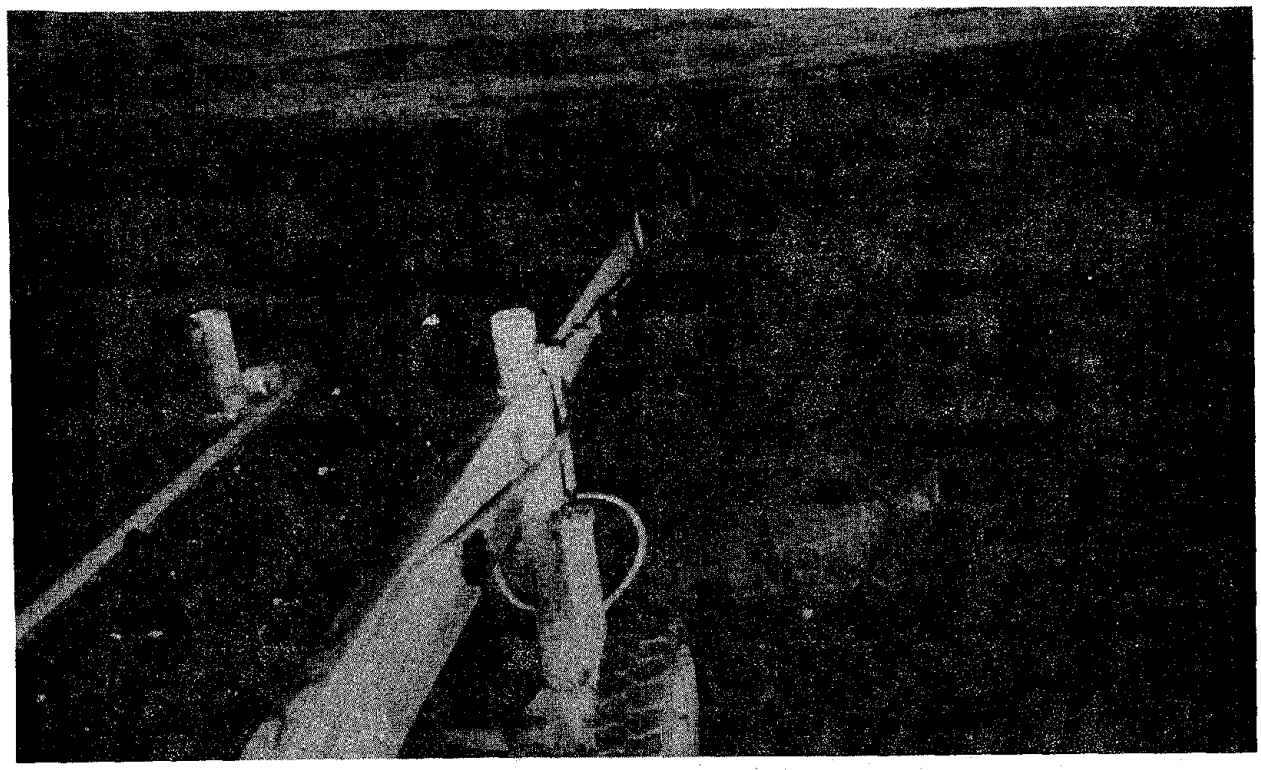


Figure 44 ADJUSTABLE, POWERED UNDERCARRIAGE ON ENTRY CONVEYOR

One design concept proposed to Ireland Mine called for a mechanical connecting device between cutterhead and following flight which would have to be secured by the operator or a mechanic. Because of current MSHA rulings, Ireland Mine felt that making and breaking this connection manually in each borehole would compromise safety and production to an intolerable degree. FMC concurred, and it was finally decided to edge bind the scroll portion of two auger flights with a steel wear bar sized out to nearly the borehole gauge. Normally following flights were a full 2 inches smaller in diameter than the cutterheads. These special flights were purchased, but when it was attempted to use them, it was found that they bound up in the borehole and prevented further drilling. Use of the special flights was abandoned; however the theory is sound and had the diameter of these flights been even a half-inch smaller, they may have worked.

8.1.19 Methane Monitor

Ireland Mines had requested that the second prototype auger miner come equipped with a Bacharach "low-profile" methane monitor. Pragmatically, from the mine's point of view, they were the standard unit on nearly all their other equipment; they were familiar with their operation, calibration,

and repair; they had calibration kits, and the Preiser-Mineco unit on the original Coalbadger had never worked satisfactorily. The rebuilt auger featured the low-profile methane monitor adjusted to alarm at 1 percent methane concentration, and shut off power to the machine at 2 percent concentration per Federal regulations.

8.2 ADDITIONAL MODIFICATIONS AND ADDED FEATURES

The preceding discussion deals only with the 19 specific recommendations made at the conclusion of the first prototype demonstration and their embodiment on the second prototype.

Several other modifications, improvements, and accessories were added to the auger miner system during the redesign and manufacturing stages of the development of the second prototype. These are discussed in detail in the following sections.

8.2.1 Extractor Frame

Production figures from the first prototype were disappointingly low and, although much of the problem was accountable to the various downtimes attributed to the machine and the mine's equipment, the Coalbadger machine's normal cycle of operation, when critically examined, was found to have inherently counterproductive elements.

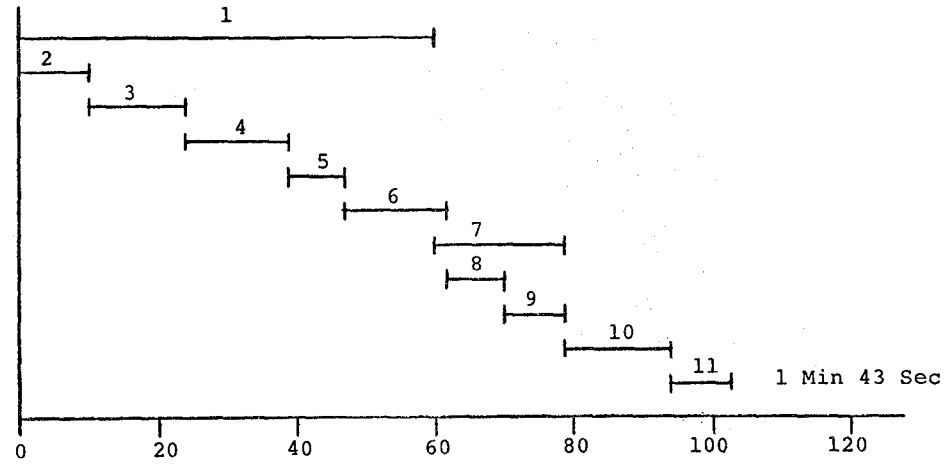
As discussed in Part 1, the drilling cycle of the Coalbadger required that the entire upper frame of the machine be shifted one hole spacing back and forth while retrieving flights from the previous hole. Even when done at normal operating rates, much time passed from the time one flight was unchucked and the next flight was chucked and ready to drill.

The obvious conclusion was that if this sequential flight retrieval operation could somehow be made simultaneous with drilling, production would rise significantly.

For comparison purposes, typical drill cycles outlining all the cycle elements for both the Coalbadger and the anticipated extractor frame concept machine were plotted. As shown in Figures 45 and 46, the overall cycle time on the Coalbadger was 2 minutes and 39 seconds, while the predicted cycle time for the second prototype was 1 minute and 43 seconds. This represents a 35-percent decrease in cycle time and, exclusive of other machine changes, could be expected to increase production by 54 percent. The promise of such a large production increase made it imperative that the new machine have a separate extractor device.

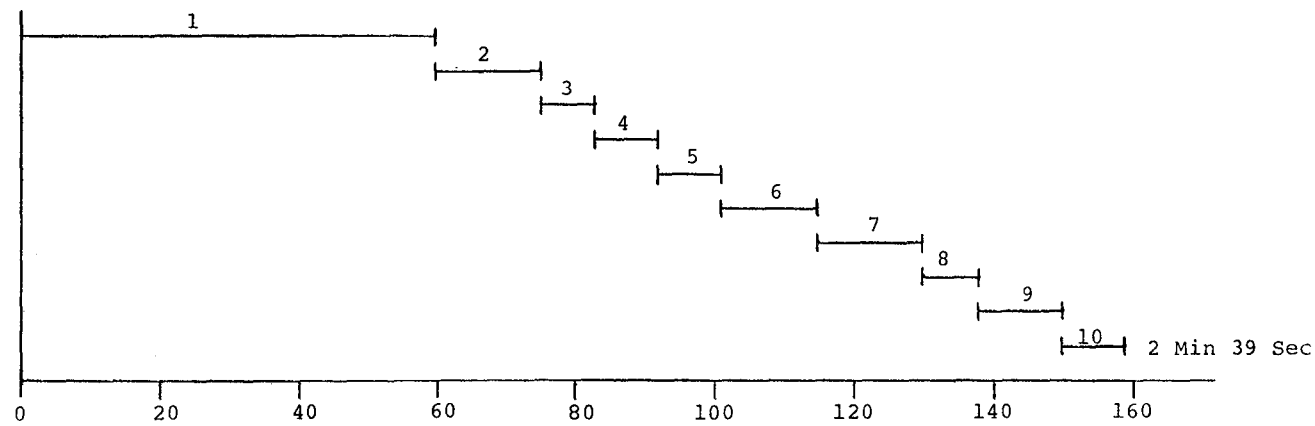
Mechanically this could not be done using the drilling carriage for extracting flights, so the concept of an extractor carriage on the auger main frame was developed. The extractor carriage was designed to have a high-torque hydraulic rotary actuator capable of turning the auger string to align the box connections or unjam the string and a tang similar to that on the drilling carriage to engage the flights.

The extractor carriage is visible in Figure 35. Figures 47 and 48 show the extractor carriage in use underground. The center distance between



	OPERATION	ELEMENTAL TIME (SEC.)
1.	Cut	60
2.	Connect Extractor	10
3.	Withdraw Flight	14
4.	Pull Pin	15
5.	Disengage Flight	8
6.	Pull Pin (New Flight)	15
7.	Withdraw Drive Chuck	19
8.	Disengage Extractor	8
9.	Prepare Flight for Transfer	9
10.	Transfer Flight	15
11.	Connect Flight and Retract Transfer Mechanism	9

Figure 45 TYPICAL AUGER CYCLE (EXTRACTOR FRAME CONCEPT)



	OPERATION	ELEMENTAL TIME (SEC.)
1.	Cut	60
2.	Pull Pin	15
3.	Withdraw Chuck	8
4.	Shuttle to Previous Hole	9
5.	Connect to New Flight	9
6.	Withdraw Flight	14
7.	Pull Pin	15
8.	Disengage New Flight	8
9.	Shuttle to New Hole	12
10.	Connect New Flight	9

Figure 46 TYPICAL AUGER CYCLE (EXISTING COALBADGER SHUTTLE FRAME)

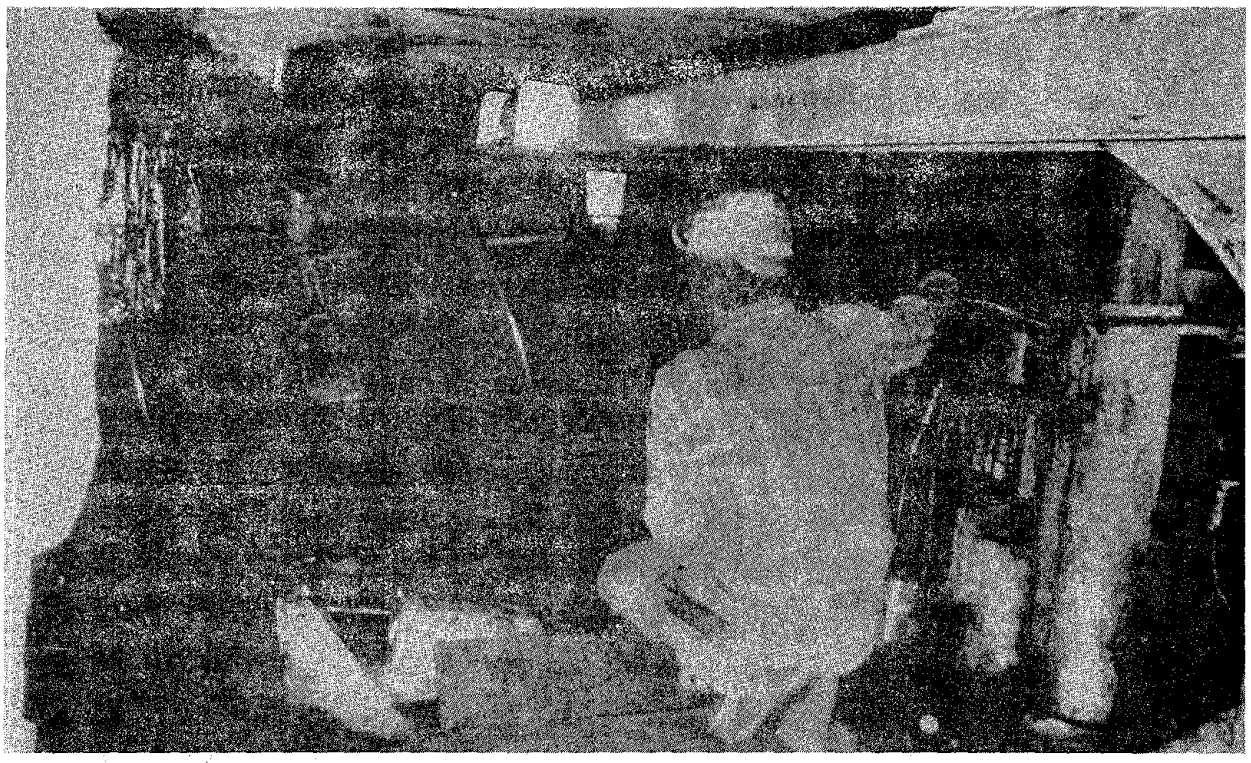


Figure 47 EXTRACTOR OPERATOR'S CONTROL STATION

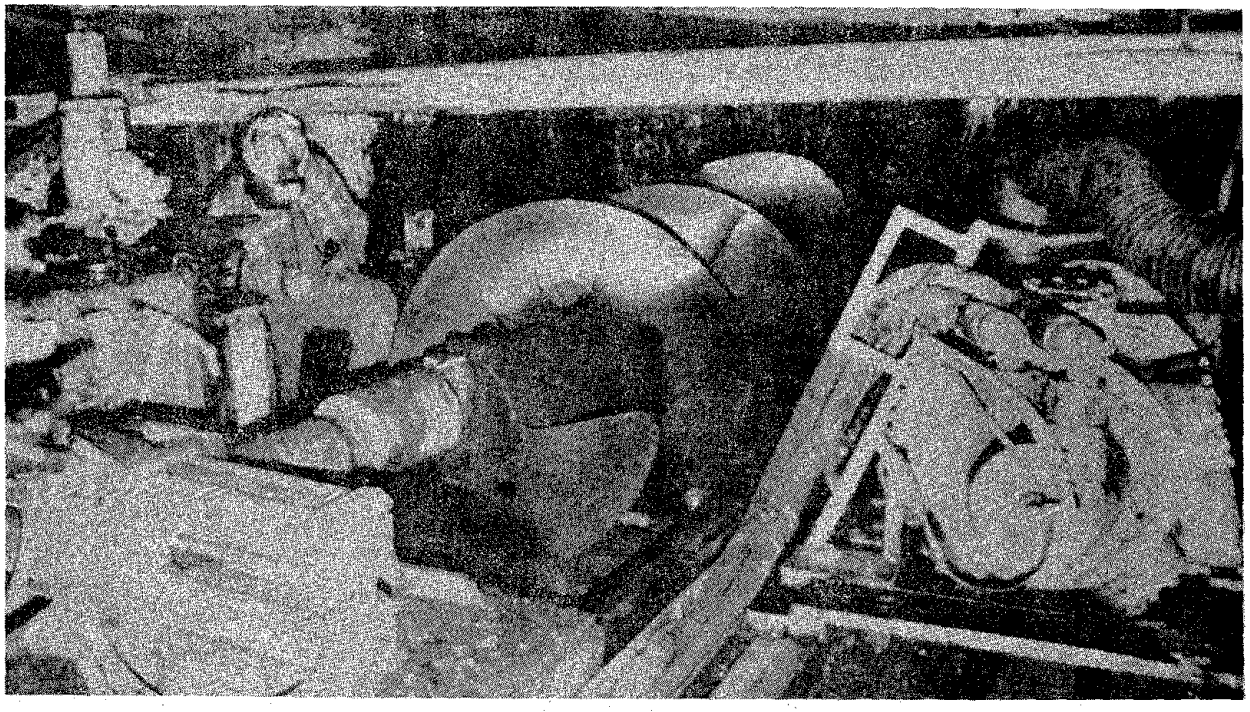


Figure 48 EXTRACTOR WITHDRAWING A FLIGHT (Note drill carriage has just completed drilling a flight.)

carriages was 57 inches which allowed a minimum 15-inch web between boreholes. It was possible to shift the carriages slightly farther apart and adjust the stops on the walking shoes to match the center distance.

As will be discussed in Section IX, the existence of the separate extractor did result in significantly improved production performance. It was usually the case that the extractor operator was able to withdraw a flight and have it on the monorail hoist before the drill operator had finished sumping the previous flight. Changeover was accomplished in seconds instead of a minute or more on the Coalbadger.

8.2.2 Backreamer

Both Bureau of Mines (while the project was under their sponsorship) and DOE project personnel had been interested in enhancing the inherently low recovery of boring auger miners by investigating the principle of back-reaming, or enlarging a borehole while withdrawing the flights and cutter-head from it.

Rather than expend a great deal of design effort on attempting a complex auto-expanding cutterhead or some other equally complex device, FMC fabricated a simple, fixed-diameter backreaming cutterhead as shown in Figures 49 and 50. This head was designed for experimental use only to test the concept of backreaming. It was intended that a borehole be drilled with the auger and that the regular cutterhead be pushed far enough through so that it could be disconnected and removed. The backreaming head would then be attached and the drillstring powered in reverse thrust. FMC's backreamer provided a 48-inch diameter back bore which against the 42-inch primary bore resulted in a 31-percent increase in recovery from a single hole. The ideal diameter ratio for backreamers is 1.41:1 which results in equal production from both the first borehole and the backreaming. Consideration must be given to allowances for web thinning. The overall recovery increase from a backreaming scheme can ultimately fall to 10 percent or less, so it is as yet unproven that a backreaming mining plan would be economically justifiable.

FMC agreed with Ireland Mine that if an opportunity arose, the backreamer would be tried. It was expected that the backreaming experiment would cause some production loss on the shift in which it was attempted and likewise that a single attempted backreamed hole would be sufficient to prove or disprove the feasibility of the practice.

As will be discussed in Section IX, a single hole was successfully backreamed before the demonstration ended.

8.2.3 Improved Lighting

The original Coalbadger had four 150-watt incandescent floodlights. Although the illumination was adequate to work safely, the second prototype was required to meet the present regulations which called for .06 foot lamberts in the working area. In addition to the four 150-watt incandescent floodlights, two 40-watt fluorescent area lights were added to the machine, one on each side. The isointensity diagrams predicted compliance with the regulations and when MSHA checked the machine in place underground with a "go/no-go" light meter, compliance was verified.

8.3 GENERAL SPECIFICATIONS

The general specifications of the improved auger miner, second prototype, are given in Table 8.

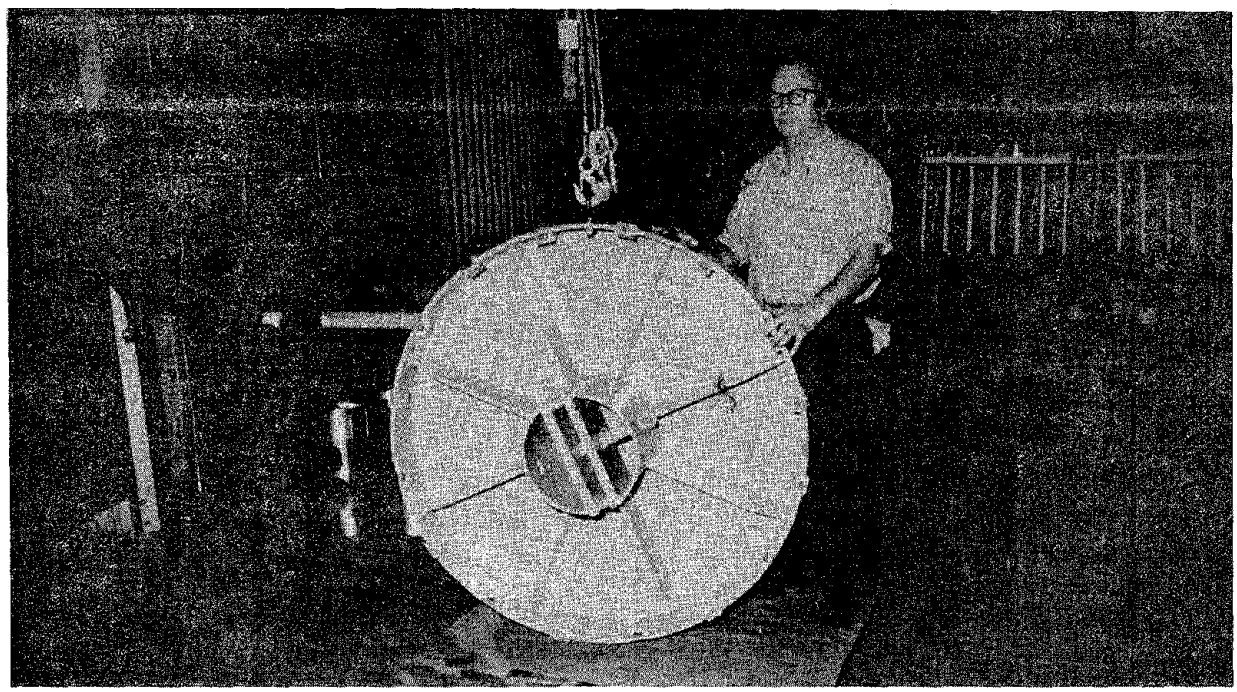


Figure 49 BACKREAMING CUTTERHEAD

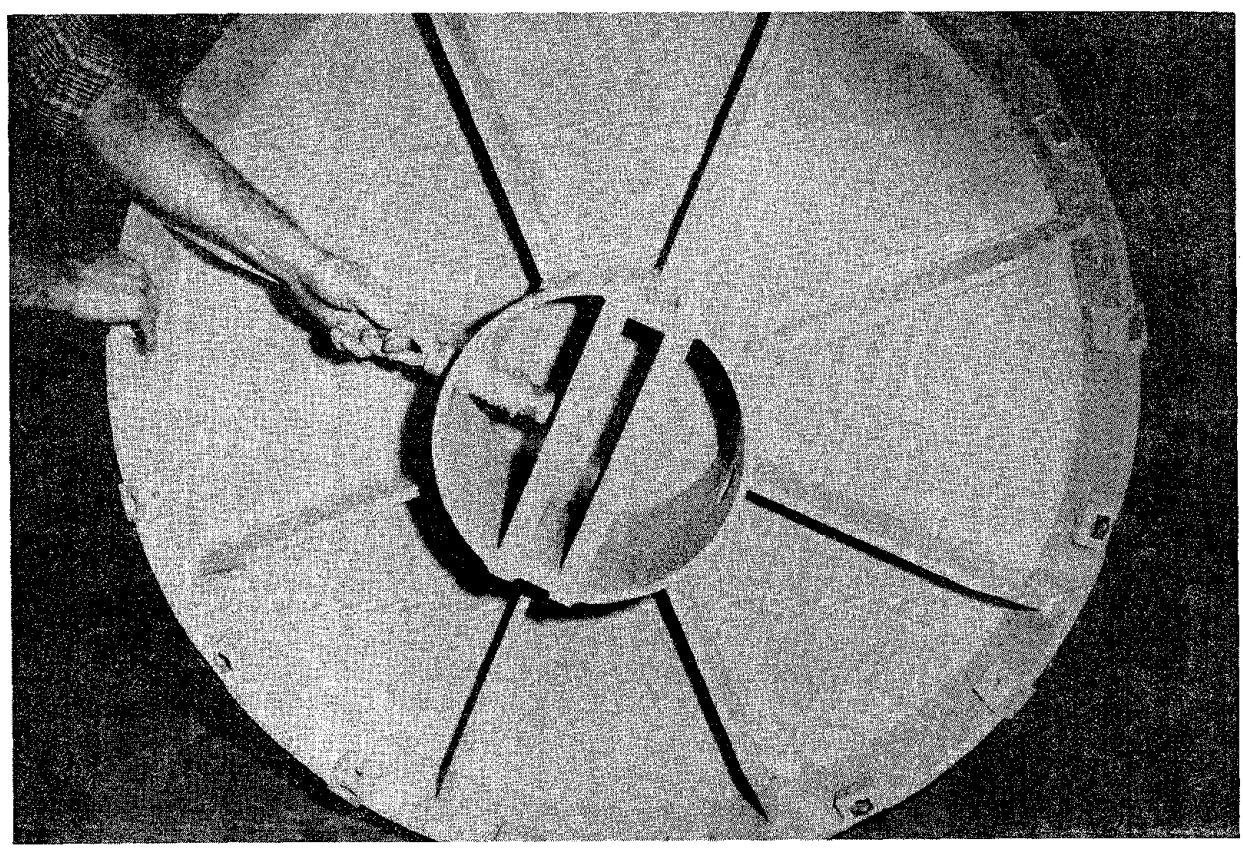


Figure 50 LATCH PIN TO ATTACH BACKREAMER TO NORMAL SCROLLED FLIGHT

Table 8 SPECIFICATIONS FOR AUGER MINER SYSTEM – SECOND PROTOTYPE

Specifications

Overall dimensions:

Length:	27'0"
Width:	13'6"
Height:	6'1"
Reducable to:	
Length:	18'3"
Width:	9'0"
Height:	4'10¼"
Total weight: (with 1st stage conveyor)	30,550 lbs.
Reducable to:	11,750 lbs.
Walking shoe stroke:	57 inches
Walking shoe bearing area:	1,340 sq. in. each 2,680 sq. in. total
Floor jack bearing area:	109 sq. in. each 436 sq. in. total
Ground pressure:	
Walking shoes:	11.4 psi
Floor jacks:	70 psi
Roof jack stroke:	30 in.
Roof jack thrust:	variable to 25,000 lbs. max. each 100,000 lbs. max. total

Power and Performance:

Auger Main Drive:

440 VAC 3Ø 60 HZ 200 KVA: Louis Allis Electric Motor:	1750 RPM 150 HP continuous 196 HP intermittent
Sunstrand hydraulic pump Series 26, variable displacement:	100 GPM @ 3000 psi
Sunstrand hydraulic motor 60 series mod 60-K001 P/N MH373JC 10 cylinder radial:	150 HP @ 60 RPM 13,500 ft-lb torque
Rotational speed:	infinitely variable to 62 RPM reversible
Auger sump rate:	
Forward (with conveyor running):	13.2 ft/min
Reverse (without conveyor running):	42 ft/min
Auger sump force:	
Forward:	19,600 lbs @ 1000 PSI
Reverse:	29,500 @ 2000 PSI
Auger auxiliary drive:	
440 VAC 3Ø 60 HZ 70 KVA: Louis Allis Electric motor driving two, 2-section Commercial Shearing Co. hydraulic pumps totaling 65 GPM:	1750 RPM 50 HP continuous
Rotary actuator:	360° rotation
Hydra power TR360-15011:	8,333 ft. lbs. torque
Extractor sump force:	
Forward:	39,000 lbs @ 2000 PSI
Reverse:	29,500 lbs @ 2000 PSI
Extractor sump rate:	
Forward:	22.3 ft/min
Reverse:	29 ft/min

Fire suppression system:	Dry chemical, manual discharge
Methane detection system:	Bacharach
Lighting:	4 incandescent floodlights 2 fluorescent area lights
Borehole ventilation through hollow auger stems (CSE Centrifugal Fan):	1100 CFM in 84-foot hole
Bore hole rockduster (MSA Batam 400)	Dust 84-foot hole in 10 min

Sub Auger Conveyor:

Dimensions:	
Width:	32 inches
Length:	23'7"
Overal height at discharge boom:	36 inches
Weight:	2400 lbs.
Drive, Char-Lynn hydraulic motor:	17.6 HP @ 18 GPM and 1750 PSI
Belt:	
Type:	Long-Airdox Poly-Vinylad Single ply
Speed:	400 ft/min
Width:	20 inches
Capacity:	5.5 TPM

Entry Conveyor:

Dimensions:	
Width over wheels:	54½ inches
Width over siderails:	30½ inches
Length:	27 feet 3 inches
Overal height at discharge boom:	variable, 3' 7" to 6' 4"
Weight:	2950 lbs.
Drive, Char-Lynn hydraulic motor:	8.8 HP @ 14 GPM and 1300 PSI
Belt:	
Type:	Long-Airdox Poly-vinylad Single ply
Speed:	400 ft/min
Width:	24 inches
Capacity:	6.5 TPM

Recovery:

Tonnage per 1 ft. of hole:	785 lbs.
Tonnage per 42 in x 5 ft auger flight (based on 80 lb/cu ft in place):	1.9 tons
Tonnage per 84 ft hole:	31.4 tons
Tonnage per panel (13 hole pattern):	408.2 tons

9.0 DEMONSTRATION RESULTS (Part 2)

9.1 PREPARATION FOR OPERATION

The auger miner, as shown in Figure 51, was completed in February 1978; however, its shipment to Ireland Mine was delayed until September. The 5R-8N test area, as described in the mine map (Figure 52), had not been completed due to the UMWA strike which preceded the eventual settlement of the 1978 national contract. Development work on the 5R-8N area proceeded for several months after the UMWA contract settlement in March 1978, and in August FMC received notification stating that the test section was complete and ready for the auger demonstration.

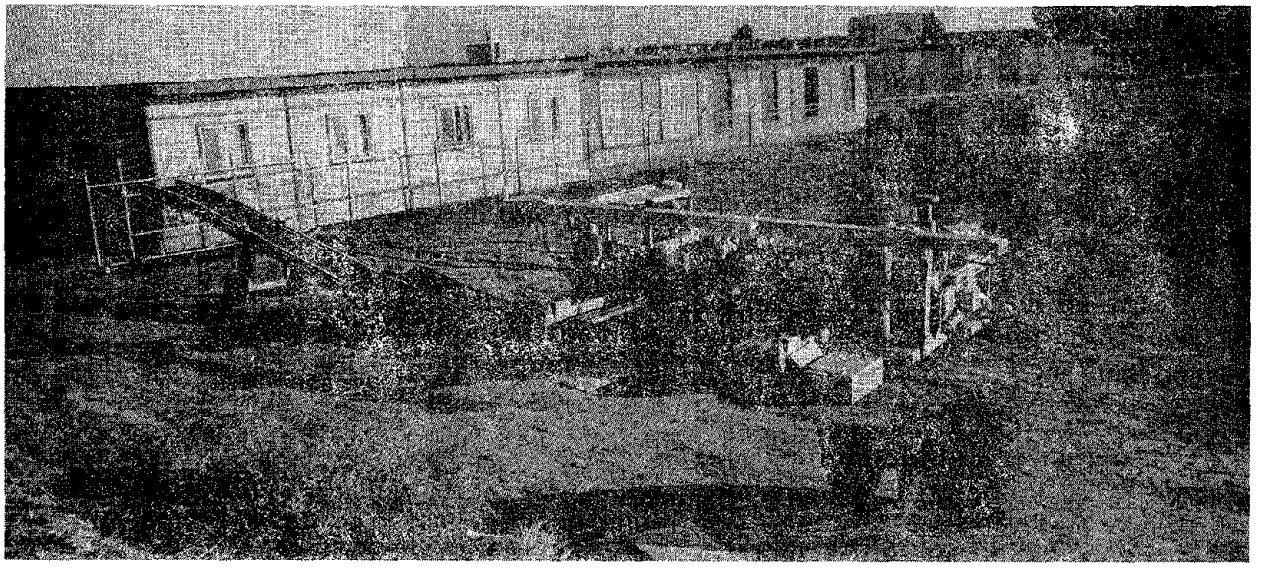


Figure 51 SECOND PROTOTYPE AUGER MINER ASSEMBLED FOR TEST PRIOR TO SHIPMENT TO IRELAND MINE

Two separate test areas of the 5R-8N development were allocated to the demonstration, one on the north side of the main entries containing 20-square blocks approximately 84 feet on a side, and one in the south side containing 17 similar blocks for a total of 37 blocks. A roof fall on the north side shortly after development rendered two blocks potentially unrecoverable depending on the direction of drilling chosen.

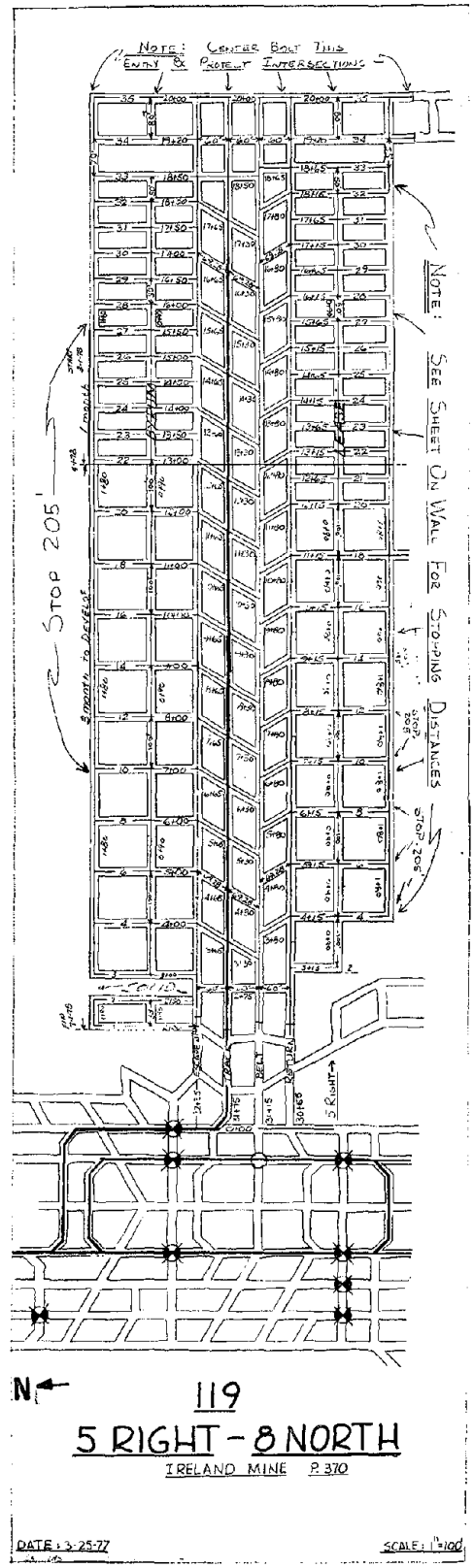


Figure 52 MINE MAP OF 5R-8N TEST SECTION IN IRELAND MINE

Ireland mine had devised a mining plan which would have called for the recovery of all blocks on the south side first, then one cross-entry move followed by the recovery of all blocks on the north side. When considering haulage and ventilation, it was necessary to have the auger drill from north towards south on the south side, and 90 degrees away from that, from west towards east on the north side. Typical equipment setups for these areas are shown in Figures 53 and 54. It was decided that each block would be drilled with 12 or 13 holes, depending on the actual size of block and the appearance of the local roof, ribs, etc. A drilling diagram of a 12-hole block, shown in Figure 55 and Table 9, indicates the nominal dimensions of blocks with varying numbers of holes and the percent recovery from each pattern. The Pittsburgh Number 8 seam in this area (and virtually everywhere else in the mine) has a distinct set of cleats running north-south and east-west. The north-south system of cleats is called the face cleat, and the east-west system is known as the butt cleat. It was later discovered that the auger drilled faster and straighter on the south side and produced better lumps most probably because the cleating promoted better breakage of the coal. On the north side, the coal did not drill as easily, the consist was smaller, and there were many hard rock or clay bands in the seam which dulled the bits and caused many abandoned holes. If these rock bands existed on the south side, the cleating presumably aided in their breakage.

Early in September the equipment was shipped from Santa Clara, and by mid-month it had arrived, been partially disassembled, loaded on flatcars and was ready to take to the section. Figure 56 shows the auger miner as it was about to be dropped down the slope.

Assemblying the equipment only took about four shifts compared to 3 weeks for the Coalbadger. However, mining did not begin until the last day of October. The delay was due to an MSHA ruling which called for changes to the section ventilation, and which required the construction of about 20 block stoppings and the replacement of about 700 feet of track and conveyor belt. At this time it was also learned that MSHA would require that the line of retreat move westward towards the neck of 5R-8N with some mining done on one side and then the other. This would require several cross-entry moves instead of one and the associated lost production, but it was a necessary concession to gain approval of a mining plan.

The numbers in the blocks on Figure 57 indicate the actual sequence of mining and the drilling direction of each block.

By late October, Ireland Mine had rehabilitated the section in accordance with MSHA's requests which included the installation of extra roof bolts, cleanup of spilled coal in the entries, in addition to the stoppings and track and belt work mentioned earlier.

9.2 DEMONSTRATION DATA

On October 31, 1978, the auger drilled a test hole near the end of the shift. Between November 1, 1978, and March 15, 1979, 53 test shifts were conducted. Figures 58, 59, 60, 61, and 62 show the auger in operation. Figure 63 shows a typical drilled-out panel. In the first 13 shifts of operation before

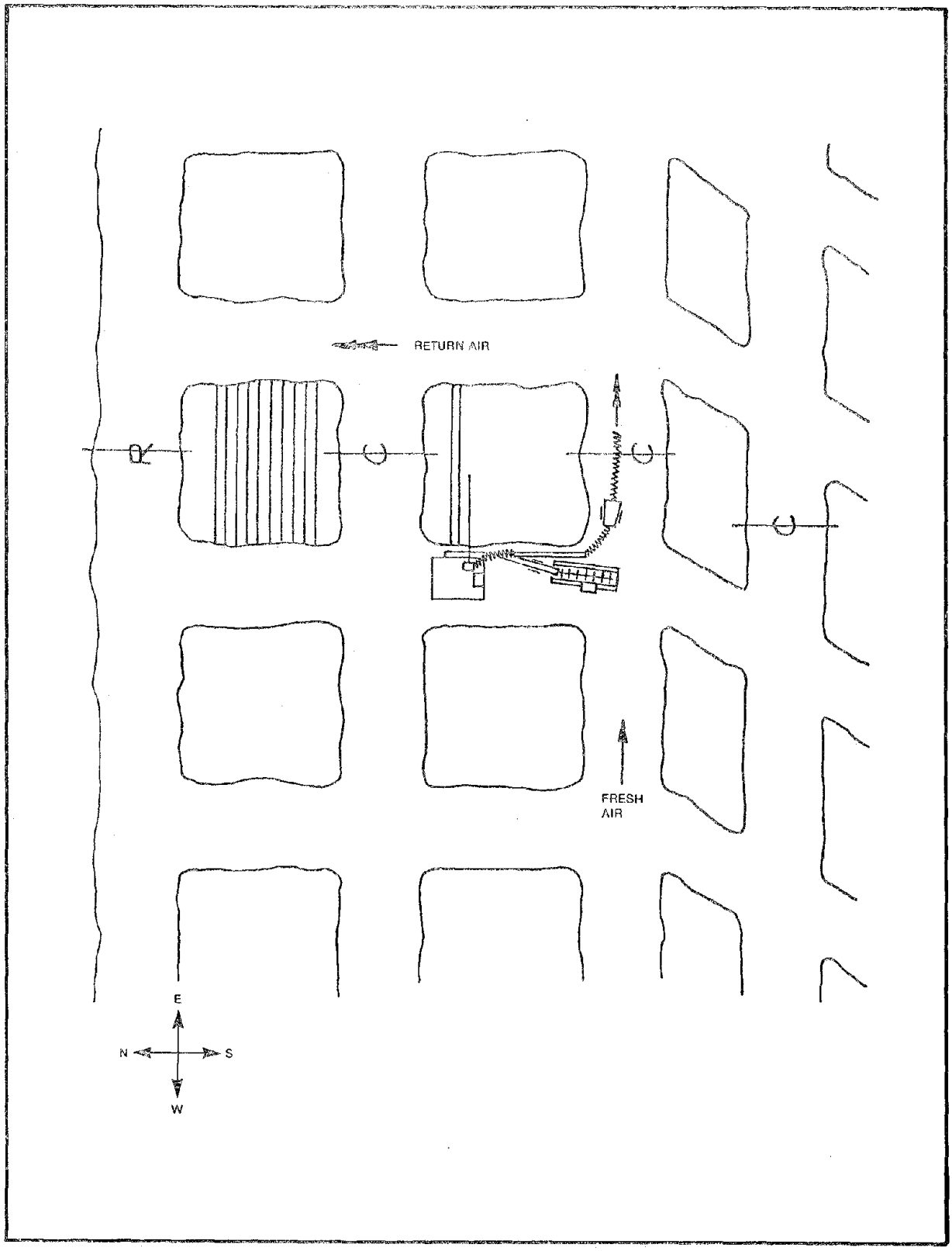


Figure 53 TYPICAL EQUIPMENT SETUP ON THE NORTH SIDE OF 5R-8N

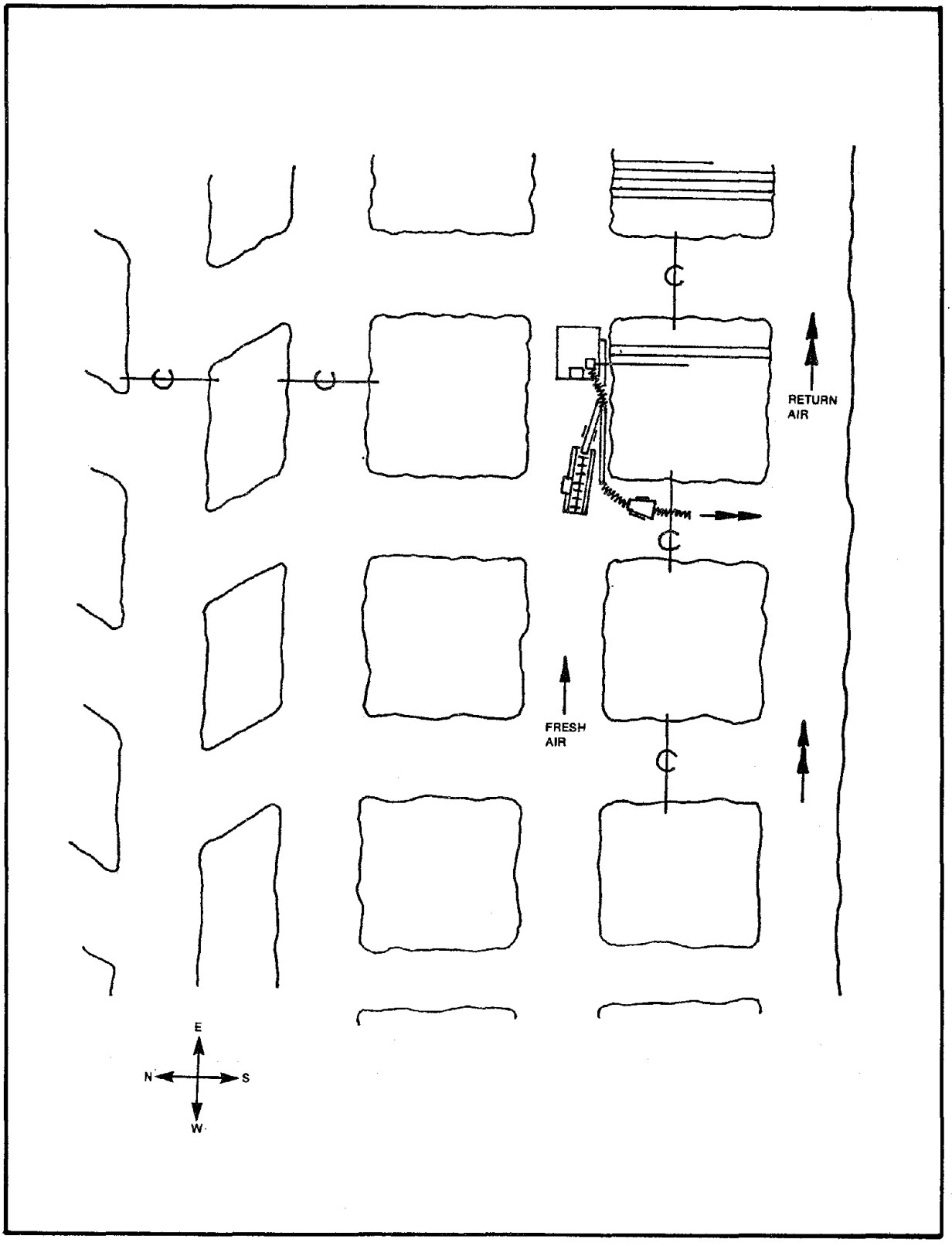


Figure 54 TYPICAL EQUIPMENT SETUP ON THE SOUTH SIDE OF 5R-8N

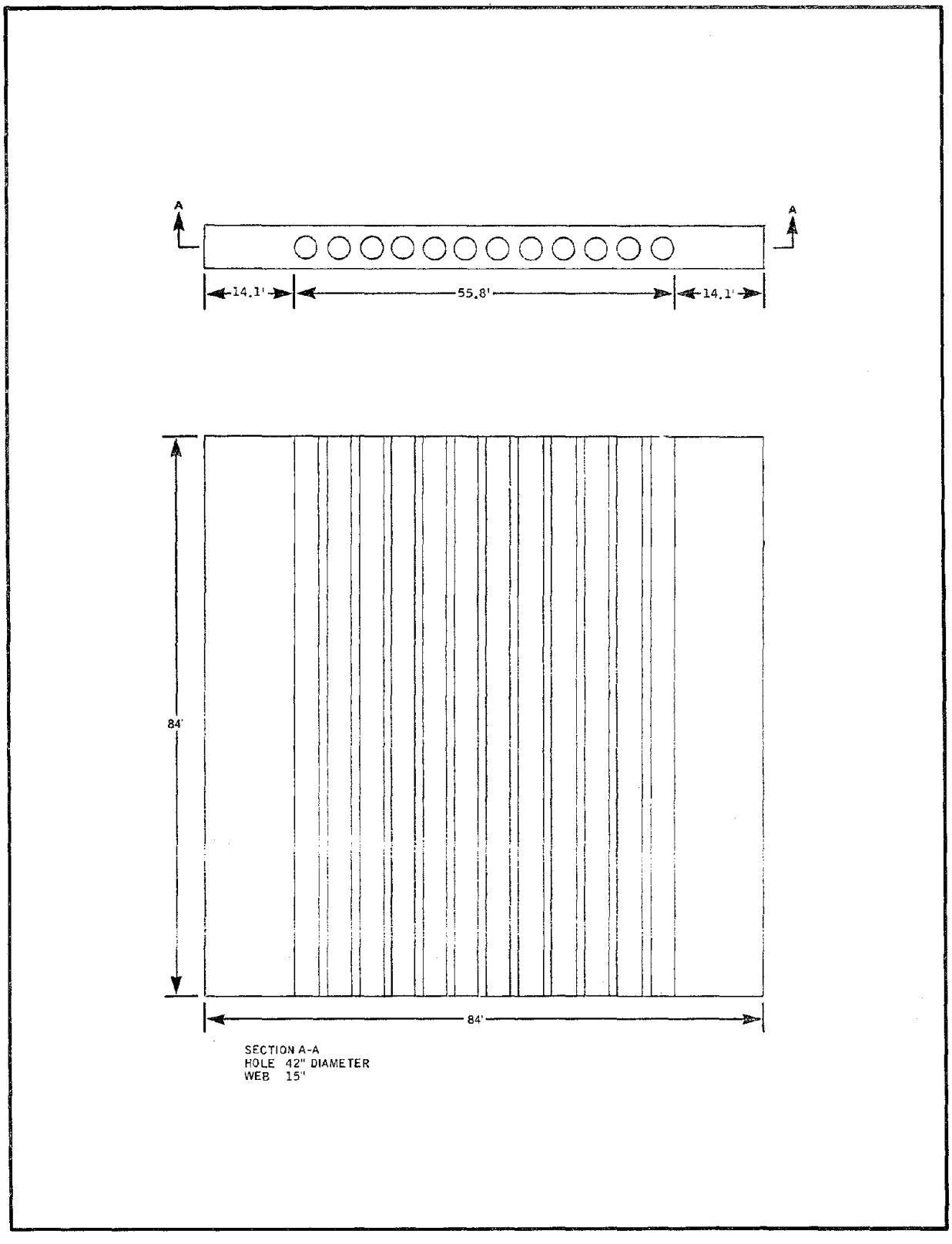
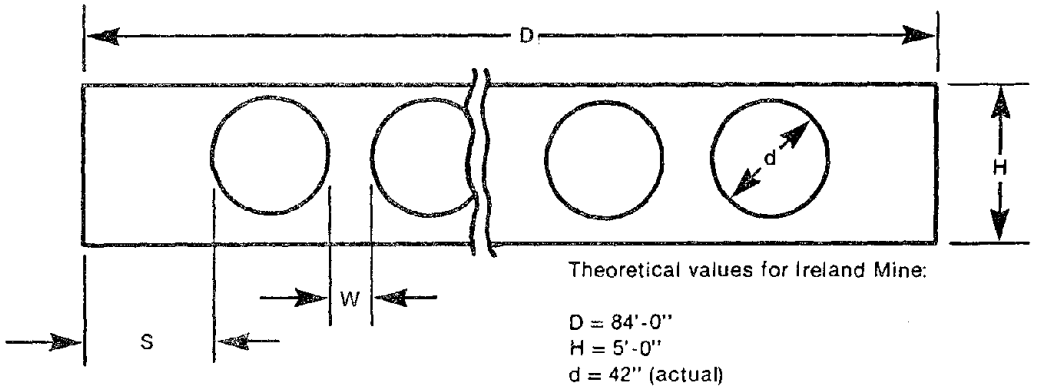


Figure 55 TYPICAL BOREHOLE PATTERN

Table 9 BLOCK DIMENSIONS AND THEORETICAL RECOVERY



Theoretical values for Ireland Mine:

D = 84'-0"
H = 5'-0"
d = 42" (actual)
w = 15" (actual)
n = Number of boreholes
S = Standoff

n	S	Recovery, percent
10 holes	18'-11"	23
11 holes	16'-6"	25.2
12 holes	14'-2"	27.5
13 holes	11'-9"	29.8
14 holes	9'-5"	32.1

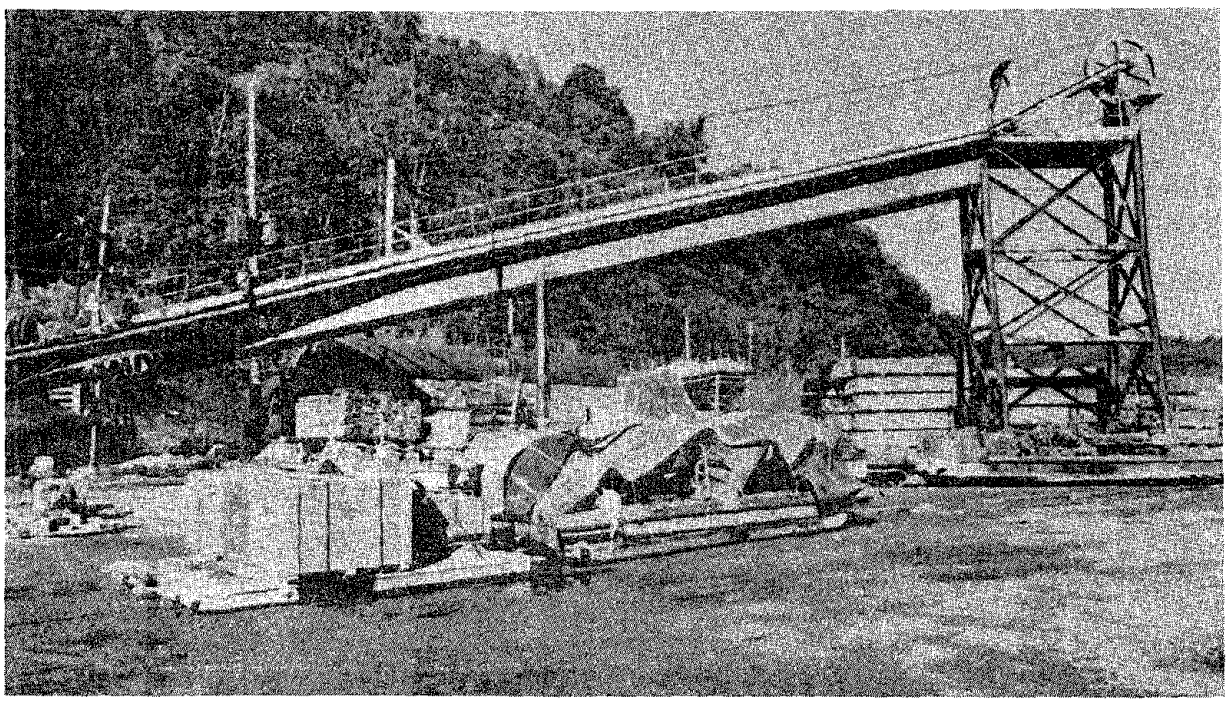


Figure 56 AUGER EQUIPMENT PREPARING TO GO DOWN THE SLOPE AT IRELAND MINE

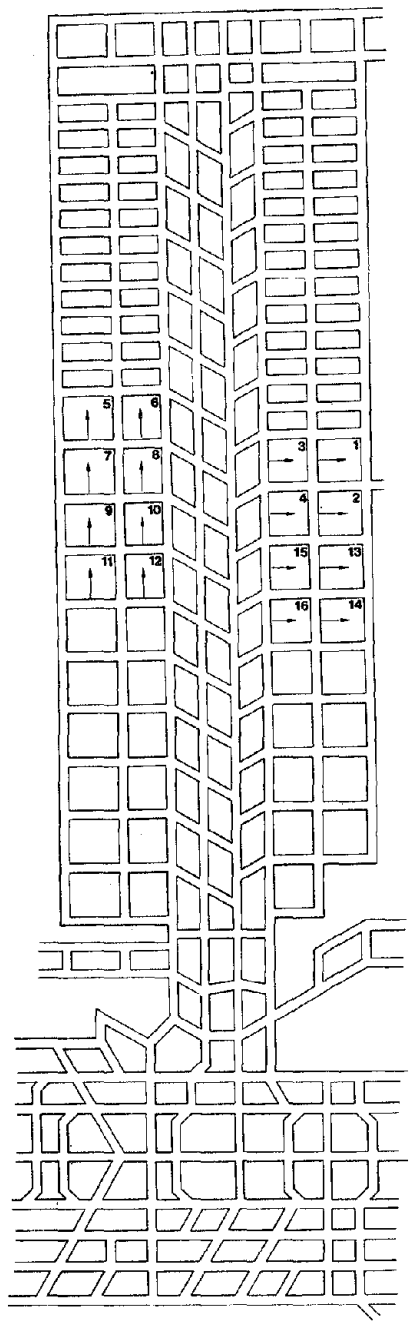


Figure 57 DRILLING SEQUENCE OF THE SECOND PROTOTYPE AUGER MINER AT THE 5R-8N TEST AREA

Figure 58 UNDERGROUND OPERATION (Note operator's compartment is uncrowded and offers good visibility)

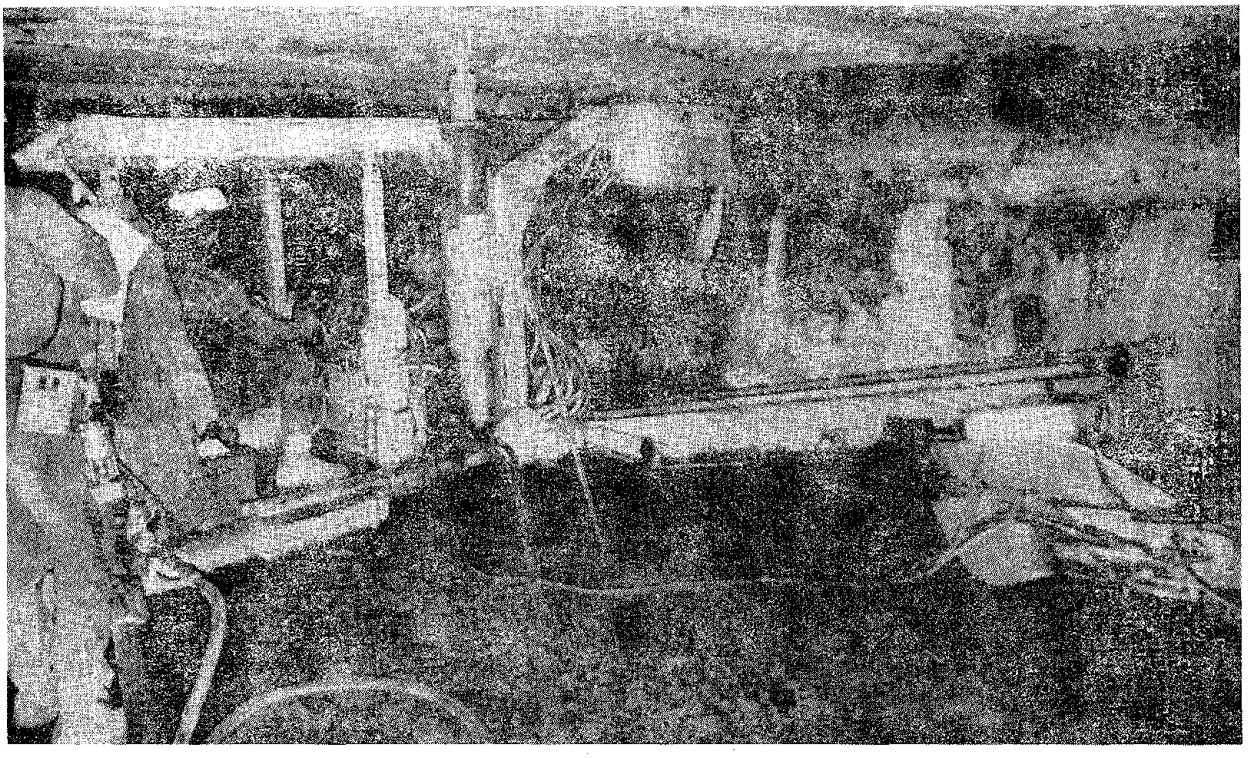


Figure 59 UNDERGROUND OPERATION (Note spill pan on entry conveyor)

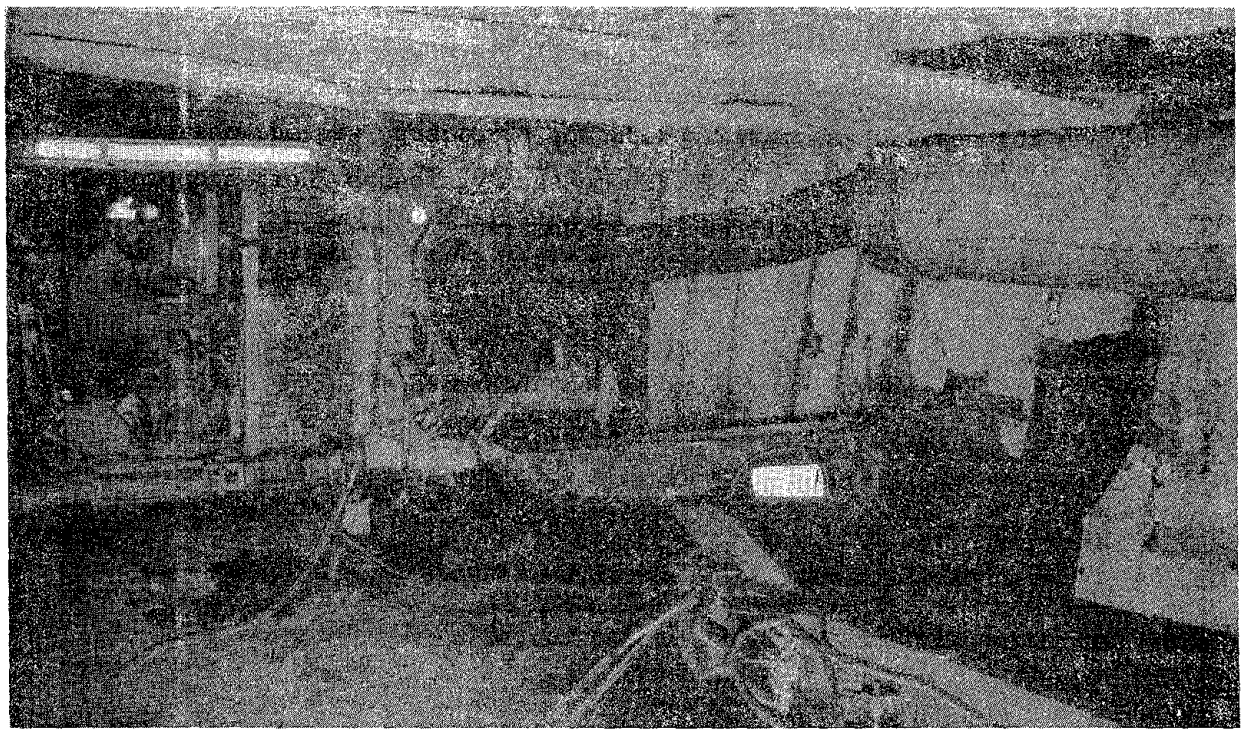


Figure 60 UNDERGROUND OPERATION IN PROGRESS. CONTROLS ON ENTRY CONVEYOR (in foreground) SELECT CONVEYOR LIFT OR RUN.

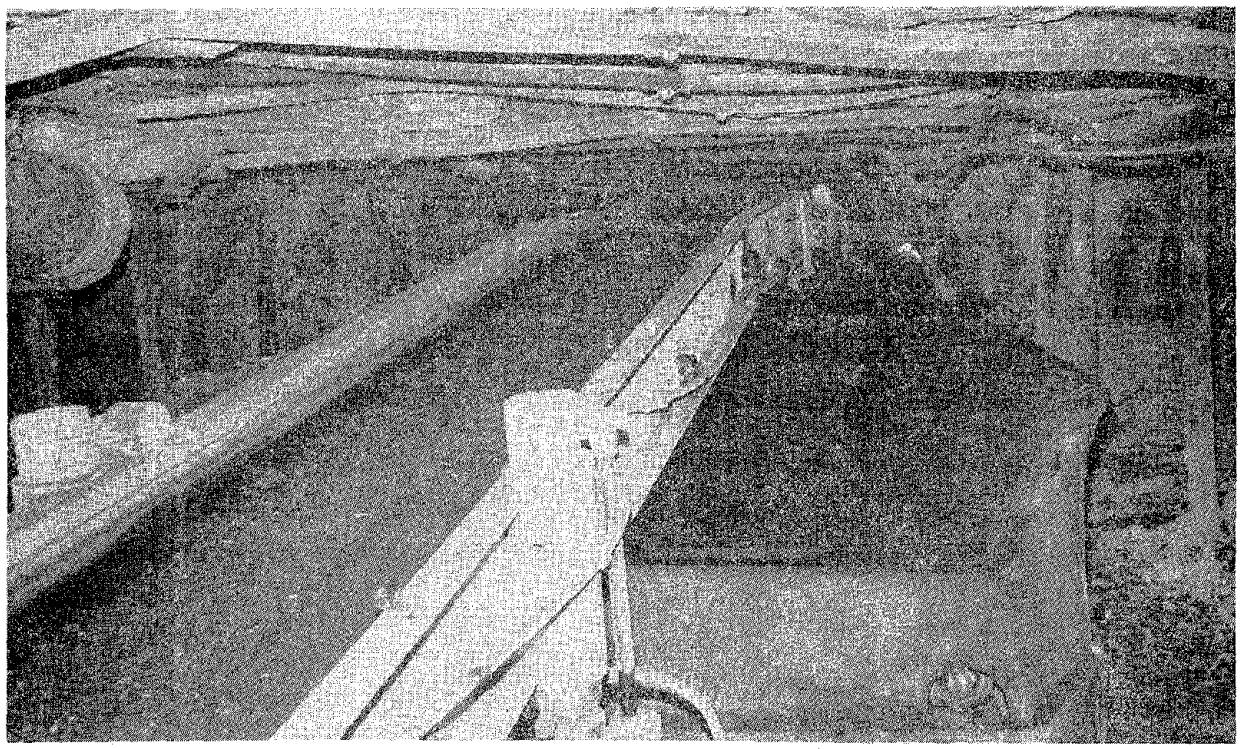


Figure 61 COAL DISCHARGE INTO SHUTTLE CAR

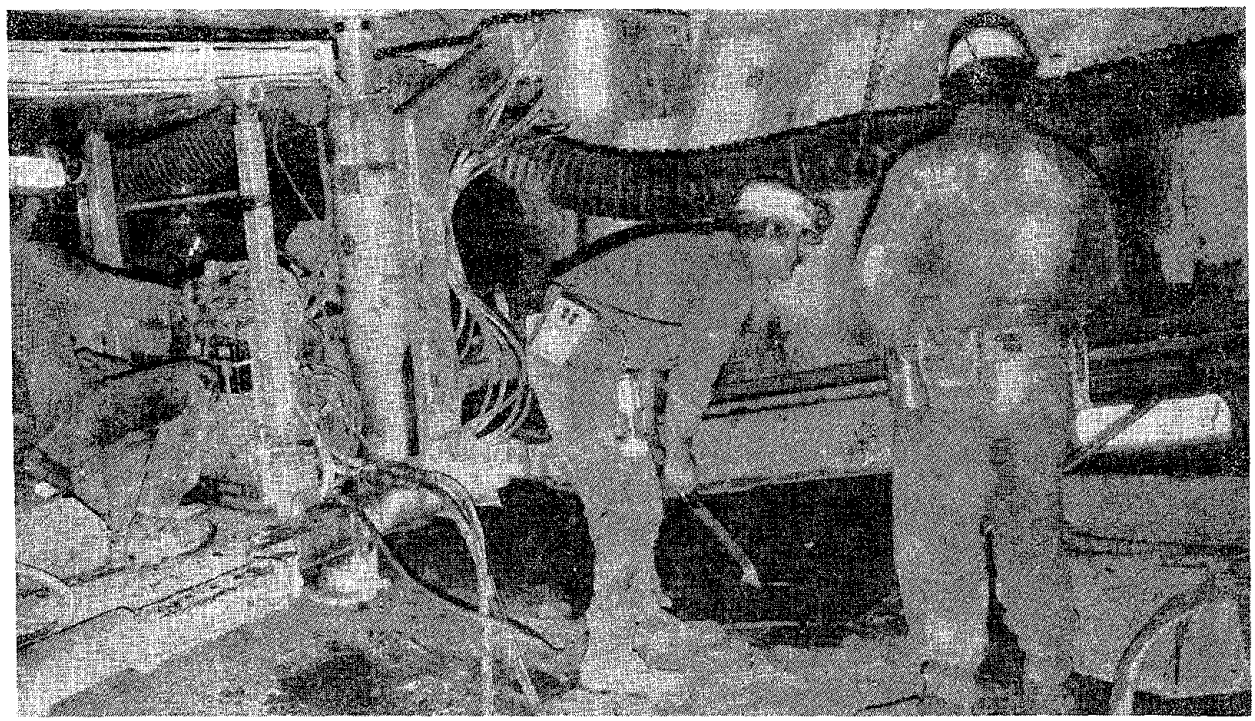


Figure 62 UNDERGROUND OPERATION – SECTION FOREMAN AND ASSISTANT MINE FOREMAN OBSERVING CUTTINGS

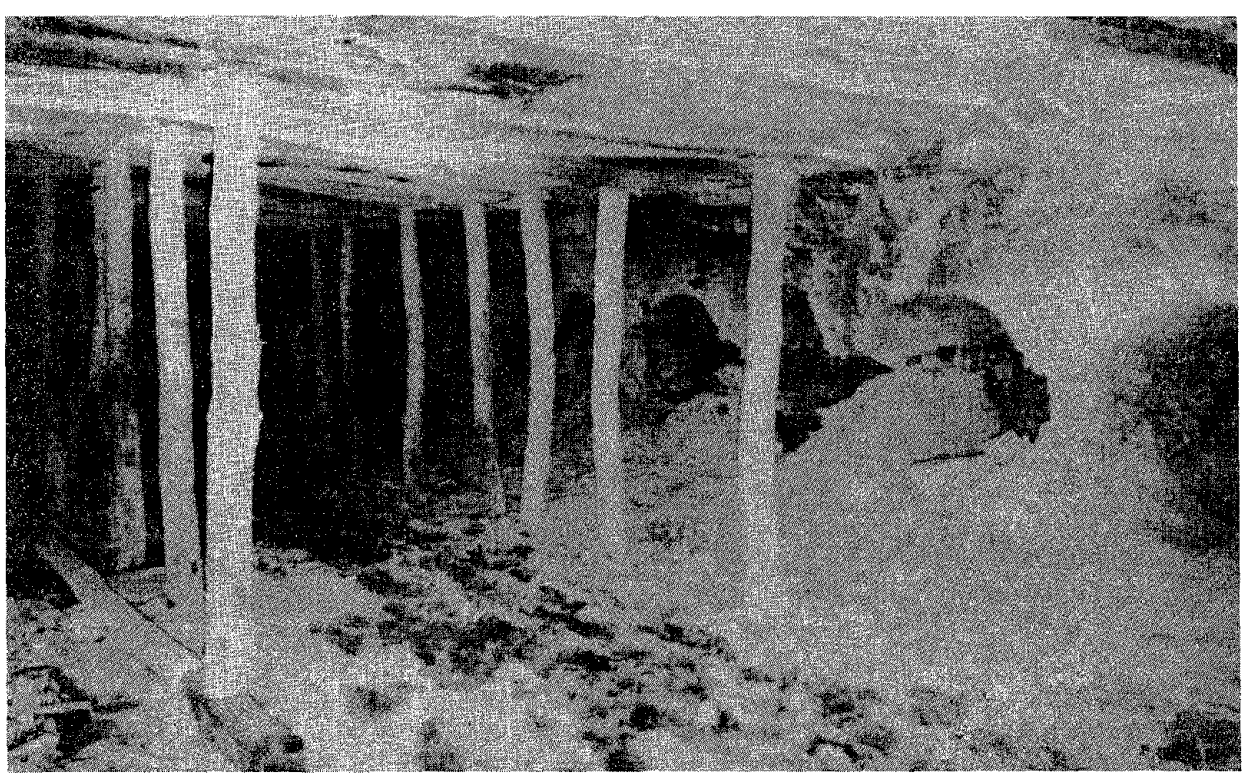


Figure 63 TYPICAL MINED OUT AREA. POSTS SET IN ACCORDANCE WITH WITH MSHA – APPROVED ROOF CONTROL PLAN.

the Thanksgiving holidays, the section averaged 94 tons per shift and had four shifts over 100 tons. Ireland Mine had received FMC's prediction by means of a computer simulation that the average production would be 123 tons per shift.

It was expected that as the crew became more familiar with the equipment, production would increase. In actuality, average shift production diminished slightly. A synopsis of the demonstration data is shown in Tables 10 and 11. Although average shift production increased 125 percent over the first prototype, it did not reach the target of 123 tons per shift and the record shift of 182 tons fell 10 percent short of our target of 200 tons. It should be noted that the crew on the auger mine varied almost daily, but it averaged at six people consisting of one each of the following:

- Foreman
- Auger operator
- Extractor operator
- Shuttle car operator
- Shuttle car operator/mechanic
- Helper.

The helper was often used for setting posts behind the machine or for other section housekeeping tasks. In other mine plans the helper would not be required. Productivity figures cited in Part 2 of this report are based on a six-person crew.

A critical analysis of why production fell short of the target is somewhat subjective, but an inspection of Table 12 is indicative of where the improvements (or failures) occurred.

Table 12 presumes that time saved in a nonproductive element has the potential of adding to productive (drilling) time. It compares the average shifts from both demonstrations. On the average, there was a time savings in moving hole-to-hole and block-to-block (cross-entry moves were not considered in calculating the average). This could be expected to yield about a 7-ton increase in shift production if no other changes in the machine had taken place. This indicates that the adoption of the walking shoe tram was beneficial. Likewise, delays due to the auger equipment were slightly reduced resulting in an expected tonnage boost of about 5 tons per shift. This was not as good a performance as was expected from the auger equipment. The singular element most at fault was the sub-auger conveyor. About 30 minutes per shift (average) were spent restringing the belt under the hold-down rollers, scooping coal rubble from under the belt and unjamming the drive. Had these 30 minutes been available for production, an extra 10 tons could have been added to the average shift tonnage. Although mechanical downtime can never be eliminated, additional improvements to the conveyor and other machine elements are necessary and possible if this type of auger is to become a viable production machine.

Table 10 PRODUCTION DATA

Total shifts considered in demonstration	53	
Total face time (including moving)	11,412	minutes
Total of all delays	14,028	minutes
Total tonnage for demonstration	4,675	tons
Average tonnage per machine (crew) shift	88.2	tons
Best shift tonnage	182	tons
Average tonnage per man shift	14.7	tons

Table 11 BREAKDOWN OF AVERAGE SHIFT ACTIVITIES

Item	Minutes (nearest tenth)	Percent (nearest tenth)
Total shift time (8 hours)	480.0	100
General delays	113.9	23.7
Auger equipment delays	101.1	21.1
Mine equipment delays	49.7	10.4
Moving delays	76.7	16.0
Operating time (drilling)	138.7	28.9

Table 12 PRODUCTION COMPARISON

Item	First prototype	Second prototype	Expected production increase (tons)
Moving	98 minutes	76.7 minutes	7.1
Auger equipment delays	117 minutes	101.1 minutes	5.3
Mine equipment delays	32 minutes	49.7 minutes	[5.9]
General delays	115 minutes	113.9 minutes	0.4
Net	362 minutes	341.4 minutes	6.9
Actual shift production	39.2 tons	88.2 tons	49.0 actual
Tons per operating minute	0.332	0.636	-

Tonnage increase attributable to machine improvements

49.0 tons -6.9 tons = 42.1 tons

Mine delays actually increased during the second demonstration, which all else being equal, acounted for a loss of production of nearly 6 tons per shift. General delays, such as travel, safety discussions, dinner, etc., remained virtually the same.

The net changes in expected shift production due to delays is only about 7 tons, but the actual increase was 49 tons. Therefore, 42 tons worth of production is attributable to improvements in the basic operation of the machine. The main improvement quite obviously came from the addition of the extractor. Other contributions came from improvements to the leveling system, ventilation tubing handling, rib alignment techniques, better availability of haulage, and shorter drilling depths.

Figure 64 is an inventory of boreholes indicating the number of boreholes which were drilled through or left blind and the numbers of the holes which had defects and the types of defects. As can be seen, 61 percent of the 163 boreholes penetrated the blocks entirely in coal compared to 53 percent in the previous demonstration. The improvement in borehole guidance due to the frame leveling system appears insignificant. It is interesting to note, however, that few, if any boreholes ever intersected an adjacent hole at a depth less than about 30 feet. Most errant trajectories revealed themselves at much greater depths. This may mean that future generations of underground augers should limit their penetration from 35 to 40 feet. This may yield much better production of clean coal per shift even considering the more frequent moves.

9.3 SIGNIFICANT TESTS AND EXPERIMENTS

Although the emphasis during the second demonstration was on production and the agreement with Ireland Mine called for a minimum of disruptive experiments, the following interesting tests were performed:

- Drilling/production power
- Methane buildup
- Backreaming.

9.3.1 Drilling/Production Power

In the first demonstration it had been discovered that the electric motor on the Coalbadger dedicated to the main drive system was at depths over about 80 feet producing in excess of its nominal horsepower. While this is acceptable for some duty cycles it was also found that drilling efficiency dropped at the deeper depths, and this combination of facts caused FMC to opt for smaller panels to drill in the second demonstration.

Power absorption curves were plotted based on data taken from drill motor pressure gages and an add-on tachometer. These data and curves appear in Appendix I. The horsepower curves again indicate that about 50 horsepower is lost in drive train inefficiency and friction just as drilling begins. At no time did the hydraulic horsepower exceed 125 and, with a conversion efficiency of 80 percent, electric motor horsepower did not exceed 156 for a full depth hole.

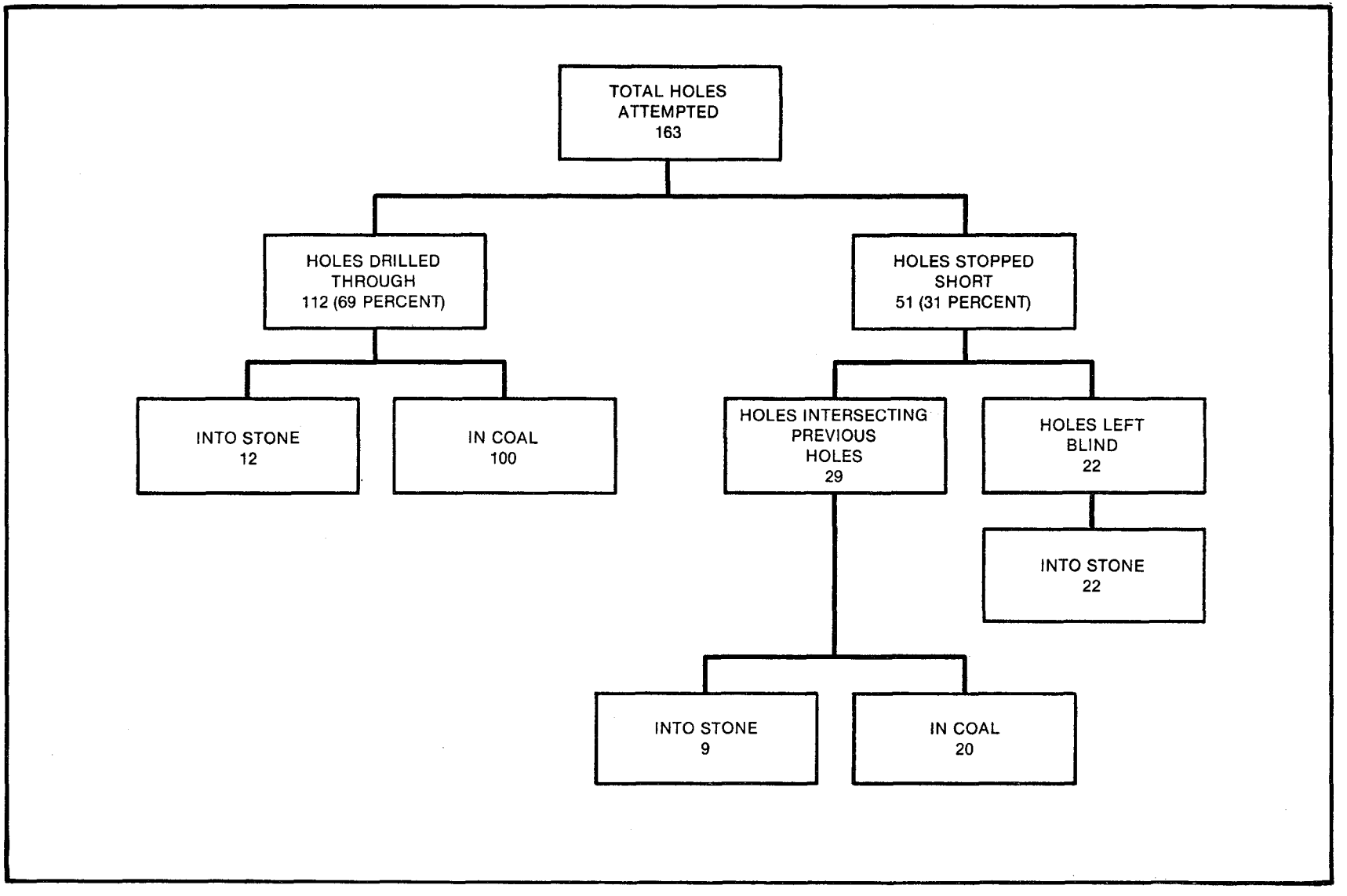


Figure 64 BOREHOLE INVENTORY

9.3.2 Methane Buildup

The question of just how much methane remained in the coal blocks and how this remaining methane would seep into the boreholes has been the subject of much debate. Methane in the entries surrounding a block was virtually zero indicating that all methane near the rib surface had been liberated. It was wondered if the entire blocks had drained in the 7 to 8 months they had stood prior to drilling.

Methane readings taken by inserting a rubber tube several feet into through boreholes, or by sampling the air on the downstream side of a block of through holes, revealed essentially no methane -- perhaps 0.1 percent at the most. It was decided to intentionally leave a blind hole which was in coal only (not wandered into top or bottom) and seal it with a methane probe inside. The first hole drilled into Block 12 was drilled approximately 70-feet deep and the drillsteels were removed as the adjacent hole was drilled. A methane probe consisting of copper tubing attached to a stand made of thin wall conduit was inserted approximately 25 feet back into the borehole. The intake end of the tube was approximately 24 inches off the floor. The borehole was then sealed with concrete blocks and plaster with the copper tube and a 1/2-inch capped pipe vent passing through the seal. A photo of the test borehole just as the seal was being constructed appears as Figure 65. An immediate methane check indicated that there was no methane. One day later the methane content had risen to 1-1/2 percent. Five days from the sealing, the methane content was 4-1/2 percent, and after 12 days it was still at 4-1/2 percent. The next reading was taken on the 15th day and the methane content had fallen to 2-3/4 percent. The drop in methane content gave rise to the theory that the web between the test hole and the adjacent hole (which was a through hole) had crushed out in some places and allowed fresh air to enter the test hole. No more readings were taken. The fact that a sealed blind hole could fill with methane to a mixture near the lower explosive limit cast some doubt on the MSHA requirement that blind holes be sealed. It was suggested that Ireland Mine petition MSHA for permission to instrument an unsealed blind hole in a similar manner to determine whether methane would build up in a similar manner or whether local barometric changes and entry air movement would keep an open hole flushed.

Unfortunately this test was not accomplished before the demonstration was terminated.

9.3.3 Backreaming

The backreaming device was described previously in Section VIII. An opportunity to try the backreamer did not arise until the final hole of block 12. The normal cutterhead was removed and the backreaming head was installed easily and without incident. As the head was withdrawn back through the block, the flights were removed one by one and offloaded from the machine in preparation for a cross-entry move. The cuttings from the backreaming operation were mostly fist-sized. Fines and gravel-sized coal were left behind in the borehole after having spilled through the opening between the bits and the barrel of the backreamer. Ireland Mine commented that future backreamers should have a skirt or band added to the head to prevent these fines from accumulating in the hole.

Figure 65 BOREHOLE INSTRUMENTED TO DETECT METHANE BUILDUP

The web between the adjacent hole and the backreamed hole was thinned to about 12 inches and it was thought that the reduced support might promote collapse of the hole or at least severe web crushing. The hole was observed routinely for several months and no such degradation was noted.

The backreaming experiment proved that backreaming could be done without binding, jamming, etc. More backreaming of a series of adjacent boreholes must be done though to prove that the practice is safe. Time and production data will determine if it is cost effective. Although prior backreaming by auger has been done in setting culverts beneath roadways, we believe that the backreaming experiment at Ireland Mine marked the first time a hole has been so enlarged in an underground mine.

9.3.4 Surge Car Haulage

Although not a test or experiment in the true sense, FMC noticed partway through the demonstration that haulage delays were occurring all too frequently and degrading production. It was also noticed that one of the two shuttle cars assigned to the section was down more than the other -- usually due to cable or cable-reel problems. FMC suggested that the ailing shuttle car be positioned beneath the entry conveyor tailboom to collect coal from the auger mine and pass it through to the better car which, when full, would trip to the belt feeder. Thus, when the second car was gone, mining could

continue with the output of the auger being stored in the first car. This surge car technique did not eliminate haulage delays but they did decrease enough that the practice was continued for the rest of the demonstration.

9.3.5 Cutterstring Thrust

It had been theorized as discussed in Section 4.4.1 that a forward thrust force was generated during cutterstring rotation by the scrolls of the auger being in contact with the bottom of the borehole. The existence of this thrust force was verified every time a borehole penetrated a block and rotation was continued. With cutting resistance removed, and the sump carriage stationary, the auger string would continue to advance until the last flight disengaged itself from the drive chuck. Quite obviously then this linear thrust existed and was large enough to move a string of flights weighing perhaps 9,000 pounds.

9.4 NOTABLE EQUIPMENT FAILURES

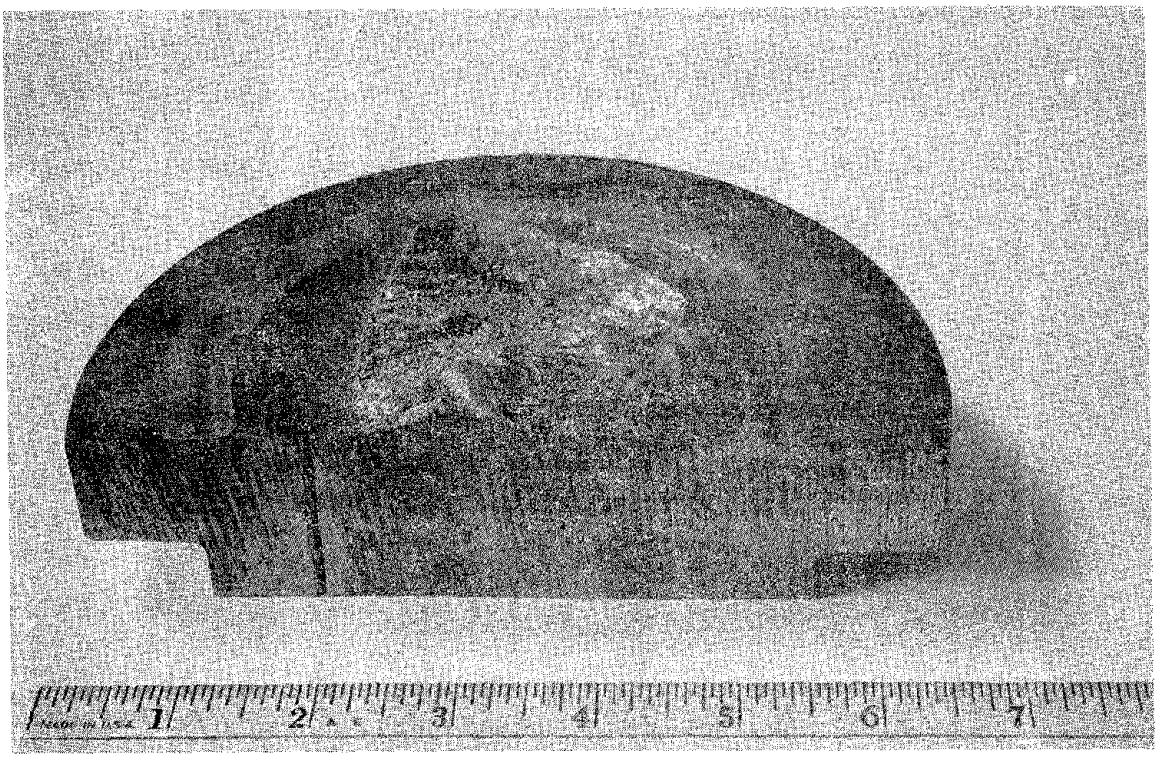
Several instances of mechanical failures which resulted in scheduled downtimes for repairs are worthy of discussion here. These failures in the aggregate caused several shifts to be lost for test, and although the auger crew was dispatched to other work in the mine and these shifts were not counted in the test data, the mine was unhappy with the lost test shifts and ultimately, with the final failure of the Sundstrand drive motor which was the cause of the early termination of the demonstration.

9.4.1 Auger Drive Shaft

The auger drive shaft was an approximately 4-inch diameter steel shaft which attached directly to the cutterhead drive motor through a flexible coupling and passed through a bearing box which reacted all the radial and axial drilling loads. The chuck attached directly to the front of the drive shaft and the chuck was surrounded by the airbox. This drive line can be seen in Figure 39. In Block 5, while beginning the fourth borehole and having abandoned the three previous boreholes in which a very hard rock band was encountered, the drive shaft failed in torsion. The failure occurred at a step in the shaft near the bearing at the front end of the bearing box and between the bearing box and the drive chuck. The bearing box and chuck with the drive shaft pieces still attached were removed and taken to the surface for examination. It was apparent that the failure was the culmination of a crack which had begun near the blend radius of the step in the shaft and had progressed nearly completely around the perimeter of the shaft and about an inch into its interior until only a core of competent metal was left to carry the drilling torque. A new shaft was begun immediately at FMC using a higher alloy heat-treated steel. The blend radius was increased and was shot peened to stress relieve the area and elimiminate stress raisers. This shaft was air freighted to Ireland Mine, reinstalled on the auger and performed satisfactorily for the balance of the demonstration.

The failed portion of the shaft was sawed off the old shaft and returned to FMC for analysis. The failed area is shown in Figures 66 and 67.

A cursory stress analysis indicated that with the relatively low number of stress cycles which the shaft had been subjected to, the failure was not one



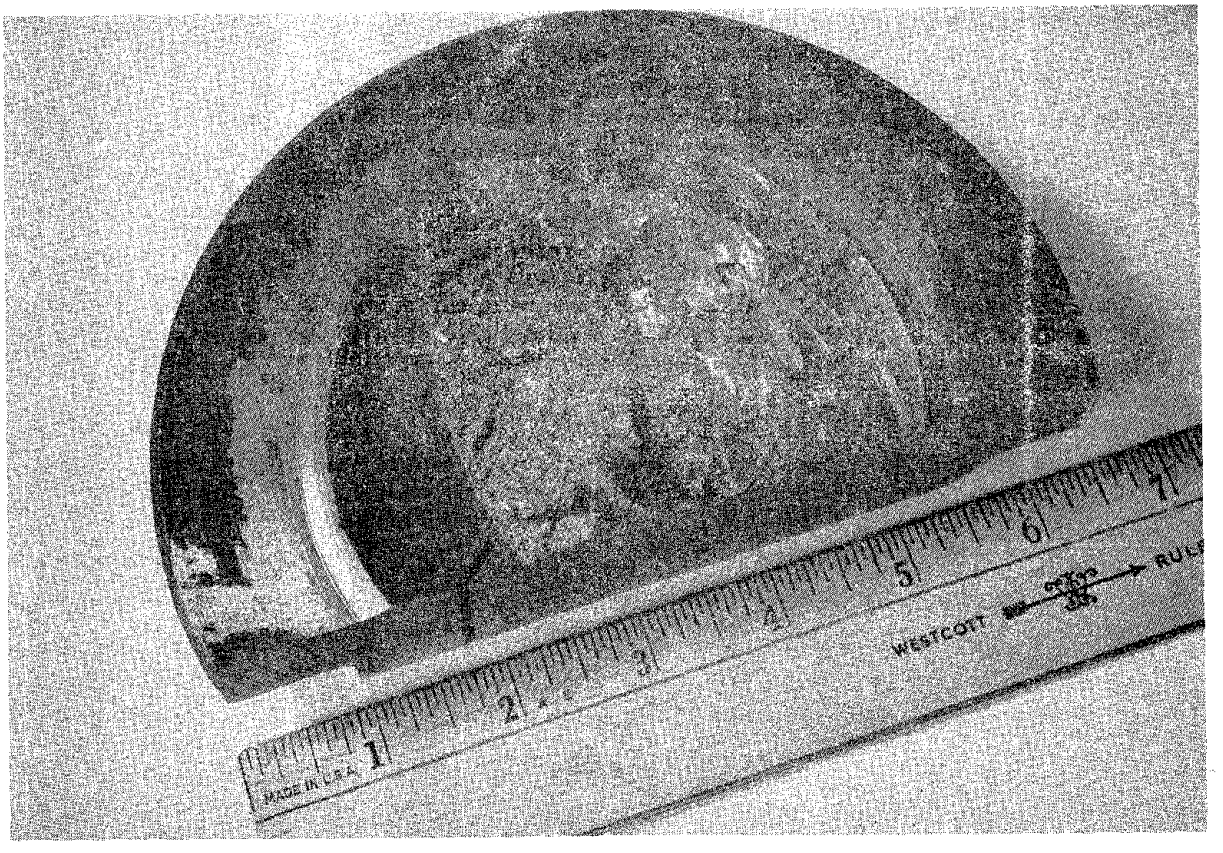


Figure 66 and 67 FAILED DRIVESHAFT

of pure fatigue. It was surmised that during machining, tool marks had been left on the blend radius thus instigating an early crack and an accelerated fatigue failure. The specimen was sent to FMC's Materials Laboratory in Santa Clara, California for metallurgical analysis. The materials laboratory report, which appears as Appendix J, confirmed that tool marks were present on the part and that crack initiation began on a tool mark. Also, a chemical analysis indicated that the raw stock supplied by a local vendor did not conform to those of 1045 steel as had been specified on the drawing, but rather was of an inferior grade. Had the proper material been supplied, and had the blend radius been increased and locally stress relieved by shot peening, the failure would probably not have occurred.

9.4.2 Monorail Drive Shaft

The hoist mechanism on the monorail was operated by a splined shaft which ran the full length of the monorail and was powered by a hydraulic motor. A ball-bearing nut which would slide linearly on the shaft and still rotate with the shaft was used to transmit torque into the hoist sprockets.

When the splined shaft was being selected, only one vendor offered a bid. Two vendors refrained from bidding stating that they were unable to produce a splined shaft of the required length. The successful bidder, the Beaver Company, suggested a two-piece assembly with the two pieces assembled with hardened steel shear pins.

The assembly worked properly during trials at FMC and well into the demonstration. However a hoist failure occurred when the shear pins, either sheared or backed out, and the ball nut was run over the now out-of-alignment spline. The end caps of the ball nut, which were thought to be metal, but which proved to be a nylon-like plastic, failed as can be seen in Figure 68. The ball nut was repaired by installing new end caps and ball bearings, and the drive shaft was repaired underground by reassembling it with new shear pins.

The hoist was put back into operation, but an identical failure occurred a short time later. The drive shaft was removed from the monorail and taken to the Ireland shop where two new shear pins, in addition to the two already there, were installed. The drive shaft was replaced on the monorail, a rebuilt ball nut was installed in the hoist and no further problems were encountered. This failure of a vendor part caused several test shifts to be lost. Ironically one of the vendors who had originally declined to bid, the THK Company, subsequently developed the capability to produce the shaft in the required length in one piece.

9.4.3 Hydraulic Drive Motor

The main drill string drive motor on the second prototype was a Sundstrand Series 60, 10 cylinder radial -- the same motor which had been used on the first prototype Coalbadger.

The motor uses a ported shaft which, as it rotates, allows high pressure oil to reach the cylinders and low pressure oil to return to the pump. Since the oil inlet ports are fixed on the motor, a labyrinth spool called a distributor ring is mounted over the shaft inside the motor case. It is fitted

with seals and passages which allow oil to reach the radial inlet/outlet ports on the shaft while it rotates. It is thus exposed to high hydraulic pressure at all times the motor is running.

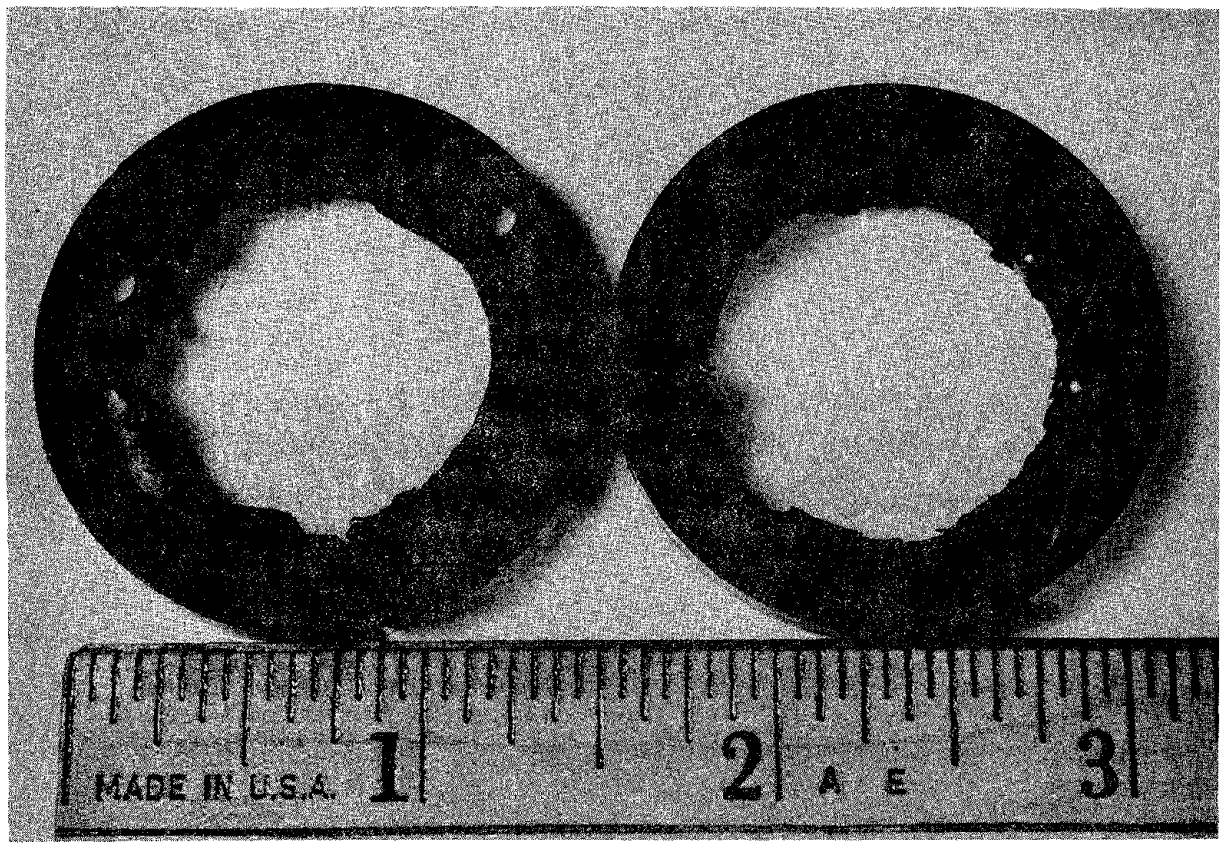


Figure 68 FAILED BEARING END CAPS ON MONORAIL HOIST

Partway through the shift on March 16, 1979, this distributor ring fractured and a portion of the ring weighing about 20 pounds was ejected violently through the rear of the motor case. The area behind the drill carriage was a popular place for the section foreman and/or FMC on-site representative to stand while viewing the operation or taking data. Fortunately no one was standing there at the time of the motor explosion, or a serious accident would have no doubt occurred.

The catastrophic nature of the failure and the fact that repair would take several weeks prompted Ireland Mine to request the termination of the demonstration. It was decided that the motor should be repaired in preparation for the possibility of a follow-on demonstration at another site.

To determine why the distributor ring had failed, FMC sent the remnant of the failed ring to the FMC Materials Laboratory for analysis. Their preliminary findings as are detailed in Appendix K were that a less-than-ideal material had been chosen and x-rays revealed that the casing contained much

porosity. Photomicrographs of the part showed that failure probably initiated at the site of one of these inclusions. For comparison purposes, the new replacement part was x-rayed and it too was found to contain porosity.

In addition, a physical examination of the replacement part showed that the material had been thickened in the area where our failure had occurred. This extra material coupled with lower hydraulic force on the remaining area of the land meant that the local stress in the part was reduced to about half its previous value.

FMC reasoned that these design changes had been brought about by earlier failures such as the one we experienced. Sundstrand cooperated with FMC in checking with their domestic network and their European counterparts (since the motor had been designed in England and the designs were later sold to a firm in Germany). It was discovered that other identical failures had indeed occurred, but that this had not resulted in a recall of all motors in the field. Rather, when motors were eventually sent in for rebuild the improved distributor ring would be routinely installed.

It is unfortunate that this information came to light only after the failure of the motor.

By the end of March 1979 the equipment was removed from the mine and stored on the surface.

After several months of attempting unsuccessfully to arrange a second demonstration of the auger, the equipment was shipped to a storage area at the DOE facility in Bruceton, Pennsylvania.

10.0 CONCLUSIONS AND RECOMMENDATIONS (Part 2)

The most clear-cut observation of both auger demonstrations is that substantial improvements in borehole guidance must take place if long hole is to become a viable production method. FMC's attempted means of accurately and repeatably setting the initial trajectory was not adequate. Internal forces within the coal block have an overwhelmining influence on trajectory. An active system -- perhaps a steerable cutterhead -- will be required to prevent so many intersected and blind holes. While a cutterhead steering system can be conceived, the challenge will be to make it affordable to the mining industry. Nonetheless it is a worthwhile research and development goal.

Downtime must also be reduced. On future machines of this general design, the first stage or sub-auger conveyor must be modified in the area of the upturn on the belt. The original rollers were replaced with flat steel hold-down shoes near the end of the demonstration and no belt "pop-out" occurred in the three shifts before the failure of the drive motor. Possibly this "fix" represents the proper design for future conveyors.

Some downtime can also be eliminated by choosing more reliable purchased components.

Consideration must be given to the advisability of long hole drilling. It appears that drilling shorter holes through shallower blocks may be more productive. Computer simulation using actual test data and projections from the two demonstrations could predict the optimum depth of borehole considering such aspects as aborted holes, coal lost when boreholed intersect, etc. On the subject of coal lost into intersecting boreholes, it became obvious that when mining from left-to-right as was done in 5R-8N, the auger scrolls should be left hand pitch and the drivestring should rotate counter-clockwise. This will allow for less coal spillage from slightly intersecting boreholes because the coal will tend to pile against the solid side and not the web side. When the web is missing, cut coal is pushed into the adjacent empty borehole.

As charted in Appendix L, the procedure for acquiring MSHA approval for electrical systems is lengthy. It is usually necessary to submit an application for investigation prior to the design of the system. In the case of the second prototype auger, 15-1/2 months elapsed between initial application submittal and approval. The auger was on route to Ireland Mine when the approval letters were received. Serious delays can occur to programs where extensive electrical design changes are required by MSHA and this information comes after the bulk of the design and fabrication effort is completed.

More timely MSHA approvals are needed, and it is our understanding that MSHA is streamlining its approval process and encouraging the use of experimental permits in lieu of full approvals - an option which may shorten the approval process by several months.

An interesting data analysis to detect learning on the part of the auger crew appears as Appendix M. The study uses traditional mathematic manipulation of the data from 50 representative shifts to plot a trend curve of operating minutes per ton of coal produced. As expected, the trend showed a rapid drop-off in the early part of the demonstration as the auger crew learned all the obvious operational shortcuts. The latter three quarters of the demonstration showed only a slight falling trend, and by the end of the 50 shifts the curve was virtually flat. The important discovery of this analysis is that in predicting future production, quite a large number of shifts must be allowed to occur such that the effects of learning are already accounted for.

For the auger this appears to be about 40-50 shifts. This number may or may not be relevant for other types of equipment. Learning curve analyses such as these should probably be a part of all future underground demonstrations, both of auger equipment and other machinery. In this way a guideline for demonstration durations can be established.

11.0 SUBJECT INVENTIONS

By definition, "Subject Inventions" means any invention, discovery, improvement, or development (whether or not patentable) made in the performance of the experimental, developmental, or research work under this contract.

No subject invention claims are being made by FMC.

Appendix A

METHANE CONTENT AND VENTILATION AIR FLOW FOR IRELAND MINE

FMC Advanced Products Division
San Jose

Interoffice

To Memo for Record

Date Nov. 5, 1975

cc

From J. Ritchey

Subject Methane Emission - Ireland Mine

In 1972 the Bureau of Mines published Information Circular IC 8558 titled, "Methane Emission From U.S. Coal Mines, A Survey". In this study measurements were made in a great number of Eastern coal mines to determine both the average daily methane emission (cu ft/day) and the average methane emission per ton of coal mined. One of the mines surveyed was the Ireland Mine of Consolidation Coal Co. It is the intent of these calculations to determine from the data in IC 8558 the methane emission to be expected when operating the underground auger miner system at Ireland Mine.

From the survey, the average daily coal production at Ireland Mine in 1972 was 9,666 tons. The average daily methane emission was 1.8 million cu ft. The average quantity of gas per ton of coal mined was, therefore, 188 cu ft/ton.

The IC is quick to point out that, "...this ratio (cu ft gas/ton of coal) is invariably higher than the actual gas content of a ton of coal in place, for methane enters the ventilating air from the adjacent strata and the unmined coal as well as from that which is mined." The article goes on to present a curve which shows that for an unnamed mine in Pennsylvania, the methane emissions dropped only about 30% when the mine was idle during a long strike. This would indicate that 70% of the metnane produced in the mine came from old workings, entries, mined out areas, etc. while only 30% was produced at the face--areas of active excavation.

Using this data as guidelines along with some physical data obtained from Ireland Mine, it is possible to estimate the methane to be expected for freshly excavated Ireland Mine coal.

Assumptions:

Seam Height	$5\frac{1}{2}$ ft (on the average)
Entry Width	$15\frac{1}{2}$ ft (on the average)

Number of Entries Being Developed	40 (on the average)
Coal Density (in place)	85 lb/ cu ft

In mining an average of 9666 tons per day a $15\frac{1}{2}$' x $5\frac{1}{2}$' entry system would have to be advanced a sum total of 2667 ft.

i.e. $$\frac{(5\frac{1}{2})(15\frac{1}{2})(l)(85)}{2000} = 9666 \text{ t}$$

$$l = 2667 \text{ ft}$$

The average entry, then, would be advanced $66\frac{1}{2}$ ft.

i.e. $(2667)/(40) = 66\frac{1}{2}$ ft

The total freshly exposed area, then, would be 115,130 sq ft.

i.e. $40\left[2(5\frac{1}{2} \times 66\frac{1}{2})+2(15\frac{1}{2} \times 66\frac{1}{2})+(15\frac{1}{2} \times 5\frac{1}{2})\right] = 115{,}130$ sq ft

The total methane contribution introduced through this area of freshly excavated coal would be 540,000 cu ft/day.

i.e. 1,800,000 x 30% = 540,000 cu ft/day

The hourly rate is then 22,500 cu ft/hr.

i.e. 540,000/24 = 22,500 cu ft/hr

And the rate in CFM is 375 CFM.

i.e. 22,500/60 = 375 CFM

The methane emission rate per square ft of area is then:

$$Rm = \frac{375 \text{ CFM}}{115{,}130 \text{ sq ft}} = 0.00326 \text{ CFM/sq ft}$$

If this 0.00326 CFM/sq ft rate is considered to be indicative of the methane to be found in the borehole produced by the auger miner, the dilution air required to keep the borehole's methane/air ratio below 1% can be calculated as follows:

The maximum borehole surface area to be produced with the auger miner will be from a hole 42 inches in diameter ($3\frac{1}{2}$ ft) by 100 ft deep.

$$\text{Area} = \pi(3\tfrac{1}{2})(100) + \frac{\pi}{4}(3\tfrac{1}{2})^2$$

$$\text{Area} = 1109 \text{ sq ft}$$

Methane production = 0.00326 CFM/sq ft x 1109 sq ft

= 3.62 CFM

One hundred times this amount of air is required to dilute the methane in the borehole so that the methane/air concentration is always below 1%.

Fresh Airflow = 362 CFM through the auger.

FMC Advanced Products Division
San Jose

Interoffice

To Memo for Record

Date Dec. 10, 1975

cc

From J. Ritchey

Subject Methane Emission - Ireland Mine

On Dec. 5, 1975 a meeting was held at the MESA offices at the Morgantown Energy Research Center between FMC and Mr. Paul Componation, Chief of the coal mine ventilation group.

The purpose of the meeting was to obtain the results of some face ventilation studies which are ongoing at Ireland Mine in the general vicinity of the planned auger miner tests--i.e. the 8-North area.

They had available some raw data from recording methanometers which indicates that methane emission from the face measures from 10 to 12 CFM per 100 sq ft of face area or 0.10 to 0.12 CFM/sq ft.

In outlining the operation of the auger miner system to Mr. Componation, it became apparent that methane calculations should consider the active excavation at the back of the borehole as the major source of methane liberation. The calculations which follow treat the entire area of borehole exposed in one sump as an equivalent area of "face". The Coalbadger auger miner as modified by FMC can sump 5 feet before a new flight must be added, and for this demonstration the auger diameter is 42 inches.

Total area exposed in one sump is:

$$A = \pi D\,(D/4 + L)$$

$$A = 64.6 \text{ sq ft}$$

Using the 0.12 CFM/sq ft figure of measured methane emission in the 8-North area of Ireland Mine, the total methane production would be:

$$M = 0.12 \times 64.6 = \underline{7.75 \text{ CFM}}$$

To maintain the methane/air concentration below 1%, 100 times this amount of fresh air is required.

Fresh Airflow = 775 CFM through the auger.

Appendix B

RECOVERY CURVES FOR AUGER BOREHOLES

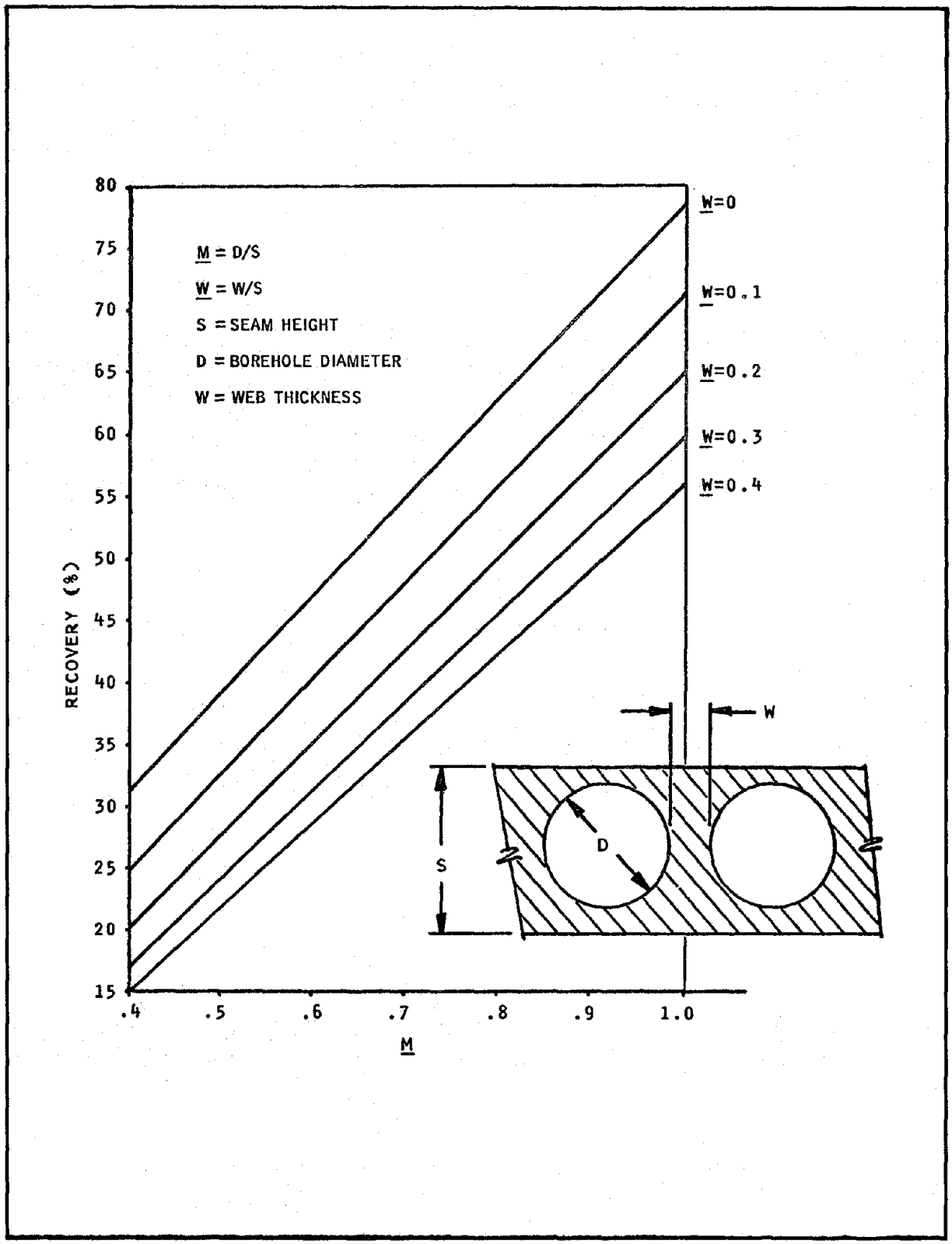


Figure B-1 COAL RECOVERY AS A FUNCTION OF SEAM HEIGHT, BOREHOLE DIAMETER, AND WEB THICKNESS

Appendix C

CONSOL DRILLING AND ROOF CONTROL PLAN

APPENDIX C

IRELAND MINE DRILLING PLAN

-NOTE-

This Technical Service Report No. 12-1312 was compiled solely for use by Consolidation Coal Company, Ireland Mine. Its development was not supported in any way by USBM Contract H0252076 and the views and conclusions contained in this report are those of the authors and should not be interpreted as necessarily representing the official policies or recommendations of the Continental Oil Company, Consolidation Coal Company, or Ireland Mine. This report is reprinted here by permission of Continental Oil Company, Research and Development Department, Mining Research Division.

CONTINENTAL OIL COMPANY

Research and Development Department

MINING RESEARCH DIVISION

TECHNICAL SERVICE REPORT

NO. 12-1312

Address Inquiry To:
H. A. von Schonfeldt or
D. S. Choi

TO: Hugh H. Lucas, Consol, Moundsville, West Virginia 26041

SUBJECT: Underground Auguring at Ireland Mine

DATE: December 21, 1975

RECOMMENDATIONS:

In reference to our recent telephone conversation on the subject matter, we suggest an augur hole pattern as illustrated in the attached figure. The minimum thickness of the web should be 15 inches, and after a series of 8 or 7 holes, respectively, a pillar of approximately 12 feet should be left. It is also recommended to leave about 13 feet of coal on either side of the 200-foot long pillar because of the higher pressure in the outer skin of the pillar.

The auguring should be done in such a way that the equipment always moves away from the gob or abandoned parts of the mine.

It is further suggested that convergence monitoring be done in the holes by the Ponca City group. Suitable convergence meters are being built at this time.

DISCUSSION:

The project was discussed with Messrs. Cosgrove and Richey from the F.M.C. Corporation, and we agreed that the augur holes could be drilled with a 15-inch web initially. Their experience shows that 36-inch diameter augur holes on a 6-inch web have remained stable for two years in the Alyce Mine of Electra Fuels Company in Maidsville, West Virginia. It should be pointed out, however, that the cover was only 250 feet and only one pillar was augured.

If all pillars between two mined panels, for example, were to be mined, the entire overburden weight would have to be taken up by the web and this would obviously create very serious squeeze conditions and most likely lead to failure of the web. To obtain stable condition for this case, the web thickness would have to be of the order of two to three times the opening radius.

Since the augur is always working against the solid portion of the pillar, it should be possible to anticipate squeeze conditions; signs of squeezing should first be noticeable in the holes furthest away from the augur.

D. S. Choi

H. A. von Schonfeldt

mc
Attach.

133 APPROVED:

D. L. McCain, Director
Engineering Section

DIST: HDD BGB DLM JGD HAvS RWU EFP RLM WNP DGW WBJ DRH Joe Richey, FMC

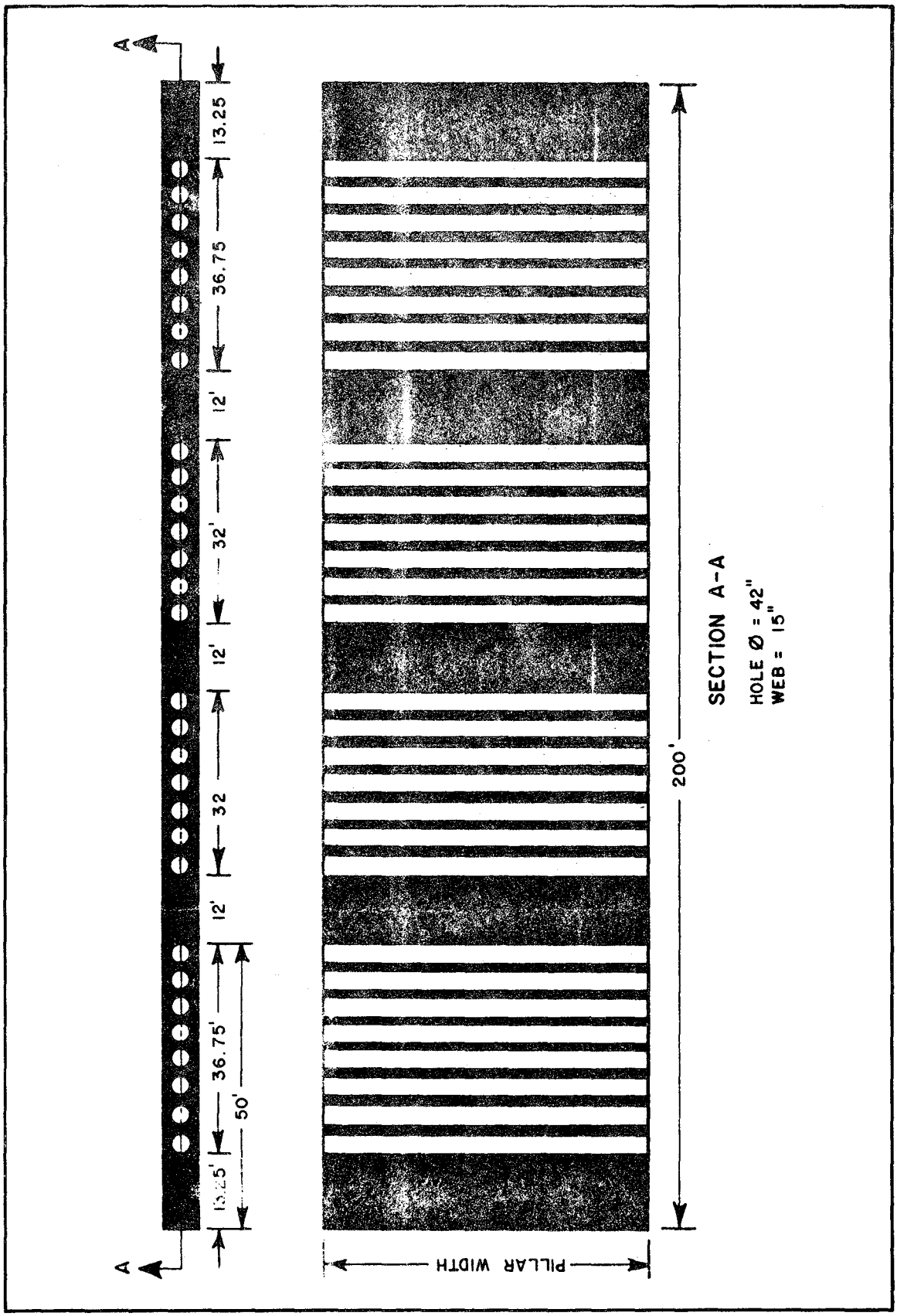


Figure C-1 SPACING OF AUGERING HOLES IN PILLAR

Appendix D

POWER-DEMAND CURVES

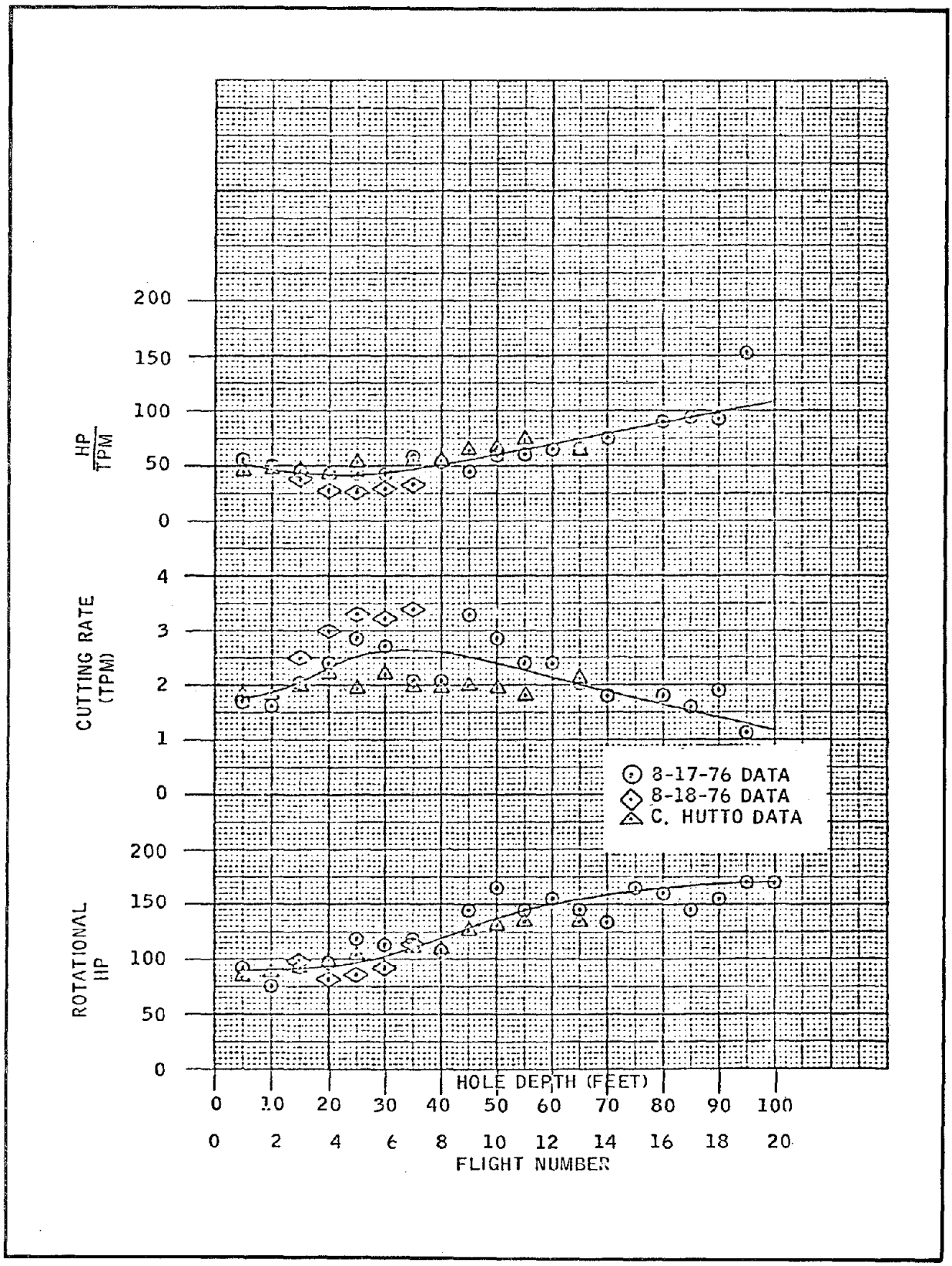


Figure D-1 POWER-DEMAND CURVES

Appendix E

PRODUCTION AND DELAY SUMMARY

SHIFT NO.	DATE	PANEL DEPTH	PRODUCTION			DELAYS								
						MACHINE SET-UP		MECHANICAL/ELECTRICAL			FUNCTIONAL			
			TONNAGE	TIME ②	TONS/MINUTE	57 IN. MOVE ③	15 FT MOVE	CONT. BADGER	CONVEYORS	OTHER	AUGER GUIDANCE	CONVEYORS	TRAVEL, LUNCH, FIREBOSSING, ETC	GENERAL MINE
1	5-5-76	50 FEET	10	58	.17			105		156	41		120	
2	5-6		10	73	.14			128	63	112		14	90	
3	5-7		20	102	.20	50(1)		145				8	105	70-NO WATER
4	5-10		10	75	.13	135(1)		48		93		9	120	
5	5-11		20	87	.23		235	40			28		90	
6	5-12		20	123	.16	75(1)			75	35		52	120	
7	5-13		20	97	.21	55(1)		70		63	80	15	100	
8	5-14		30	88	.34	60(1)		110		9		38	174	
9	5-17		10	50	.20	35(1)		50					100	245-BUILD STOPPING
10	5-18		20	96	.21	45(1)		90		35	30		114	70-NO COAL CARS
11	5-19		50	122	.41	60(2)		14		145			115	24-" " "
12	5-20 ④		60	175	.34	65(2)		50		45			145	
13	5-21		40	80	.50	110(3)	180	10					100	
14	5-24		40	95	.42	20(1)		15		140			110	100-NO WATER
15	5-25		40	110	.36	120(2)				120			130	
16	5-26		70	170	.41	120(3)	65 ⑤	15					110	
17	5-27		0				340 ⑤						140	
18	5-28		0				195 ⑤	157					128	
	6-1		CREW ABSENT											
19	6-2	75 FEET	30	95	.32	45(1)		160	30	50			100	
20	6-3		2	5	.40			355 ⑥					120	
21	6-4		0					375 ⑥					105	
22	6-7		40	170	.24	35(2)		25		20		15	140	15-NO COAL CARS
23	6-8		80	285	.28	75(2)							120	
24	6-9		30	125	.24	30(1)	125	20		45 ⑦			135	
25	6-10		0							480 ⑦				
26	6-11		0							480 ⑦				
27	6-14		18	55	.33		120						125	180-NO WATER
28	6-15		130	280	.46	70(2)							130	
29	6-16 ⑧		50	205	.24	100(2)		30		10			135	
30	6-17		30	105	.29	30(1)	210					15	120	
31	6-18		60	215	.28	30(1)	30			80			105	
32	6-21		50	200	.25	50(1)		45			60		125	
33	6-22		10	25	.40			33			60		130	230-NO COAL CARS
34	6-23		30	60	.50	40(1)			120		135		125	
35	6-24		0				280 ⑨						80	120-ELEVATOR DOWN
	6-25		CREW ABSENT											
	6-26 THRU 7-9		MINE VACATION											
36	7-12		0				350 ⑩						130	
37	7-13		0				380 ⑩						100	
38	7-14	100 FEET	60	120	.50	30(1)				125		95	110	
39	7-15		70	285	.25	35(2)		20				10	110	
40	7-16		60	115	.52					105			70	190-ELEVATOR DOWN
41	7-19		80	255	.31	50(2)		20		30			125	
42	7-20		70	165	.42	50(2)		80		75			110	
	7-21		MINE ON STRIKE											
43	7-22		80	205	.39	45(2)		15		10	75	15	120	
44	7-23		100	195	.51	15(1)	170						100	
45	7-26		70	180	.39	30(1)		115 ⑫			30		125	
46	7-27		0					480 ⑫						
47	7-28		0					480 ⑫						
48	7-29		0					480 ⑫						
49	7-30		0					480 ⑫						
50	8-2		0					270 ⑫					70	140-ELEVATOR DOWN
51	8-3		40	85	.42	40(1)		210		25			110	
	8-4 THRU 8-10		MINE ON STRIKE											
52	8-11		60	115	.52	20(1)				15		25	125	180-NO COAL CARS
53	8-12		0							480 ⑪				
54	8-13		100	305	.33	45(2)				40	10		80	
55	8-16		70	185	.38	65(2)		100		40			90	
56	8-17		60	110	.55	135(2)				115			100	
57	8-18		30	110	.27		160 ⑬	30					120	60-NO A.C. POWER
58	8-19 ⑬		20	50	.40								100	
TOTAL			2000	5916		2135(53)	2840	4867	288	3178	549	312	5801	1624
AVERAGE			34.5 TONS/SHIFT	102 MIN./SHIFT	.34	40 MIN./MOVE		84 MIN./SHIFT	5.0 MIN./SHIFT	55 MIN./SHIFT	9.5 MIN./SHIFT	5.4 MIN./SHIFT	114 MIN./TRIP	28 MIN./SHIFT

NOTES:

1. ALL TIMES IN MINUTES.
2. INCLUDES HAULAGE DELAYS.
3. INCLUDES SETTING BITS AS REQUIRED. NUMBERS IN PARENTHESIS INDICATE NUMBER OF MOVES.
4. SHUTTLE CAR HAULAGE BEGINS.
5. WITHDRAW FLIGHTS AND MOVE TO 75 FOOT PANEL.
6. REMOVE REPAIR AND REPLACE AUGER DRIVE PUMP.
7. REMOVE FAILED CUTTERHEAD FROM MINE AND MAKE REPAIRS.
8. NEW (PERMANENT) CREW BEGINS.
9. MOVE TO 100 FOOT PANEL.
10. HYDRAULIC AUGER DRIVE MOTOR FAILED.
11. AUXILIARY VENTILATION FAN DOWN.
12. 100 FOOT PANEL COMPLETE. MOVE TO END OF PANEL TO BORE INTERSECTING HOLE.
13. INTERSECTING HOLE COMPLETE BEFORE END OF SHIFT. BALANCE OF SHIFT (930 MIN.) NOT INCLUDED AS DELAY.

FMC Corporation, Engineered Systems Division, Santa Clara, California

PRODUCTION AND DELAY SUMMARY-UNDERGROUND AUGER PHASE II DEMONSTRATION

USBM CONTRACT H0125076

Figure E-1 PRODUCTION AND DELAY SUMMARY

Appendix F

PRODUCTION AND DELAY TABULATION BY PANEL

Table F-1 PRODUCTION AND DELAY TABULATION BY PANEL

Item	Production	Production time	Machine setup	Mechanical/ electrical delay	Functional delay	Travel, lunch, firebossing	General mine delay	Total time
50-foot panel	470 tons, total 29.4 tons/shift, average .29 tons/minute, average							
Time (minutes)		1,601	1,430	1,981	316	1,843	509	7,680
Percentage of total		20.85%	18.62%	25.79%	4.12%	24.00%	6.63%	100%
75-foot panel	560 tons, total 32.9 tons/shift, average .31 tons/minute, average							
Time (minutes)		1,825	2,615	2,517	285	2,293	545	10,080
Percentage of total		18.1%	25.94%	24.97%	2.83%	22.75%	5.41%	100%
100-foot panel	970 tons, total 46.2 tons/shift, average .39 tons/minute, average							
Time (mihutes)		2,490	930	3,835	260	1,665	570	9,750
Percentage of total		25.54%	9.54%	39.33%	2.67%	17.06%	5.85%	100%
Total demonstration	2,000 tons, total 34.5 tons/ shift, average .34 tons/minute, average							
Time (minutes)		5,916	4,975	8,333	861	5,801	1,624	27,510
Percentage of total		21.50%	18.08%	30.29%	3.13%	21.09%	5.90%	100%

Appendix G

PRODUCTION/TIME CHARTS

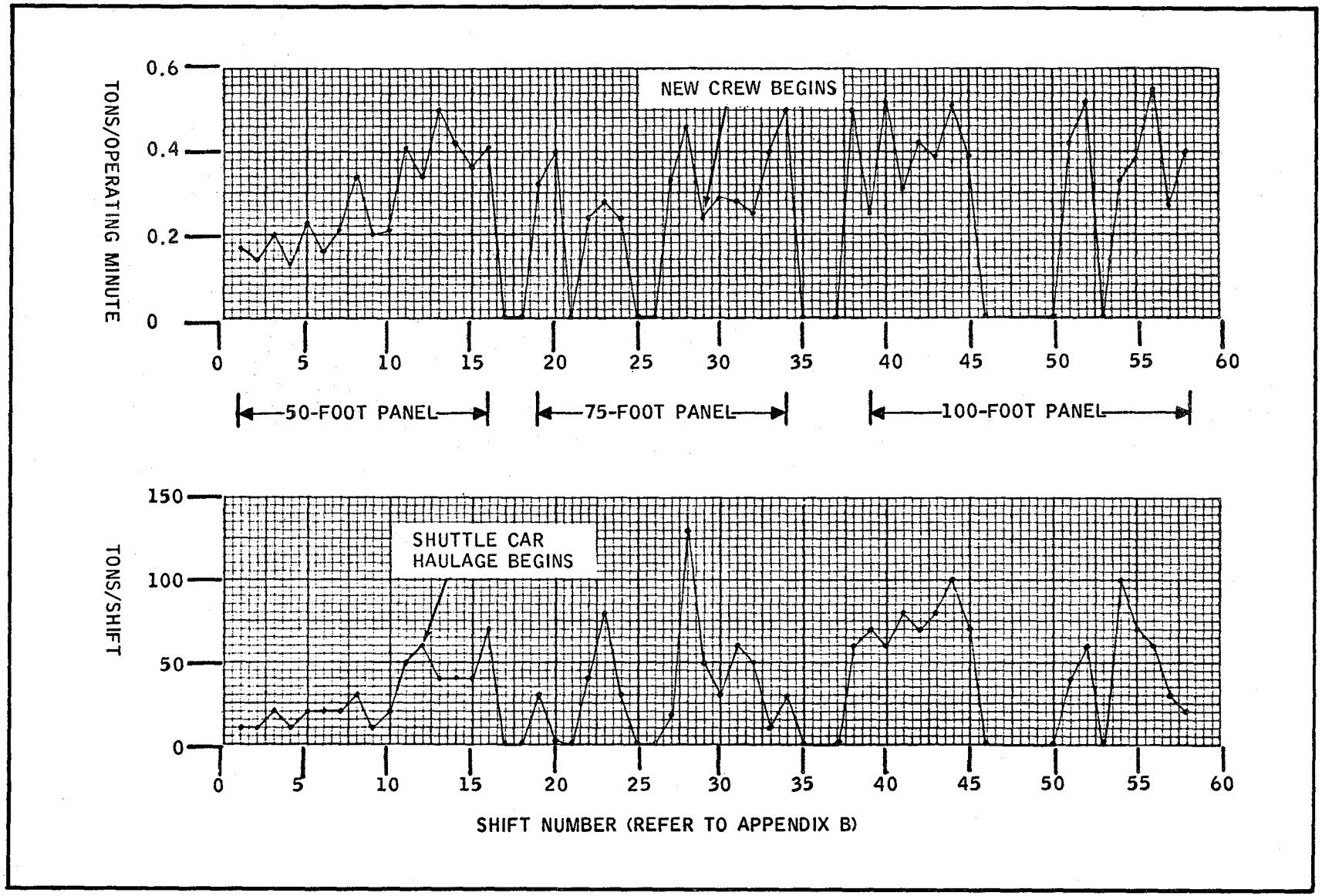


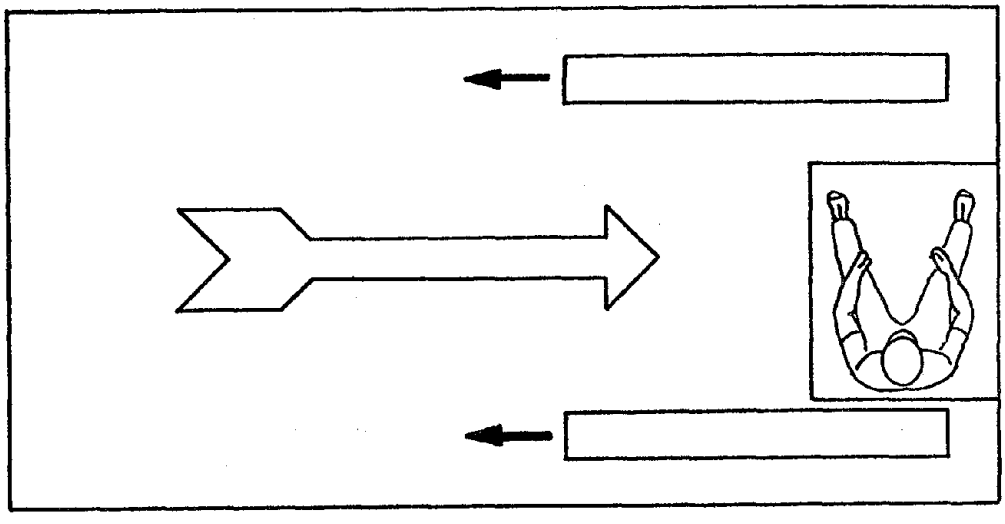
Figure G-1 PRODUCTION/TIME CHARTS

Appendix H

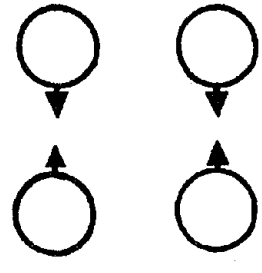
CONTROL LAYOUTS AND TRAMMING INSTRUCTIONS

WALKING MACHINE RIGHT

CAUTION: Retract conveyor before attempting to move machine



Valve Handle Sequence

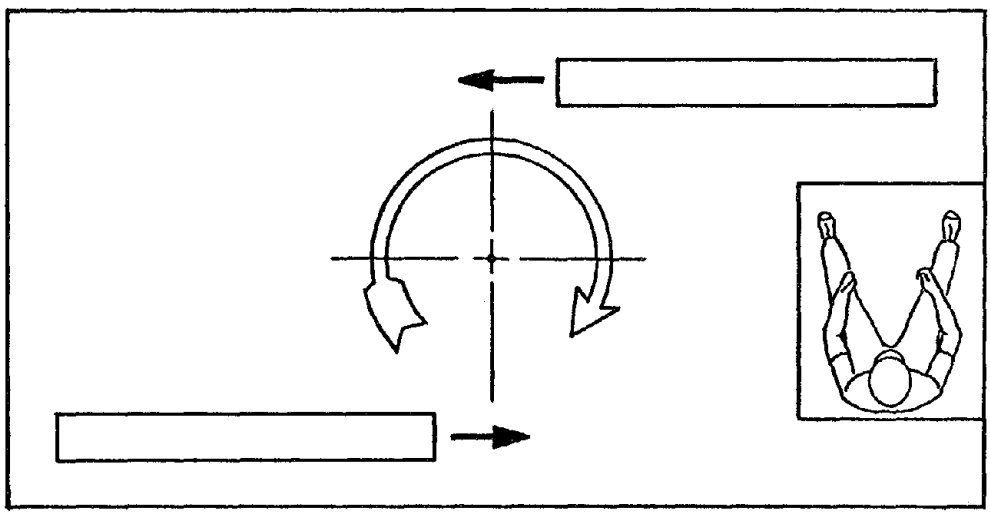


1. Raise frame on floor jacks
2. Front and rear shoe right
3. Lower frame on walking shoes
4. Front and rear shoe left

Figure H-1

STEERING MACHINE CLOCKWISE

CAUTION: Retract conveyor before attempting to move machine



VALVE HANDLE SEQUENCE

① Raise frame on floor jacks

② Front shoe right, rear shoe left

③ Lower frame on walking shoes

④ Front shoe left, rear shoe right

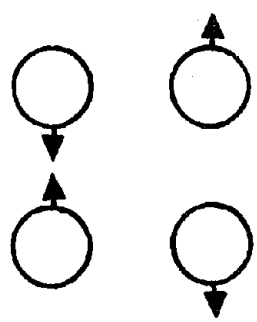
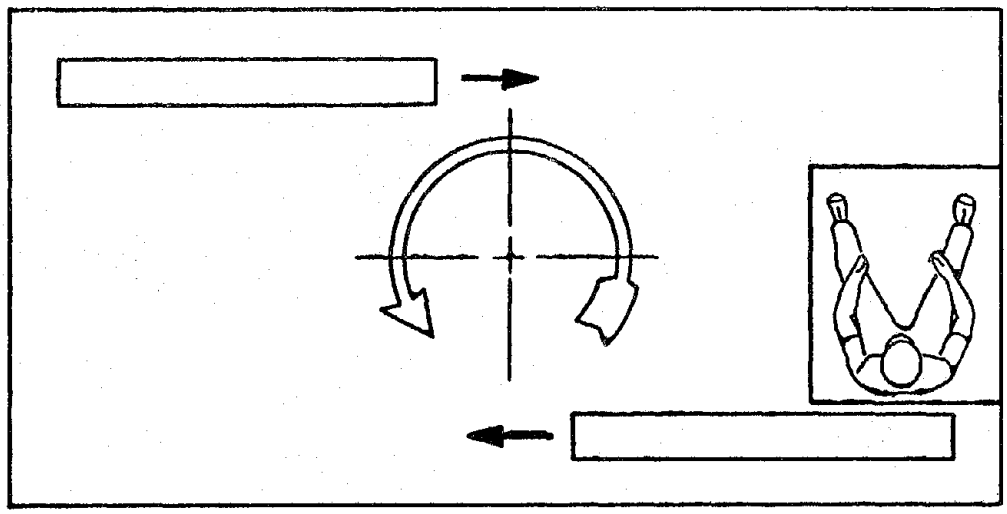


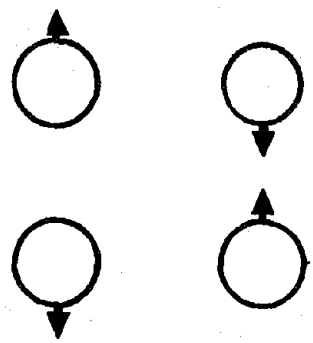
Figure H-2

STEERING MACHINE COUNTERCLOCKWISE

CAUTION: Retract conveyor before attempting to move machine



VALVE HANDLE SEQUENCE

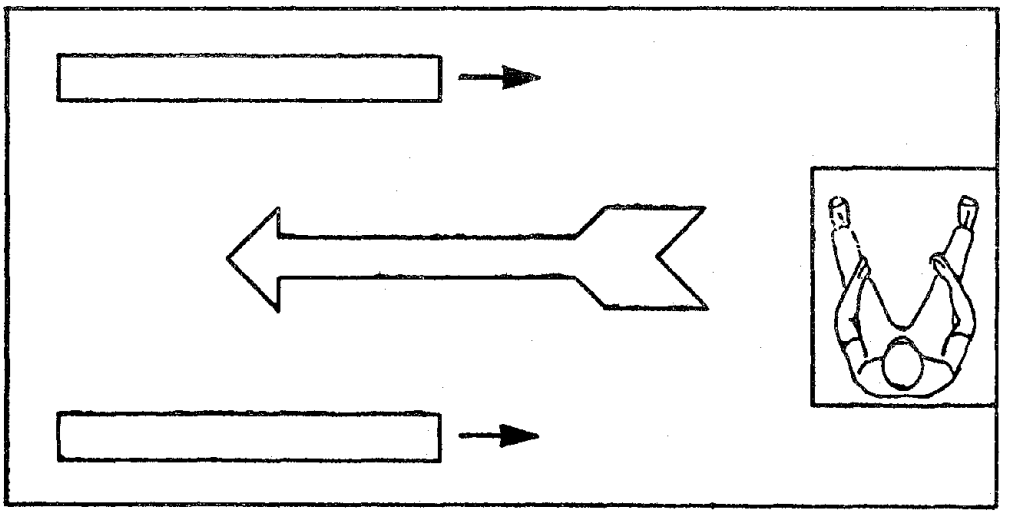


1. Raise frame on floor jacks
2. Front shoe left, rear shoe right
3. Lower frame on walking shoes
4. Front shoe right, rear shoe left

Figure H-3

WALKING MACHINE LEFT

CAUTION: Retract conveyor before attempting to move machine



VALVE HANDLE SEQUENCE

1. Raise frame on floor jacks
2. Front and rear shoe left
3. Lower frame on walking shoes
4. Front and rear shoe right

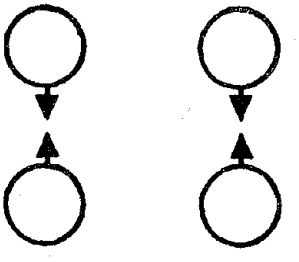
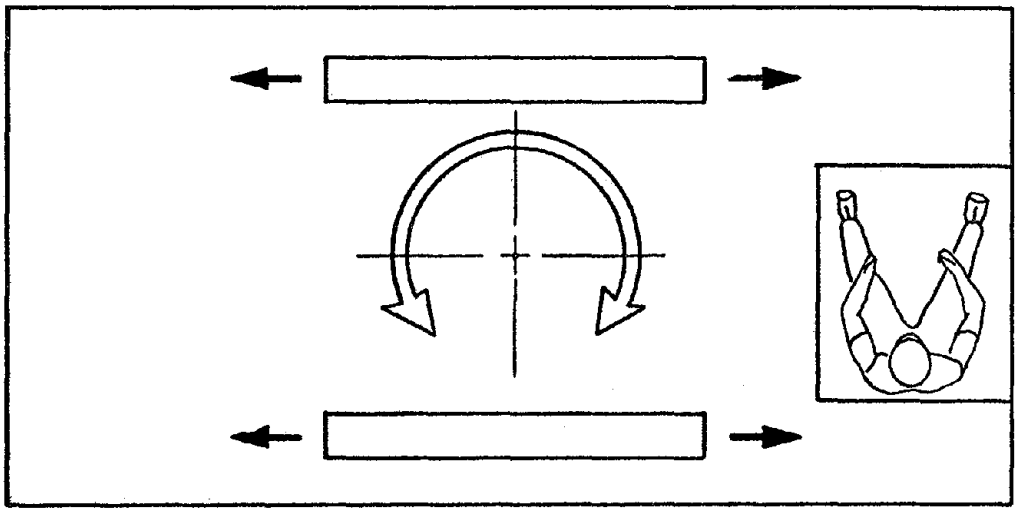


Figure H-4

PIVOT STEERING MACHINE

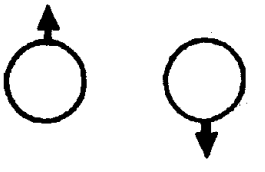
CAUTION: Retract conveyor before attempting to move machine



VALVE HANDLE SEQUENCE

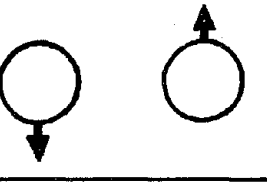
① Raise frame on floor jacks

② Center front and rear shoes
Note: To center walking shoes under frame time a full stroke and divide by two



③ Lower frame on walking shoes

④ To pivot clockwise, front shoe left, rear shoe right



⑤ To pivot counter clockwise, front shoe right, rear shoe left

Figure H-5

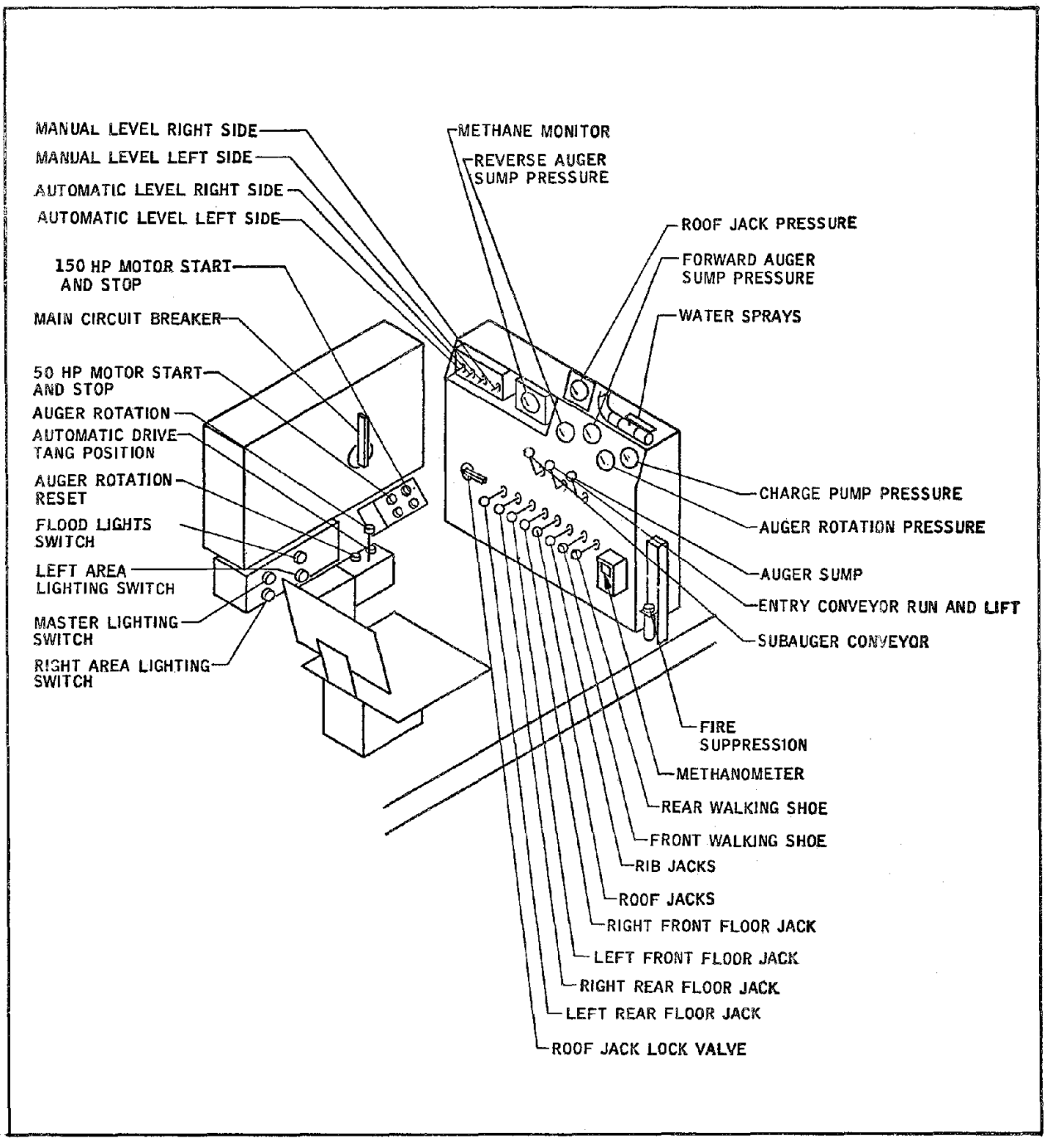

MANUAL LEVEL RIGHT SIDE
MANUAL LEVEL LEFT SIDE
AUTOMATIC LEVEL RIGHT SIDE
AUTOMATIC LEVEL LEFT SIDE
150 HP MOTOR START AND STOP
MAIN CIRCUIT BREAKER
50 HP MOTOR START AND STOP
AUGER ROTATION
AUTOMATIC DRIVE TANG POSITION
AUGER ROTATION RESET
FLOOD LIGHTS SWITCH
LEFT AREA LIGHTING SWITCH
MASTER LIGHTING SWITCH
RIGHT AREA LIGHTING SWITCH
METHANE MONITOR
REVERSE AUGER SUMP PRESSURE
ROOF JACK PRESSURE
FORWARD AUGER SUMP PRESSURE
WATER SPRAYS
CHARGE PUMP PRESSURE
AUGER ROTATION PRESSURE
AUGER SUMP
ENTRY CONVEYOR RUN AND LIFT
SUBAUGER CONVEYOR
FIRE SUPPRESSION
METHANOMETER
REAR WALKING SHOE
FRONT WALKING SHOE
RIB JACKS
ROOF JACKS
RIGHT FRONT FLOOR JACK
LEFT FRONT FLOOR JACK
RIGHT REAR FLOOR JACK
LEFT REAR FLOOR JACK
ROOF JACK LOCK VALVE

Figure H-6

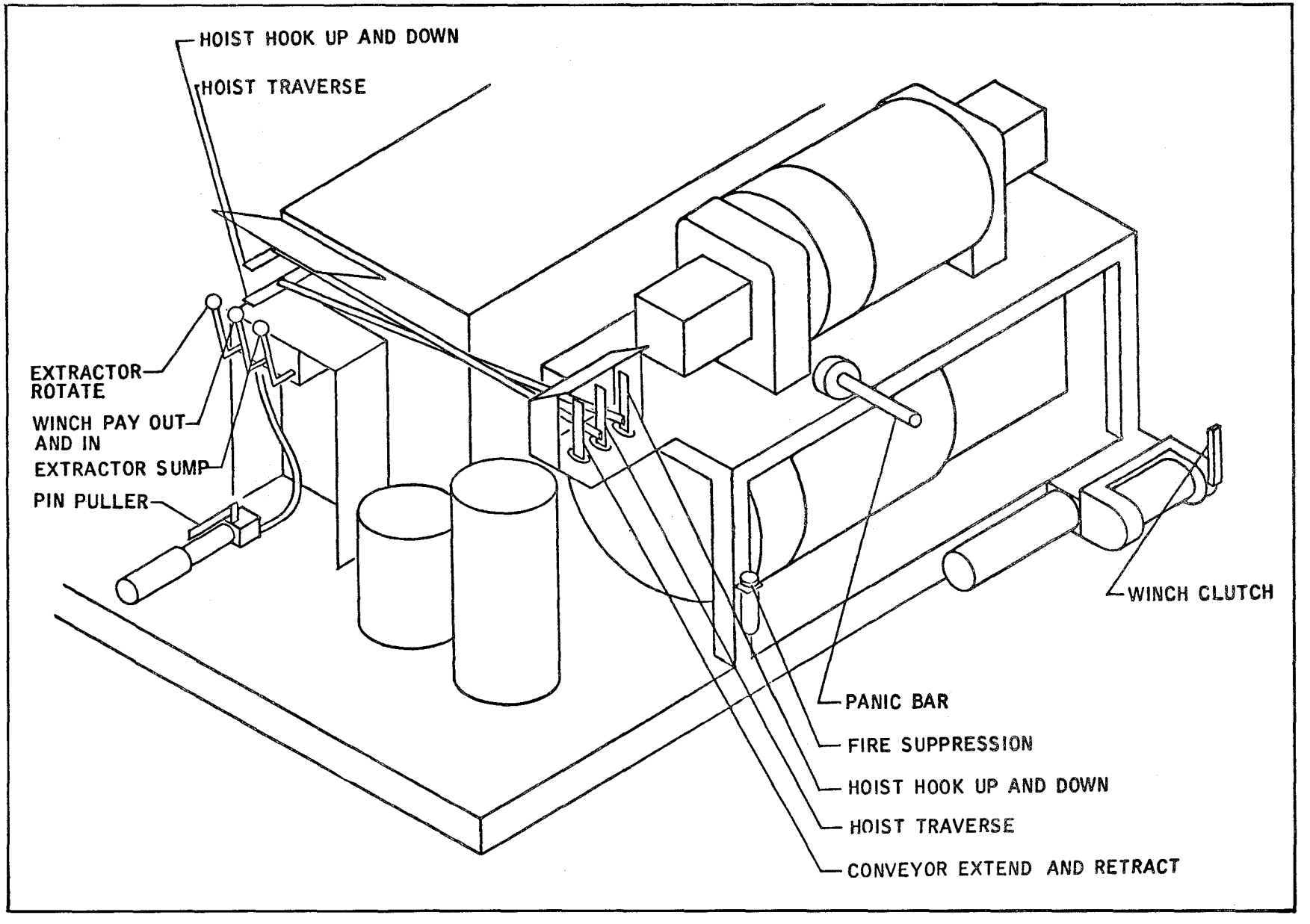

HOIST HOOK UP AND DOWN
HOIST TRAVERSE
EXTRACTOR ROTATE
WINCH PAY OUT AND IN
EXTRACTOR SUMP
PIN PULLER
WINCH CLUTCH
PANIC BAR
FIRE SUPPRESSION
HOIST HOOK UP AND DOWN
HOIST TRAVERSE
CONVEYOR EXTEND AND RETRACT

Figure H-7

Appendix I

TORQUE AND HORSEPOWER CURVES

TORQUE AND HORSEPOWER STUDY FOR UNDERGROUND AUGER PANEL EXTRACTION SYSTEM

by Ed Giroux

2-28-79

This study includes torque, rpm, pressure and horsepower measurements, calculations and graphs for the Underground Auger Panel Extraction System. Graphs include torque vs. borehole depth, horsepower vs. borehole depth, and a comparison graph of these two.

Torque was calculated by using the measured rpm and a Sundstrand speed vs. torque curve for the MH373 motor Catalog #1074 Bulletin 9682. A standard 13,500 ft-lbs was used as the theoretical torque due to the small range of rpms. Assuming the performance varied linearly with pressure, an operating torque was obtained.

$$\frac{\text{(Pressure Observed -50) psi}}{\text{3000 psi}} \times \text{13,500 ft-lbs} = \text{Operating Torque}$$

Horsepower was calculated by using the equation,

$$\frac{\text{Operating Torque (in-lbs)} \times \text{RPM (revs./min)}}{\text{63,000}} = \text{Horsepower}$$

The combined data collected here as shown in graph no. 3 exhibits a range of torque and horsepower probably attributed to the variance in coal hardness. It is a reliable indication, however, of the band width in which the underground auger miner operates.

TEST HOLE #1

BOREHOLE DEPTH		C'HEAD RPM	DRIVE TORQUE (FT-LBS)	DRIVE HP	NOTES
FLITE #	CUM. DEPTH				
C'Head	5	Not Recorded			
1	10	Not Recorded			
2	15	46.3	7,975	69	
3	20	46.3	9,675	85	
4	25	46.2	11,025	97	
5	30	45.5	11,925	103	
6	35	45.5	12,375	107	
7	40	46.2	12,825	113	
8	45	40.0	13,275	101	
9	50	41.7	13,725	109	
10	55	44.1	14,175	119	
11	60	43.5	14,625	121	
12	65				Stalled
13	70	45.8	14,625	128	
14	75	44.8	14,625	125	
15	80				Stalled
16	85				Holed Thru

Figure I-1

TEST HOLE #2

BOREHOLE DEPTH		C'HEAD RPM	DRIVE TORQUE (FT-LBS)	DRIVE HP	NOTES
FLITE #	CUM. DEPTH				
C'Head	5	Not Recorded	7,650	-	
1	10	46.9	7,875	70	
2	15	46.9	8,775	78	
3	20	47.6	9,900	90	
4	25	47.6	9,900	90	
5	30	46.9	11,025	98	
6	35	46.9	12,150	108	
7	40	46.9	13,275	119	
8	45	45.5	13,275	115	
9	50	44.8	13,275	113	
10	55	46.2	13,275	117	
11	60	46.2	13,275	117	
12	65	42.9	13,275	108	
13	70	44.8	13,275	113	
14	75	44.8	12,150	104	
15	80	43.5	13,275	110	
16	85	43.5	13,275	110	
17	90				Holed Thru

Figure I-2

TEST HOLE #3

BOREHOLE DEPTH		C'HEAD RPM	DRIVE TORQUE (FT-LBS)	DRIVE HP	NOTES
FLITE #	CUM. DEPTH				
C'Head	5	46.2	7,650	67	
1	10	58.8	5,400	61	
2	15	46.9	6,525	58	
3	20	46.9	6,525	58	
4	25	46.9	7,650	68	
5	30	46.2	7,650	67	
6	35	46.2	8,775	77	
7	40	46.2	8,775	77	
8	45	45.5	11,025	95	
9	50	45.5	11,025	95	
10	55	44.8	11,025	94	
11	60	44.1	12,150	102	
12	65	44.8	12,150	104	
13	70	43.5	12,150	101	
14	75	43.5	13,275	110	
15	80	41.1	13,275	104	
16	85	41.1	13,275	104	
17	90				Holed Thru

Figure I-3

TEST HOLE #4

BOREHOLE DEPTH		C'HEAD RPM	DRIVE TORQUE (FT-LBS)	DRIVE HP	NOTES
FLITE #	CUM. DEPTH				
C'Head	5	46.2	7,650	67	
1	10	46.9	8,775	78	
2	15	45.5	8,775	76	
3	20	45.5	8,775	76	
4	25	45.5	11,025	95	Holed Thru #3
5	30	45.5	11,025	95	
6	35	45.5	11,025	95	
7	40	46.2	5,400	47	Coal Spilling into #3
8	45	46.9	5,400	48	

Figure I-4

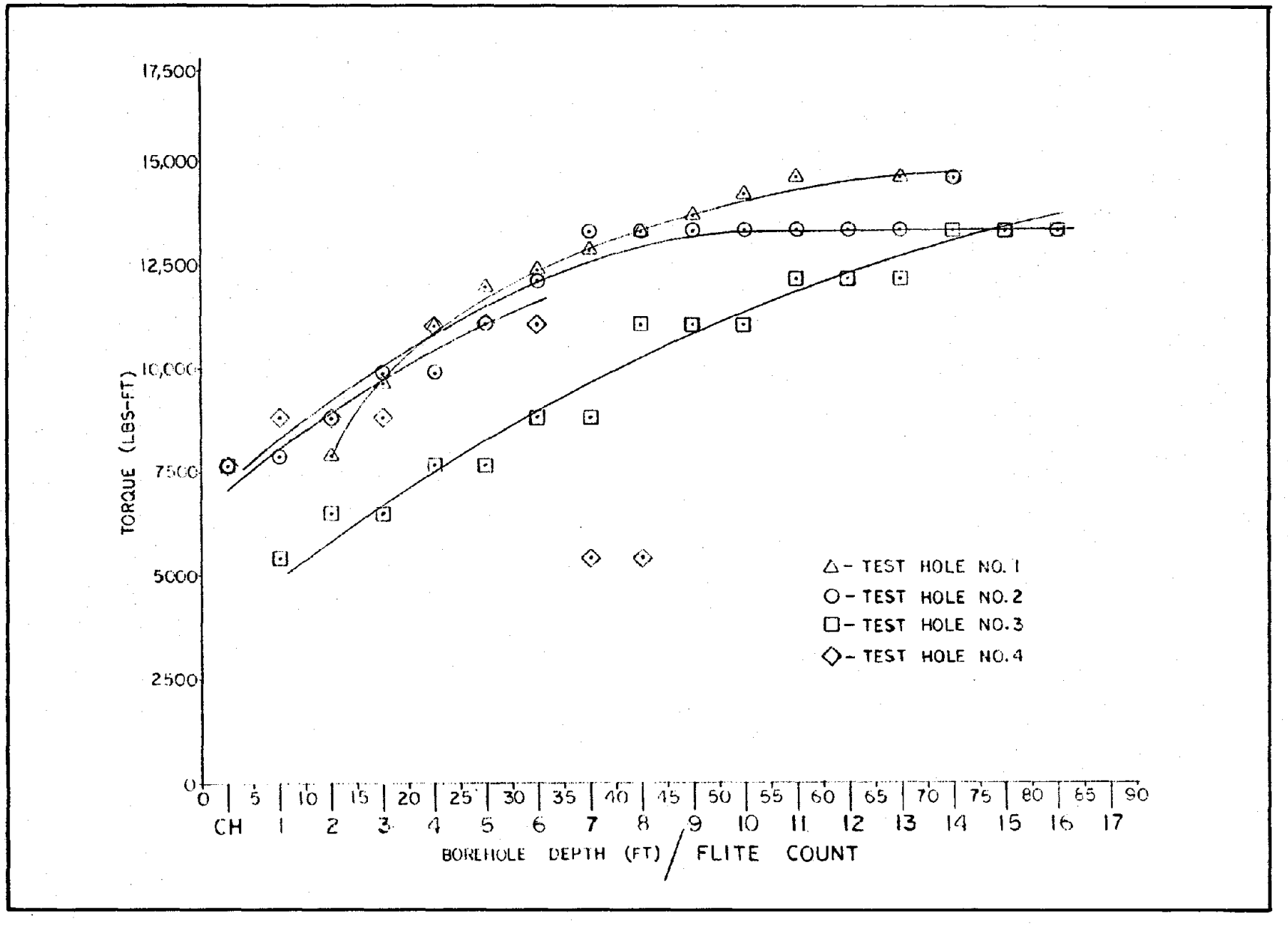


Figure I-5 GRAPH NUMBER 1

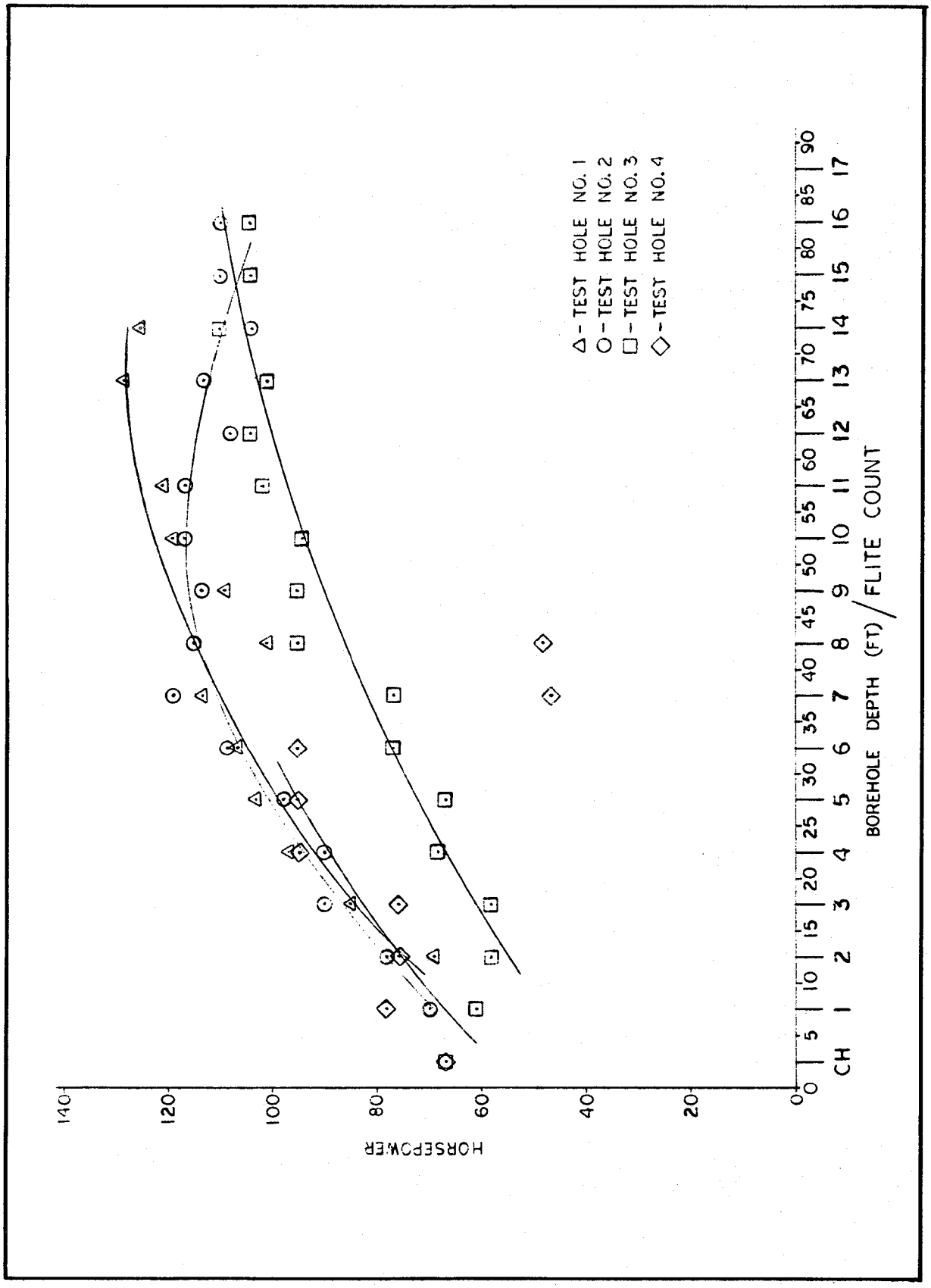


Figure I-6 GRAPH NUMBER 2

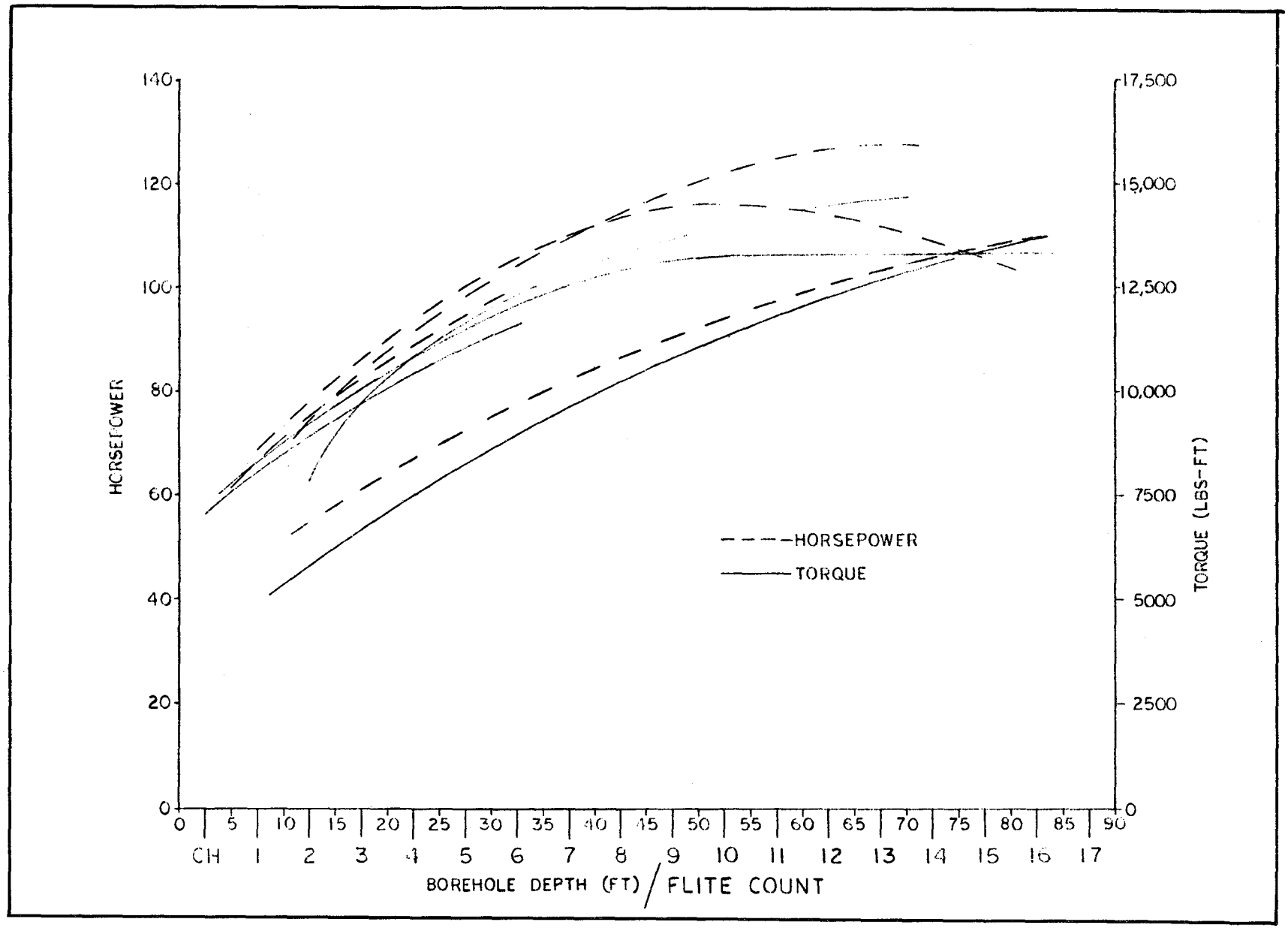


Figure I-7 GRAPH NUMBER 3

Appendix J

DRIVESHAFT FAILURE ANALYSIS

FMC Central Engineering Laboratories
Santa Clara

MATERIALS LABORATORY TEST REPORT

Lab No: 732784

Order No:	8483-140-33	Part Name:	Drive Shaft, Auger Drill	Part No:	
Date:	5/1/79			Lot Size:	
Req By:	Joe Ritchey	Heat No:		P.O. No:	D-09771
Phone:	3275	Size:		Rec. No:	
Div/Plt:	ESD	Mfgr:		No. Test Pcs:	1

Specification: 1045 CR Steel

Information Desired:

- ☐ Conformance of Material to Specification
- ☐ Conformance of Certification to Specification
- ☐ Process Analysis, Bath/Panel Conformance
- ☒ Other — Explain

Brief Failure Analysis----failed at approximately 60,000 cycles.
(Look for machining discontinuities)

Results: 5-14-79

CUSTOMER REQUEST

Perform failure analysis of drive shaft that achieved 60,000 cycles before breaking.

INVESTIGATION

MACROANALYSIS

The fracture surface of the broken shaft is shown in Figure 1. The fracture surface revealed a oval-shaped final fast fracture region (light-colored) indicating that unequal perpendicular bending stresses were applied. Several ratchet marks are present on the periphery of the fracture surface in Figure 1, indicative of numerous initiation points. A machining groove on the shaft radius served as the site where the fatigue crack initiated is shown in Figure 2 and indicated in Figure 1.

MICROANALYSIS

A photomicrograph of the microstructure is shown in Figure 3 and consists of an equiaxed grain structure containing both pearlite and ferrite which indicates the part was not heat treated.

CHEMICAL ANALYSIS

The chemical analysis did not concur with the steel carbon 1045 material specified. The chemical analysis (page 2) more nearly conforms to a 1020 grade.

Exhibit J-1 (Sheet 1 of 2)

Continuation Sheet

MATERIALS LABORATORY TEST REPORT

Lab No: 792784

CHEMICAL COMPOSITION

Element	Part	STANDARD 1045	STANDARD 1020
C	.21	.43-.50	.18-.23
Mn	.73	.6 -.9	.6 -.9
P	.02	<.04	<.04
S	.02	<.05	<.05
Si	.22	-	-
Ni	.16	-	-
Cr	<.05	-	-
Mo	.05	-	-

HARDNESS

The hardness was BHN 140 (3000) which is equivalent to a tensile strength of ≈70 KSI.

CONCLUSIONS & RECOMMENDATIONS

The shaft failed in a bending plus torsional fatigue mode. Cracking initiated in a machining groove in the reduced radius region of the shaft. The material was not steel 1045 as specified but steel 1020.

The reduced radius should have been uniformly shot peened to remove machining grooves and defects that create stress raisers. Substitution of a heat treatable alloy steel will promote increased fatigue resistance in the shaft, alloy steels such as 4140 and 4340 hardened to Rc 30-35 will more than double the fatigue strength.

Tom Kenfield

Tom Kenfield

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Exhibit J-1 (Sheet 2 of 2)

Figure J-1 1/2 scale

Fracture surface showing (a) oval shaped, final fast fracture region (b) ratchet marks indicative of numerous crack initiation points and (c) identification of primary initiation site.

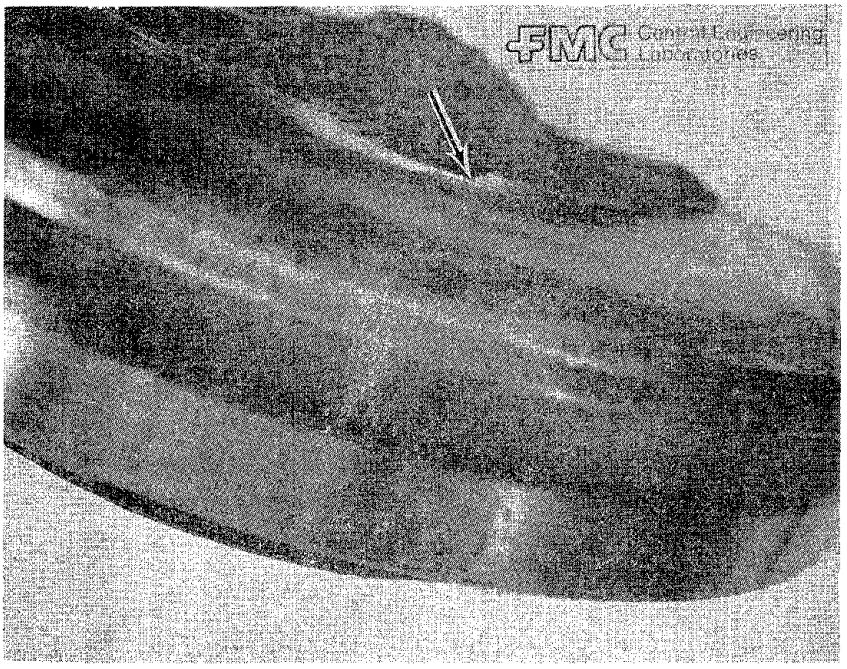

Figure J-2 Full Scale

Machined groove fatigue crack initiation site (arrow) which served as primary initiation point for crack propagation.

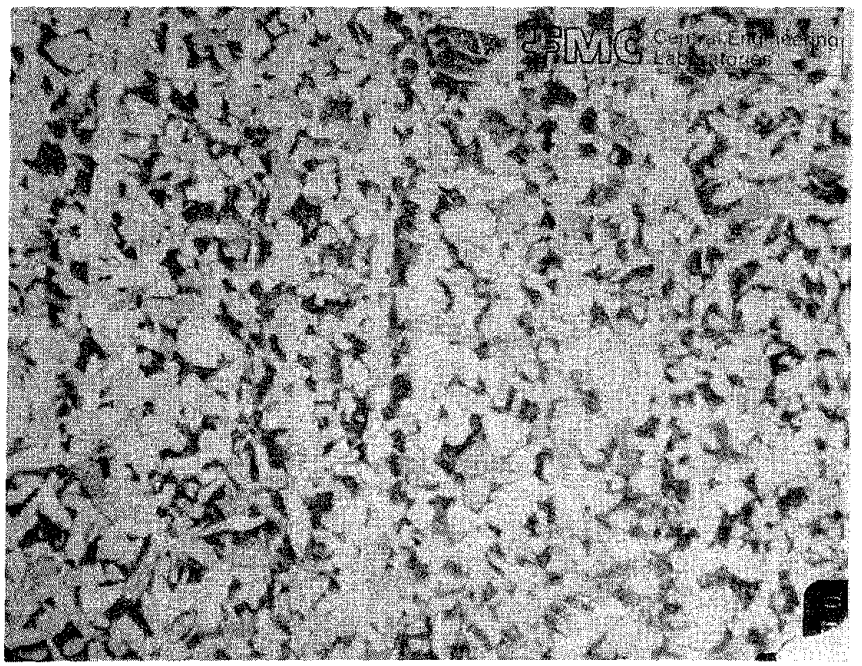

Figure J-3 MAG=60X

Photomicrograph showing equiaxed grain structure containing both ferrite and pearlite.

Appendix K

DISTRIBUTOR RING FAILURE

FMC Central Engineering Laboratories
Santa Clara

MATERIALS LABORATORY TEST REPORT

Lab No: 792785

Chg No:	8483-141-33	Part Name:	Distributor	Part No:	
Date:	5/1/79			Lot Size:	
Req By:	Joe Ritchey	Heat No:		P.O. No:	D-09771
Phone:	3275	Size:		Rec. No:	
Div/Plt:	ESD	Mfgr:	Sunstrand Co.	No. Test Pcs:	1

Specification:
Cast Iron

Information Desired:
☐ Conformance of Material to Specification
☐ Conformance of Certification to Specification
☐ Process Analysis, Bath/Panel Conformance
☒ Other — Explain

Subjected to 5000 psi across the lands. (intermittent hydraulic shock load)

Results: 5-25-79

CUSTOMER REQUEST

Determine reason for catastrophic failure of distributor casting.

BACKGROUND

The distributor serves as a hydraulic slip ring, diverting oil for a rotating shaft mechanism. The distributor receives a constant 3,000 to 3,500 psi hydraulic pressure across the lands (intermittently loaded to 5,000 psi for several seconds) as shown in Figure 1.

INVESTIGATION

Macroanalysis

The fracture occurred through the holed section between the lands, Figure 2. There were no obvious areas where initiation of the failure may have occurred.

Microanalysis

A photomicrograph revealed a fairly typical gray iron microstructure of a pearlitic matrix with graphite flakes (Type C) dispersed throughout. SEM fractography, Figure 4, shows essentially a brittle fracture surface with some indications of porosity.

X-ray Analysis

X-ray analysis of a replacement casting disclosed regions of shrinkage, Figure 5 & 6, in the vicinity of the holed section between the lands. There is some shrinkage also

Exhibit K-1 (Sheet 1 of 2)

seen in the center ribbed region, Figure 6. A sound region of the casting is shown in Figure 7.

Hardness

Hardness of BHN 179 (3000 kg) was obtained at various locations on the casting.

Chemical Analysis

The chemical composition (Table 1) of the casting, except for the total carbon content and carbon equivalent, along with knowledge of the microstructure and hardness appear to be most representative of a class 30 gray iron.

TABLE 1 CHEMICAL COMPOSITION (% Wt)

Element	Casting	Class 30
Total Carbon	2.8	3.25 min.
Carbon Equiv.	3.6	4.25 max.
Si	2.4	2.4 max.
Mn	0.9	0.7-1.0
P	0.1	0.1 max.
S	0.2	.15 max.
Ni	0.1	-
Cr	0.05	0.4
Mo	0.1	-

CONCLUSIONS AND RECOMMENDATIONS

The primary reason for failure of the casting was from the shrinkage in the region between the lands where the load is applied. The shrink section will not be able to withstand any sustained loading before yielding would occur. The total carbon content and carbon equivalent should be higher to promote increased graphitization and thereby reduce the shrinkage tendency.

Tom Kenfield

Tom Kenfield

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Exhibit K-1 (Sheet 2 of 2)

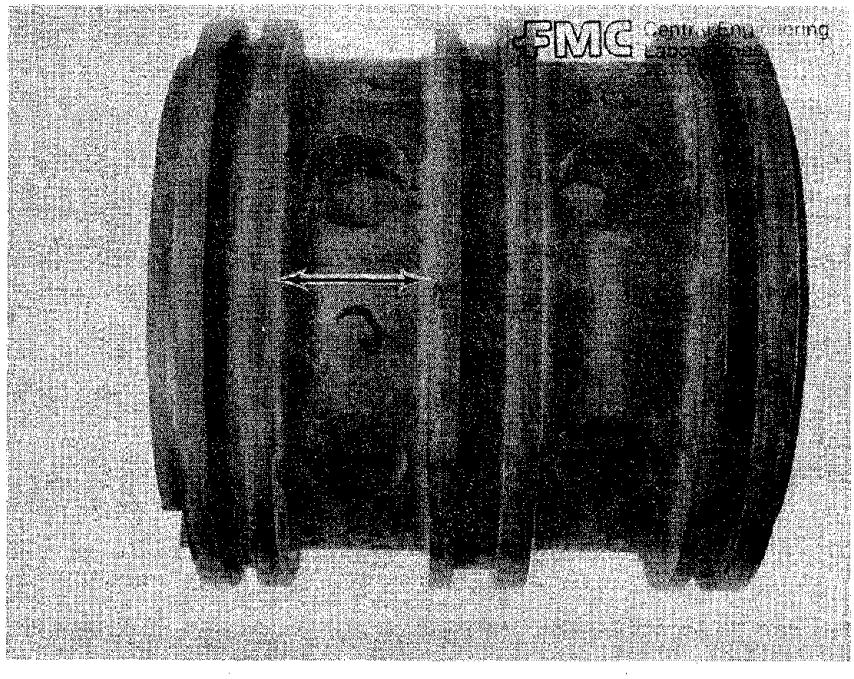

Figure K-1 **Half Scale**

Compressive loading pattern during operation of distributor casting.

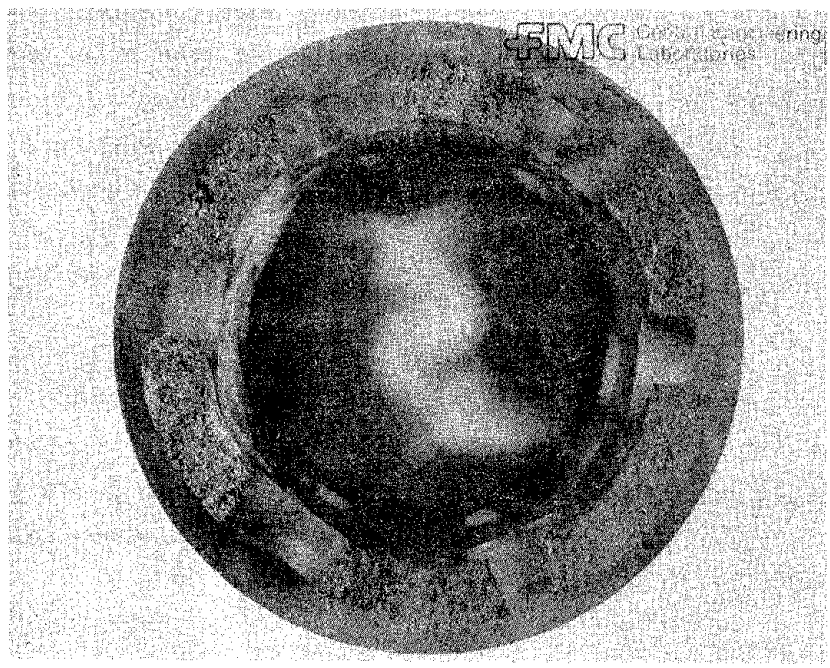

Figure K-2 Half Scale

Failed section of distributor casting through holed section.

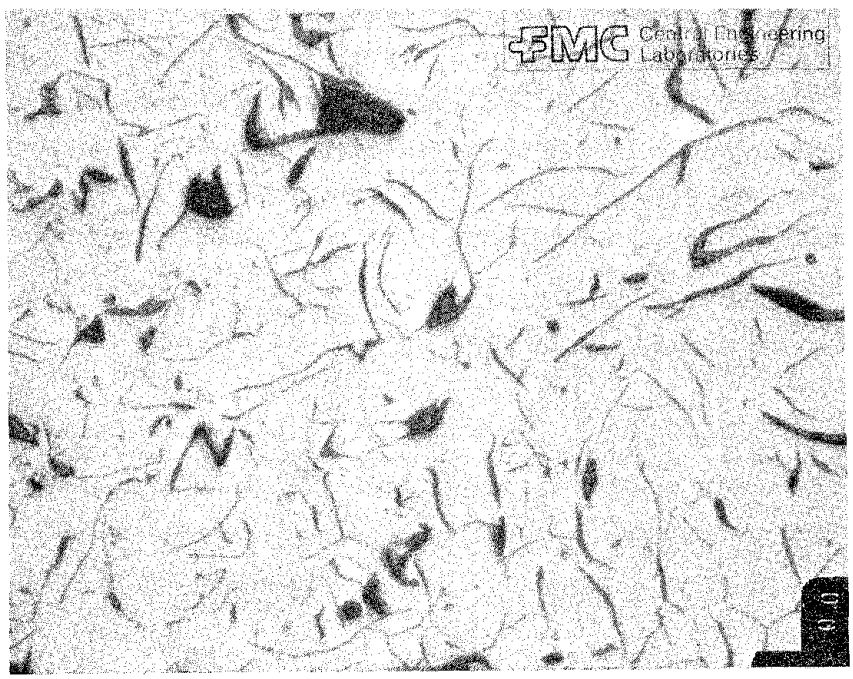

Figure K-3 MAG=100X

Microstructure of gray iron distributor displaying a Type C flake distribution.

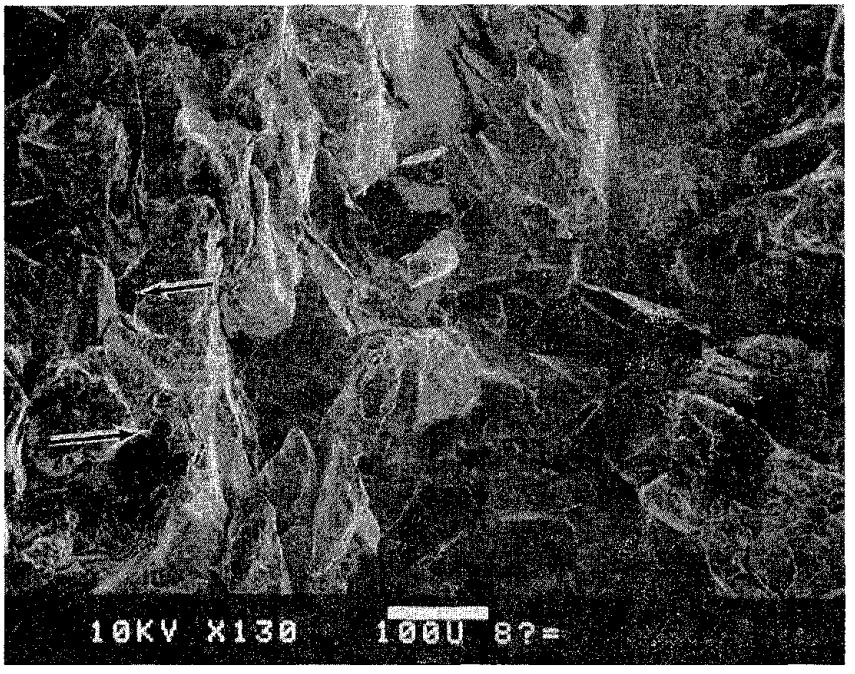

Figure K-4 MAG=130X

SEM fractograph showing an essentially brittle fracture surface with some evidence of porosity (arrows).

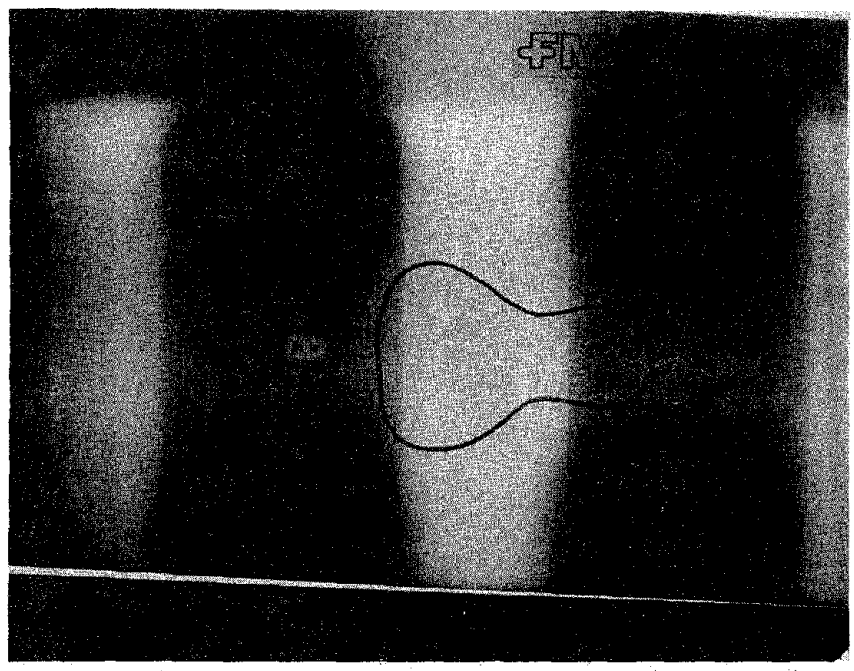

Figure K-5 Full Scale

Replacement casting showing regions of shrinkage porosity (circled area).

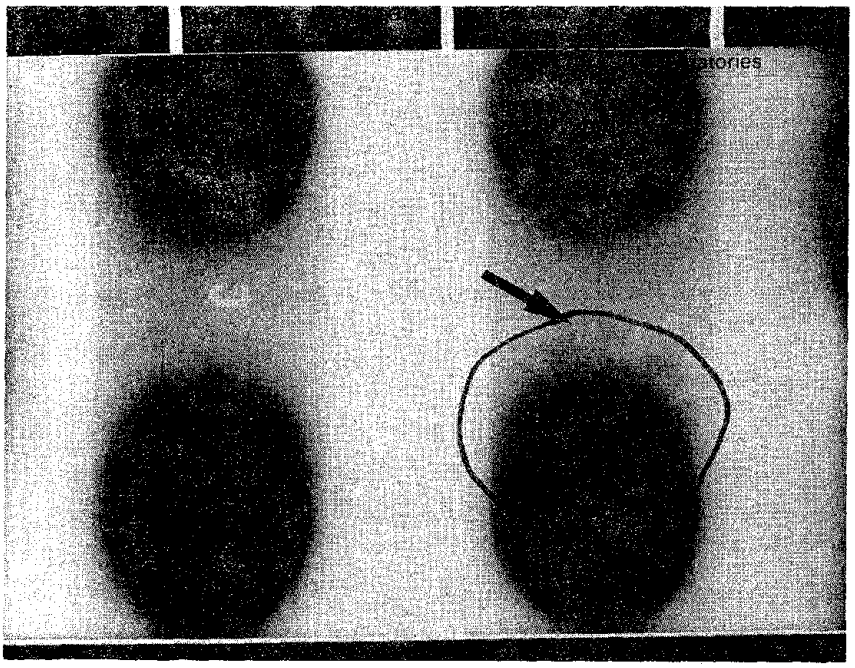

Figure K-6 Full Scale

Replacement casting showing regions of shrinkage porosity (circled area).

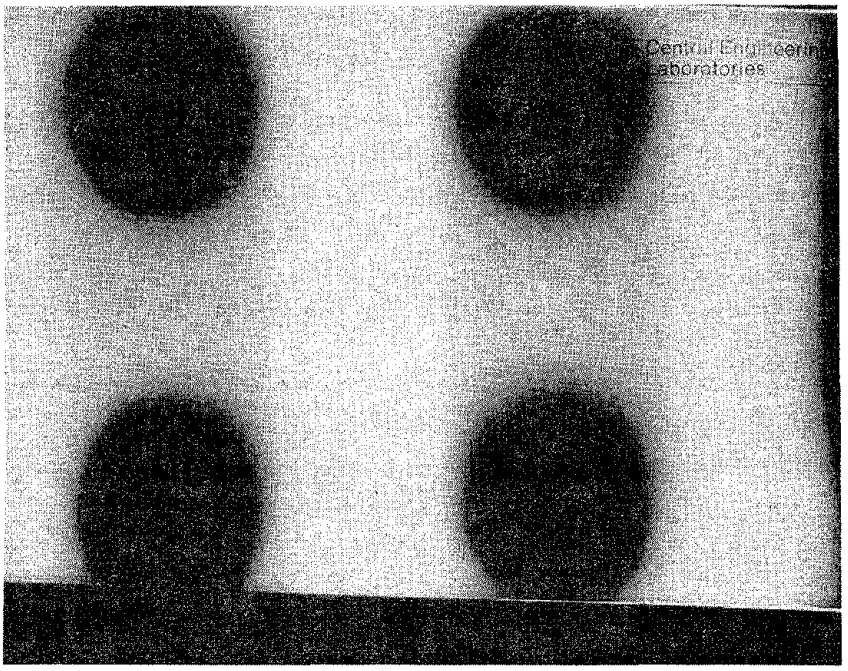

Figure K-7 Full Scale

Sound region of replacement casting.

Appendix L

CHRONOLOGY OF MSHA-APPROVED ACTIVITIES

APPENDIX L
CHRONOLOGY OF MSHA-APPROVED ACTIVITIES

Although approvals were already granted for the first prototype auger mine (the Coalbadger) when it was purchased, the second prototype required a field modification and an extension to the original approval. Since new intrinsically safe circuits were added to the machine, MSHA investigation and acceptance was required. Outlined as follows is a chronology of the activities which were required to obtain the acceptances.

Date	Event
May 16, 1977	Submitted initial letter to MESA on the speed control system.
June 23, 1977	Submitted initial letter to MESA on the leveling system.
January 13, 1978	Updated the drawings on both.
January 13, 1978	Submitted status report to the Technical Project Officer of USBM on I. S. Approvals.
April 14, 1978 (2 separate letters)	Updated drawings by adding drawing 5135610 describing the source of the 110-volt AC input power to the DC power supply.
April 24, 1978	Outlined electrical field changes on the auger to Consolidation Coal Company.
May 26, 1978	Updated circuits to use the Honeywell servo valve as an alternate to the Moog.
June 23, 1978	Updated all drawings and sent parts to the MESA investigator for physical investigation.
July 5, 1978	Further information sent to MESA to support the investigation.
July 26, 1978	Further information sent to MESA to support the investigation.
August 1, 1978	Further information sent to MESA to support the investigation.
August 21, 1978	Sent new sets of drawings incorporating changes desired by MESA.

Date	Event
August 23, 1978	Sent new sets of drawings incorporating changes desired by MESA.
September 6, 1978	Received favorable evaluation of rotational control system from MSHA. #1A-235 MSHA: A&CC: PAR #0019068.
September 7, 1978	Received favorable evaluation of leveling system from MSHA. #1A-173 MSHA: A&CC: PAR #0020299.
October 19, 1978	Sent letter with final data items for file and corrected typo on acceptance letter.
October 31, 1978	Received new evaluation on 1A-173 correcting contract number and typo errors.
January 9, 1979	Corrected drawings after the fact for Lambda Power supply modification error.
April 10, 1979	Received SNAP APPROVALS #0752 and 0753 on circuit modification.

Appendix M
LEARNING CURVE

FMC Engineered Systems Division
Santa Clara

CALCULATION SHEET

Project No.								
5	2	5						

Client DOE Date March 1981

Sheet 1 of 2 Sheets

Project Title Auger Panel Extraction System

Calculated By L. McDonald

Subject Learning Curve

Checked By

Data analysis was performed in order to determine whether a learning curve was present in the production data of the in-mine-demonstration of the auger panel extraction system. Evidence of learning must be removed in order to predict the average production rates to be expected after familiarity with the system is complete.

The sample data used for analysis consists of fifty production shifts. Operating time per shift is defined as 480 minutes minus all delays per shift. Delays are defined as general delays, face delays due to auger downtime, and face delays due to downtime not attributable to auger operation. Moving time is included in operating time, since learning is expected to occur in the moving function.

Table 1 lists the production data used for this analysis. Summary data is shown below.

Average tons mined per shift	94
Average operating minutes per shift	225
Average tons per operating minute	.4154
Average operating minutes per ton	2.4074

The time required per unit of production (tons) was normalized, divided by average operating minutes per ton. Figure 1 illustrates the slight downward trend in time required per production unit. The large dispersion around the mean time (2.4074 operating minutes per ton) is explained by the fact that production is influenced by a great many factors other than operating time. The large unexplained variance represents factors such as crew composition and size, efficiencies of other mine equipment, mine and face conditions, etc. In order to determine whether learning was responsible for the decrease in operating times during the in-mine test, a regression line was fitted to the data. Results are shown in Figure 2. Logarithms were computed for the data in order to detect the slope of the learning curve line. This regression line was then used to illustrate the downward trend in the original data. Figure 3 shows the learning function and production rates. The average production rate (with learning removed from the test data) approaches 2.1666

operating minutes per ton (2 minutes, 10 seconds). The figure also illustrates, as expected, that learning is virtually complete at the end of the test (50 production days). The resulting prediction for production rate with the auger system is 2.1666 operating minutes per ton, or .46 tons per operating minute.

	Production Rates	
Element	Average	Learning Removed
Operating Minutes per Ton	2.4074	2.1667
Tons Per Operating Minute	.4154	.4615

FMC Engineered Systems Division
Santa Clara

CALCULATION SHEET

Project No.

Client _____ Date _____

Project Title _____

Subject _____

Sheet 1 of 2 Sheets

Calculated By _____

Checked By _____

TABLE 1

PRODUCTION DAY	TONS MINED	CUM. TONS MINED	PRODUCTION DELAYS	OPER. MIN.	TONS PER OPER. MIN.	OPER. MIN. PER TON	NORMALIZED OPER. MIN. PER TON
2	56	56	246	234	.2393	4.1786	1.7356
3	84	140	285	195	.4308	2.3214	.9643
4	98	238	197	283	.3463	2.8878	1.1996
5	42	280	341	139	.3022	3.3095	1.3747
6	126	406	198	282	.4468	2.2381	.9297
7	42	448	302	178	.2360	4.2381	1.7604
8	98	546	263	217	.4516	2.2143	.9198
9	70	616	371	109	.6422	1.557	.6468
10	56	672	129	351	.1595	6.2679	2.6036
11	168	840	168	312	.5385	1.8571	.7714
12	84	924	316	164	.5122	1.95238	.8110
13	154	1,078	246	234	.6581	1.5195	.6312
14	140	1,218	195	285	.4912	2.0357	.8456
15	56	1,274	289	191	.2932	3.4107	1.4168
16	56	1,330	381	99	.5657	1.7679	.7344
17	98	1,428	317	163	.6012	1.6633	.6909
18	154	1,582	178	302	.5099	1.9610	.8146
19	42	1,624	264	216	.1944	5.1429	2.1363
20	56	1,680	342	138	.4058	2.4643	1.0236
21	98	1,778	233	247	.3968	2.5204	1.0469
23	84	1,862	221	259	.3243	3.0833	1.2808
24	112	1,974	168	312	.3590	2.7857	1.1571
25	98	2,072	240	240	.4083	2.4490	1.0173
26	70	2,142	302	178	.3933	2.5429	1.0563
27	42	2,184	325	155	.2710	3.6905	1.5330
29	70	2,254	302	178	.3933	2.5429	1.0563

FMC Engineered Systems Division
Santa Clara

CALCULATION SHEET

Project No.

Client _____ Date _____

Sheet 2 of 2 Sheets

Project Title _____

Calculated By _____

Subject _____

Checked By _____

TABLE 1

PRODUCTION DAY	TONS MINED	CUM. TONS MINED	PRODUCTION DELAYS	OPER. MIN.	TONS PER OPER. MIN.	OPER. MIN. PER TON	NORMALIZED OPER. MIN. PER TON
30	140	2,394	156	324	.4321	2.3143	.9613
32	98	2,492	185	295	.3322	3.0102	1.2504
33	140	2,632	138	342	.4094	2.4429	1.0148
34	182	2,814	143	337	.5401	1.8517	.7692
35	112	2,926	190	290	.3862	2.5893	1.0756
36	84	3,010	294	186	.4516	2.2143	.9198
37	112	3,122	285	195	.5744	1.7411	.7232
39	42	3,164	279	201	.2090	4.786	1.9880
40	84	3,248	357	123	.6829	1.4643	.6082
41	112	3,360	209	271	.4133	2.4196	1.0051
42	84	3444	287	193	.4352	2.2976	.95439
45	84	3,528	318	162	.5185	1.9286	.8011
46	112	3,640	305	175	.64	1.5629	.6490
49	70	3,710	358	122	.5738	1.7429	.7240
50	70	3,780	197	283	.2474	4.0429	1.6794
51	70	3850	245	235	.2979	3.3571	1.3945
52	70	3,920	351	129	.5426	1.8429	.7655
53	98	4,018	268	212	.4623	2.1633	.8986
54	56	4,074	169	311	.1801	5.5536	2.307
55	154	4128	250	230	.6700	1.4935	.6204
56	168	4,396	236	244	.6885	1.45238	.6033
57	84	4,480	254	226	.3717	2.6905	1.1176
63	98	4,578	328	152	.6447	1.5510	.6443
65	98	4,676	348	132	.7424	1.3469	.5595
	4,676		12,743	11,257			

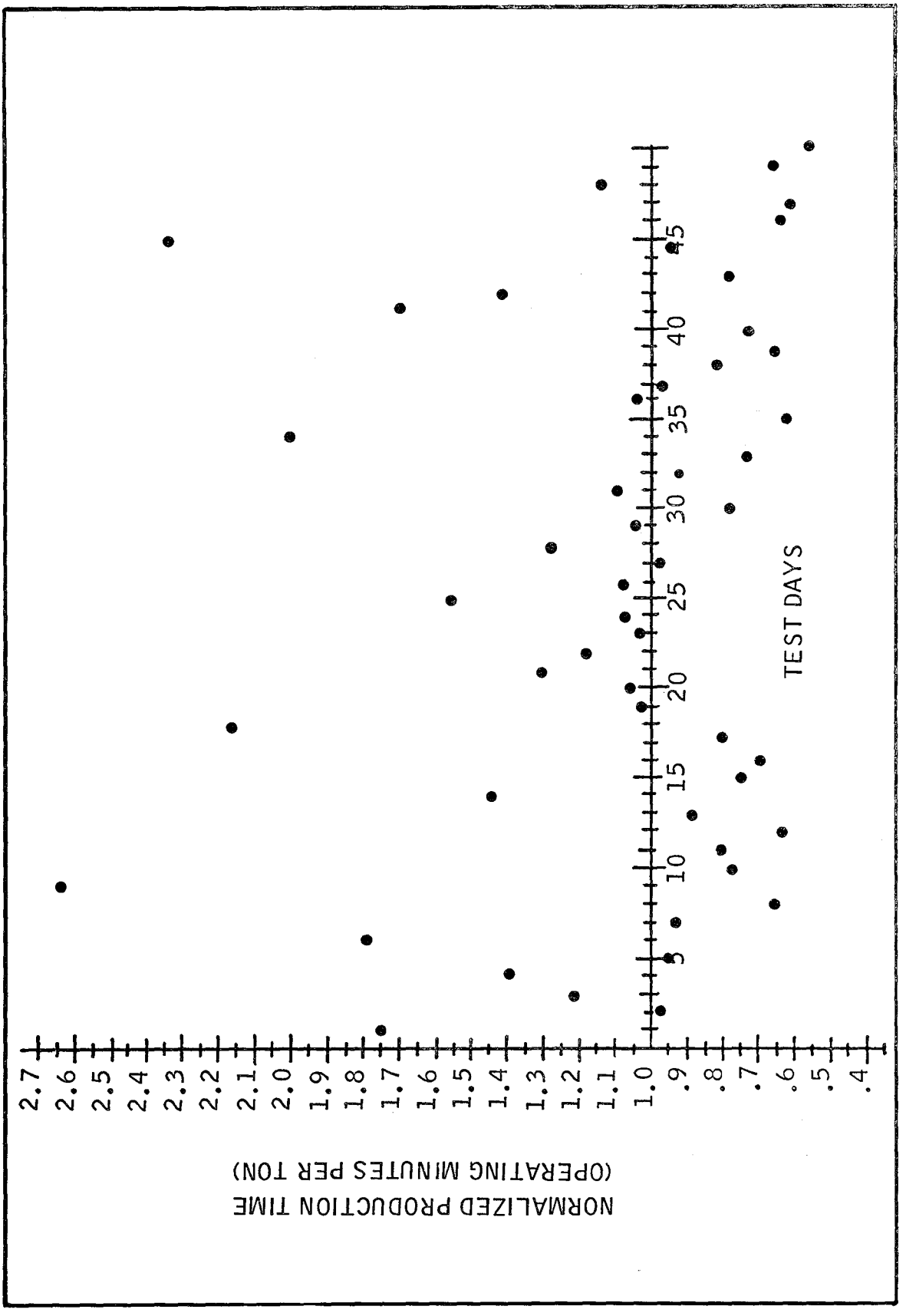

Figure M-1 NORMALIZED PRODUCTION RATES

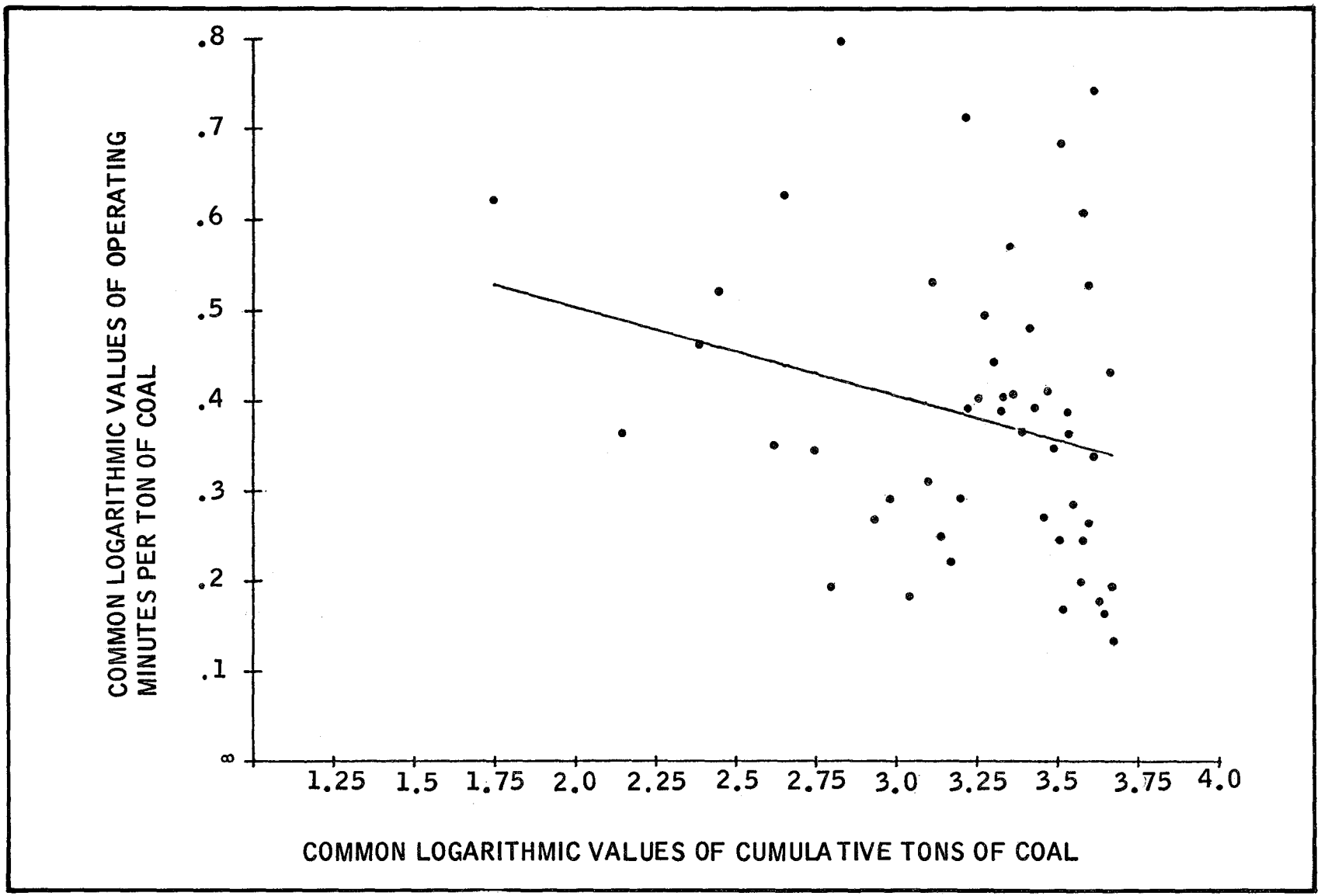

Figure M-2 LINEAR REGRESSION OF PRODUCTION RATES PLOTTED ON LOG/LOG SCALE

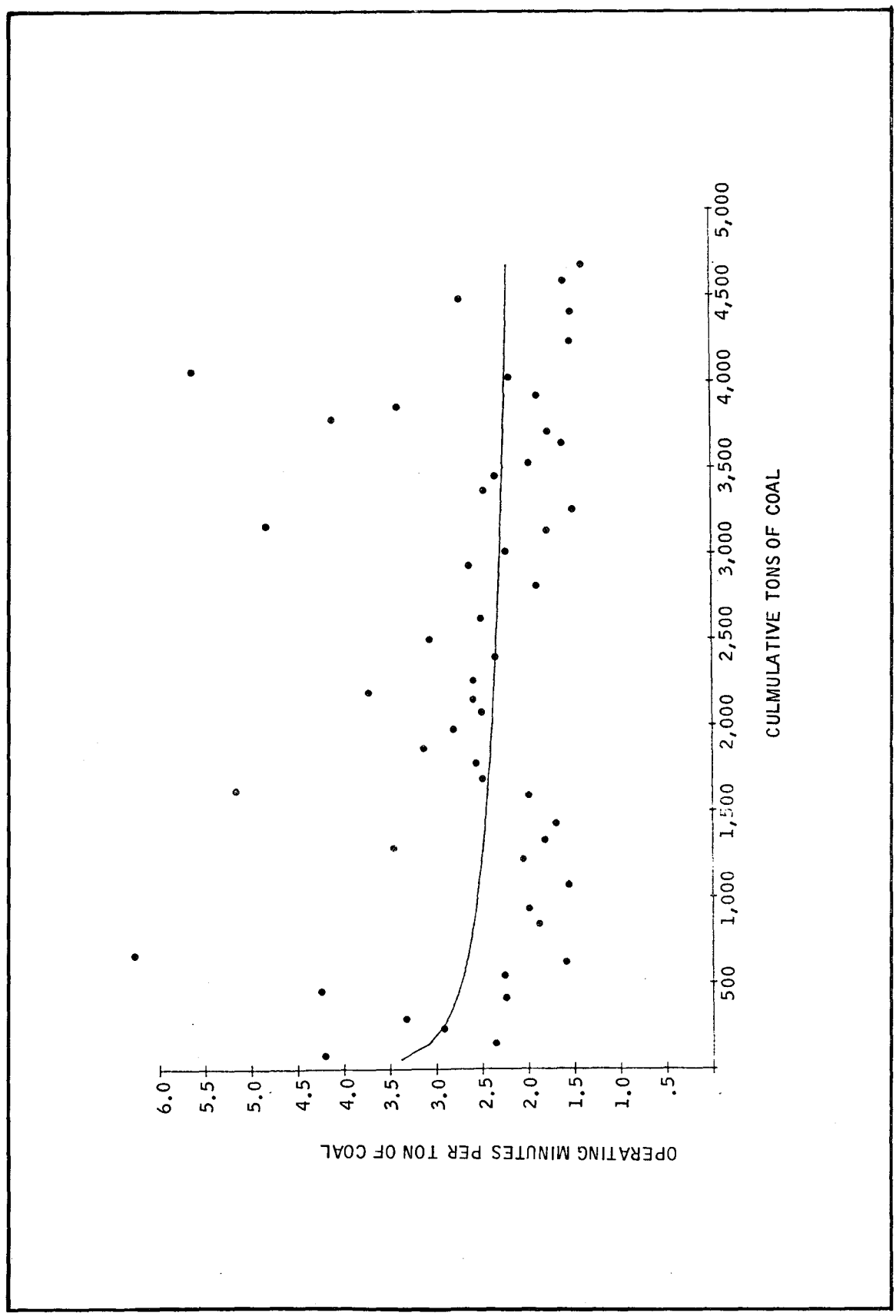

Figure M-3 PRODUCTION RATES AND LEARNING CURVE

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