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REPORT OF INVESTIGATIONS/1996

New Techniques for Mining Thin-Seam Mountaintop Coal Reserves

UNITED STATES DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES

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Rhea Lydia Graham, Director

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

cm^3/min	cubic centimeter per minute	min	minute
km	kilometer	mm	millimeter
kN	kilonewton	N	newton
kPa	kilopascal	$N \cdot m/cm^3$	newton meter per cubic centimeter
kW	kilowatt	s	second
m	meter	t	metric ton
m/min	meter per minute	Tt	trillion metric tons
m/s	meter per second		

NEW TECHNIQUES FOR MINING THIN-SEAM MOUNTAINTOP COAL RESERVES

By Joseph P. DuCarme,[1] Jasinder S. Jaspal,[2] and August J. Kwitowski[3]

ABSTRACT

The U.S. Bureau of Mines explored new methods and technology for extracting coal from the Appalachian Mountain region of the Eastern United States. These methods were aimed at extracting coal from thin seams and focused on the use of "roof-fall-tolerant" (RFT) mining concepts. These methods could be used on existing, abandoned highwalls or in conjunction with new highwall mining. The research was conducted using small-scale and quasi-full-scale physical models to obtain experimental results. Details are provided on (1) the quasi-full-scale model for cross-ridge mining, (2) the quasi-full-scale model for down-ridge mining, (3) the test results of these models, and (4) recommendations for future research. Significant results include demonstration of the RFT concept, a discussion of expected coal produced from these methods, and recommendations for future research.

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INTRODUCTION

The United States is estimated to have 10.5 Tt of coal resources. About 40% of this resource is located in seams less than 1 m thick, that often cannot be recovered economically. About 70% of the underground coal in the United States is produced from 1- to 4.5-m-thick seams. Thinner seams account for markedly less coal production, where 25% of U.S. underground coal is obtained from 700-mm to 1-m seams, and 5% from 400- to 700-mm seams (1).[4] The 30% ratio of coal production from seams less than 1 m is not consistent with the large resources that are available in thin seams. This shows that thin seam resources are not utilized to the extent of their availability (2). It also shows that a cost-competitive technology for extracting coal from thin seams is not available to the mining industry. In 1985, the U.S. Bureau of Mines (USBM) conducted a review of thin-seam coal mining. A conclusion was that the development of an attractive mining method to allow additional production from thin seams would benefit the export and domestic power generation market, as much of the thin-seam coal is of premium quality (2).

USBM estimates show that about 10% of the total thin seam resource is contained in seams between 350 to 700 mm thick; this resource is, for the most part, uneconomically recoverable with present conventional technology (*3*). Thin-seam coal deposits are particularly prevalent in the mountains of the Appalachian region of the Eastern United States. Many thin seams in this region have been partially recovered using highwall and auguring technology. However, much of the remaining reserve, along with over 40,225 km of unreclaimed highwalls, still exists in this region.

Of course, thin-seam coal extraction needs to be economically and environmentally attractive. A 1993 cost estimation, completed by the Washington Office of the USBM, determined an operating cost of $10.31 per short ton of clean coal. Because thin-seam extraction affects more surface area per unit of coal produced than thicker seam extraction, an acceptable thin-seam mining method should produce minimal surface and other environmental effects.

In 1987, the USBM began a project aimed at developing technology for a low-cost, high-recovery, environmentally acceptable mining system to recover thin-seam mountaintop reserves (*4*). Twelve different concepts were considered during the initial phase of the project. From these, a "roof-fall-tolerant" (RFT) method was chosen for development. This mining system concept does not require any roof support, nor are any personnel required underground. For RFT mining, a thin, flexible cutting string would be pulled through the coal seam, cutting and conveying the coal away. All equipment and personnel would remain in safe locations on the highwall benches.

CONCEPT DEVELOPMENT

An extensive search of foreign and U.S. literature and patents was made to obtain information on thin-seam extraction technology. Also, visits and contacts were made with many people in the mining industry. This information showed that thin seams were being mined in many countries. For instance, in Korea, a 75-mm wire rope and a 50-mm chain with attached cutters make up a flexible cutter string, which is used in a thin-seam mining system. This system is shown in figure 1. The coal is mined by drawing the cutter string across the 15-m-wide face in a reciprocating fashion. The cutter string extracts a 100- to 200-mm slot in 510-mm-thick seams. Because of the inclination of the seam, the cut coal and the coal remaining in the roof cave and fall to the lower roadway. From the lower level, the coal is hauled out. A similar system is also in use in Russia. Since these systems are used in sloped seams, the gravity-based coal transport feature would not work in the nearly level seams prevalent in the Appalachian Mountain regions. It did appear that some features of these systems (i.e., flexible string cutting, etc.) could be used in developing a mining system for the thin seams in the target region of the Appalachian mountaintops.

The main elements of underground coal mining are coal cutting, ground control, and haulage. Because the Appalachian mountaintop ridge configuration presents the possibility of mining from bench to bench, a longwall-type mining system seemed logical. However, the high cost and the physical size of the longwall shields were the main reasons for exploring alternative techniques for mining these thin seams.

Coal Cutting: Thin-seam shearers (as low as 760 mm) are available in the industry, but their use would preclude the mining of thinner seams most prevalent in the target region. Plows were found to be better for application to

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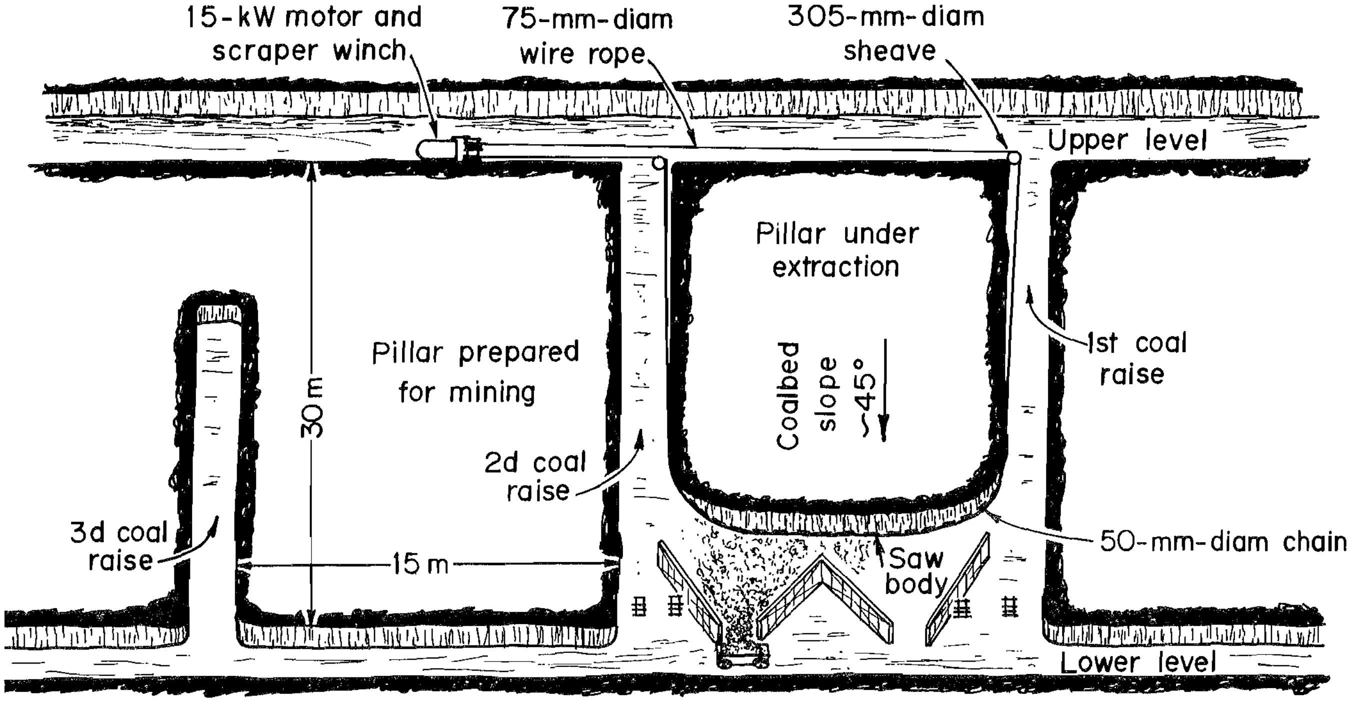
[4] Italic numbers in parentheses refer to items in the list of references at the end of this report.

Figure 1

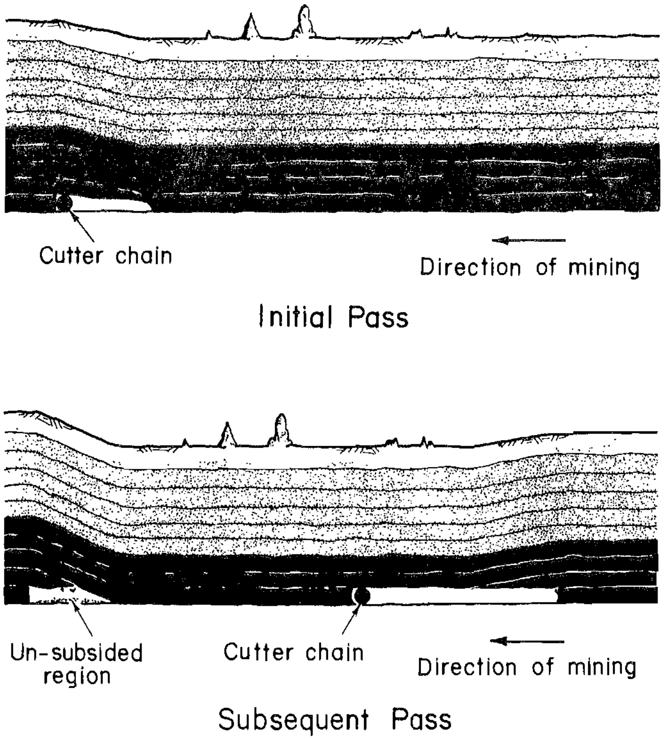


Korean mining system.

thin seams, but not ideal for use in thin-seam mountaintop reserves. Cable-drawn cutting systems like scraper boxes were also investigated. Flexible cutter strings appeared to be the most versatile and cost-effective extraction method for mining thin coal seams in the mountaintop ridges of Appalachia.

Ground Control: The success of a mining system depends largely on ground control. For thin seams, it is very difficult to find suitable roof supports to fit in the low seam heights. Additionally, roof-support systems like hydraulic jacks, backfilling, and air bags are expensive, and would make the thin-seam mining system less economically attractive. A technique to induce controlled caving, behind the cut face and without dedicated roof-support members, appeared as an attractive alternative to the costly mechanized roof-support systems. In this theoretical system, a thin section of the coal seam is removed by a flexible cutting string. As the mining advances, the rest of the coal seam outby subsides until it is supported by the floor. The collapsing roof leaves a small area open at the immediate face so that the cutter string can continue to operate. At some point, the mining process reverses, thus cutting another section. This process is repeated until the entire seam is mined. This concept was designated the RFT system. An artist's illustration of the RFT mining system is shown in figure 2.

Figure 2



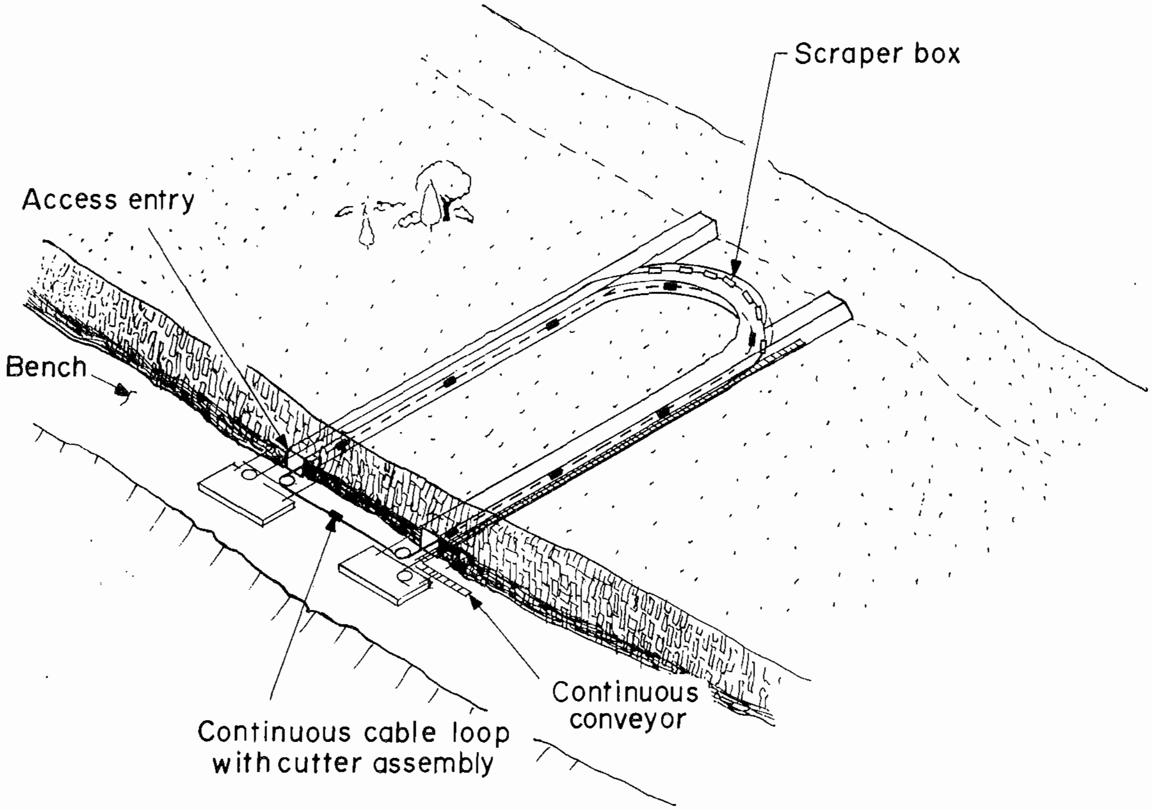
Initial and subsequent passes of RFT mining.

Haulage: Some version of a regular longwall armored face conveyor could be feasible, but it would cause problems if used with the caving technique described above. It would also increase the system cost. Thus, the research concluded that the flexible cutter string would have to both cut and transport the coal.

Two methods using the RFT concept were investigated. The cross-ridge mining method would require two parallel access holes from one bench through the mountain to the other bench. A drawing of this method is shown in figure 3. The distance between these access holes would dictate the face width. Cutting action would proceed from one bench to the other, hence the term "cross-ridge." Mined coal would be removed by conveyors installed in the parallel access holes. The entire seam would be mined in multiple passes. All equipment would be placed on the bench toward which the face would advance.

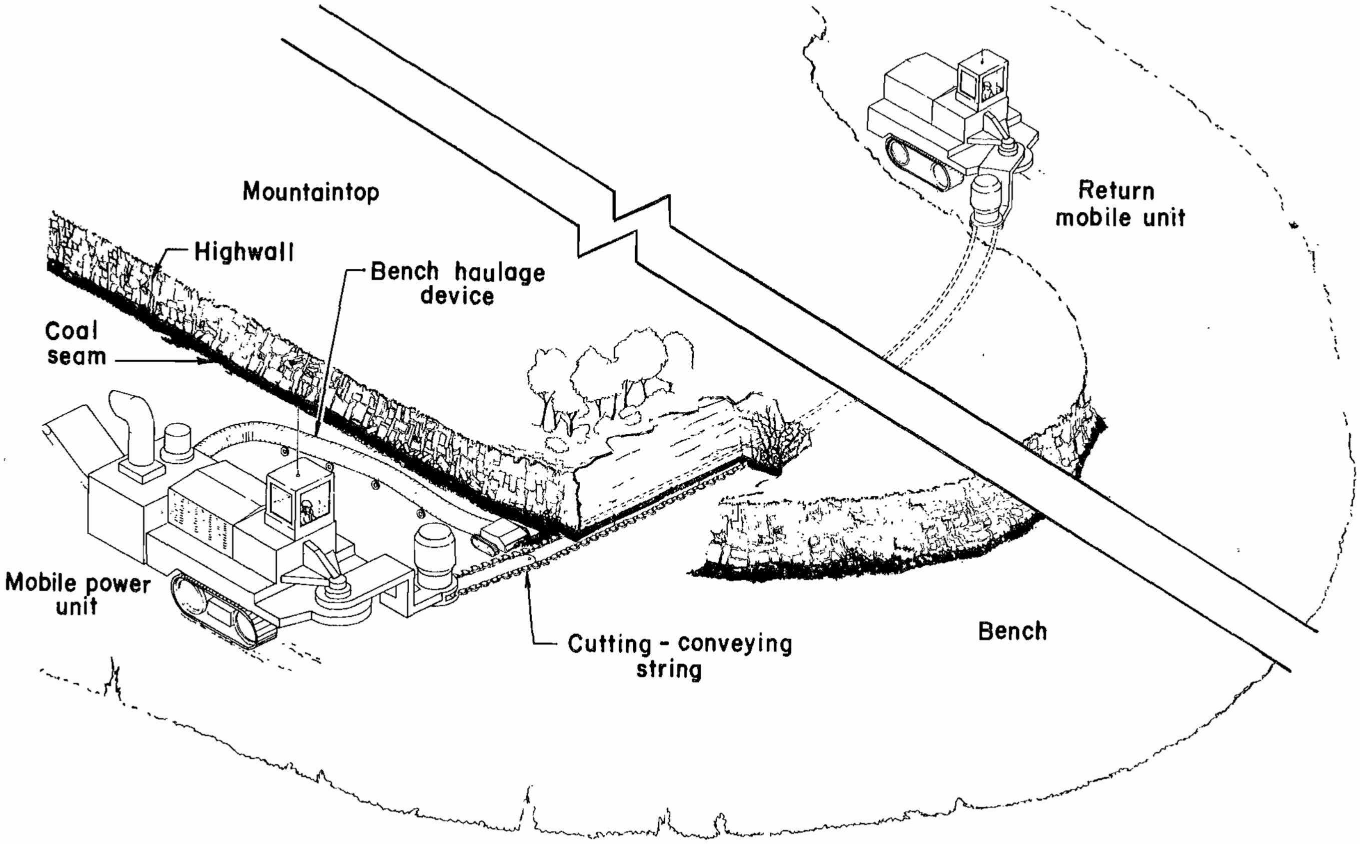
The other method investigated is referred to as "down-ridge" and is shown in figure 4. For this method, two machines with a loop of flexible cutter string carried between them proceed on opposing benches down the ridge, cutting a small slot as they advance. Because the cutter string is a continuous loop, the cutting action is continuous. To mine the entire coal seam, the machines simply reverse direction and cut into the subsided coal. This process is repeated as often as necessary to mine the entire seam.

Figure 3



Cross-ridge mining system.

Figure 4



Down-ridge mining system.

PHYSICAL MODEL TEST PROGRAM

Physical modeling was used to investigate the feasibility of the RFT mining concept. Previous work, where a small-scale simulation model was used to study overburden movement for a shallow longwall mining operation, had been conducted at the West Virginia University by Syd Peng.[5] Thus, modeling seemed an appropriate approach to evaluate the RFT hypothesis for the mountaintop mining project.

A variety of physical scale models were used by the USBM at its Pittsburgh Research Center (PRC) Mine Equipment Test Facility (METF) to test and verify the validity of the RFT concept. The first of these were a series of 1/16-scale models on the down-ridge mining method. The later versions of these models added an overburden simulation. All tests conducted with these models showed the RFT concept to be valid. The success with the 1/16-scale models led to the development of quasi-full-scale models. These quasi-full-scale models were actual size with respect to the cutter elements used; other dimensions were reduced to fit within the physical confines of the METF.

Researchers conducted an investigation into simulating overburden pressure on the quasi-full-scale models. The

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[5] Private communication from S. Peng, West Virginia University, Morgantown, WV, 1988.

study concluded that, because of the extremely high forces required, there was no economical way to simulate overburden pressure at PRC's METF. Only the weight of the artificial coal (coalcrete) block above the slot being cut served to cause "roof falls" during operation. Even without the overburden pressure simulation, both quasi-full-scale models exhibited RFT behavior as described in the following sections.

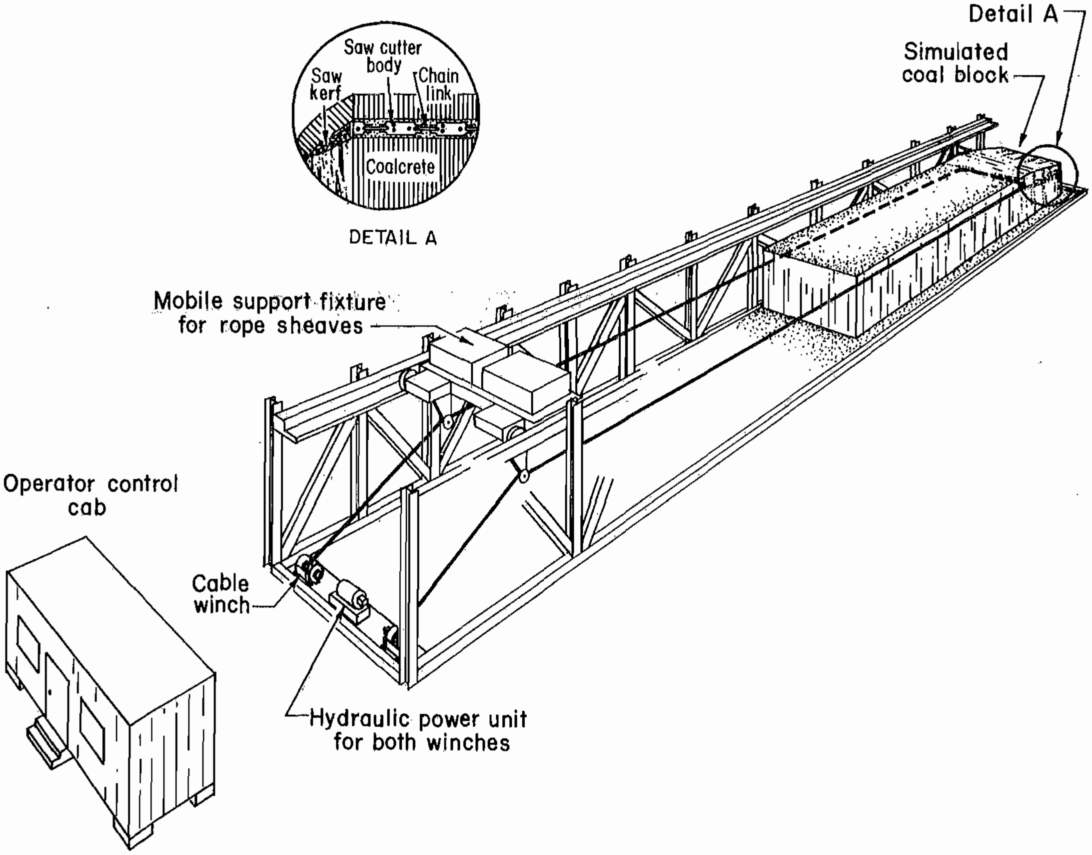
QUASI-FULL-SCALE MODEL FOR CROSS-RIDGE MINING

A quasi-full-scale test model was constructed at PRC's METF. This model was full size with respect to the cutting elements and was designed to emulate a cross-ridge mining system (5). A drawing of the model is shown in figure 5. The model was designed using data previously collected from tests conducted on the linear cutting apparatus (LCA). The LCA is a large machine designed for testing cutter bits in coal and other types of material. The LCA tests determined what forces prototype cutters required while cutting at various depths through coal (2).

Creation of a large block of coalcrete was the first step in constructing the quasi-full-scale model. The coalcrete material, developed by the USBM, is a mixture of lump coal, fly ash, and portland cement. Many of its physical properties closely match those of natural coal. Because of physical restrictions in the test facility, the coalcrete block was only 5 m in width. This is far short of the hundreds of meters width that is envisioned for an actual mining system. Also, the pulling capacity of the model was limited to 57.8 kN, which is sufficient for the 5-m-wide face of the model, but is much less than what would actually be required for a complete mining system.

The model was instrumented and equipped with a personal computer-based (PC) data acquisition system. This system consisted of an 80386-based, IBM-compatible PC with a data translation model 2801-A data acquisition card. The software used was the Global Lab data acquisition package from Data Translation. The hydraulic power unit was equipped with pressure transducers and a wire rope speed sensor was designed using a magnetic pickup. These sensors enabled the data acquisition system to record the model's winch pressures and speeds during

Figure 5



Quasi-full-scale model for cross-ridge mining.

testing. The data gathered were analyzed and graphed using a PC spreadsheet program. A linear relationship between the hydraulic pressure at the winch and the amount of pull on the cable was established during commissioning.

Two closed-circuit television (CCTV) cameras were positioned at the rear of the block, where the cutting action took place, and two monitors were placed at the operator's console. The cameras were connected to video cassette recorders (VCR's) so that the tests could be recorded.

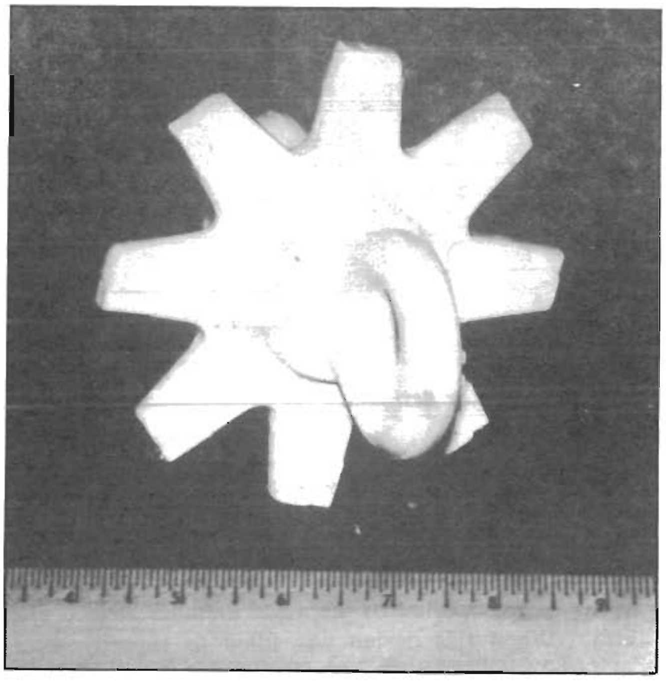
Early investigations determined that an entirely new type of coal cutter element would be required. Several innovative cutter elements were designed and fabricated. Some of these used standard conical carbide-tipped bits; others used carbide-tipped cutters from road-paving equipment; and one used a carbide-tipped bit developed by the USBM's Twin Cities Research Center. All of these are described in detail as follows.

The testing program for each type of cutter consisted of two parts. The first part was aimed at establishing a baseline for the cutter being tested. For each different cutter design, this baseline was established by installing a single cutter on the cutter string. A 30-s test was conducted with data on the cutting force being acquired at a rate of 100 samples per second. The duration was sufficient to allow four passes of the cutter across the face of the block. The second part consisted of a 5-min test with the sampling rate set to 10 samples per second. The total face advance was measured during this test. With this information, a specific energy requirement (work required per volume removed) for each cutter tested was calculated. Average winch pressure was determined from analysis of the acquired data from the first part of the test. The hydraulic power unit was designed to supply constant flow to the winches until the upper limit on line pull was met. This design provided for a constant line speed independent of line pull. For these tests, a line speed of 91.4 m/s was used. Since the relationship between hydraulic pressure and line pull is linear, multiplying the pressure value by the conversion factor determined during model commissioning gave an average line pull value. The total volume of coalcrete cut was determined by measuring the face advance and the slot height. This was converted to a rate by dividing by the duration of the test. Multiplying the average required force by the cutter string speed and then dividing by the rate of the volume cut gave the specific energy requirements for each cutter. Additional cutters were then added to the cutter string, one at a time, and the testing program was repeated until the required pull exceeded the models' capacity or the number of cutters was sufficient to span the entire face width. This testing program was designed to allow fair comparisons to be made between the different cutter designs.

Initial testing was conducted with a cutter string consisting of 115-mm-diameter star bits previously tested on the LCA. This cutter bit is shown in figure 6. It became apparent that, in the softer sections of the coalcrete block, the star cutters took a large depth of cut. Since there was no way to control the crowd force, a means was needed to limit physically the star cutter's depth of cut. This was done by welding a short length of tubing (shroud) to the cutters, as shown in figure 7. The diameter of the tubing was 13 mm smaller than the cutter diameter. This allowed a maximum of 6.5-mm depth of cut. This shroud created a cavity that increased the conveying action of the cutter string.

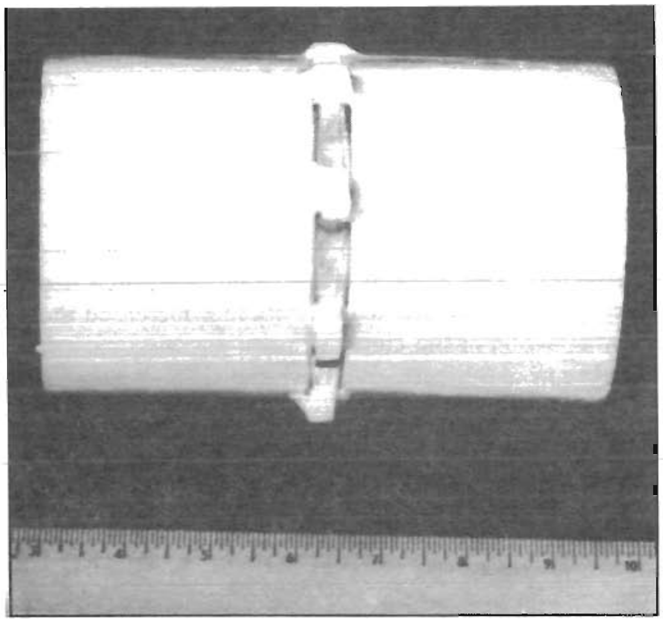
The need for increased conveying action became apparent as the cutting tests progressed. The cavities created by the shrouds installed on the star bits did not have sufficient volumetric capacity to convey all of the cut material from the slot. Significant amounts of material were left behind as the cutter string advanced. Several designs of conveyor flights and scrapers were installed and tested in an attempt to clear the slot of all the cut material. Most of these designs consisted of a scoop or flight bar attached to a hardened shoe that slid along the face. Some of the designs tested are shown in figure 8. Where possible, the conveyor elements were incorporated into the design of the bit holders (see figures 12 and 13). Observations showed that with the most efficient of these designs, which was the one fabricated from a flight bar conveyor, about

Figure 6



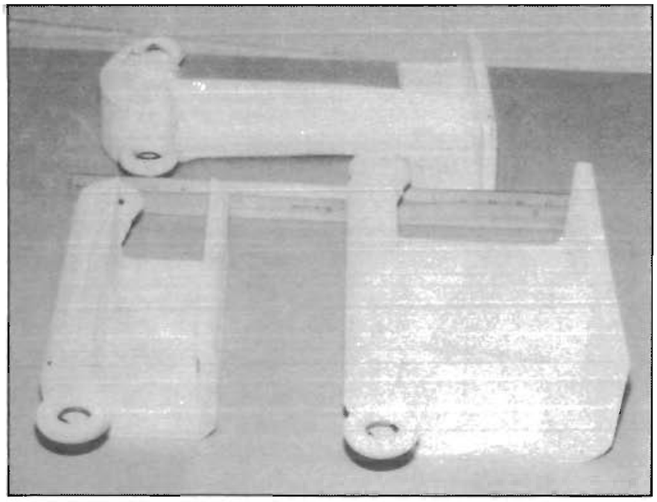
Star cutter (115 mm diameter).

Figure 7



Shrouded star cutter (115 mm diameter).

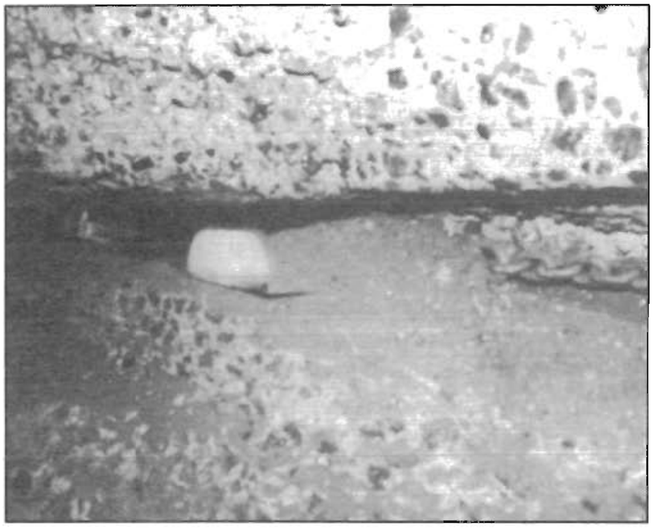
Figure 8



Various conveyor designs.

90% of the cut material was conveyed out of the slot. When the other scoop designs filled to capacity, because of the curvature of the face, the remaining coal was swept outby. This action was prevented in the flight bar conveyor design because of the angle member welded to its end. When this design was filled to capacity, excess material would flow over the flight bar and be gathered by the next conveyor element. Figure 9 shows this action as the conveyor element is being pulled from the slot in the coalcrete block.

Figure 9



Flight bar conveyor at work.

An important fact about the conveying action in this mining system is that every conveyor design has a finite capacity. The cutting elements take nearly a constant depth of cut across the entire face width. Therefore, cutting elements generate an essentially fixed volume of material for each unit of face width. This means that a certain number of conveyor elements are required to carry the material generated by each cutter element. Thus, successful conveying depends upon balancing the conveyor capacity to the number of cutter elements and the face width.

In all, approximately 25 different cutter string configurations were tested for the cross-ridge mining configuration. Table 1 shows specific energy requirements calculated for the various cutter designs. The cutter designs are listed in order of increasing specific energy requirements. The dual star cutter, shown in figure 10, was designed to eliminate the problem of "tracking" that was observed to occur with the single star cutters. As the tests on the single star cutters progressed, observations of the face revealed that each cutter was following in the grooves cut by the preceding cutter. This led to decreased cutting efficiency because of increased friction. The dual star cutter eliminated this problem by fixing an offset between the teeth on two adjacent cutters. The Kier cutter, shown in figure 11, is a design similar to one in use in the Korean slot mining system. In the Korean mining system, gravity provides the conveying action because of the seam inclination. In our model, as in most mountaintop seams in the United States, the coal seam is level, thus the need for conveying elements on the cutter string. The box cutter, shown in figure 12, is a design created by project personnel using standard conical bits. A flight bar, acting as a conveying device, is welded onto the bit holder.

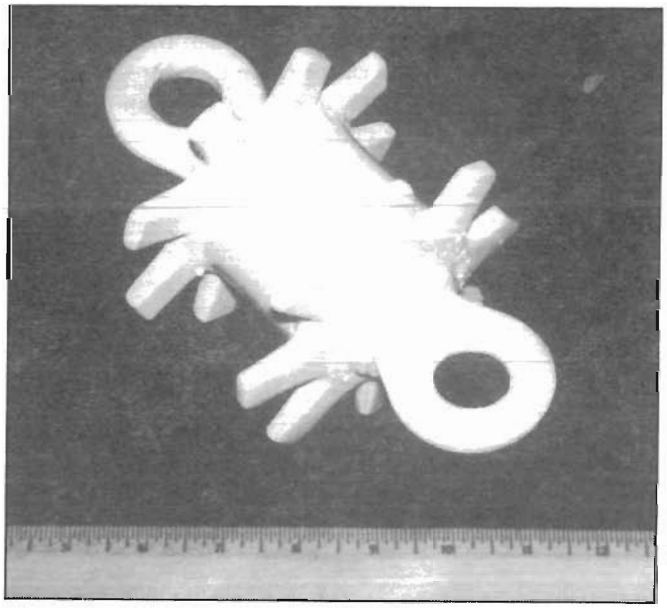
Table 1.—Test results for cross-ridge mining

Cutter	Average force, N	Volume cut, cm^3/min	Specific energy, N•m/cm^3
Concave	45,800	41,100	102
Box	35,230	20,895	154
Kier	38,750	18,300	193
Dual star ...	25,540	11,210	208

Research conducted at the USBM's Twin Cities Research Center developed an alternate design to the standard conical carbide-tipped bits commonly used on underground mining equipment (*6*). This bit has a 10° concave head, 57 mm in diameter. By using the information provided in the paper, a similar design was created and a small number of these carbide-tipped cutters bits were fabricated by an outside vendor. A bit holder was developed that incorporates a flight bar for conveying the cut coal from the slot. Figure 13 shows this completed cutter, which proved to be the most efficient design tested.

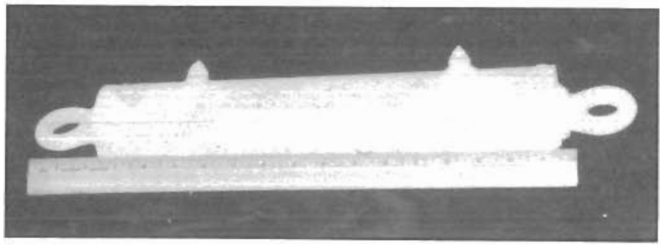
As for roof control, the cross-ridge mining model exhibited RFT behavior throughout the testing. The first tests initiated the slot at the rear of the coalcrete block, about halfway up the block. The slot progressed about 2 m

Figure 10



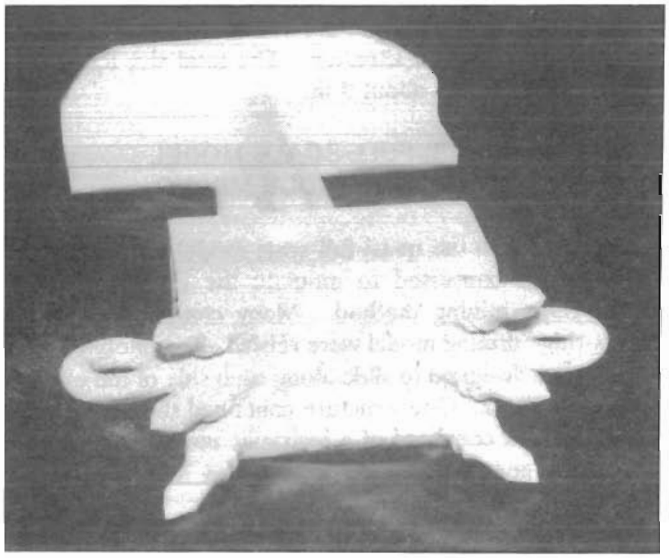
Dual star cutter (115 mm diameter).

Figure 11



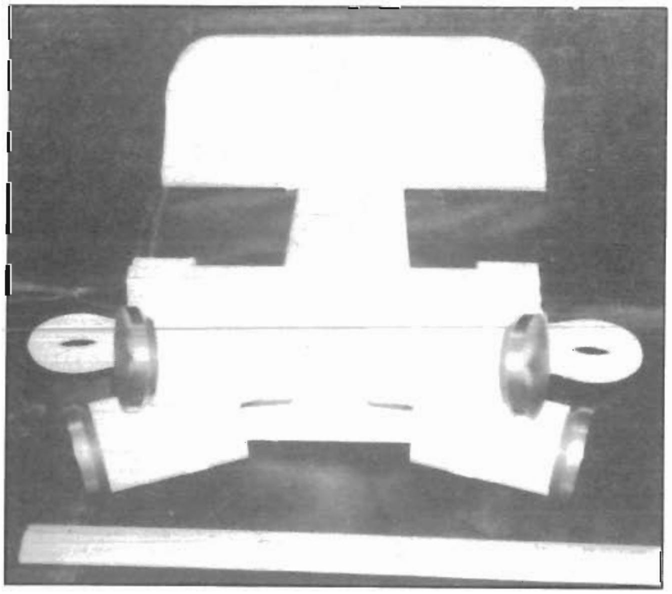
Kier-style cutter (115 mm diameter, 390 mm long).

Figure 12



Box cutter with conveyor flight (115 mm high).

Figure 13



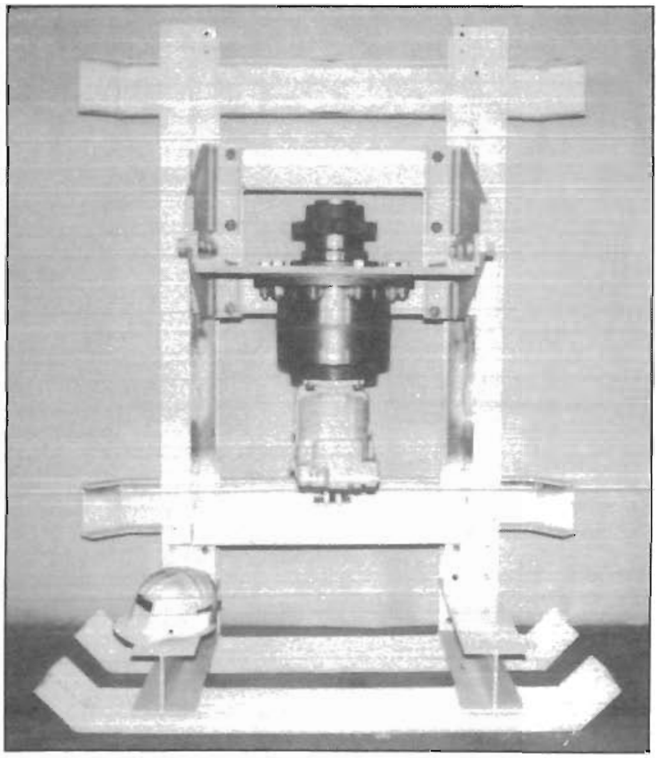
Concave bit cutter with conveyor flight. Bits are 57 mm diameter.

into the block before the first roof failure occurred. The fracture line was slightly outby the cutter chain when failure occurred. Examination showed that the coalcrete was of particularly poor quality in the fracture region. No additional sudden failures occurred during the entire remainder of testing. However, as the slot progressed into the coalcrete block, the cantilevered section of the roof outby had gradually settled down and became supported on the floor of the slot. This closed off the slot several meters outby, but a small wedge-shaped open area was left adjacent to the immediate face. The thickness of the coalcrete above the slot was about 1 m and thus simulated a substantial solid roof situation. The total slot length cut during all tests was about 9 m.

QUASI-FULL-SCALE MODEL FOR DOWN-RIDGE MINING

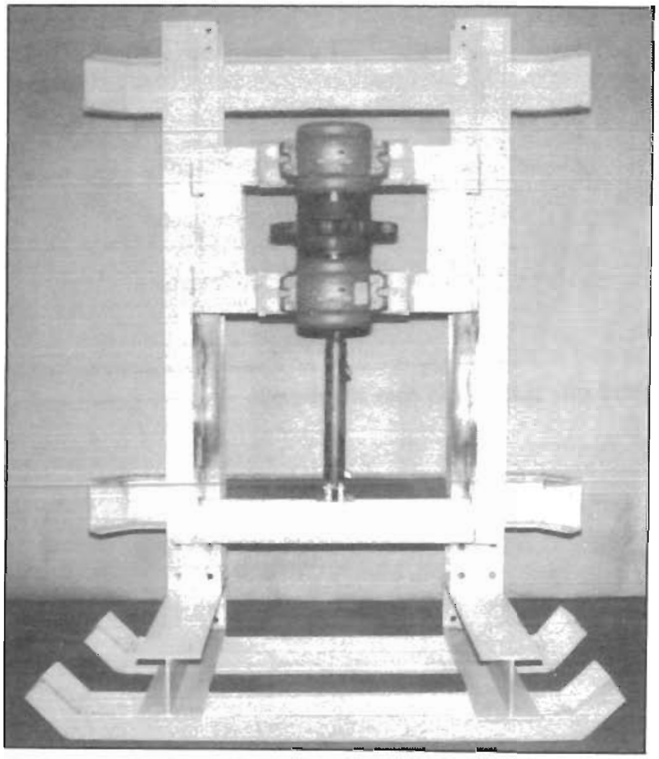
In late 1992, the quasi-full-scale model for cross-ridge mining was converted to emulate the continuous-loop, down-ridge mining method. Many components of the cross-ridge mining model were reused. Two mobile structures were designed to slide along each side of the existing coalcrete block. One structure contained the drive mechanism, which consisted of a hydraulic motor coupled to a planetary gear drive. A sprocket was mounted on the output shaft of the gear drive and a similar sprocket was mounted on the other mobile structure. These structures are shown in figures 14 and 15. A continuous loop of 78.1-mm pitch, rollerless engineering chain was used for the flexible cutting string. This type of chain is commonly used on shuttle car conveyors and is available with many different styles of sideplates for different attachments. For cutting bits, a 19-mm shank by 85-mm-long conical bit was chosen. Bit holders were fabricated from mating bit blocks welded to steelplate bolted to the cutter chain. This bit set is shown in figure 16. The loop of chain was 10.67 m long with a single cutter bit installed every 305 mm. There were seven complete sets of five individual bits installed on the cutter chain loop. The offsets in this set of bits was designed to cut a 152-mm-wide slot. The hydraulic power unit, used in the cross-ridge mining model, was used to power the cutter chain drive. The hydraulic winches were used to provide the required crowd force on the mobile structures and were powered by a small auxiliary hydraulic unit. A drawing of the complete model is shown in figure 17. The PC-based data acquisition system was reused in the cross-ridge mining model. Pressure transducers were installed so that the data acquisition system could monitor and record the winch and cutter chain drive pressures. A magnetic pickup was installed on the drive sprocket to monitor the speed of the cutter chain.

Figure 14



Drive side mobile structure.

Figure 15

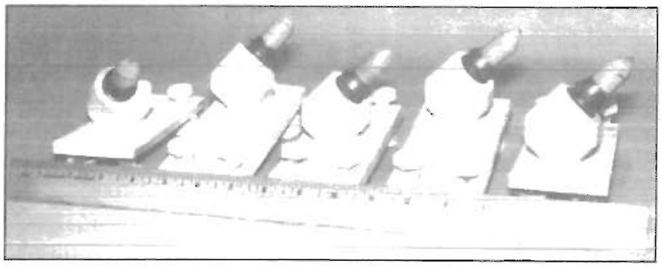


Idler side mobile structure.

The operator panel and controls for down-ridge mining model are shown in figure 18. This panel was ergonomically designed for operator comfort and ease of use. The gauges on this panel provided the operator with information on chain speed, winch pressure, and drive pressure. The motor controls, along with the emergency stop button, were grouped in the lower left-hand corner of the panel. The lever on the right-hand side controlled the speed and direction of the cutter chain drive. The monitors and VCR's for the CCTV system were located on the top of the panel enclosure, in the operator's line of sight.

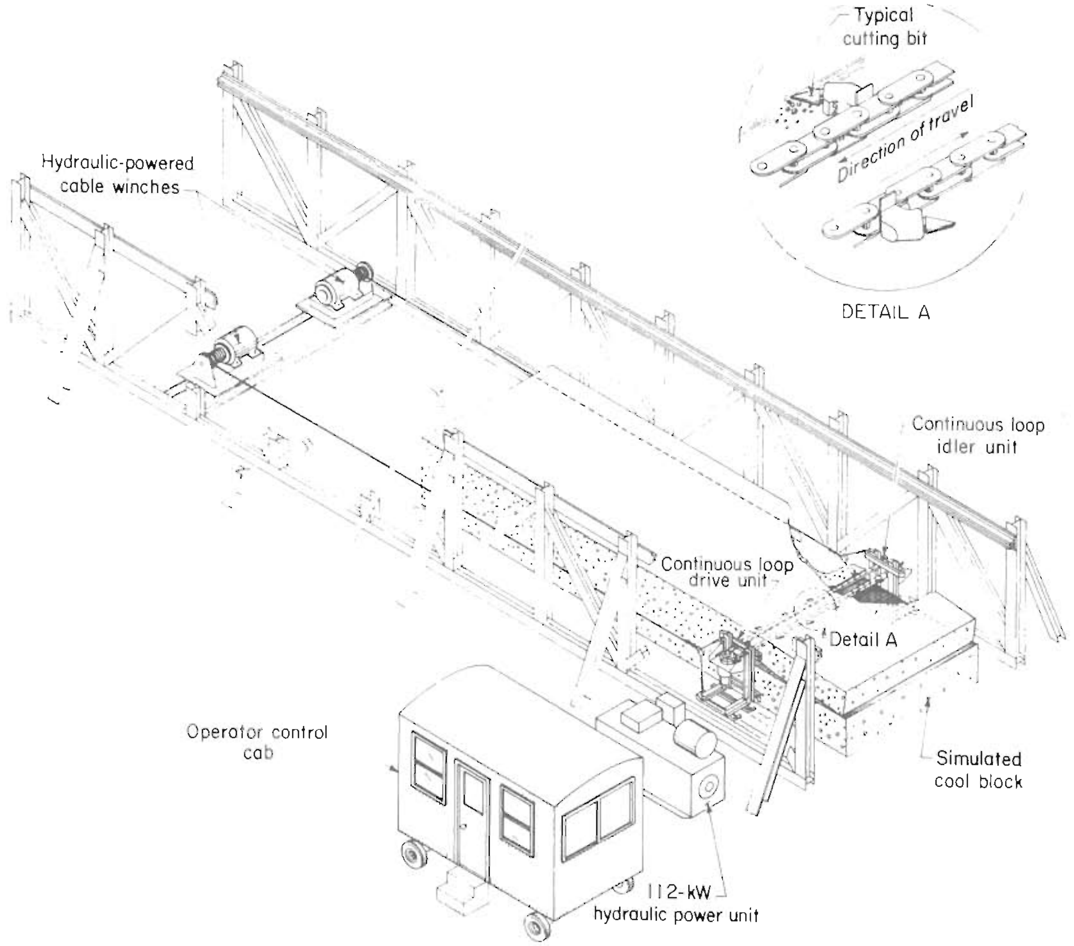
During the initial operation of this quasi-full-scale model, temporary supports were provided for the mobile drive and idler structures. These supports were maintained until a slot of sufficient depth was cut to allow the units to become self-supporting against the sides of the coalcrete block. The geometry of the model provided an easy means of developing tension in the cutter chain: Advancing the drive unit while leaving the idler unit stationary increased the center-to-center length between the sprockets, because the block width was constant. Thus, the cut slot was not at right angles to the sides of the block, but was slightly diagonal across the block. Figure 19 shows the drive and idler structures at the beginning of the cutting operations.

Figure 16



Cutter bit set for down-ridge mining model. Bits are 19 mm diameter by 85 mm long.

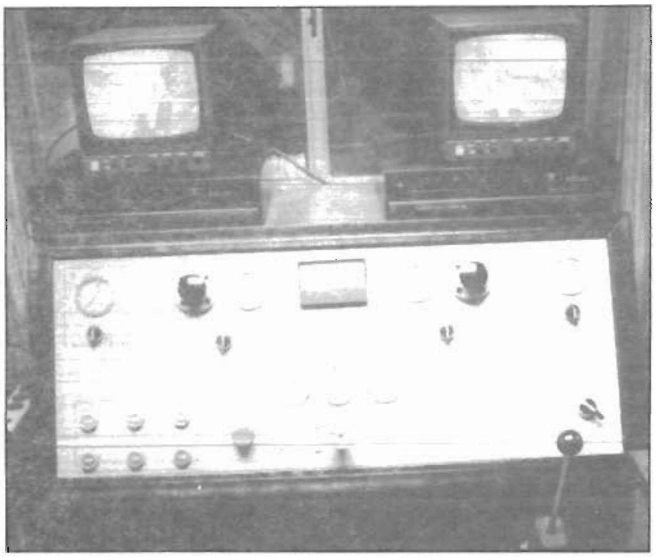
Figure 17



Quasi-full-scale model for down-ridge mining.

The testing program consisted of 30-min tests with the data acquisition system monitoring and recording all operating parameters. Initial tests were conducted with the operator attempting to control the advance of both the drive and idler units. This proved to be a difficult task since it was impossible to maintain the proper chain tension. Thus, a different procedure was developed because operators on opposing benches of a highwall would never coordinate their efforts to achieve equal advances. This procedure, which proved to be successful, involved maintaining a fixed tension on the drive side unit and intermittently advancing the idler side structure. Starting with some slack in the loop of cutter chain, a fixed tension was applied to the drive side structure. As this side advanced, the slot was cut into the block and, since the idler side unit was not being moved, the slack was taken out of the cutter chain. At some point, the drive side unit ceased advancing because of tension building up in the chain. This was seen on the control panel as a decrease in required drive power. Visual observations from the CCTV system also showed that cutting had ceased. A small amount of tension was applied to the idler side and caused it to advance, adding slack to the cutter string. The application of tension to the idler side was stopped when about 200 mm of advance had occurred. This small advance maintained sufficient tension to prevent the return strand from contacting the cutting strand. Since the drive side was under constant tension, it began to advance and cut when slack was introduced onto the cutter chain. This cycle was repeated as required during the test and total advance was recorded. Numerous tests were run with crowd forces ranging up to the winch's maximum capacity of 77.1 kN.

Figure 18



Operator's panel for down-ridge mining model.

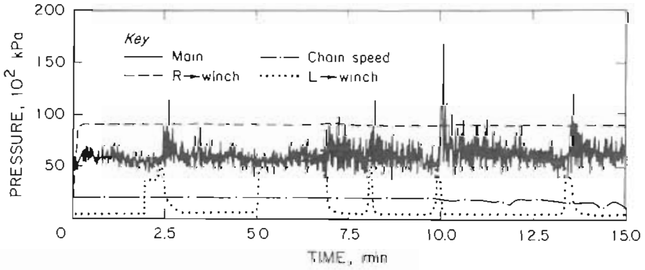
Figure 19

Beginning cutting on down-ridge mining model.

Figure 20 shows the raw data collected during a typical test. The graph shows the constant pressure supplied to the right winch. This caused a constant crowd force to be applied to the drive mobile structure. The graph also shows the intermittent pressure applied by the operator to the left winch. This caused an intermittent force to be applied to advance the idler mobile structure. The drive motor curve shows the pressure required to maintain constant chain speed at the cutter chain drive. This pressure measurement can be translated into the required torque at the drive sprocket. Because the chain speed was constant at 91.4 m/min, this torque can be further converted to kilowatt units.

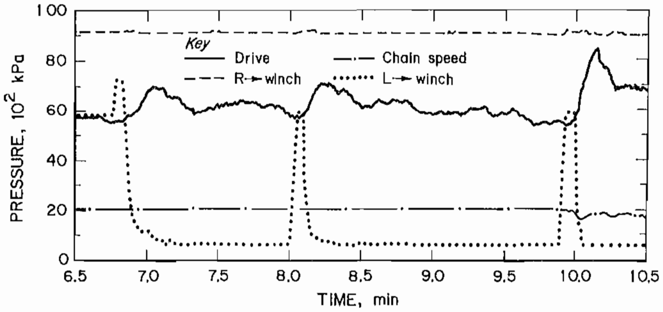
By processing the data on the required torque curve with a moving average function, a graph was generated that is much easier to comprehend. This is shown in figure 21. During this test, a constant pressure of 9,000 kPa was applied to the drive side winch. This created a crowding force of 50 kN on the drive mobile structure. As seen in the figure, a short interval of pressure was periodically applied to the idler winch, causing the idler

Figure 20



Raw data from down-ridge mining test.

Figure 21



Processed data from down-ridge mining test.

mobile structure to advance. The operation of the model continued as described above.

Referring to figure 21, there is an immediate increase in required torque after each advance of the idler mobile structure. Naturally, this is because the cutter string was being pulled by the drive side, cutting into the face. Then the curve gradually falls off to a minimum value, around 5,170 kPa. Visual observations, with the CCTV system, also showed cutting action slowed as the drive advanced, taking the slack out of the cutter string. During this test, the average cutter string force recorded was about 18.2 kN; this translates to 27.8 kW of required power.

As stated before, various tests were run with different constant forces being applied to the drive side mobile structure. The results of these tests showed that a minimum force of 38.5 kN was needed to cause efficient cutting. If about 55 kN or more was applied, the drive side advanced rapidly when slack was introduced by advancing the idler side, and massive torque peaks and shock loadings occurred at the cutter chain drive. Some of this action was caused by large portions of the face cutting loose and being jammed into the slot by the conveying action of the cutter chain. Therefore, in an actual mining system, crowding force would need to be limited to avoid damage and excessive wear on the equipment. Table 2 summarizes various data on these tests. Overall, the tests conducted on the quasi-full-scale model for down-ridge mining showed very promising results.

Table 2.—Test results for down-ridge mining

Test	Crowd force, N	Average power, kW	Peak power, kW	Volume rate, cm^3/min
1	28,900	25.2	44.5	5,030
2	38,545	25.2	53.4	11,323
3	48,185	29.7	74.2	25,531
4	57,825	37.1	89.0	15,420

Several items concerning roof behavior were noted during the testing program. Roof falls occurred at regular intervals of between 1.8 and 2.4 m of advance. Each fall that occurred behaved in exactly the same manner: The section outby fell and was supported by the floor. Fracturing occurred between 50 and 127 mm outby the leading section of the cutter string. The fallen section self-supported at the fracture because of compressive forces. This action left the immediate face area open, allowing the cutter string to continue operating. This behavior, which has been noted in other small-scale and quasi-full-scale models, supports and validates the RFT concept, which is central to these new mining methods. Other researchers at PRC note that this behavior is observed on longwall faces, except at shallow depths.

The cutter string exhibited very efficient conveying action without any auxiliary scrapers or conveyors being installed. Examination of the slot after each test revealed that nearly all of the cut material had been removed from the slot. The majority of the material was moved to the drive side; a small amount was conveyed by the return strand to the idler side. Investigations on larger face widths will be required to determine if some type of conveyor element is needed on the cutter string for field deployment of the down-ridge mining method.

It was originally anticipated that different bit configurations would be installed on the cutter string for the quasi-full-scale model for down-ridge mining. A particular desire was to test the concave bit, which was so successful on the cross-ridge mining model. Unfortunately, the prototype concave bits were very expensive and the continuous loop of cutter chain on the down-ridge mining model required many more cutters than did the cutter string for cross-ridge mining. Budget constraints forced the testing of different bit configurations to be curtailed and ultimately caused the project to end prematurely.

EXPECTED COAL PRODUCT

Concerns were raised about the generation of coal fines from these mining methods. From the onset of the project, it was realized that the crowd force generated by the mining equipment would naturally be distributed over the entire face. Conversely, in traditional mining the crowd force is applied to a small area; i.e., the face of the cutting drum. With only a small crowd force available per unit area, it is obvious that the production from these methods will be finer in nature than what is produced by traditional mining methods. Also, more fines will be produced because of the narrow slot height being cut and the conveying distance and method causing a grinding action to occur. Because the coal found in the mountaintop settings is generally of high British thermal units per pound values and low sulfur content, it should be a marketable product even if it contains a large percentage of fines. Discussions with representatives of local coal-fired powerplants revealed that the amount of fines acceptable is greatly dependent upon the environment surrounding the plant, with more rural-located plants being able to tolerate a higher percentage of fines in the coal they purchase. The technical problems associated with coal fines can be overcome by blending at preparation plants and utilizing different handling methods.

Core samples taken from the coalcrete block showed that the compressive strength is a good match for coal, about 6,200 kPa on average. Physical examination showed the block consisted of lump coal, about 25 mm and smaller, with stratification layers held in random orientations by the surrounding cement mix. Figure 22 shows two core samples taken from the coalcrete block. This construction explains, in part, the reason for the generation of the large percentage of fines produced during cutting operations on the quasi-full-scale models. Since the coal is held in place by the surrounding cement, it can only be ground away by the passing cutters. Without consistent orientation of the stratification layers of the coal, there exists no tendency for the coal to shear in large pieces, as would in-situ coal. At present, there is no way to confirm how an actual coal seam would behave, with respect to fines generation, when subject to these mining methods. It is assumed that the amount of fines generated by an actual mining system would be less than what was generated from the quasi-full-scale model. This was influenced by factors that include that during successive passes, the coal would be fractured because of subsidence and that the coal would tend to break out of the face by the cutting action, as opposed to being held in place by surrounding cement.

Figure 22

Core samples from full-scale model.

RESEARCH FINDINGS AND CONCLUSIONS

CROSS-RIDGE MINING METHOD

This research determined that an inherent problem with the cross-ridge mining system is that the crowd force applied to the cutter bits cannot be actively controlled. The relationship between the crowd force and cutting force is dependent on the radius of curvature of the face. This radius is the one the cutter string naturally makes as it traverses the face, from one entry to the other. Obviously, face width determines the radius generated. From the results of our models, it appears that the radius tends to be about 60% of the face width. As long as the material being cut is of uniform hardness across the entire width of the face, the radius remains constant. The smaller the radius, the larger the crowd force becomes in proportion to the cutting force. Because of this, the radius becomes self-correcting if a small section of the face width is of harder material than the rest. The softer material is cut away first, leaving a much smaller radius at the harder section. This causes increased crowd force at the harder material and subsequently causes it to be cut away at a faster rate. The depth of cut is dependent on crowd force, material hardness, and bit design. If the material is soft enough and the crowd force large enough, this system acts like a self-energizing brake. Aggressive cutter bits dig into soft material; this requires more cutting force to be applied. The larger required cutting force generates more crowding force. This causes the depth of cut to increase and again forces the required cutting force to increase. At some point, the required cutting force exceeds the available cutting force and stalling occurs. Because of the geometry of this method, the crowd force cannot be directly controlled. Therefore, the only solution to this problem is to limit the depth of cut or increase the face width. Since the quasi-full-scale model was constructed with a relatively small face width, this problem was magnified. To determine the extent of this problem, investigations into the behavior of this mining method should be conducted using larger face widths. In an actual mining system, the face width would be much larger, on the order of hundreds of meters, and this problem would most likely be less severe. For this system to be successfully implemented, the cutter bit's characteristics, coal seam hardness, and face width (and therefore the face radius) would have to be carefully balanced. The information and data obtained from the quasi-full-scale model cross-ridge mining will be of great value in defining the parameters of a field prototype system.

Additional technical problems with this mining method include maintaining the cross-ridge mining entries, removing the coal from them, and resetting the cutter string for multiple passes. While these problems do not appear insurmountable, no investigations to date have been conducted to find viable and cost-effective solutions.

DOWN-RIDGE MINING METHOD

Although the down-ridge mining method appears to require a larger capital equipment cost than would the cross-ridge mining method, these investigations indicate that the down-ridge mining method is probably superior. The down-ridge mining method promises to have a higher production rate for each unit of power consumed. Additionally, control over the mining process in the down-ridge mining method is greatly improved because of the separate control of cutter and crowd forces.

ROOF-FALL-TOLERANT MINING

The RFT concept was demonstrated during operations on both small-scale and quasi-full-scale models. This behavior was noted on the quasi-full-scale models even though these models lacked any overburden pressure simulation. Only the weight of the coalcrete block above the slot served to demonstrate this behavior. The cross-ridge mining model exhibited a semiplastic, gradual mode of roof collapse while the down-ridge mining model showed a more sudden, repeatable mode of roof failure. These differences should not be attributed to the method of mining, but to the thickness of the immediate roof strata above the slot. In the cross-ridge mining model, the depth of coalcrete above the slot being cut approached 1 m. This simulated a rather substantial and solid immediate roof strata. The slot cut with the down-ridge mining model was near the top of the coalcrete block, thus simulating a roof strata thickness of 300 mm. In either case, cutting operations were unaffected by roof collapse.

RECOMMENDATIONS FOR FUTURE RESEARCH

The new mining methods that were developed offer a great potential for extracting the vast thin-seam reserves that exist in the Appalachian Mountains. Although the validation of these methods has not progressed beyond quasi-full-scale modeling, the tests conducted on the down-ridge mining method have shown very encouraging results for the RFT concept. Additionally, all the modeling results have supported the viability of the RFT mining method.

It is recommended that future research on this subject be directed at further development of the down-ridge mining method. The down-ridge mining method appears to be a more efficient method, than the cross-ridge mining method, in terms of power consumption. Implementation of this method would require a larger capital equipment cost than implementation of a cross-ridge mining system. However, cost-benefit analysis completed during the project shows the costs to be well within reason. Future investigations with models are recommended and should be conducted with larger face widths and with overburden pressures.

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