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# Sublevel Caving By Pillar Extraction

Final Report
Contractor— R.J. Bowen Mining Engineering Consultant

May 1977

MASTER

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## U. S. Department of Energy
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### Division of Fossil Fuel Extraction
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# SUBLEVEL CAVING BY PILLAR EXTRACTION

Prepared for

UNITED STATES DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

by

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MASTER



Final Report
on
Contract No. J0265009
Sublevel Caving by Pillar Extraction

*Design and Feasibility Study*
*of a Method for Underground Mining*
*of*
*Thick Seam Western Coal*

May 25, 1977

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16. Abstract This report describes the design and feasibility analysis of the Sublevel Caving by Pillar Extraction mining method which is a conceptual mining method devised toward the objective of improving mining techniques for extraction of deep coal occurring in seams of thicknesses greater than 12 feet.

The concept envisions that advance openings of the mine will be driven along the bottom of the thick seam with top coal being extracted incrementally during retreat mining, or in "falls", shot down from the roof. Loading of fallen top coal will be accomplished with the loading machine operator remaining in a protected position.

The conclusions of the study are that the Sublevel Caving by Pillar Extraction mining method is suitable for test application in a working coal mine in the immediate future: that it offers improvement in resource recovery, competitive productivity, and improvement in health and safety for mine personnel over previous thick seam underground methods. Analysis of the economic performance indicates a price of $13.22 per ton to sustain 15% return on investment for 20 years.

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## - FOREWORD -

This report was prepared by R.J. Bowen, consulting mining engineer, of 1879 Delann Lane, Salt Lake City, Utah 84121, under USBM Contract J0265009. The contract was initiated under the Advancing Mining Technology/Coal Program. It was administered under the technical direction of DMRC with Mr. Richard Oitto acting as the technical project officer. Mr. William Case was the contract administrator for the Bureau of Mines.

This report is a summary of the work recently completed as part of this contract during the period June 1976 through May 1977. This report was submitted by the authors on May 25, 1977.


## ACKNOWLEDGEMENTS

The authors wish to acknowledge the very helpful guidance of the Bureau of Mines technical and administrative personnel at the Denver Mining Research Center throughout the duration of Contract No. J0265009 and for suggestions they have made toward the refinement of the "Sublevel Caving by Pillar Extraction Mining Method". For example, the suggestion to utilize angled blast holes was theirs. This resulted in a significant improvement of the mining concept.

Acknowledgement is also extended to Mr. John Peparakis, consulting mining engineer, of Salt Lake City, Utah, with whom discussions were held at the very early stages of conceptualization of this mining method and to whom credit is due. The expert assistance of project affiliates, specifically, Mr. Lars Olavsen, Dr. William Hustrulid, Mr. Russell Ramey, and Mr. Bernard Ludwig, is also greatly appreciated as are the many helpful comments and suggestions from numerous other mining experts with whom this project has been reviewed.

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## SECTION 1

## INTRODUCTION AND SUMMARY

Coal seams which exceed 9 feet in thickness may be regarded as exceptional. Since the vast majority of coal mined underground is produced from coal seams of less than 9 feet of thickness, coal industry techniques and equipment have quite naturally been developed which favor such thicknesses. Some present standard face mining machines have a maximum range to about 12 feet above the floor. Coal above 12 feet cannot be reached by ordinary continuous miners or longwall equipment. Neither can mining at a height in excess of 12 feet be safely conducted with standard conventional or continuous mining equipment and presently used methods at a reasonable cost. In the Western United States there exist, however, substantial reserves of deep coal which exceed 12 feet in thickness. These reserves cannot be mined efficiently with present underground mining techniques. The conceptual mining method entitled, "Sublevel Caving by Pillar Extraction", was developed under the sponsorship of the United States Department of Interior, Bureau of Mines, toward the objective of providing a means by which coal from thick seams can be produced with levels of mining performance, "equal to or greater than the levels obtained when standard pillar mining or longwall mining methods are used to mine coal seams seven feet thick..." (1)

There are four important challenges to mine any thickness of coal, namely; to mine safely, with good recovery, at a competitive rate of production, and at a profit. All of these challenges can be met in mining relatively thin seams. In mining thick seam coal with present techniques, there has to be some compromise. All of the challenges cannot be met. Since safety must by law be maintained, and profit is necessary for survival of the venture, either productivity or recovery must be sacrified. Good productivity, however, is essential to profitability. Therefore good recovery is usually sacrificed. It is also true that as the thickness of the coal bed increases, percentage recovery continues to decline. Recovery percentages below 30% might be expected when present equipment and techniques are applied to a coal seam twenty feet thick or more. The remainder, or 70% or more of the coal in place, of course, is lost forever. The effort to find a better way to mine thick coal seams is timely and important from the standpoint of increased production of coal, improved resource recovery and the economic values to be obtained.

(1) U.S. Bureau of Mines Request for Letter Proposals, Synopsis No. 101, March 18, 1975.

Artist's Conception

Loading of Fallen
Top Coal Under Brow
(Seen From Cave Side)

DWG-1

The Sublevel Caving by Pillar Extraction mining method is a modification of the normal pocket and stump method of pillar mining which includes recovery of top coal.

This method envisions that the mine openings will be driven against the bottom of the seam. Top coal recovery is to be accomplished on retreat by drilling and shooting the entire thickness of top coal remaining above 8-foot high openings in increments of "falls" and loading out the fall of top coal with a narrow, drum-type continuous miner.

Advance and retreat mining are to be kept strictly in balance under the concept and the cave-to-solid-coal distance kept as small as possible to facilitate ground control. Also, narrow, "yielding pillars" are proposed to minimize bumping under heavy cover and assist in preserving the competency of the mine roof. With the relatively small cave-to-solid-coal distance and the "yielding pillars", the overburden weight is believed to be cantilevered over the pillars and working places to the solid coal.

The sequence of mining is important to this concept since it enables the balance of advance and retreat mining progress and provides that all men will be under an 8-foot high primary support roof, or an 8-foot high roof with intense temporary support, at all times.

The 8-foot high roof minimizes danger from sloughing and enables ventilation to be accomplished effectively with proven techniques. Working against the bottom of the seam reduces problems with coal dust and thereby contributes to better health and safety conditions for the miners.

The key features and considerations for this mining method are:

- o Retreat slicing of top coal
- o The mining sequence.
- o Ground control accomplished by small cave-to-solid-coal distance, and thin, "yielding pillars", plus maintaining solid coal blocks to receive abutment pressures.
- o Presently available mining equipment.
- o Recovery of thick coal under heavy cover without exposing personnel to danger from high ribs and roof.
- o Competitive rate of production and cost per ton.
- o Minimum time to application.
- o Applicable to seams of varying thickness.
- o Highly effective ventilation.

- o Modified or standard mining practices.
- o Wide range of application.

The Sublevel Caving by Pillar Extraction mining method was developed and evaluated on the basis of a hypothetical mine with a capacity of approximately one million tons per year from one 20-foot thick seam. This is an arbitrary capacity. The concepts utilized in the mining method would be limited in mining capacity only by considerations such as coal reserve, local conditions, etc.

This concept, or essential elements of it, can be utlized and may be advantageous under any conditions suitable to room and pillar mining. Although the method has been developed on the basis of a 20-foot thick seam, it is believed to be applicable to underground thickness of seams 40 feet in thickness. Detailed study of the ultimate thickness to which this method may be applicable proved to be beyond the scope of this original design study. Mining very thick seams with this method would require substantial modification of the mining cycles and dimensions of pillars and openings shown. Application of this method to very thick seams may constitute a worthwhile study. The minimum seam thickness in which this method would be expected to find application is 12 feet. The ground control method proposed under the Sublevel Caving by Pillar Extraction method is also advantageous for seams thinner than 12 feet under heavy cover where safety and recovery are overriding considerations. The ground control scheme proposed under this concept may be applicable to thick or to thin seams at mining depths somewhat greater than 2,000 feet.

The mining concept, Sublevel Caving by Pillar Extraction, is shown under two differing sets of mining conditions. The first condition is that in which the mine roof is not self-supporting and would require intensive support. This condition is termed "adverse" and would be expected to be encountered under heavy cover or whenever the top coal being mined is weak and friable, or extensively fractured. The second condition under which the concept is described is that in which the top coal roof has been determined to be reliably self-supporting. This condition is termed "favorable" and would be expected to be encountered under light to medium cover.

Section coal recovery at 75 percent and mine coal recovery at 64 percent are estimated. Based upon the mining conditions described and applying the operational cycles used in this report, productivity under "adverse" conditions is calculated to be approximately 434 tons per unit shift. Productivity under "favorable" conditions is calculated to be approximately 554 tons per unit shift. Productivity is favorably influenced by the large tonnages of top coal made available for the continuous miner to load from one position. Expected selling price necessary to sustain a 15 percent return on invested capital using this method is calculated to be $13.22 per ton.

Development and evaluation of the Sublevel Caving by Pillar Extraction mining method is based upon the experience of the writer in mining coal ranging in height from 7 to 19 feet at Columbia and Horse Canyon, Utah, and the 22-foot B-Seam at Somerset, Colorado and upon the experience of consultants to this project who likewise have direct and lengthy experience in mining thick seam Western coal.

## Present Methods for Mining Thick Seam Coal

The problems associated with mining thick seam coal are not new to the mining industry and numerous measures have been taken in the past to achieve reasonable recovery and productivity when mining thick seams. The most important of these are discussed below.

It was once an acceptable practice to shoot down top coal over long distances in room and crosscuts and then to support the roof with long props while the fallen coal was recovered. This method, which was practiced in the mines at Hiawatha, Utah, fails to meet present requirements for safe roof support and would also pose problems in ventilation.

Another method of mining thick coal may be called top slicing. In this method, rooms and crosscuts are driven against the bottom of the seam. Roof coal is then shot down and enough coal loaded out to enable the roof rock to be bolted. Pillar ribs are then slabbed and the remaining broken coal loaded out with the men under the bolted roof. In this method the men are exposed to high ribs. It is also necessary that the roof be very strong for this method to succeed.

Some thick coal is mined with continuous miners by a method known as bench mining. Bench mining simply means that rooms, crosscuts and pillar splits are driven against the top of the seam which is then bolted. The top portion of the remaining pillar is then mined by additional cuts that can be reached with the operator under bolted roof. The continuous miner is thereafter used to ramp down and recover the bottom coal in the room near the pillar split and in the split itself. The main disadvantages to bench mining are exposure to high roof and ribs, erratic recovery and difficult control of ventilation and dust.

In each of these methods, the cost of making high ribs safe and of providing good ventilation along high ribs is prohibitive and detracts from overall productivity. Experience has shown that difficulties associated with these methods are severe enough to inhibit their widespread use and to induce operators mining in thick seams to abandon the possibility for good recovery and concentrate on higher productivity and safer operation by selectively mining just a portion of the thick seam.

## GENERAL MINE DESIGN

### Main Entries

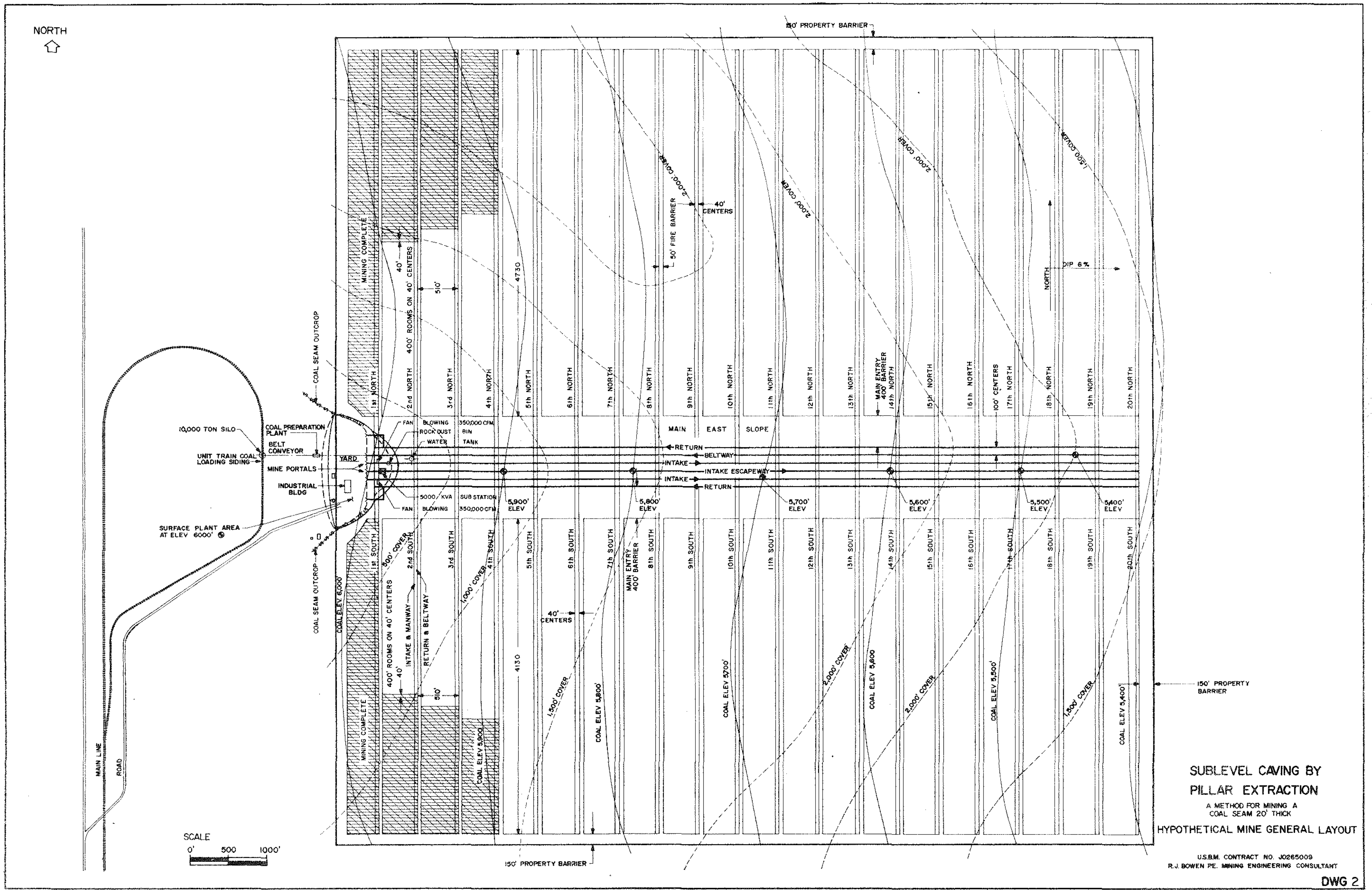
Six main entries would be driven into the coal seam from the portals at the outcrop to the property boundary or limits of mining. The following arrangement is designed to provide maximum strength and protection for these entries. Crosscuts will be staggered on 200 foot intervals except for the turnouts to the respective section entries.

Barrier pillars 400-feet wide will be left on each side of the main entries. These barriers will protect the entries against abutment pressures from the mined out sections. They will also furnish two substantial coal blocks to provide a source of coal and protection during final retreat operations.

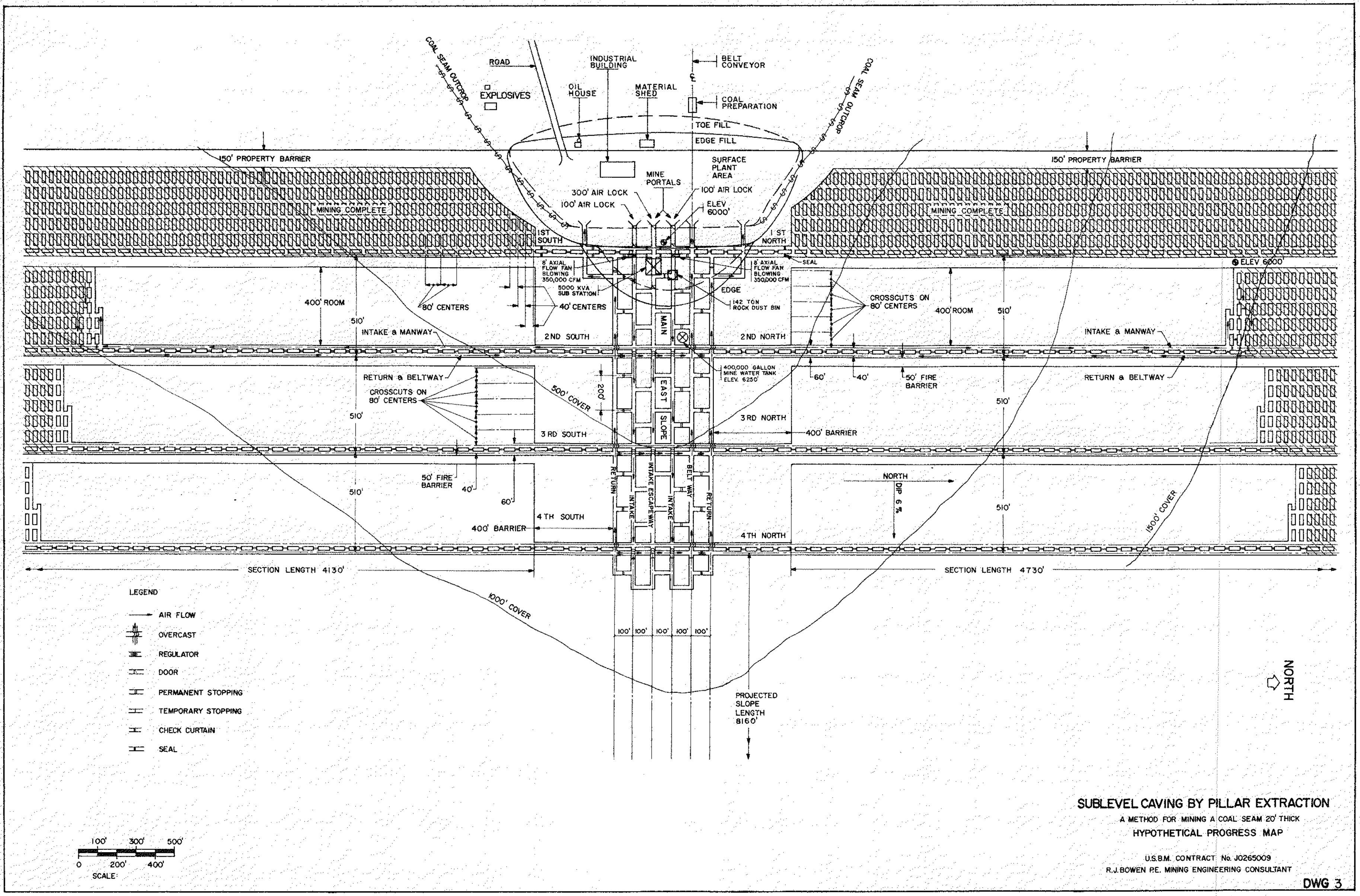
All openings of the mains would be driven 20 feet wide by 10 feet high giving a cross-sectional area of 200 square feet. Ventilation would be by the double split method: i.e. the two outside entries would be return airways. Three entries would serve as intake airways with one being designated as a smoke free intake escapeway. Power cables and sectionalizing circuit breakers, switches, etc. would be installed in one of the other intake entries. Water mains for sprinkling and/or mine drainage would be installed in the beltway where it would be protected from freezing by the separate split of warm air flowing toward the portal. The main water line would be under a minimum of 100 p.s.i. pressure and equipped with outlets and valves to accommodate fire hose stations in accordance with MESA standards. Fire outlets would be installed at each crosscut.

### Mining Section Entries

Two section entries would be driven from the main entries and extended to a distance of approximately 4,400 feet at intervals of 510 feet when measured between the center lines of the beltway entries. One entry will serve as an intake and the other as a return in which the 42-inch conveyor belt would be installed. The return entry would also contain a 4-inch water line with fire hose connections every third crosscut along the entire length of the beltway. All advance openings will be driven against the bottom of the seam 8 feet high and 20 feet wide. Entries and rooms will be driven on 40-foot centers with crosscuts on 80-foot centers. Rooms will be 400 feet long as measured from the centerline of the intake entry to the centerline of the last crosscut. This would result in five 20-foot by 62-foot pillars. Crosscuts are 18-feet wide.



NORTH
150' PROPERTY BARRIER
MINING COMPLETE
400' ROOMS ON 40' CENTERS
40'
510'
4730
4130
1st NORTH
2nd NORTH
3rd NORTH
4th NORTH
5th NORTH
6th NORTH
7th NORTH
8th NORTH
9th NORTH
10th NORTH
11th NORTH
12th NORTH
13th NORTH
MAIN ENTRY 400' BARRIER 14th NORTH
15th NORTH
16th NORTH
100' CENTERS
17th NORTH
18th NORTH
19th NORTH
20th NORTH
1st SOUTH
2nd SOUTH
3rd SOUTH
4th SOUTH
5th SOUTH
6th SOUTH
7th SOUTH
MAIN ENTRY 400' BARRIER
8th SOUTH
9th SOUTH
10th SOUTH
11th SOUTH
12th SOUTH
13th SOUTH
14th SOUTH
15th SOUTH
16th SOUTH
17th SOUTH
18th SOUTH
19th SOUTH
20th SOUTH
50' FIRE BARRIER
40' CENTERS
2,000' COVER
1,500' COVER
1,000' COVER
500' COVER
DIP 6%
NORTH
MAIN EAST SLOPE
RETURN
BELTWAY
INTAKE
INTAKE ESCAPEWAY
5,900' ELEV
5,800' ELEV
5,700' ELEV
5,600' ELEV
5,500' ELEV
5,400' ELEV
COAL ELEV 6,000'
COAL ELEV 5,900
COAL ELEV 5,800
COAL ELEV 5,700
COAL ELEV 5,600
COAL ELEV 5,500
COAL ELEV 5,400
INTAKE & MANWAY
RETURN & BELTWAY
FAN BLOWING 350,000 CFM
ROCK DUST BIN
WATER TANK
5000 KVA SUB STATION
FAN BLOWING 350,000 CFM
COAL SEAM OUTCROP
10,000 TON SILO
COAL PREPARATION PLANT
BELT CONVEYOR
UNIT TRAIN COAL LOADING SIDING
YARD
MINE PORTALS
INDUSTRIAL BLDG
SURFACE PLANT AREA AT ELEV 6000'
MAIN LINE
ROAD
SCALE
0' 500 1000'
SUBLEVEL CAVING BY PILLAR EXTRACTION
A METHOD FOR MINING A COAL SEAM 20' THICK
HYPOTHETICAL MINE GENERAL LAYOUT
U.S.B.M. CONTRACT NO. J0265009
R.J. BOWEN P.E. MINING ENGINEERING CONSULTANT
DWG 2


COAL SEAM OUTCROP
ROAD
EXPLOSIVES
INDUSTRIAL BUILDING
OIL HOUSE
MATERIAL SHED
BELT CONVEYOR
COAL PREPARATION
TOE FILL
EDGE FILL
150' PROPERTY BARRIER
SURFACE PLANT AREA
MINE PORTALS
300' AIR LOCK
100' AIR LOCK
ELEV 6000'
MINING COMPLETE
1ST SOUTH
1 ST NORTH
8' AXIAL FLOW FAN BLOWING 350,000 CFM
5000 KVA SUB STATION
SEAL
ELEV 6000'
EDGE
142 TON ROCK DUST BIN
400' ROOM
510'
80' CENTERS
40' CENTERS
CROSSCUTS ON 80' CENTERS
MAIN
INTAKE & MANWAY
2ND SOUTH
2ND NORTH
RETURN & BELTWAY
400,000 GALLON MINE WATER TANK ELEV. 6250'
60'
40'
50' FIRE BARRIER
200'
EAST
SLOPE
500' COVER
3RD SOUTH
3RD NORTH
400' BARRIER
RETURN
INTAKE
INTAKE ESCAPEWAY
BELT WAY
NORTH
DIP 6 %
1500' COVER
4TH SOUTH
4TH NORTH
SECTION LENGTH 4130'
SECTION LENGTH 4730'
1000' COVER
100' 100' 100' 100' 100'
PROJECTED SLOPE LENGTH 8160'
LEGEND
AIR FLOW
OVERCAST
REGULATOR
DOOR
PERMANENT STOPPING
TEMPORARY STOPPING
CHECK CURTAIN
SEAL
NORTH
100' 300' 500'
0 200' 400'
SCALE
SUBLEVEL CAVING BY PILLAR EXTRACTION
A METHOD FOR MINING A COAL SEAM 20' THICK
HYPOTHETICAL PROGRESS MAP
U.S.B.M. CONTRACT No. J0265009
R.J. BOWEN P.E. MINING ENGINEERING CONSULTANT
DWG 3

Drawing No. 4 shows a plan of rooms and entries for the section. Fire barriers separate each section from the adjoining section or the mined out and caved area of the previous section. The fire barriers facilitate ventilation control, and make it possible to expeditiously isolate the mined and caved sections from the rest of the mine. Also, in case of emergency, an operating section can be quickly isolated.

## Bleeder Entries

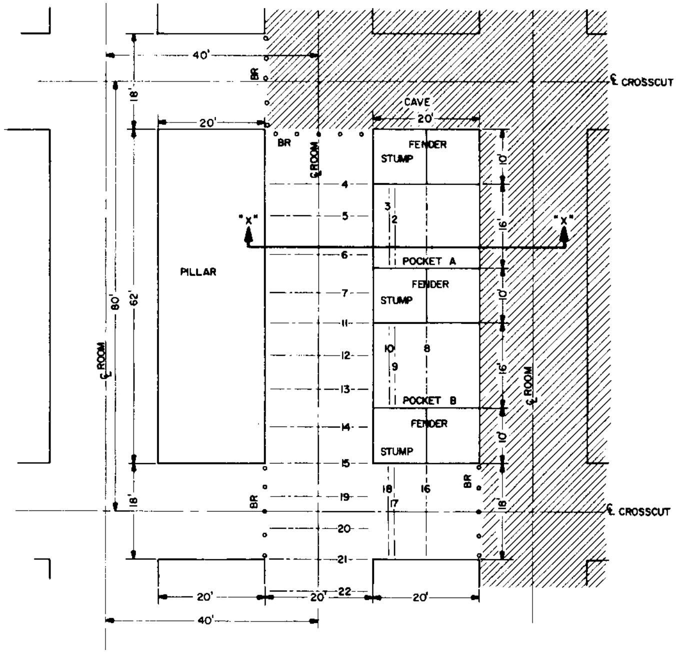
A request for an exception to eliminate bleeder entries is recommended due to the provisions for safely sealing the mined out sections as explained above. If required, however, two bleeder entries would eventually extend around the perimeter of the mine. They would be interconnected and continuous to two openings at the outcrop. Since the mine is to be on blowing ventilation, any breakthrough to the surface would assist in returning mine air to the outside atmosphere. Initially, the bleeder entries would be driven along the property barrier concurrently with and parallel to the first section entries to the end of the section or to the limits of the property barrier. The section mining unit would then drive the two end bleeders and connect them. In retreat mining of the first section, a 50-foot barrier pillar would be left to protect the bleeders. Each of the section entries and the last crosscut of the first room would connect to the bleeder. The bleeder would also be connected at intervals of 480 feet by extending a room of the first section through the barrier pillar. Three bottom, or last section entries, would be driven. In retreat mining, the chain pillars would be left and the remaining two entries would serve as bleeders. This plan was discussed with MESA and verbal concurrence indicated. (See Drawing No. 9, Alternate Plans.) Bleeder entries adversely affect recovery, productivity, ground control, spontaneous combustion and ventilation. They also generally present unacceptable hazards to personnel engaged in maintaining them.

## Pillaring

Pillars would be mined by making splits or pockets in the pillar in the sequence and dimensions as shown by "Plan of Pillaring", Drawing No. 4. The top coal within the room and adjacent to split A would be recovered before making split or pocket B.

Top coal would be broken down for loading by shooting rows of $60^0$ angled holes spaced about 5 feet apart in the numbered sequence as shown using millisecond delays.

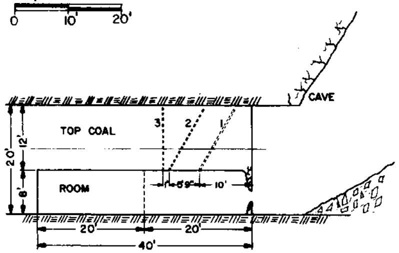
The pillar dimensions recommended are 20 x 62 feet. The fenders on each side of the pillar splits are 10 feet thick. These pillar dimensions should expedite mining of the pillar and attendant top and thereby avoid accumulating weight as much as possible.




## PLAN OF PILLARING

SCALE

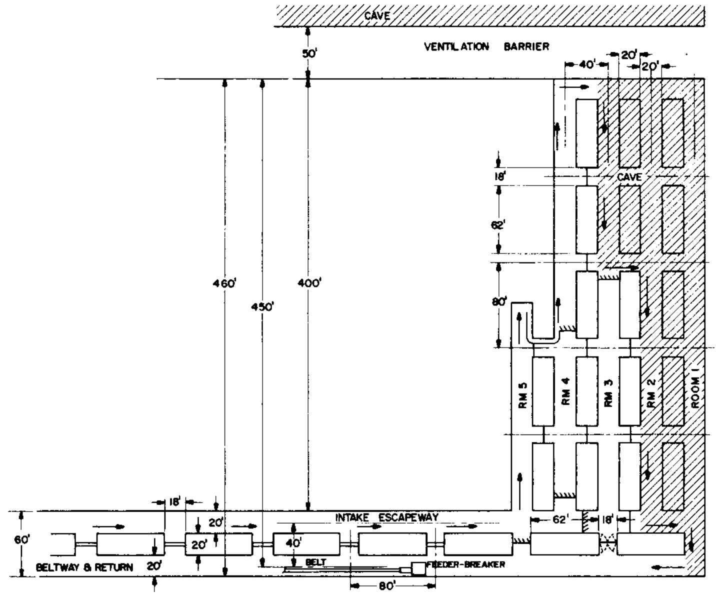
0 10' 20'




SECTION "X"-"X"

SHOOTING SEQUENCE

| FALL | ROW |
|---|---|
| 1 | 1, 2, 3 |
| 2 | 4 |
| 3 | 5, 6, 7 |
| 4 | 8, 9, 10 |
| 5 | 11 |
| 6 | 12, 13, 14 |
| 7 | 15 |
| 8 | 16, 17, 18 |
| 9 | 19, 20, 21, 22 |




## PLAN
## ROOMS & ENTRIES

SCALE

0 50' 100'

LEGEND

TEMPORARY STOPPING

STOPPING

CHECK CURTAIN

VENTILATING AIR FLOW

BR BREAKER ROW OF 20 TON HYDRAULIC JACKS

1,2,3 etc SHOOTING SEQUENCE OF BLAST HOLE ROWS

SUBLEVEL CAVING BY PILLAR EXTRACTION

A METHOD FOR MINING A COAL SEAM 20' THICK

SECTION PLAN FOR MINING UNDER BOTH FAVORABLE AND ADVERSE CONDITIONS

U.S.B.M. CONTRACT No J0265009

R.J. BOWEN P.E. MINING ENGINEERING CONSULTANT

DWG 4

The time element, which is an important factor in successfully extracting a pillar, is kept to a minimum.

The end of the pocket is mined through in order to provide better ventilation. The continuous miner will cut a hole in the fender about five feet above the floor to expose the cave and facilitate ventilation of air through the pocket into the return against the cave. Protection against a possible hazard of cascading coal or rock from the cave while drilling and loading blast holes would be provided by the remaining coal in the fender. The $60^{o}$ angled holes place the drillers and shotfirers back 10 feet from the brow.

With the selection of the continuous miner, recovery of the inside ends of the side fenders and some of the top coal above the fender is feasible. No shooting of the fender would be required.

## The Ground Control System

The best and most economical means of supporting the ground is by careful arrangement of the mining operation so that the integrity of the ground is preserved as much as possible without support. The nature of the support depends upon the ground conditions and the length of time required to maintain the openings. As has been noted by many, it is important to plan operations so that the length of time that an opening must remain open is a minimum. This is due both to a time deterioration of the mine roof as well as a shifting of the loading.

The basic method to be employed is a caving system, and therefore the presumption is that the layers making up the immediate roof will cave in a predictable manner after the coal has been removed. The system can be characterized from a ground control view point as being similar to a retreating longwall. Because of the thickness involved, the mechanical props are replaced by pillars. As in longwall mining, there will be a front and rear abutment. The position and nature of the rear abutment will be determined by the type of the roof strata, the yieldability of the pillars and the depth and thickness of the seam.

The distance between the front and rear abutment could be of the order of several hundred feet. Figures 1-1 and 1-2 show sectional views illustrating British and French opinions regarding pressure distribution. Published American opinion is not as applicable to Sublevel Caving by Pillar Extraction as are these observations adapted from Woodruff. (2)

(2) Woodruff, S. D., "Methods of Working Coal and Metal Mines", Volume I, Pergamon Press, 1966.

However, the observations of Stephanko regarding pressure distribution and caving action for longwall mining are informative. (3)

The solid unmined seam supports the front end of the "roof beam" or "arch" and is therefore subjected to pressure (abutment pressure) greatly in excess of the depth pressure (estimates place the maximum values at four to six times the normal depth pressure). At depths of more than a few hundred feet, the abutment pressures are of sufficient magnitude to induce shearing of the softer roof rocks overlying coal seams and fracturing of the coal in the seam. The depth at which it occurs depends upon strengths of roof rocks, span of the opening and physical properties of the seam. Induced cleavage in the coal ahead of the face reduces the coal from an elastic solid to a fractured material and the position of the maximum abutment pressure moves to a distance ahead of the face such that lateral confining stress is sufficient to prevent further shearing.

The fact that the main roof load can be transferred to abutments suggests that by carefully designing the pillars needed to maintain openings over a limited area, the main load carrying function could be transferred to solid coal barriers provided on the flanks designed for the loads. Thus over a limited area the vertical load is reduced and conditions are created which are similar to those in shallow workings.

The principle of yielding pillars involves the selection of dimensions for pillars and roadways so that the stress on pillars will be sufficient to create a small amount of yielding which by allowing a slight downward deflection of the roof transmits more vertical stress to the abutments. (See Figure No. 1-3).

The use of large pillars within the excavation increases the intensity of stress in the area of support. The increase in stress concentration at the edges of such support creates localized shear stresses which can fracture the roofs of rooms.

When pillars are made small enough to yield, the intensity of stresses at edges is reduced with the result that roofs of openings remain intact. However, if dimensions of pillars are made too small then the main span of the beam or arch may collapse with the result that the pillars are crushed.

(3) Stephanko, Robert, "Coal Mining in the U.S.A.", Pennsylvania State University, College of Earth and Mineral Sciences, 1976.

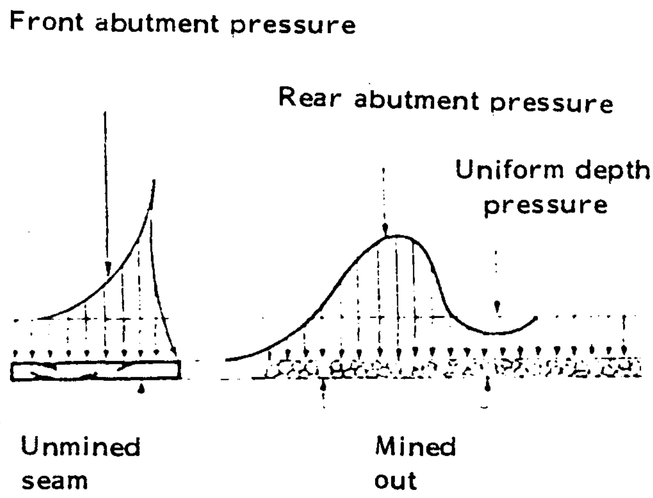



Fig. 1-1 Pattern of pressure distribution indicated by British Observations. Woodruff. (4)

British Observations References:

Alder, H., Potts, E. L. J., and Walker, A., Research on strata control in the northern coalfields of Great Britain, International Conference on Strata Control and Support in the Workings, Organized by INICHAR, Liege, April 1951, p. 106.
Evans, W. H., Hogan, M. A., and Vallis, E. H., An investigation of the loads on packs at moderate depths, Trans. Inst. Mining Engrs., London, Vol. 100, Part 12, pp. 340-361, 1941 (Paper 3003).
Evans, W. H. and Jones, T. J., An investigation of the loads on packs at moderate depths. Part II, Trans. Inst. Mining Engrs., London, Vol. 105, Part 6, pp. 286-302, 1946.

(4) Woodruff, p. 19.

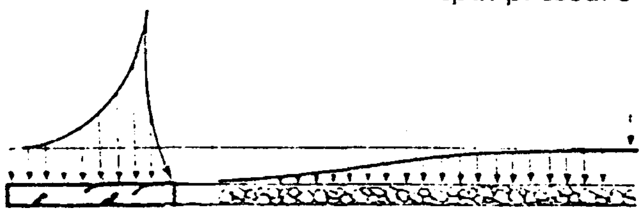



Fig. 1-2 Pattern of pressure distribution indicated by German Observations. Woodruff. (5)

German Observation Reference:

Jacobi, O., The pressure on seam and goaf, Presented at the International Strata Control Congress, Essen, October 17 and 18, 1956; Published by Steinkohlenbergbauverein, Essen.

(5) Woodruff, p. 19.

The following hypothetical case (taken from Woodruff) (6) involving a seam situated at a depth of 1,800 feet illustrates the type of design that might be used to mine thick coal. For this depth, the distance between abutments is about 330 feet. If the pillared area consists of six rooms each 13 feet wide separated by pillars 50 feet wide, then the total width of the developed zone will be about 380 feet and the front abutment of the pressure arch will rest on pillars in the developed area as shown in Figure 1-4 resulting in floor heave and fracture of the roof adjacent to the pillars. The pillars are too wide to yield and therefore carry load which could otherwise be transferred to the abutments. Excessive stresses may cause shearing of the roof adjacent to the pillar sides.

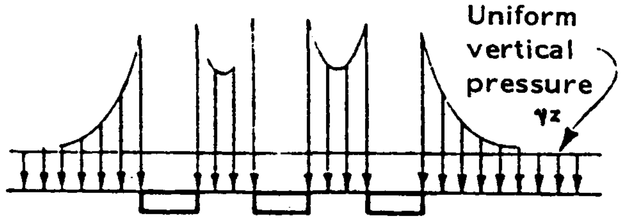
The abutment zones are believed to be a result of strong rock strata acting as cantilever beams which transfer weight from the area over the cave to areas behind and in front of the cave. It is to be expected, however, that the section entries will be subject to some abutment pressures from the cave advancing from the rear. The two section entry plan of this mining method, with yielding pillars, maintains roof strength at a maximum and thus provides maximum safety from roof falls and bumps, especially under conditions imposed by heavy cover.

If pillar widths are reduced to 16 feet then the same number of working places occupy a total width of only 180 feet and the entire working area is well inside the pressure abutments. (See Figure 1-5.) The narrow pillars can yield enough to avoid excessive stress build up in the intermediate pillars and transfer the pressures to a front abutment on the solid coal ahead of the developed area. This situation is illustrated by citing the example of the French potash mines (depths 2,000 - 3,000 feet) in which pillars were initially made 25 feet in width. However, severe roof problems were experienced. When pillar widths were reduced to 10 feet the difficulties disappeared. (7)

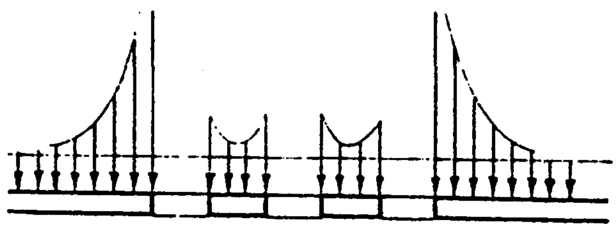
Particular attention is called to the fact that driving the rooms and extracting the pillars and top coal under heavy cover makes it necessary to keep the distance between the cave and the solid coal block as small as possible. Attention should also be given to drawings provided which show the sequence of top coal removal. This sequence enables mining to proceed in a thick seam without high ribs. It also provides for overburden load distribution and small cave-to-coal block separation.

(6) Woodruff, p.

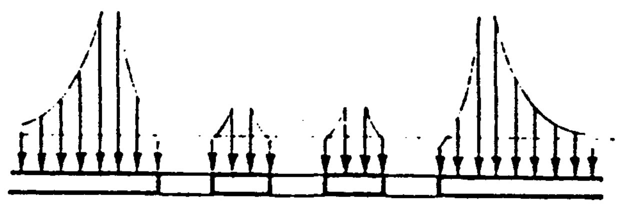
(7) Collardey, J., "Criteres d' application de la methode des chambers et piliers", Ann. Mines Belg., Dec. 1958, pp. 1110-1127.




Distribution of pressure on pillars in a homogeneous, elastic material.



Distribution of pressure when pillars yield elastically (without fracturing).



Distribution of pressure when pillars yield by fracturing.

Yielding Pillars

Fig. 1-3

Therefore, the advance mining of the rooms and the retreat mining of the pillars together with the top coal recovery must be kept in balance. Only one room would be driven during the extraction of one row of pillars and the accompanying top coal. This balance is an important part of the ground control. For example, Drawing No. 4 shows room 5 advanced a little beyond the second crosscut, with retreat mining ready to recover coal from the third pillar between room 3 and mined room 2. Room 6 would not be started until pillaring had begun between rooms 3 and 4 and the entry chain pillar below room 2 had been removed.

The mining method design has been modified from the originally proposed three section entry plan to a two entry plan. This modification yields the advantage to ground control of having to maintain as few openings as possible, thus preserving a maximum size block of undisturbed coal to absorb abutment pressures. The two section entry system also furthers the objective of keeping the cave-to-solid-coal distance at an absolute minimum.

Various room lengths were considered in an effort to achieve an optimum balance between ground control considerations and the necessity of achieving competitive rates of production. Rooms shorter than 400 feet were considered to effect shorter shuttle car haulage and to reduce the time the room and pillar openings must stand open. The experience of the writer in mining thick coal argues, however, against a design for this concept with rooms of less than 400 feet. The effect of the abutment pressure zone in many instances reaches out 300 to 350 feet beyond the advancing cave line. This increased pressure effect has been observed on section entries during retreat mining, first with shaker conveyors, and later with continous miners using the same system of ground control. The 400 foot room length is a design feature selected to give some measure of assurance that the side abutment pressure zone from the adjoining mined section will fall on the solid block instead of on the section entries.

## Roof Support

The standard plan for roof control for those conditions described as "adverse" will be wooden props supporting timber crossbars on 4-foot centers in the main entries and in the section entries. In the room and pillar areas, the wooden crossbar sets would be supplanted with aluminum structural crossbars supported by hydraulic jacks.

In the main and section entry development the method of placing the wooden crossbar sets would be to use the miner cutter head to lift the crossbar to the roof until the legs or posts are placed by the miner helper and timber utility man and other available section crew members. The miner would not be withdrawn from the face during roof support operations and the roof support would be placed as quickly after mining

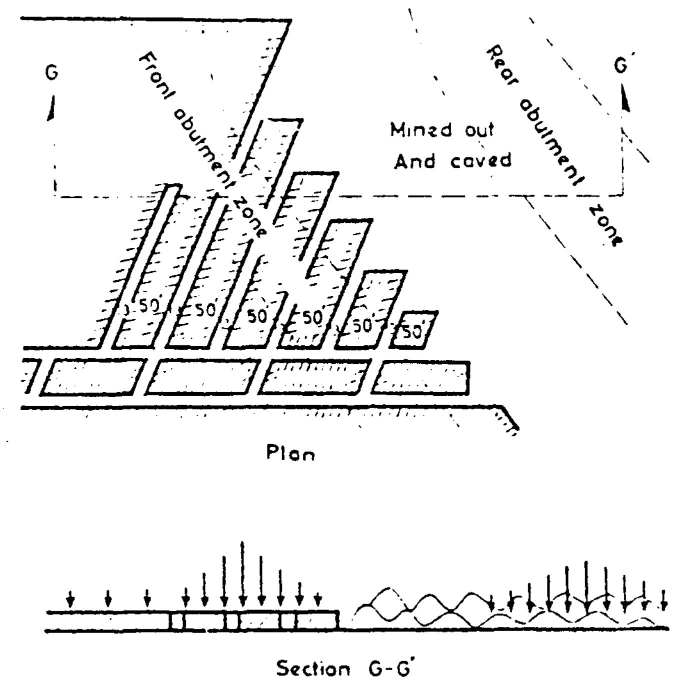



Fig. 1-4 Pillars are too wide to yield and too much area has been developed. The front abutment will damage the development openings.

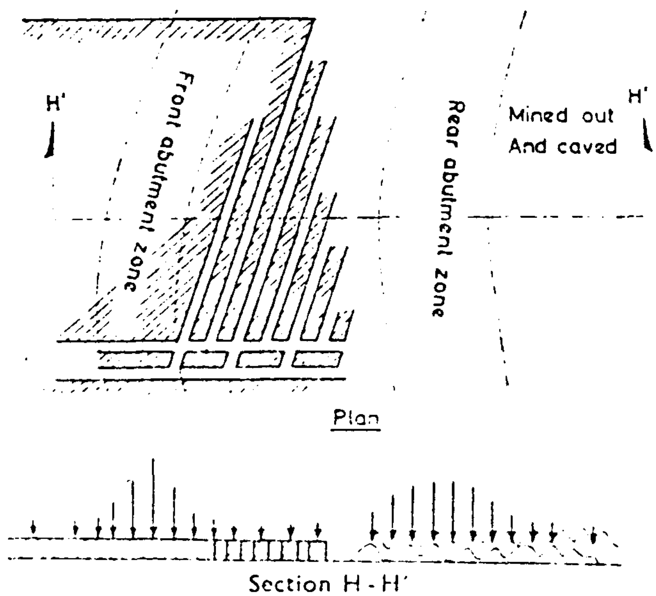


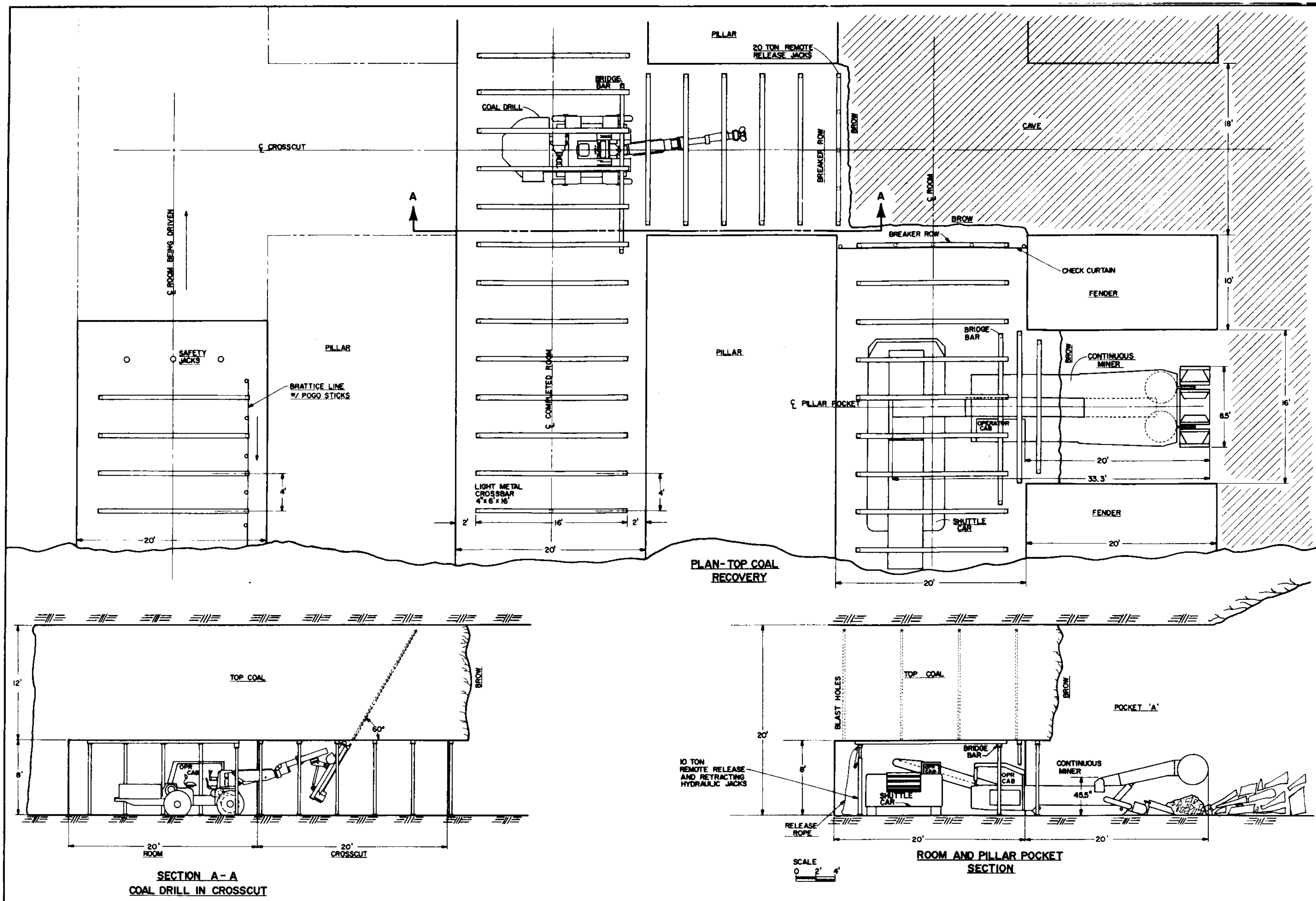

Fig. 1-5 Pillars are narrow enough to yield and to allow the "main roof load" to transfer to the abutments. The developed zone is narrow so that it lies entirely between the abutments.

as possible, thus preventing roof sag and helping to maintain roof competency. Wooden props and crossbars would be used in the mains and the section entries because they are much cheaper than the metal sets. As the entry supports are expected to stand a long time before removal, no significant advantage in time saved during placement would be achieved to offset the much greater cost of the metal crossbar and jack sets. Of course, additional support would be installed when necessary.

By contrast, "favorable" roof conditions would be where the top coal is reliably self supporting over the span of the opening. It is the type of roof expected where the coal is relatively strong and under light cover. The support plan for favorable roof would consist of one row of props on 6-foot centers set 3 feet from the rib opposite the continuous miner operator. The props would be installed as the miner advances, keeping the timbering progress ahead of the operator's cab position. Of course, the roof would be watched and additional support supplied as conditions warrant under both favorable or adverse conditions.

Under adverse conditions prefabricated light weight aluminum crossbars will be installed on 4 foot centers in the rooms using 10-ton capacity hydraulic jacks for posts. These sets will be placed in a similar manner as described above. These bars and jacks will be retrieved as the top coal and pillars are mined and used over again in the advancing room.

Aside from safety, the efficiency and productivity of an underground mining operation is tied closely to roof support. It is essential that the roof support system be compatible with the mining method used. In the case of Sublevel Caving by Pillar Extraction, the roof support requirements take on special significance. Standard expansion type roof bolts cannot be used because they will not anchor satisfactorily in coal, and they would have to be removed before shooting top coal. Resin bolts will anchor, but there is still the problem of them interfering with the loading operation. Pumpable bolts are not available from a practical standpoint at this time. In addition to the above disadvantages, any type of bolting would require removal of the miner from the face and the installation of safety props before the bolts can be installed by the roof bolting machine. Bolting, therefore, adds three functions to the mining cycle. Self advancing roof support systems and mobile roof support systems have not yet found widespread acceptance in room and pillar mining. These systems are not deemed to be sufficiently flexible to support widespread application of the concept. Also, they are sufficiently "new" to the industry that their inclusion in the design would add some uncertainty to this concept. Crossbar sets for roof support under "adverse" top coal conditions serve the concept support requirements better than roof bolts or other methods in view of studies made.




SUBLEVEL CAVING BY PILLAR EXTRACTION

A METHOD FOR MINING A COAL SEAM 20' THICK

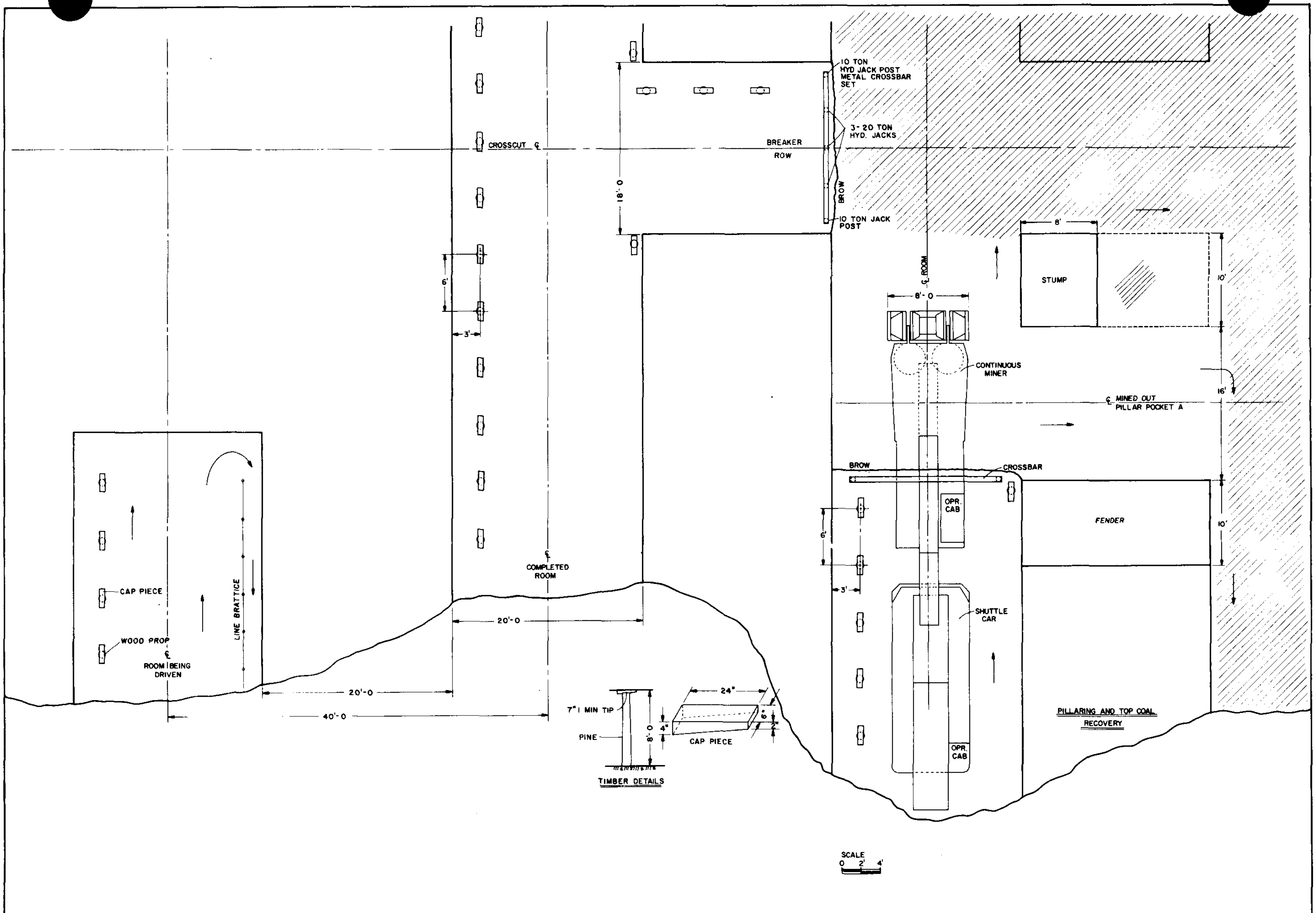
PLAN OF ROOF SUPPORT AND OPERATIONS

UNDER ADVERSE CONDITIONS

U.S.B.M. CONTRACT No J0265009

R.J. BOWEN P.E. MINING ENGINEERING CONSULTANT

DWG 5




SUBLEVEL CAVING BY PILLAR EXTRACTION

A METHOD FOR MINING A COAL SEAM 20' THICK

**PLAN OF ROOF SUPPORT AND OPERATIONS UNDER FAVORABLE CONDITIONS**

U.S.B.M. CONTRACT No J0265009

R.J. BOWEN P.E. MINING ENGINEERING CONSULTANT

**DWG 6**

## Mine Ventilation

The supply of mine ventilating air would be furnished by two 8-foot axial flow fans operating as blowers. Each fan would blow approximately 350,000 CFM of fresh air into the three intake airways which would be distributed as required throughout the mine. The theory behind the choice of blowing ventilation and actually established by Kaiser at the Sunnyside Mine, is that the whole mine is maintained above atmospheric pressure. This aids in keeping the gasses released by retreat mining back within the mined out areas. In other words, the gasses are pushed back into the cave instead of being sucked out of the cave. The above advantage also aids in combatting and controlling spontaneous combustion.

## Section Ventilation

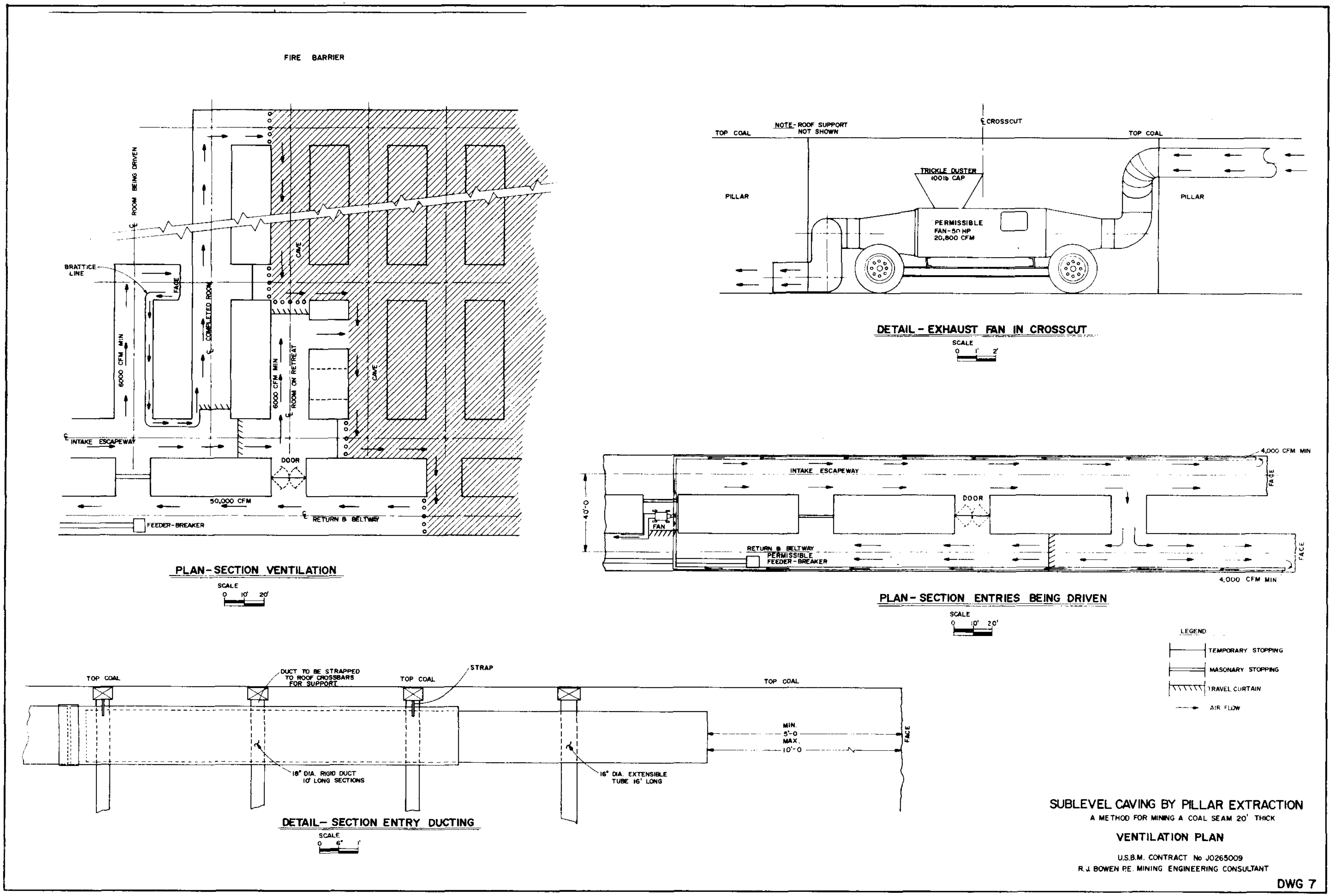
In section entry driving, effective face ventilation will be accomplished with the aid of a portable fan operating as an exhauster. The fan will be permissible. It will be located in the return at the first crosscut out by the feeder breaker and exhaust air from the face through an 18-inch diameter rigid tube. The tube comes in 10-foot lengths and will be extended as the face advances. A 16-inch tube 16 feet long will be inserted in the end of the 18-inch tube in such a manner that it can be pulled out to remove the dust-laden air near the cutter wheels well in advance of the operator's position. There will always be a current of fresh air traveling toward the intake of the vent tube. (See Drawing No. 7.)

The fan will serve two vent tube lines, one to each entry face, and have the capacity to exhaust 4,000 CFM from each face when each line is 350 feet long. It is estimated a 50-HP fan will be required. A trickle duster will be installed on this fan to control float dust. Wet scrubbers and noise attenuators have been checked into and found too ineffective and impractical in their present stages of development.

Ventilation for the room and pillar work will be furnished by line brattice, and regulated by check curtains. Due to the fact that the void between the cave serves as the return for both the rooms and pillar retreat, very little work is required to provide effective positive ventilation with the brattice lines.

## Mining Machinery Application

Mining machinery application is the aspect of the mining method design changed most dramatically as a result of investigations toward the contract objectives of achieving the greatest degree of practicality, adapting the concept to as wide a range of conditions as possible, and the utilization, if feasible, of machinery likely to be found in an existing


FIRE BARRIER
ROOM BEING DRIVEN
BRATTICE LINE
FACE
COMPLETED ROOM
6000 CFM MIN
CAVE
6000 CFM MIN
ROOM ON RETREAT
INTAKE ESCAPEWAY
DOOR
50,000 CFM
RETURN & BELTWAY
FEEDER-BREAKER
PLAN-SECTION VENTILATION
SCALE
0 10' 20'
CROSSCUT
NOTE-ROOF SUPPORT NOT SHOWN
TOP COAL
TOP COAL
PILLAR
TRICKLE DUSTER 100lb CAP
PERMISSIBLE FAN-50 HP 20,800 CFM
PILLAR
DETAIL-EXHAUST FAN IN CROSSCUT
SCALE
0 1' 2'
4,000 CFM MIN
INTAKE ESCAPEWAY
FACE
40'-0
DOOR
FAN
RETURN & BELTWAY
PERMISSIBLE FEEDER-BREAKER
FACE
4,000 CFM MIN
PLAN-SECTION ENTRIES BEING DRIVEN
SCALE
0 10' 20'
LEGEND
TEMPORARY STOPPING
MASONARY STOPPING
TRAVEL CURTAIN
AIR FLOW
TOP COAL
DUCT TO BE STRAPPED TO ROOF CROSSBARS FOR SUPPORT
TOP COAL
STRAP
TOP COAL
MIN. 5'-0
MAX. 10'-0
FACE
18" DIA. RIGID DUCT 10' LONG SECTIONS
16" DIA. EXTENSIBLE TUBE 16' LONG
DETAIL-SECTION ENTRY DUCTING
SCALE
0 6" 1'
SUBLEVEL CAVING BY PILLAR EXTRACTION
A METHOD FOR MINING A COAL SEAM 20' THICK
VENTILATION PLAN
U.S.B.M. CONTRACT No J0265009
R.J. BOWEN P.E. MINING ENGINEERING CONSULTANT
DWG 7

mine. All of these considerations point toward the continuous miner unit for the concept. Accordingly, mining cycles applying the continuous miner to the concept were developed and productivity analyzed. Surprisingly, the predicted production for the continuous miner is somewhat better than that expected for the originally proposed selection of equipment.

Some disadvantages for the continuous miner as compared to the conventional mining unit with diesel-driven LHD's are obvious. The LHD's have superior mobility and are able to clean up the work place and under favorable conditions can load out the coal faster than the continuous miner.

The disadvantages notwithstanding, the continuous miner unit has been found to be the best choice of equipment. The ability of the continuous miner to remain in the face to accomplish coal removal and loading makes it better suited than conventional equipment for mining with the relatively small number of working places dictated by the ground control method. The miner unit is an all-electric system which is not subject to UMWA and National Institute of Occupational Health and Safety (NIOSH) objections to diesel equipment. Added to this consideration is the fact that continuous mining is the most widespread underground method in the United States. Continuous mining equipment would, therefore, be most likely to be that found in existing mines. If the concept is presented with a selection of continuous mining equipment, it biulds upon a large base of common experience among miners and mine management personnel, those who must in the end judge the validity of the method.

As was emphasized in the original proposal, the efficient loading of the fallen top coal is the critical production element for this mining method. A primary consideration in the original selection of a rear-operated LHD as a basic and enabling machine was its length, and its ability to load efficiently and yet keep the operator at a safe distance from the brow. Closer investigation of continuous miners has shown that most of the top coal loading cycle will be accomplished with the operator remaining in his safety cab under supported roof. This will be possible with no modifications to presently available continuous miners. For those times when advance of the operator's station beyond the brow is necessary, the miner will be equipped with remote controls. Loading of fallen top coal with the miner operator in his cab would be considered to be the normal operating procedure.

The selection of the continuous miner as the machine to be utilized for both advance mining and for top coal loading keeps the cost of the section complement of equipment within accepted norms.

The advantages of the originally proposed section complement as regards dust control compared to a continuous miner are, of course, lost.

However, the small number of working places which effect fundamentally better ventilation mitigate against this loss. The continuous miner also offers the advantage to ventilation of being able to mine through the pocket until it is breached by a small hole. This is an improvement over the original concept as regards ventilation in the retreat rooms and pillars. The advantage of providing at least a partial shield against cascade of roof rock from the cave area into the pocket is achieved.

The option of leaving the fenders in the pillar pockets intact or mining them with the continuous miner after the top coal in the pocket has been shot down and loaded gives the operating crew an important advantage in making an on-the- spot judgment as to the competency of the roof rock above the pocket and the advisability of attempting recovery of the fender. This option was not available under the original proposal which envisioned shooting of the fenders concurrently with top coal falls.

The recommended section complement of equipment for both advance and retreat mining with this method is then as follows:

1 - Narrow, drum-type continuous miner equipped with remote controls.

2 - Shuttle cars (10-ton capacity)

1 - Roof coal drill (modified roof bolting machine)

1 - Utility front-end loader with fork lift attachment

1 - Power load center

1 - Pressurized rock dusting system

1 - Belt feeder breaker

Six considerations guided the selection of a basic complement of section equipment. First the equipment must be effective within the overall concept. Loading must be accomplished with the operator at a safe distance from the brow. Second, the machinery selected should be likely to be found in existing mines. Third, modifications to machinery should be known to be feasible and constitute a minimal departure from presently available equipment. Fourth, the equipment should integrate well to allow efficient application and economies in support systems. Fifth, the equipment should be acceptable by MESA standards in union and non-union mines. Sixth, the complement of equipment should be reasonable in cost.

Materials Handling and Transportation

Efficient and timely flow of men and materials into and out of the mining areas constitutes one of the most important mining support functions. It is recommended for the Sublevel Caving by Pillar Extraction method that this function be accomplished with battery-powered tractors and trailers. The ideal method would be to apply diesel-driven tractors to this service.

It is also recommended that the material haulageways be concreted. Concreting these roads would reduce rock dust usage and favor maintenance of both the roadways themselves and the equipment used upon them. In addition a measure of comfort would be gained for the underground personnel and transport would be faster. Concrete roadways would be especially useful in the event of a soft mine floor.

Rock dust would be handled in bulk and supplied into the sections by pressurized pipelines from which it would be applied to the mine surfaces by hose. Entries would be machine rock dusted with a battery-powered unit.

## SECTION 2

## ENGINEERING FEASIBILITY

The Sublevel Caving by Pillar Extraction mining method has been examined thoroughly in regard to engineering feasibility of the overall concept and each of its unique design features. The conculsion of this evaluation is that this method is feasible from the standpoints of health and safety, productivity, recovery, overall adherence to mining regulations, availability of required mining machinery, compatibility with standard coal industry labor practices, suitability to geologic conditions generally prevalent in coal fields of the Western United States, and environmental effect. The economic feasibility analysis shown in Section 4 of this report, leads to the preliminary conclusion that this method will be feasible from the standpoint of economics also.

This mining method can be competitive with room and pillar mining in coal seams of ordinary thickness (6 to 8 feet) under similar geological and economic conditions. It is expected that comparable results can be maintained in the performance categories mentioned above. This is due in large part to the conclusion reached that variations in mining conditions would effect the Sublevel Caving by Pillar Extraction method in thick seams in much the same manner as they would ordinary room and pillar mining in a thinner seam of 6 to 8 feet. Under ideal conditions favorable results can be expected from both methods. Under severely adverse conditions, recovery, productivity, and cost may become marginal or unsuccessful for both methods of mining. As safety imposes an absolute standard, it becomes a determining factor under marginal conditions for the other performance categories for either Sublevel Caving by Pillar Extraction in thick coal or standard room and pillar mining in ordinary coal of 6 to 8 feet in height. If the venture cannot be made safe, then another mining method must be found or the mining venture abandoned.

The ground control method planned for the Sublevel Caving by Pillar Extraction method in thick seams is applicable to ordinary room and pillar mining in 6 to 8 foot seams under heavy cover. In fact it is essential. Room and pillar mining under heavy cover where a large number of rooms are driven and multiple rows of pillars are extracted simultaneously is a very difficult operation to sustain. Reasonable performance becomes more and more unlikely as more area is mined. Either dangerous and erratic ground control is tolerated or recovery of pillars is abandoned in large areas of the mine. In a thick coal seam, an expansive developed area under heavy cover virtually precludes recovery of an acceptable percentage (60% at a minimum) of the seam.

Application of the Sublevel Caving by Pillar Extraction method would necessitate strict adherence to the ground control principles of balanced advance and retreat mining, limitation of the number of rooms (size of the developed area open at any given time) to maintain a minimum cave-to-solid block distance, and yeilding pillars. It is the concensus of project affiliates that the Sublevel Caving method would not be successful unless this ground control method were faithfully executed.

## Geological Feasibility

Geological conditions in Western coal seams as they relate to mining conditions are so variable that a complete discussion would be excessive in volume. No single set of geological conditions was identified in the examination of this subject as being typical of conditions likely to be encountered with the Sublevel Caving by Pillar Extraction method. The most likely general condition for roof and floor is believed to be a shale roof and sandstone floor. The second most likely condition would be the reverse, a shale floor and sandstone roof. Many variations are expected to be encountered. Some of the most important geological factors as they affect the feasibility of the method are discussed below.

A friable, weak coal, even if the roof rock were strong and reliable, would require placement of more intensive roof support than that shown for this method. The cost of labor and materials required to support the top coal would be dramatically higher. Such a roof would need to be lagged and crossbar sets might be required on two-foot centers. Top coal shot holes could be spaced further apart and drilling and shooting time lessened, thus compensating somewhat for increased cost of roof support. The rate of production during advance mining could be expected to be severely impaired. During top coal retreat, the problem of broken shot wires would be aggravated. And the roof support, including lagging, which would have to be left in place to prevent broken wires, would encumber the loading operation.

The presence of large "pots" or slicken sided rock masses embedded in the roof rock would preclude mining of a seven foot thick seam by room and pillar methods. The danger of these masses falling on workers is too great and the predictability, and therefore manageability of the hazard, insufficient. With the Sublevel Caving by Pillar Extraction method the top coal would hold these masses in place. However, during loading of top coal these masses would be free to cause unacceptable damage to the continuous miner. The continuous miner would not only be frequently damaged, but might be trapped requiring a very dangerous recovery operation.

An exceptionally strong roof stratum, such as a massive sandstone, could cause the condition known as a "hanging fall". Under this condition, the weight from the cave may be carried over onto the pillar being extracted. This weight may cause fenders to crush, the top coal to fracture extensively, and sloughage beyond the limit of practical roof control to occur. To mine successfully under such a condition, the fenders of the pillars or the pillars themselves might be enlarged. This enlargement could, however, set the stage for bumps. If the fenders were enlarged enough to break the roof, they would have to be reduced in size before leaving the place in order to prevent them from acting as a supporting column. Under these conditions Sublevel Caving by Pillar Extraction would probably be more successful than standard room and pillar methods in seams of ordinary thickness due to the fact that the removal of the top coal would create a much higher fender. The "slenderness ratio" of the column would be reduced and the fender would be more likely to fail in the desired fashion. Also, under the Sublevel Caving method no one would be exposed to the fender when it started to fail.

On the other hand, a strong roof stratum (short of the point of causing "hanging falls") is conducive to reducing dilution of top coal by roof rock and to machine safety. A reasonably strong roof allows sufficient time for loading of the entire fall of top coal. A strong roof would be reliable and would extend the cave some distance into the mined out area. This would allow recovery of coal from the fenders left from the pillar. A strong roof would tend to enhance the desired yeilding action of the pillars and preserve the strength of the top coal roof, thus facilitating the entire retreat operation. The yielding of the pillars would be expected to increase rib sloughage somewhat but not beyond tolerance or control by methods such as pinning with wooden dowels. Return ventilation along the cave would also be enhanced by a strong roof. Ideally, the rock roof would hold for a few hours for a distance of about 20 feet into the mined out area and then would cave to a point within about 10 feet of the pillar. Although this is admittedly an ideal condition, such roof conditions are not uncommon.

A very weak roof stratum may be incompetent to the extent that it would cave upon the fallen top coal before it could be loaded. Under this condition recovery of the full seam thickness would not be possible with this method. If, however, the coal itself were strong, the uppermost portion of the seam could be left to support the weak roof rock. The practice of leaving top coal to support weak, sloughing shale bands or other unreliable roof rock is presently applied when necessary in Western mines. Roof bolts which hold steel mats against the roof often aid in supporting weak roofs above seams of ordinary thickness. This technique, of course, could not be used with the Sublevel Caving by Pillar Extraction method. If top coal were left for support, careful shooting would be required so that the support top coal would not be fractured by the explosives. Recovery would be diminished significantly by this practice.

It is not uncommon in Western coal fields to encounter interbedded lenticular members of shale, carbonaceous shale and sandstone immediately above the coal seam. When this occurs, the composition or the roof rock may change within short distances (100 to 300 feet). These lenses of roof material appear to have been interchangeable in deposition. If a firm sandstone or shale lens occurs, it is likely that good roof conditions will be experienced. Where either sandstone or shale is mixed with carbonaceous material in the roof rock, sloughage will take place when the top coal is withdrawn. Generally the thickness of this type of sloughage is less than one foot. Where present, this type of sloughage gets into the run-of-mine coal regardless of measures taken against it and the quality of the run-of-mine coal is reduced. The only practical way of treating this problem is to accept the sloughage along with the coal and then to remove it in a coal washing plant. Where roof sloughage of this type occurs, Sublevel Caving by Pillar Extraction would have a relative advantage over mining operations in seams of ordinary thickness because a greater amount of coal would be produced for a given amount of sloughage.

In some mines, the coal rests upon a fire clay bottom which may range in thickness from approximately 3 inches to 24 inches. As much as 6 inches of fire clay bottom can feasibly be loaded out with the coal without causing unreasonable dilution or slurry problems in a washing plant. A fire clay floor produces a very slick, sticky mud when water is introduced upon it. This mud can encumber the movement of mobile machines and personnel so severly that the entire operation bogs down. In a thick coal seam, the best method of eliminating the problems caused by a fire clay floor is to leave 2 feet of coal along the bottom of the seam. This is required even though leaving bottom coal results in problems with floor heave, additional cleanup, additional sprinkling and rock dusting and fire hazard. Although some of the bottom coal can be recovered beyond the brow with the continuous miner, leaving bottom coal is deleterious to good recovery. As mentioned previously, main entries and section entries (travelways) will be concreted. This measure would greatly reduce the problems associated with a fire clay bottom.

Some measures can be taken to reduce the amount of water on the mine floor. These are:

(a) Ensure control of the flow of water in cutting and sprinkling of the coal falls to achieve adequate control of dust and prevention of formation of puddles of water on the floor.

(b) Attempt to load excess water with the coal in the loading and cleaning work phases of the mining cycle.

(c) Coal drills could be equipped with a vacuum system which would pick up the fine coal at the collar of the hole and collect it into a dust chamber in the same manner as is used on a dry roof bolting machine.

(d) A feature of this concept is improved utilization of ventilation by reduction of the number of working places. The increased amount of air available at each working place would provide a more effective sweep of dust into return airways, thus reducing the need for spray control of dust.

Many coal mining operations are plagued with rock bands in the coal at one time or another. A continuous miner can cut through and load out rock bands up to 2 feet in thickness depending on the hardness of the rock. When a rock band over 2 feet in thickness is encountered, it would be necessary to either go over it or under it depending upon its location within the thick seam. Because of the height requirements imposed by the machinery and roof support selected for this mining method, it would be necessary that the rock band be 5 or 6 feet above the bottom of the seam in order to mine under it. On-the-spot judgment would determine whether to shoot it down, or whether to shoot up to it. The presence of rock bands should be relatively less harmful to the Sublevel Caving by Pillar Extraction method then they would to seams of ordinary thickness because of the expected greater proportion of coal.

Geological faulting is present in Western coal fields and displaces the coal seams in a variety of ways. Some faults have a sharp breakline and there is little disturbance of the coal or the roof or floor strata on either side. This type of faulting often leaves good roof conditions to the escarpment. Faulting may also occur in Western coal fields in a series of steps comprising a faulted zone. The faulted zone will contain weakened coal and roof rock.

The thick seam of coal which the Sublevel Caving method presumes offers some advantages in mining faulted seams. The coal could be displaced downward by as much as 14 feet and the mining operation could still continue through solid coal. In some broken zones the top coal supported by intensive crossbar support may provide a more easily controlled roof than would exposed roof rock in the same zone. Coal is only one-half as heavy as the rock and the roof supports would safely bear twice as much volume below the supporting arch in such a zone. The Sublevel Caving method would be much more amenable to mining thick seams in a faulted coal field than would a method utilizing longwall techniques.

The Sublevel Caving by Pillar Extraction method is quite flexible and can be applied to coal seams which are lenticular in nature. A good example of lenticular coal deposition is the Emery Field in Utah. In this field the mineable lenses of coal may vary in thickness from 5 feet to 25 feet within an area of approximately one square mile. It would be desirable in such a coal lens to be able to mine efficiently in both extremes of thickness without change to the overall mine plan and utilizing the same complement of machinery for both circumstances. The Sublevel Caving by Pillar Extraction method offers this possibility. A switch from standard room and pillar mining to Sublevel Caving could be made as the seam thickness exceeded 12 feet. In portions of the coal lens thinner than 12 feet the coal drill could easily be reconverted to be a roof bolting machine if this were necessary or desirable.

Seams which pitch as much as 30% could be mined with the Sublevel Caving method. The limiting factor in a pitching seam would be the limits of traction of the loaded shuttle cars and other mobile equipment.

Sublevel Caving by Pillar Extraction is expected, on the whole, to have good capability to adapt to geological discontinuities and variations as mentioned above. Other geological features such as rock spars, igneous dikes, areas barren of coal (wants), channels and rolls would not prevent this method from succeeding. The range of variation of geological influences is so great that any mining venture can be said to be almost unique in its setting. The applicability of any mining method, therefore, is subject to the sum of these influences and variations. Most of the factors discussed above would not by themselves preclude mining by the Sublevel Caving method unless they were severe in degree or extent. However, as would be the case with any other mining method as well, a number of these factors in common could rule out the mining method. For example, a weak roof and weak top coal in combination would make the Sublevel Caving by Pillar Extraction method unfeasible.

Environmental Impact

Underground coal mining can be quite advantageous environmentally. The single major effect is likely to be subsidence of the mined out areas which can be expected to occur unless large portions of the coal are left in place as massive pillars. As a greater volume of material is extracted by the Sublevel Caving method, subsidence can be expected to occur in greater degree. Some surface disturbance is expected to occur regardless of the depth of cover. In rugged topography, mining will shift the support balance of sandstone cliffs and cause them to slough for a period of time until a new support balance is developed. During this period rock will fall onto the talus surface. It may be necessary to repair and revegetate the areas where this occurs. It is possible that surface drainage patterns would be disturbed by the effects of subsidence and also that acquifiers would be disrupted.

The Sublevel Caving by Pillar Extraction anticipates removal of the majority of the coal seam. More complete removal of the coal may mitigate the increased subsidence somewhat by creating more uniform subsidence than would otherwise be the case. The most severe effects of subsidence would be expected to occur above the boundaries of mined and unmined coal, namely the junctures of the fire barriers and the mined out areas.

Most land under which Western coal fields lie is publicly owned and would come under federal and state jurisdictions regulating the effects of mining operations on the environment. The overall societal benefits of the energy provided from the coal and energy conservation effected by more complete recovery of the resources in a thick coal seam would have to be weighed against the damage of subsidence and the likely alternative uses of the land above the seam.

## DRILLING AND SHOOTING

Drilling and shooting of the top coal holes are two of the most critical operations of the Sublevel Caving by Pillar Extraction Method in regard to productivity, recovery and safety. Therefore, this subject has been the result of intensive and detailed study. A number of alternative procedures have been suggested and evaluated.

### Recommended Procedure

The drilling procedure recommended for the Sublevel Caving method is to drill 60-degree angled top coal holes to the top of the seam on 6 and 7 foot centers. An exception to the angled holes is one vertical row of holes at the opening to the crosscut and pillar pocket. This vertical row will trim the brow and maintain a safe and competent top coal roof in the room for subsequent extraction. This vertical row of holes also assists in balancing the coal burden between the room and pocket and crosscut shot holes.

Shot holes angled at 60 degrees can be drilled using two 6-foot 8-inch drill steels, stopping the drill once to add the second steel, to within 6 inches of the roof of a 20-foot seam. These holes are approximately 13 feet, 4 inches long. The angling of the holes will help to confine the coal fall within reach of the gathering arms and head of the continuous miner.

A very important safety advantage is realized by angling the top coal holes. The men operating the coal drill and loading the holes are removed from the dangerous immediate vicinity of the brow by the angled hole. The nearest approach to the brow or the pillar pocket break through by drillers and shot firers is 10 feet with this method. (See Figure 2-1) This safety improvement overcomes the most serious safety objection to the Sublevel Caving Method. With this hazard under control, the method now appears to offer general safety improvements over any previous underground thick seam mining method.

### Drill Hole Placement

In the pillar pocket and crosscuts, recommended spacing between the rows of 60-degree holes is 5 feet, 9 inches measured along the mine roof. This spacing gives a distance between the rows of 5 feet when measured normal to the holes. (See Figure 2-3) Within each row the distance between the two inside holes is 4 feet. The spacing between the inside sets and outside sets of holes is 5 feet, leaving a spacing of one foot between the outside drill holes and the rib. In the crosscut, the spacing

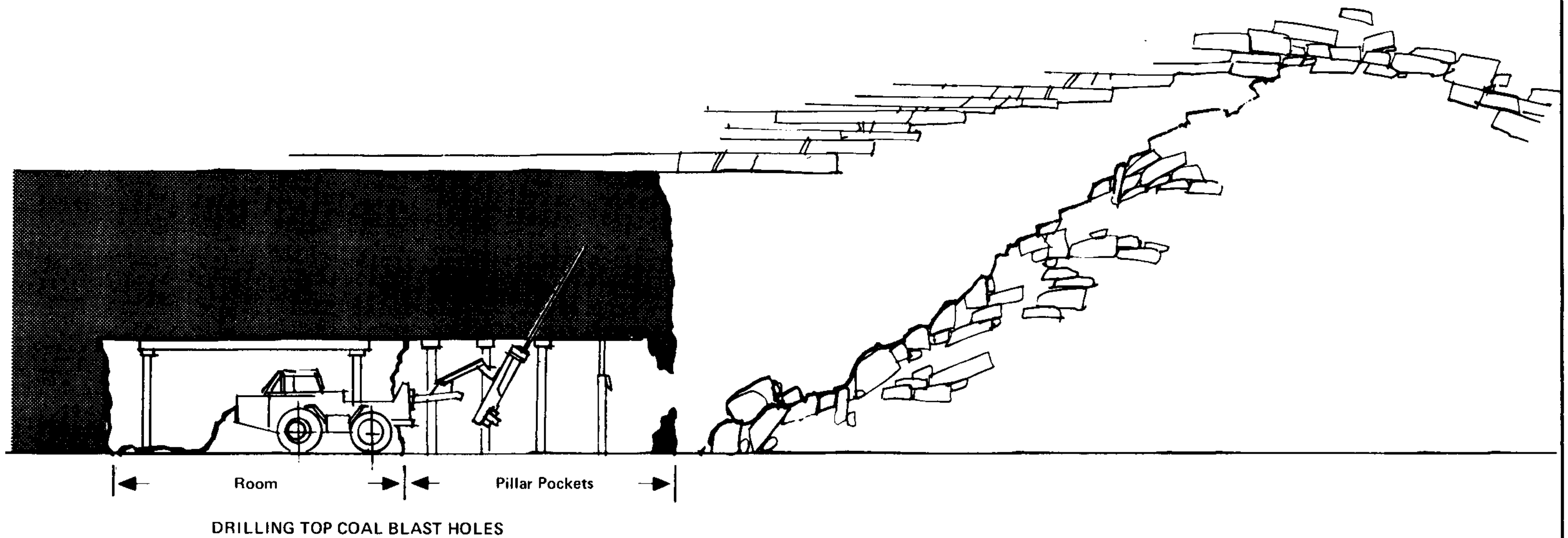



Fig. 2-1 Drilling top coal in pillar pocket
Section View

between the inside holes is 6 feet and between the inside and outside holes 5 feet. This scheme results in 88 blast holes over the room, pillar pockets and crosscut or the number associated with the extraction of one pillar. This number is to be compared with 128 holes required by the previous plan of spacing vertical holes on 4-foot centers. The fewer holes offer an increase in operating efficiency and a savings in explosive materials.

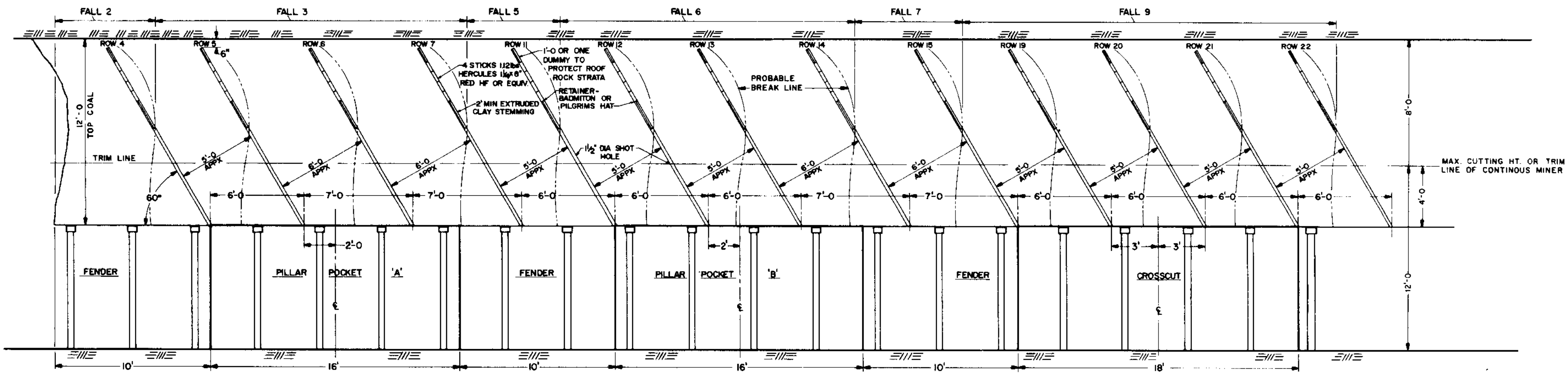
In the room being retreat mined, the spacing between the rows of top coal holes as measured along the mine roof varies between 6 and 7 feet. This variation is necessary in order to miss the crossbar roof supports. The rows of holes will be placed according to the plan shown in Drawing No. 8.

Drilling Procedure

Only drilling of the pillar pocket top coal requires high machine utilization and extraordinarily close supervision and well coordinated work by the crew. The pillar pocket must be drilled, the holes charged and shot in as rapid a sequence as possible to avoid delays to the continuous miner. (See Retreat Mining Cycles, Tables 3-4) Drilling of the top coal in the rooms and crosscuts can be accomplished on a much less stringent schedule in those periods of the mining cycle denoted as "auxilliary work".

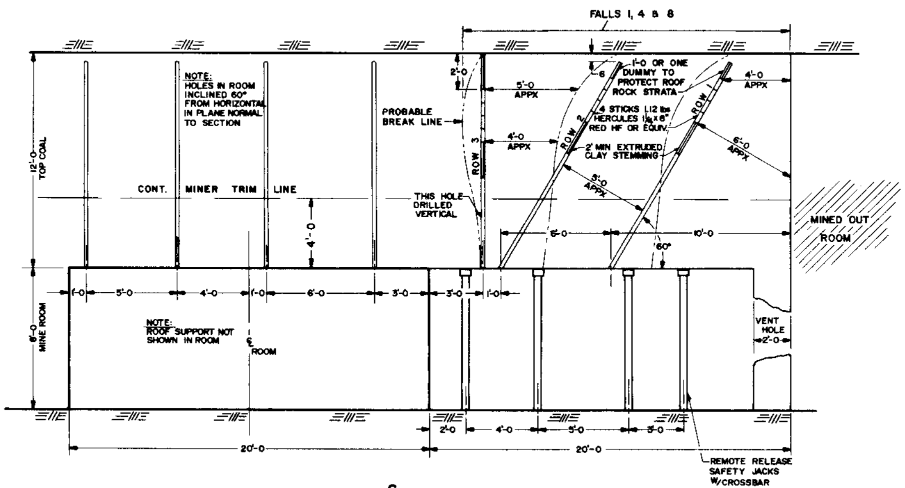
Before starting to drill top coal blast holes or any other work, the place would be checked for gas, the roof would be tested, loose coal taken down, and additional supports placed if required.

The certified shot firers operate the coal drill. While one man operates the machine, the other will stand by to perform the steel change quickly. It bears repetition that both the machine operator and the man performing the steel change will be back at least 10 feet from the brow and under primary supported roof at all times. The machine operator would also have the safety advantage of a protective cab. The hazard to which the drillers would be exposed is believed to be less than that incurred by other personnel in standard room and pillar mining. The continuous miner helper or a roof bolting crew are examples of more hazardous assignments.



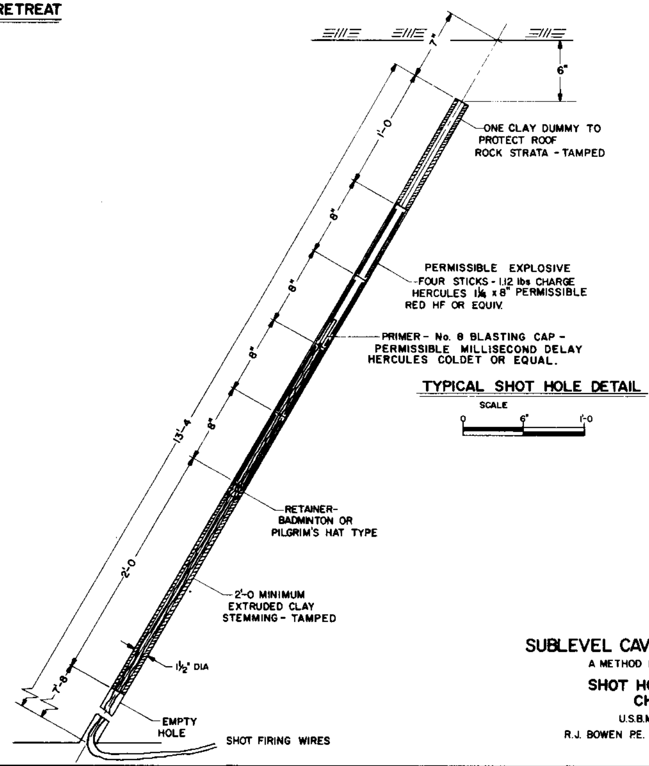

**SECTION @ ℄ ROOM ON RETREAT**

SCALE 0 3' 6'




**SECTION @ ℄ PILLAR POCKET OR CROSSCUT ON RETREAT**

SCALE 0 3' 6'




**TYPICAL SHOT HOLE DETAIL**

SCALE 0 6" 1'-0

## SUBLEVEL CAVING BY PILLAR EXTRACTION

A METHOD FOR MINING COAL SEAM 20' THICK

### SHOT HOLE LOCATIONS AND CHARGING DETAILS

U.S.B.M. CONTRACT No J0265009

R.J. BOWEN P.E. MINING ENGINEERING CONSULTANT

DWG 8

## TABLE 2 - 1

## DRILLING CYCLE

| FUNCTION | TIME |
|---|---|
| Position machine (spot hole) | 30 seconds |
| Drill 6 feet, 8 inches | 15 seconds |
| Retract steel. Add extension | 30 seconds |
| Drill 6 feet, 8 inches | 15 seconds |
| Retract steel. Remove extension | 30 seconds |
| TOTAL | 2 Minutes |

### Charging Procedure

Charging will be done by hand as mentioned previously. The shot firer would stand on a low bench and insert the cartridges into the hole. An alternative to using a bench would be to use a lightweight, U-shaped guide along which the cartridges can be pushed into the holes. A plastic retainer of the type known as a Pilgrim's Hat or the type known as a Badminton would be inserted in the hole immediately behind the last cartridge. The explosives train would then be pushed into the hole with the charging stick until the uppermost cartridge is in contact with the clay stemming. The plastic retainer will hold the cartridges in place until a minimum of 24 inches of clay stemming can be tamped against the retainer.

In charging each 1 1/2-inch diameter drill hole, one 16-inch stemming dummy of extruded clay will be placed at the end of the hole and tamped with a wooden charging stick. After tamping, the clay dummy will provide about 1 foot of stemming. In addition to the clay, there will remain about 7 inches of coal which will, together with the stemming, serve to reduce the shock of the explosion on the rock strata over the coal. Minimizing damage to the roof rock reduces dilution of the top coal by rock sloughage and also preserves the competency of the roof during loading of the fall of coal. Two stemming dummies will be tamped in against the top of vertical holes or holes which are drilled to the rock.

Each shot firer should be able to load shot holes independently and concurrently. After completion of loading of the row or rows of holes which constitute the fall of coal, the detonator wires would be connected in series in each row.

Should a break occur in the series connected row, however, a very serious condition may result. Four holes would misfire. Corrective work would probably not be possible due to the danger of working under the partially broken top coal. The entire fall of coal would have to be abandoned. If the top coal appears unusually bad, parallel connection of shot holes within the row could be applied. This would confine a misfire to one hole if one wire were broken. After connection of the rows is complete, the rows are then connected in parallel to the lead wires which are shunted. The shunt of course would remain in place until the shot is ready to be fired.

Removal of Support

After drilling, but before charging of the blast holes, the roof would be inspected and trimmed. After charging, further measures deemed necessary to protect the shot wires would be taken. Support would next be removed. After the support has been removed, three 20-ton jacks would be placed under the crossbar remaining nearest the coal being shot. These jacks and crossbar will then serve as a breaker row when the shot is fired. Subsequent to placement of the 20-ton jacks, the main shooting lead will be connected and strung to the location of the blasting machine.

The mining cycle estimated for "adverse" conditions allows time periods ranging between 5 and 30 minutes for removal of support. This period varies in accordance with the number and complexity of the support elements being taken out. In the case of a single crossbar and two support jacks, the jacks would be released from a position under supported roof and the jacks and crossbar dragged out of the way of the shot. This method is also believed to be applicable to the four crossbars and bridgebar across the crosscut and pocket openings. The jacks would be released and the crossbars and bridgebar allowed to collapse to the floor. If the pieces were connected to each other by chain, the entire set could then be dragged out of the way with a shuttle car, dismantled and moved or stored. If collapsing the bridgebar and crossbar at once from a remote position is not possible in practice, a more complicated and time-consuming procedure will be necessary. Jacks will have to be placed under the crossbar ends before the bridgebar is removed. The bridgebar is then taken out and each crossbar set is removed by collapsing it and dragging it out of the way. Placement of the four jacks needed would require approximately 2 minutes each. Removal of the bridgebar would require approximately 5 minutes. Removal of the four crossbar sets would take about 5 minutes each. Five minutes is allowed for difficulty encountered.

The breaker row of jacks is then placed and the shot fired. Ten minutes are estimated for these tasks.

Another ten minutes are allowed for smoke to clear and to sprinkle the fall of coal before loading commences. Smoke is expected to clear quickly into the return air stream flowing along the void between the pillar and the cave.

The breaker row of jacks will then be released remotely and dragged out of the path of the loading operation. If too much coal is piled around the breaker jacks it will be necessary to load some coal out before they are removed.

Figures 2-3 through 2-7 present plans on the drill holes, charges, order of firing, including specified millisecond delays, and the estimate of coal tonnages made available from each fall. Total coal made available from each fall is slightly more than the estimate of coal loaded out from each fall as indicated in Figures 2-3 through 2-9 and as shown on the Mining Cycle Charts.

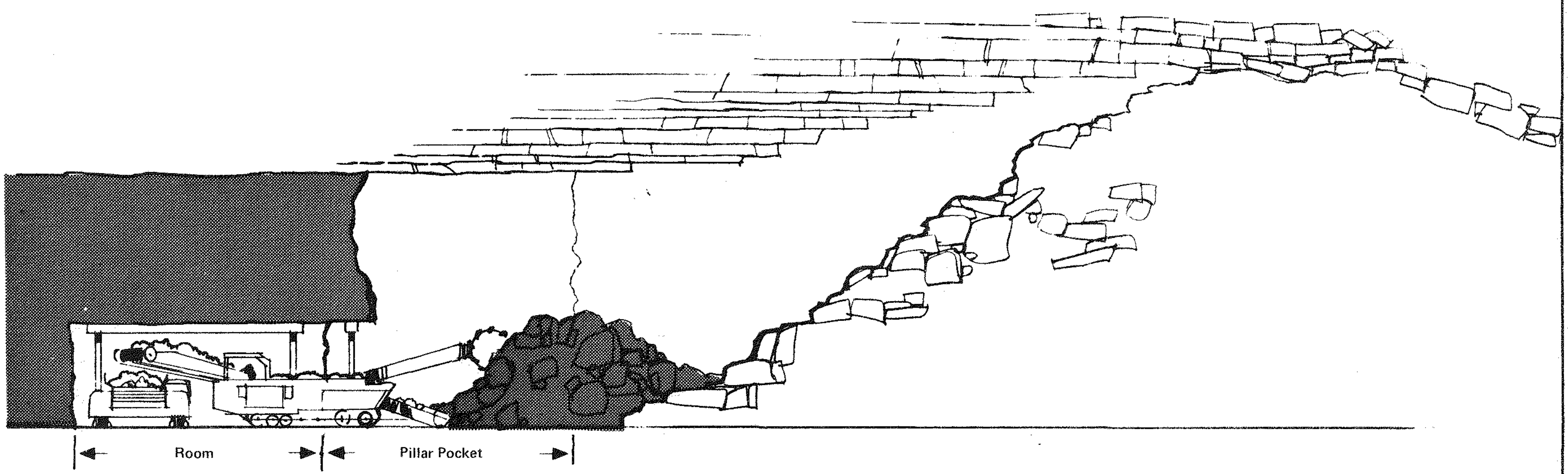
Sequence of Top Coal Extraction

In order to establish a brow for subsequent top coal recovery above the room, the top coal would be recovered from the last crosscut and across the end of the room. The brow across the room will then be in line with the ends of the pillar.

The sequence of extracting the top coal begins in Pillar Pocket A. The rows of holes designated as No.'s 1, 2, and 3 in pillar Pocket A are shot to create Fall No. 1 of top coal. Approximately 130 tons of coal will be loaded from this fall. (See Figure 2-3).

Shot No. 2 consists of one row of holes (Row No. 4) in the room opposite the outby fender of Pocket A. This fall produces approximately 50 tons of coal. (See Figure 2-4).

Fig. 2 - 2




LOADING FALLEN TOP COAL

FIGURE 2 - 3
PLAN AND SECTION VIEWS OF SHOT HOLES IN PILLAR POCKET REQUIRED BY FALLS NO. 1 AND 4

NOTE: Numbers by holes indicate rotation of firing by Hercules Coaldet millisecond delays or equivalent thereto. Charge for each hole consists of four 1 - 1/4 by 8 - inch sticks of Hercules RED HF permissable explosive or equivalent thereto. Series-parallel connection.

Estimated tons made available by these top coal falls - 144.

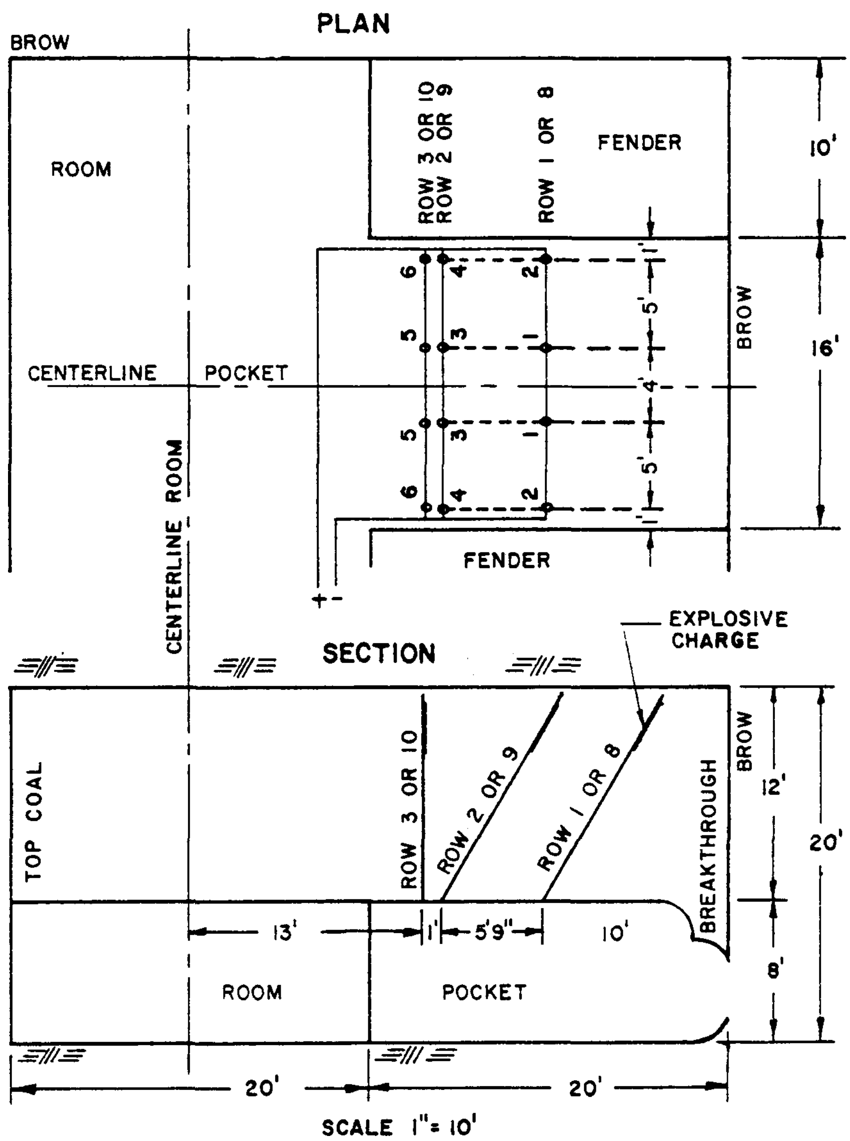
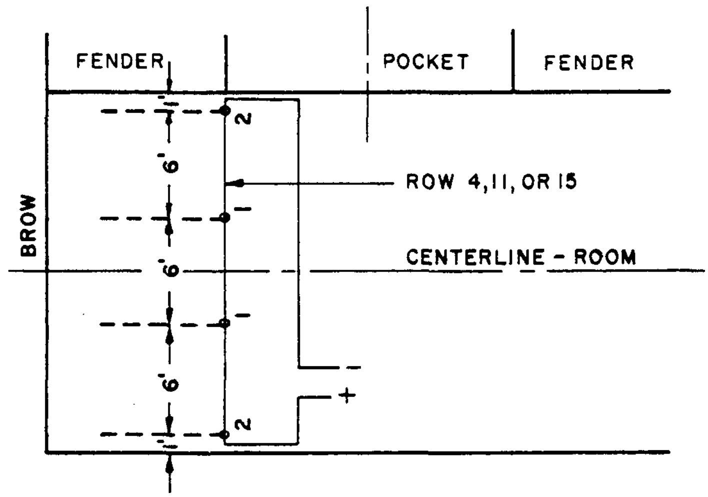
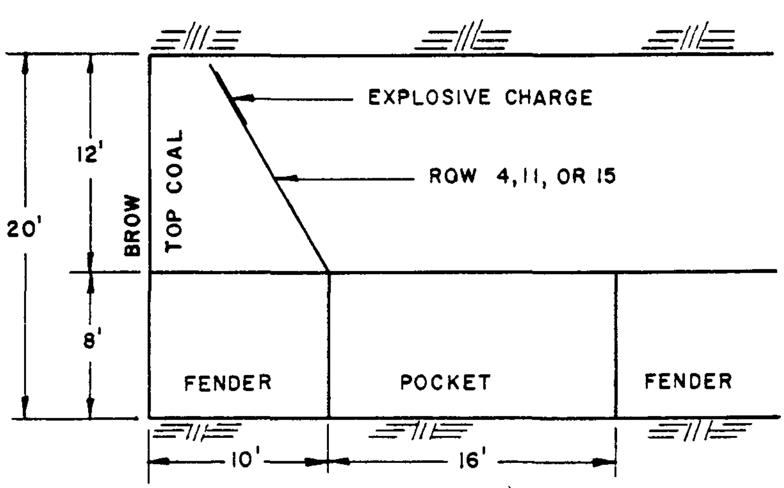


FIGURE 2 - 4
PLAN AND SECTION VIEWS OF SHOT HOLES IN ROOM
REQUIRED BY FALLS NO. 2, 5, AND 7

Estimated tons made available by these top coal falls - 58.



Shot No. 3 (rows of holes numbered 5, 6, and 7) breaks the top coal down in the room to a point near the inby edge of the entrance to Pocket A. Fall No. 3 produces approximately 190 tons of coal. Fall No. 3 will also recover the top coal left at the entrance to Pocket A. This coal is left for this fall to assist in stabilizing the top coal roof for charging and shooting of Fall No. 3. (See Figure 2-5).

Under "favorable" conditions recovery of as much coal as possible from the outby fender of Pocket A would be attempted at this point in the top coal extraction sequence. The continuous miner would be in the remote control mode for extraction of the fender. Under "adverse" conditions this probably would not be possible without unjustifiable risk of damage to the continous miner from falling roof rock.

Fall No. 4 is produced by firing Rows 8, 9, and 10 in Pocket B. The procedure for Fall No. 4 is identical to that for Fall No. 1. Again approximately 130 tons of coal will be loaded out. (See Figure 2-3).

Fall No. 5 (Row 11) is identical to Fall No. 2. Production is 50 tons. (See Figure 2-4).

Fall No. 6 (Rows 12, 13, and 14) duplicates Fall No. 3. Production is approximately 190 tons. (See Figure 2-5).

Under "favorable conditions" the inby fender of Pillar Pocket A would be mined at this point in the extraction sequence.

Fall No. 7 (Row 15) duplicates Falls No. 2 and 5. Production is again approximately 50 tons. (See Figure 2-4).

Fall No. 8 (Rows 16, 17, and 18) breaks down the top coal above the crosscut. Slightly more coal is available than from Fall No. 1 and Fall No. 4 due to the extra two feet of width in the crosscut as compared to the pillar pockets. Approximately 140 tons are produced. (See Figure 2-6).

Under "favorable" conditions the fender adjacent to the crosscut would be mined at this point in the top coal extraction sequence.

Fall No. 9 completes the extraction of top coal from the pillar pockets and crosscut associated with mining one pillar. Four rows of holes (19, 20, 21, and 22) are shot and approximately 220 tons of coal

FIGURE 2 - 5
PLAN AND SECTION VIEWS OF SHOT HOLES IN ROOM
REQUIRED BY FALLS NO. 3 AND 6

Estimated tons made available by these top coal falls - 209.

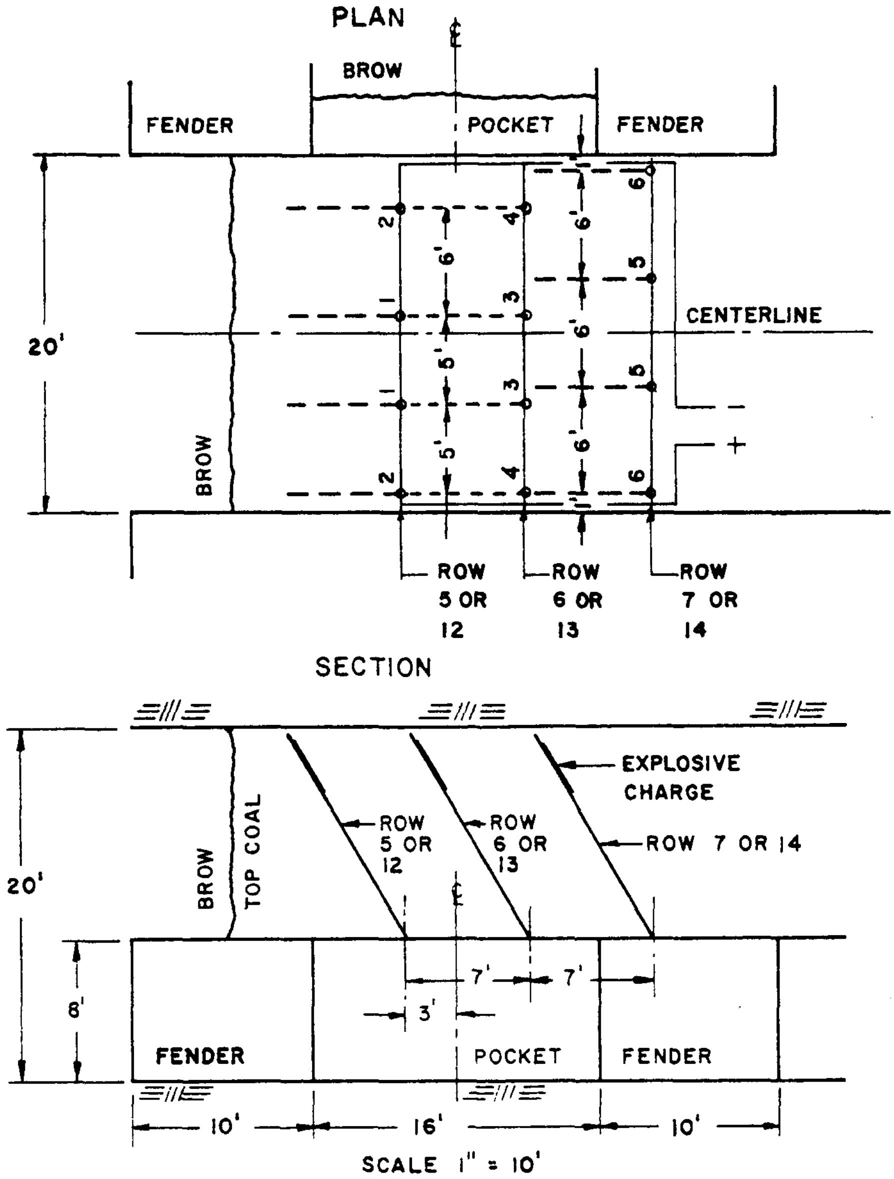
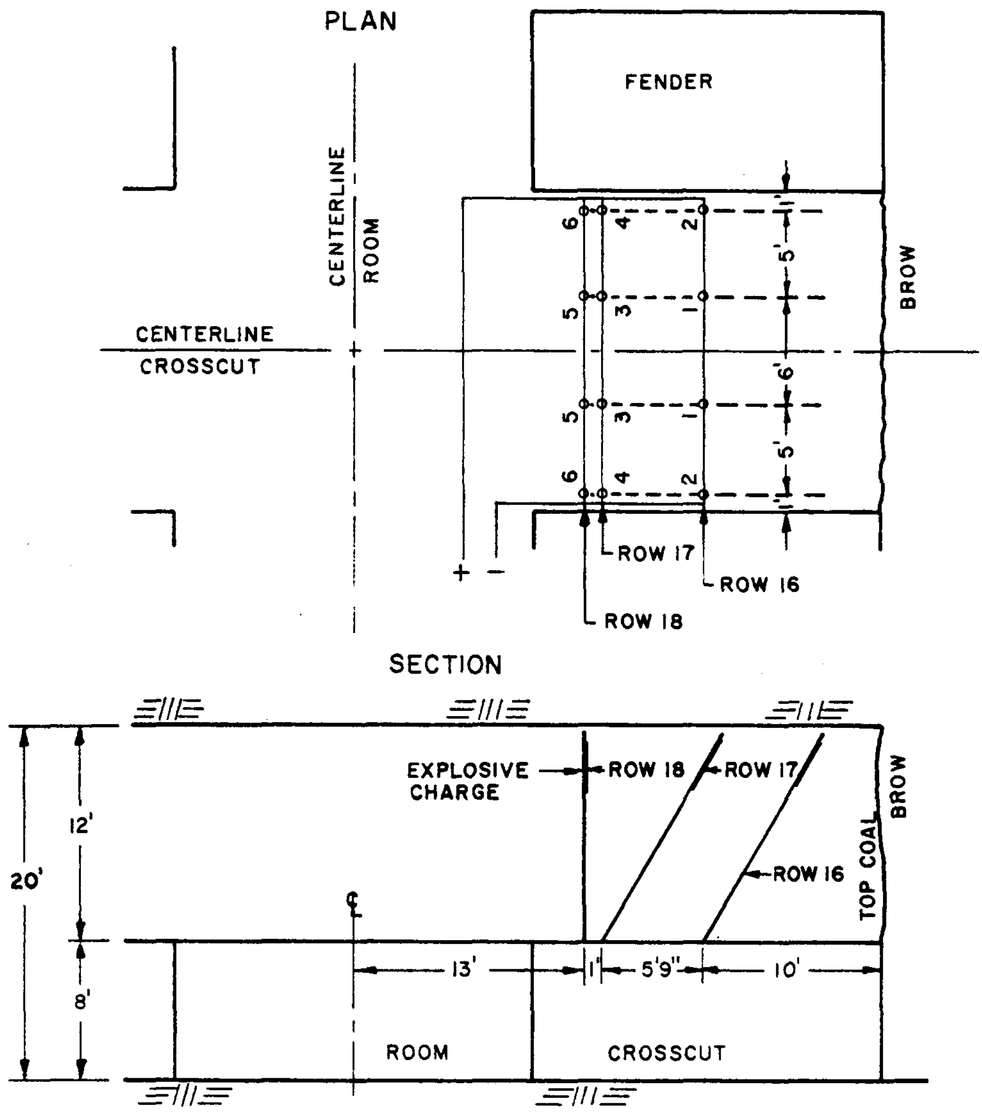


FIGURE 2 - 6
PLAN AND SECTION VIEWS OF SHOT HOLES IN ROOM
REQUIRED BY FALL NO. 8

Estimated tons made available by this fall of top coal - 156.




SCALE 1" = 10'

will be loaded out. (See Figure 2-7).

Explosives

As part of the attempt to derive a scheme for a remote machine loading of blast holes for this mining method, a thorough survey of permissable explosives was undertaken and various options including water gel explosives discussed with manufacturer's representatives and with blasting experts from MESA. The conclusions of these discussions and study were that nitroglycerin-based explosives were most suitable for this method due to their slower detonation speeds and the fact that machine loading, to which water gels are easily adapted, was found to be unfeasable.

The powder train recommended will normally consist of four 1 1/4-inch by 8-inch cartridges of Hercules Red HF permissable explosive of the equivalent thereto. In the case of a 20-foot seam, this charge would average 1.12 pounds. The number of cartridges could vary between 3 and 5 (0.84 pound charge and 1.4 pound charge) depending on the hardness of the coal, the amount of boney, the presence or absence of cleavage planes and other variables. In seams thicker than 20 feet, the charge could be as much as 3 pounds per blast hole.

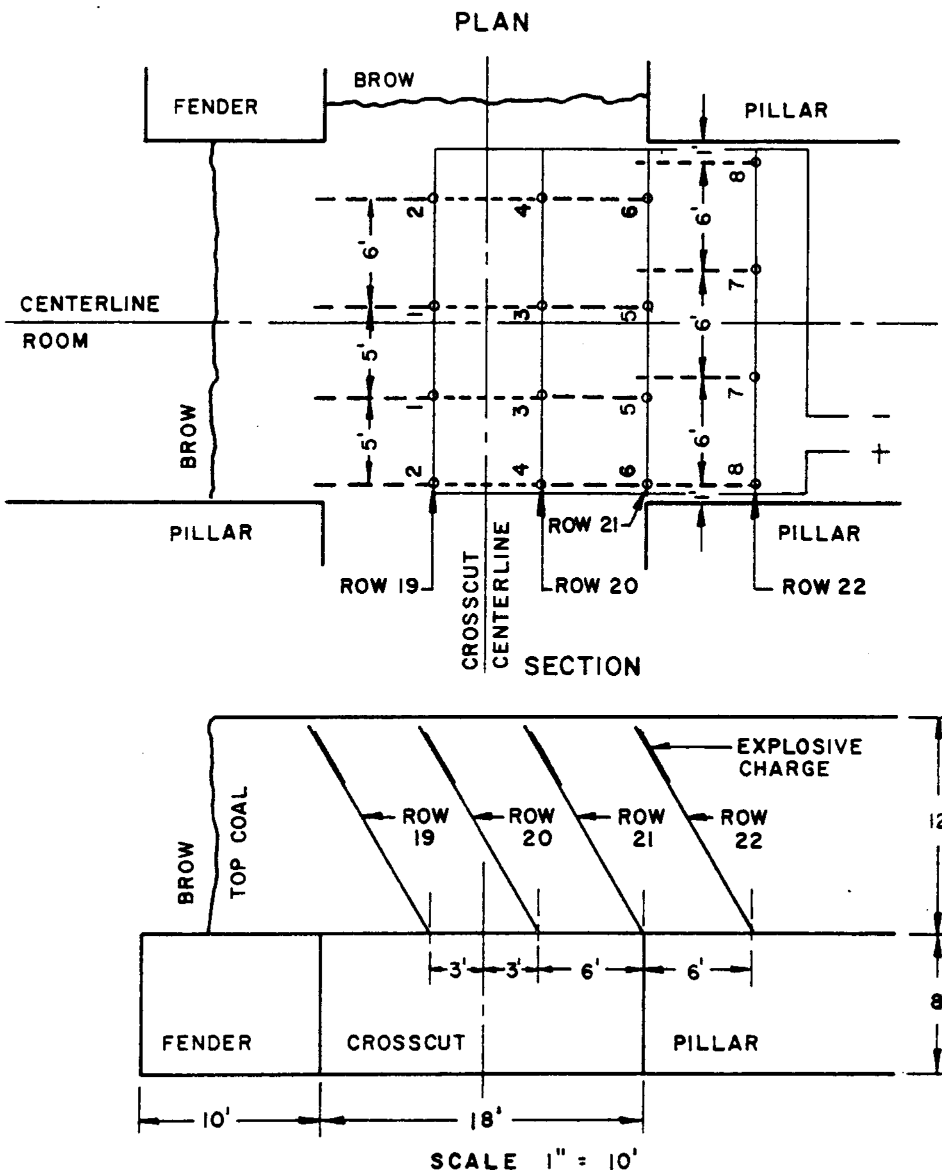
The center cartridge would contain the detonator. The detonator would be a No. 8 electric blasting cap timed by a millisecond delay appropriate to the order of firing of the holes comprising the shot. The order of firing of each hole is shown in Figures 2-3 through 2-7.

The burden of coal on the explosive will range between a minimum of 4 feet and a maximum of 7 feet. The estimated optimum burden is 5 feet. Under "favorable conditions" a uniform spacing along the mine roof of 7 feet between rows of holes will approximate the 5 foot optimum burden. Spacing between blast holes within the row is 6 feet in the room. All blast hole locations in the room are referenced from the centerlines of the rooms and crosscuts. (See Drawing No. 8).

Actual practice and on-the-spot judgement will be the best determinant of the most effective blasting charge given the characteristics of the individual mine. The ideal shot would merely move the top coal down into the mine opening without disturbing the rock strata overlying the coal seam.

FIGURE 2 - 7
PLAN AND SECTION VIEWS OF SHOT HOLES IN ROOM
REQUIRED BY FALL NO. 9

Estimated tons made available by the fall of top coal - 247



## Drilling Machinery

The drilling of the top coal holes is recommended to be accomplished with a boom-mounted-mast-type roof bolting machine such as the JOY RBD8B or the EIMCO-SECOMA 350B-2G. Both of these machines are capable of drilling 60 degree and 90 degree top coal holes.

The roof bolters would be modified to provide fast penetration rate in coal. The modification required is a minor one which increases the drill rotation from 300-400 rpm to 1,000-1,200 rpm. These drill speeds provide a coal penetration rate of approximately 27 feet per minute.

Both machines listed above permit use of a 6-foot, 8-inch drill steel under the 8-foot roof. Both vertical holes and 60 degree angled holes can be drilled with one steel change.

The boom mounted, mast type, rotary top coal drill is a very flexible machine and can be used for other work including horizontal blast holes which on ocassion may be required. It may also be used to drill holes for wooden rib pins if they are needed.

The EIMCO-SECOMA 350B-2G offers the advantage of a maximum bolting height of 20 feet, 6 inches. With this machine, bolting of the roof rock would be performed when necessary to stabilize the roof rock. This task has not been included in the mining cycle because its necessity would be very infrequent.

The coal drill trailing cable would be supported against the roof at intersections of shuttlecar roadways. These machines would be equipped and maintained to comply with MESA standards of safety including permissibility, lighting, cabs, canopy, noise, and fire suppression.

## Drill Dust Control

Dust control on the recommended drilling machines would be accomplished by water injected through the drill steel. Dust is then allayed at the point of origin and carried away with the slurry of cuttings. This measure will significantly reduce float dust and leave a clean blast hole. Routine cleaning will handle the slurry and dust before it dries.

An alternate method of coal drill dust control is by the enlarged or double vacuum system in which the cuttings and dust are pulled through the drill steel into bags which filter out the dust before the air is returned to the mine atmosphere. The vacuum bags are emptied, sprinkled, and loaded out as sludge into shuttlecars.

A third method would be to have sprays impinging on the stream of dust and cuttings at the collar as the hole is drilled.

The first, or water-on-the-bit, method of allaying the dust does not require dust sampling. The latter two methods are more erratic and do require dust sampling of the atmosphere.

Developments are presently under way which will enhance the safety and efficiency of top coal drilling in the future. The flexible shaft drill and the automatic steel change machine both appear promising for this task. Application of these improved machines would permit all top coal drilling to be accomplished by one man from within a protective cab. This would free one man to perform other tasks within earshot.

Procedures to Prevent Broken Shot Wires

The probability of shot wires being broken by coal sloughing from the roof is greatest during the interval of time between withdrawal of the roof supports and firing of the shot. In the instance of withdrawing one roof support set and shooting one row of holes, this period of time would be approximately 15 minutes. When four support sets and a bridgebar are withdrawn and four rows of holes are shot, this period would be expected to be 40 minutes.

To guard against this possibility, the following measures are generally recommended:

1. Prior to loading, the top coal would be carefully scaled and a judgement made as to its reliability after support withdrawal. If the coal is judged on the scene to be reliable, then no deterrent other than diligent observation of the top coal and the connecting wires would be needed.

2. If the coal is weak or friable, then wooded props would be left to help support the roof before the shot. These props would be destroyed and would encumber the loading operation.

3. When the connections in each row of holes are made, the wires should be trimmed and a tight connection made so the wires will be held against the roof instead of dangling.

4. Coal nails can be used to hold the wires tightly against the roof. Loose wires are much more vulnerable to falling coal or being caught by roof support during removal.

5. The lead wires forming the parallel connections to the rows of holes can be protected by fastening each lead against the roof near the rib line.

Various alternative measures have been discussed to further reduce the frequency of broken shot wires under unreliable top coal. These are:

1. Nailing a 1/2-inch plank to the top coal with the shot wires above it.

2. Half sections of plastic pipe nailed to the roof.

3. An expendable plastic netting fixed to the roof to contain small pieces of coal.

4. Parallel connection of all shot wires.

5. Redundant wiring so that no single broken wire will cause a misfire.

These alternative measures are time consuming and cumbersome. They may in infrequent cases be helpful, but are not deemed to be sufficiently useful to be recommended generally. Improvements in blasting techniques such as wireless shooting or constant monitoring of lead continuity would greatly reduce or eliminate this problem.

Until general improvements in blasting techniques are in practice, nothing will take the place of care and diligence in scaling of the coal and observation until the last possible moment of the lead wires. If care and vigilance are applied, the frequency of misfires due to broken shot wires should be very low even with unreliable top coal.

## Drilling and Shooting for Seams Thicker Than Twenty-five Feet

The maximum seam thickness to which the recommended drilling and shooting procedure can be applied is believed to be 25 feet. The Sublevel Caving Method is believed to be feasible, however in seams up to 40 feet thick.

In seams ranging in thickness from 25 to 33 feet, the alternative drilling procedure would be used which calls for shorter holes interspersed between the holes drilled to the top of the seam. The holes drilled to the top of the seam would range from 19 to 29 feet long. The holes would be loaded with 8 or 9 sticks of powder (2.24 to 2.52 pounds of explosive). The intermediate holes would range from 10 to 18 feet and would be loaded with charges ranging from 3 sticks of powder to 8 sticks (.84 to 2.24 pounds of explosives).

For seams of thickness ranging from 33 feet to 40 feet (25 feet to 32 feet of top coal), holes would be drilled varying depths into the top coal in a manner such that explosive charges would be placed at vertical intervals of 12 feet starting at the top of the seam down toward the roof. Each of these charges would be about 6 feet back from the brow.

Each hole would be charged with 4 sticks of powder. The charge nearest the bottom would be shot first and the charge nearest the top last.

As seam thickness increases the pile of broken top coal from each increment shot increases. So, too, does the amount of coal which would be expected to be lost due to cascading into the gob. This would be mitigated somewhat by shooting narrower increments (as measured along the roof line). This would permit the continuous miner to extend farther into the void. The size of the fenders would need to be increased again creating more loss.

Recommended pillar, room and pillar pocket sizes would have to be adjusted for thicker seams.

## SAFETY, HEALTH AND MINE ENVIRONMENT

The design philosophy of the Sublevel Caving by Pillar Extraction concept is predicated on the "Safety First" proposition. All aspects of the design have incorporated safety considerations and no element of the design has been considered feasible unless it could at least meet present safety standards. A consistent effort has been made to improve on present safety practices whenever possible.

### Ground Control

The ground control method of the Sublevel Caving by Pillar Extraction method is intended to improve safety when mining under heavy cover by minimizing roof falls and bumps which occur when pressure accumulates over mine openings. This is intended to be accomplished by maintaining the cave-to-solid coal block distance as small as possible. In the mining sections this is done by keeping advance and retreat mining in balance and by limiting the number of rooms to three. One room is under retreat mining; one room is to provide an intake escapeway and one room is on advance mining. Small pillars which yield and transfer weight to the solid coal are employed. In the section entries, this objective is proposed to be achieved by limiting the number of entries to two and by utilizing the small, yielding pillar concept.

In the main entries of the mine, which must stand for periods of several years to as much as forty years, the protection against roof falls and bumps is provided by massive main entry pillars and massive barrier pillars judged to be sufficiently strong to safely contain weight accumulations. The ground control method of the Sublevel Caving by Pillar Extraction concept is believed by the writers to be one of the strong points of the concept in regard to intrinsic safety.

Also, by limiting the number of rooms in the mining section to three, the amount of time the section pillar must stand before it is extracted is substantially reduced. This adds to the basic safety of the mining concept.

### Mining Height

Driving the openings along the bottom of the seam and limiting the advance mining height to eight feet is another basic design element of the Sublevel Caving method which is largely intended to increase the safety of the mining operation. Ribs eight feet high are much less likely to slough than are higher ribs. Also, if sloughing does occur, it cannot fall across the entire width of the opening trapping a miner. The eight-foot mining height facilitates installation of roof support and makes the support more effective. Falls from benches needed to install ventilation materials along high ribs are avoided. Strain injuries due to working with materials high overhead are also mitigated.

## Roof Support

Closely associated with the ground control method and mining height, from the standpoint of safety, is the roof support method. Effective roof support will help achieve the same safety objectives as will good ground control. The roof support method must meet a number of criteria; it must be strong enough to provide reliable support; it must be rugged enough to stand up under the rather severe abuse of recovery and reinstallation; it must permit rapid installation and be simple enough that errors in installation are not common. It must comply with MESA and state regulations. The support system must be light enough to allow convenient, safe handling. The support system must be compatible with the mining machinery utilized and with the various mining operations. It must also be sufficiently flexible to allow variation in its application due to localized variations in roof conditions.

The roof support systems which best meet the criteria outlined for Sublevel Caving by Pillar Extraction consist of crossbars and posts for "adverse conditions" and a single row of props for "favorable conditions". Under "adverse conditions", wooden crossbars supported by timber posts are to be installed in the main entries and the section entries. Metal crossbars supported by hydraulic jacks provide support in the section rooms, crosscuts and pillar pockets.

This roof support system should meet the important safety requirement of having all personnel remain under primary supported roof at all times. There should be no ordinary reason for varying from strict adherence to this rule. Its strict enforcement should not encumber the mining operation and should increase safety awareness.

An important feature of the roof support scheme for top coal extraction is that the sequence of top coal falls is arranged so that the top coal roof always bridges from the pillar rib to the rib of the fender or to the rib of the next pillar to be extracted. This measure helps stabilize the top coal roof and reduces dangerous sloughage while men are engaged in work preparatory to shooting of the next fall or during loading of already fallen top coal. (See Drawing No. 4.)

## Roof Support Installation - "Adverse Conditions"

Installation of roof support is recognized as one of the more hazardous tasks required in underground coal mining--a fact borne out by numerous studies and analyses of injury cause and frequency over many years. As shown in the Mining Cycle Charts, roof support is installed with the continuous miner remaining in the face area. This procedure enhances safety by providing support quickly, before the roof starts to sag. Thereby roof competency is preserved. This procedure also increases productivity as well. The procedure for installing one metal crossbar support set is described in the following steps:

1. The continuous miner stops after having advanced the face approximately 4 feet and positions the cutter head for placement of the crossbar.

2. Two men, the continuous miner helper and the timber and utility man, bring the crossbar up alongside the miner and place it in the saddles specially mounted on the cutter head frame.

3. The continuous miner operator would then raise the cutter head, center the bar and place it against the roof.

4. The continuous miner helper and the timber and utility man would step out under the support provided by the bar and the continuous miner, place a 10-ton capacity hydraulic jack under each end of the crossbar and raise the jacks completing the primary roof support set.

5. The men would step back in the clear and the miner would resume operation for another 4 feet of advance after which the next crossbar set would be installed in the same manner.

This procedure is estimated to take between 5 and 10 minutes to complete.

This same procedure would be applied to supporting the roof in the entries under "adverse conditions". Round wooden timbers 16 feet in length would take the place of the metal crossbar and round wooden posts would be installed to support the crossbar.

Roof Support Materials

Studies were made to determine the most suitable crossbars and supports for the section rooms, crosscuts and pillar pockets where rapidity of installation, ease of removal and the fact that the support would remain in place for only a short time are prime considerations. Another important factor is avoiding backstrain injuries to the personnel installing the support. Ease of placement against the roof is also important. For example, I-beams are difficult to keep from turning when placed against an uneven roof. In light of these considerations, high-strength steel beams with 4-inch wide flanges were compared with a built-up 4-inch by 6-inch by ½-inch aluminum box section beam for crossbars with spans of 14, 16, and 18 feet. The comparative yield strengths of these bars with the load concentrated at the center are shown in the table below.

TABLE 2-2

COMPARISON OF CROSSBAR STRENGTHS

| Shape | WT/FT in lbs. | Section Modulus. Cubic Inches | Material | Working Stress KSI | Beam Strength Weight in lbs. 14' | 16' | 18' |
|---|---|---|---|---|---|---|---|
| 4WF | 13.0 | 5.2 | USS X ten 50 | 50 | 13,350 / 182 | 11,600 / 208 | 10,200 / 234 |
| | | | T1 Steel | 100 | 26,700 / 182 | 23,200 / 208 | 20,400 / 234 |
| 4x6x$\frac{1}{2}$ Al. Box | 6.7 | 19.6 | 5083 Al. | 15 | 15,000 / 94 | 13,000 / 107 | 11,500 / 120 |
| 4x6x$\frac{1}{4}$ Steel Box | 17.0 | 12 | T1Steel | 100 | 61,500 / 238 | 53,300 / 272 | 47,000 / 306 |

NOTES:
1. 100% yield strength used for steel, approximately 90% yield strength used for aluminum to account for weld.
2. Strengths calculated for concentrated load capacity at center.

It is to be noted that the 5083 aluminum crossbar is somewhat stronger than the 4" wide flange bar made of X-Ten-50 steel, but not as strong as the crossbar made of T1 steel. There is a decided weight advantage with the aluminum beam. The 16-foot aluminum beam weighs 107 pounds as compared to 208 pounds for the 4-inch by 4-inch wide flange steel bar. The shape of the aluminum box section makes it easier to handle when placing it against the roof. The aluminum bar has a bearing area against the roof of 8 square feet as compared to 5.28 square feet for the steel I-beam.

The main disadvantage to the aluminum crossbar appears to be cost. A 16-foot aluminum box section beam will cost about $200 compared to about $41 for a bar made of X-Ten-50 steel and about $67 for the T1 bar. The number of bars needed for one 400-foot room and 5 crosscuts is 124. The respective material costs are:

| | |
|---|---|
| X Ten 50 Steel | $5,084 |
| T1 Steel | $8,308 |
| 5083 Aluminum | $24,800 |

## Support at Intersections, Crosscuts and Pillar Pocket Openings

A bridge or carrier bar 18 feet in length will support the roof at all intersections. It is recommended that the bar be made of a 4-inch by 6-inch by 1/4-inch box section of T1 steel. This beam can carry a maximum concentrated load of 47,000 pounds and weighs 306 pounds. It would be supported by a 20-ton-capacity jack at each end. Four aluminum crossbars would rest on the bridgebar with the other end near the pillar rib supported by a 10-ton jack. The bridgebar would be lifted into place by the utility front-end loader and held against the bars while the jacks are installed under each end.

## Support Within Rooms, Crosscuts and Pillar Pockets

Under "adverse conditions" the support system described above will be installed on 4-foot centers in the section rooms, the crosscuts and the pillar pockets. The rooms and crosscuts will require 16-foot crossbars and the pillar pockets will require 14-foot crossbars . In the pillar pocket three 14-foot crossbars will be installed and four safety jacks would be installed beyond the last crossbar. This intense support is intended to protect the driller-shooters while they are working in the pillar pocket. (See Drawing No. 5.)

## Roof Support for "Favorable Conditions"

As previously stated "favorable conditions" are those under which the top coal roof has been judged to be reliably self-supporting. Accordingly, a much less intense support system would be required. This would be comprised of a single row of props on 6-foot centers set 3 feet out from the rib opposite the miner operator's cab. These props would be set 10 feet ahead of the miner operator's cab during advance mining. They would be installed with the continuous miner remaining in the face area. In section entries under "favorable conditions", the same method would apply. The props would be withdrawn remotely as the top coal is mined. (See Drawing No. 6.)

## Retreat Mining of Chain Pillars and Top Coal Over Section Entries

Special attention is called to the fact that the front abutment pressure zone may throw weight on the section entries. This intense stress zone may be 300 to 350 feet out from the cave. The yielding of the chain pillars may not be sufficient to cause all of the abutment pressure to override onto the solid coal. Should this occur, the competency of the top coal roof in the section entries could be impaired, making recovery more difficult and uncertain. It would be necessary to bar down loose coal, to install additional crossbar sets and to replace broken timbers. In retreat mining under these circumstances, the judgement of the foreman would determine which, if any, of the crossbar sets should be removed prior to shooting a fall of top coal.

## Hazards Associated with the Top Coal Brow

It has been recognized from the outset that the most serious safety detriment unique to the Sublevel Caving by Pillar Extraction method is imposed by the top coal brow. Minimizing this hazard has been a primary objective. The roof rock may break and cause a very dangerous cascade of rock into the working area. Gas may accumulate in the void beyond the brow near the roof rock where it is difficult to detect. The top coal near the brow or comprising the top coal brow may have been weakened by previous blasting or may have been inherently weak and prone to fall. Recovery of the fenders beyond the brow cannot be accomplished except with the continuous miner in remote control. Hazards to the machine may prevent even this.

Numerous measures have been suggested to deal with this problem safely. The machines suggested for this method are all to be equipped with safety cabs which will enhance general safety, but are especially necessary when operating near the brow. The continuous miner is to be equipped with remote control in order to perform some loading tasks with the operator's station beyond the brow. Blast holes are to be angled at 60 degrees in order to remove the driller-shot firers from the immediate vicinity of the brow. Approach to the brow will be strictly limited and rigidly enforced. The pillar pockets will have only a small breakthrough to the void for ventilation purposes. The remaining coal at the end of the pocket (about 2 feet) will be left to provide a shield against falling roof rock. A minimum of explosive is used to reduce damage to the roof rock and to the remaining top coal.

Gas Test

The brow imposes a difficulty in testing for gas at the top of the void. A test for gas in this area can be made with a National Mine Service G-70ER 20-foot Telescoping Methanometer Probe System. This probe allows the tester to remain under supported roof. An angled hole would be drilled through the brow specifically for the purpose of testing the gas at the top of the void. The telescoping methanometer would be inserted along this hole into the void near the top. This procedure would permit the tester to remain well back from the hazard of the brow and under primary supported roof. Seam thicknesses greater than 20 feet may require modification of the methanometer probe to accomplish this test.

Breaker Rows

Immediately before each shot, a crossbar set (supported by two 10-ton jacks at each end and three 20-ton jacks in the middle) would be set between the last row of charged holes and the first row of holes for the next fall. The crossbar and jacks would serve as a breaker row and would reinforce the top coal roof in the vicinity of the newly formed brow. The 20-ton jacks would be removed to facilitate loading, but the crossbar would remain to provide roof support during loading and during charging of the next row of holes. The breaker row would be installed just before the lead wires are connected for firing the shot. After the shot the 20-ton jacks would be released remotely and dragged out of the way with a shuttle car, or with the continuous miner. Additional breaker rows would be installed and left in place as barriers at the brow formed by previous retreat to an intersection. (See Drawing No. 5, Plan of Roof Support.) The breaker row would assist in this case to contain cascading roof rock.

## Hazards Associated with Restricted Work Places

The limited number of work places determined by this concept are essential to the ground control method which is believed to offer substantial safety benefits. A concurrent disadvantage results from this strategy, however. The number of places in which mobile machines can travel is likewise reduced and the support system selected further reduces the effective travel space. Travel clearance is also aggravated by the handling and storage of support materials. These circumstances cause danger to walking personnel who may be struck by or trapped against a rib or another piece of equipment.

Five mobile machines must perform their respective functions in the space allotted. The two shuttle cars would be grouped with the continuous miner when the miner is operating and would also occupy two of the three rooms available in order to maintain the shuttle car changeout point close to the miner. These two rooms and the portion of the entries close to the feeder breaker would have intense traffic. Everyone within the section should be aware of these roadways and alerted to the shuttle car motions and the routes taken by each car between the miner and the belt feeder breaker. Also anyone approaching a check curtain or roadway door should do so with caution and be certain that a shuttle car is not approaching it from the opposite side. The operator of the car should sound an audible alarm when starting up, going around a pillar, or through a check curtain.

The restricted travel space would also cause a likelihood that the posts and hydraulic jacks supporting the crossbars would be struck by machines causing them to fall. This results in hazard to anyone in the vicinity of the falling support elements and may also result in deterioration of the roof due to the sudden removal of support. As the crossbar is planned to be connected to the jacks or posts to facilitate remote removal, it may be sufficient to fix the bars to the roof with coal screws and short pieces of rope in order to prevent their falling when the post or jack is struck by a machine. This work could be performed in those periods of the mining cycle denoted as "auxilliary work".

## Hazards Associated with Machinery and Machinery Modifications for Safety

The most critical machinery hazard of the Sublevel Caving by Pillar Extraction method has been discussed under Hazards Associated with Restricted Work Places. There remain hazards to personnel whenever miners are working in the vicinity of powerful machinery with many moving parts. Records show the roof bolter occupation to have the highest accident frequency in the industry. The main sources of injuries to roof bolters are roof falls and machine manipulation injuries including changing of steel, handling bolts, etc. The proposed concept does not use roof bolting as a means of support. However there is no room for complacency because a roof bolting machine is used to drill top coal shot holes. And at the present time a man is still required to make a drill steel change to reach the required depth. Both operators, however, are under primary supported roof. One man is still vulnerable to injuries associated with a working machine, including manipulation, rotating parts and the handling of drill steels, etc.

To guard against this type of hazard, two alternative machines are suggested. The flexible shaft drill, now in its final stages of field testing, has the capability to drill the top coal holes with only one man required. The operator would remain in his protective cab and drill the holes with ease and at a faster rate than can be accomplished with a machine that requires a steel change. The prototype machine has the capability to drill a shot hole 16 feet deep. The "dump" motion of the drilling unit allows an inclination of 75 degrees from the horizontal, or a total variation of 30 degrees. A minor modification would permit the machine to drill a vertical hole and holes at 60 degrees from the horizontal. The machine will be available in the near future.

The second machine alternative suggested to improve the safety of the roof drilling operation is a modification of the Eimco Secoma roof bolter to achieve automatic drill steel changes. An inquiry has been made with the Secoma Division of Eimco based in France to determine if this can be done. If this capability is available, one operator remaining within his steel cab could drill the blast holes as described above. This capability could make the top coal drilling phase of this method practically free from the two sources of injuries mentioned above, namely roof falls and moving machine parts. The Secoma machine could also drill the holes required in seams thicker than 25 feet.

As mentioned previously, all mobile machines would be equipped with protective cabs as a safety measure. Cabs would protect five of the nine man crew at least part of the time. This would include the shuttle car operators, the continuous miner operator, the coal drill operator and the man operating the front-end loader. The protective operator's cabs are an essential safety device and reduce greatly the frequency of "struck by", "struck against", and "caught between" injury types. The mobile machines would be equipped as well with other, more or less standard, safety features related to fire protection, electric shock, and protection against injury from moving parts.

As has been discussed previously, the continuous miner would be equipped with remote controls to facilitate work in the immediate vicinity of or beyond the top coal brow.

## Spontaneous Combustion

In thick coal where operating openings are driven in the upper part of the seam, spontaneous combustion is more prone to occur due to accumulations of coal on coal which cause slow eddy currents of air and set the stage for heating. This incipient accumulation of heat is also more apt to occur in places where the coal floor heaves. Heat build-up spots are also more frequent where moisture occurs along with sloughing ribs and a heaving coal floor. Spontaneous fires are more frequent in thick seam mines where the openings are extensive in the upper part of the thick seam. In such a mine, the spontaneous heating problem becomes more intense where the earth temperature is elevated especially in the proximity of retreat mining districts where the caving action aggravates floor heave and rib sloughage.

Openings driven against the bottom of the seam eliminate the coal floor as a source of fuel. The inert floor also reduces heat build-up in piles of coal. The effect of height and the proposed layout will reduce the piles of coal in which heating can occur. Also the mining plan will isolate loose coal in the cave area where the oxygen content will diminish below the combustible limit before heat build-up can reach ignition temperatures.

The design of the concept facilitates combating spontaneous combustion. Good fire bossing, alertness of supervisors and crew for early detection, removing of loose coal, and hot spots are also necessary to prevent spontaneous combustion. Water is piped throughout the mine and as mentioned fire connections are provided at every other crosscut. Of course, small fires can be put out with a fire hose. Large fires must be extinguished by sealing or flooding.

## Miscellaneous Safety Measures

The achievement of a safe mining operation under this method will involve many additional safety measures which may be considered as standard practice. These would include, but are not limited to:

1. Electric Power Distribution. An intrinsically safe mine power distribution system is fundamental to avoid fires and injuries from electric shock. The power system should be designed by an electrical engineer familiar with mining requirements. Details would be left to the design engineer, but some of the

overall parameters for the hypothetical mine would include:

a. A 5000 KVA outside substation to step the utility voltage down to 7200 volts for underground primary armored cable distribution.

b. These cables would be hung on messenger cable with a 500 MVA capacity breaker to isolate and protect the cable to each load center.

c. Each section load center would have taps for 550, 440 and 120 volt outlets.

d. Each outlet would have breaker and cable protection to the 2 shuttle cars, continuous miner, top coal drill, feeder breaker, front end clean-up loader, lighting and extras for needs that may arise such as rock dusters, pumps, fans, battery charging, etc.

2. Water Supply. A plentiful supply of water at a minimum pressure of 100 psi for fire protection, for sprays on the miner, the coal drill and the belt conveyor, and for sprinkling of fallen top coal and shuttle car roadways is necessary.

3. Safety Awareness and Safety Training. No mining operation will ever be conducted in a safe manner, no matter how well designed for safety, unless the mine management and the miners themselves are thoroughly convinced that safety pays. An active and well conducted program of safety awareness and training in safe mining practices would be carried out for mine personnel at all levels of responsibility. This would be even more important in the case of a new mining method, such as Sublevel Caving by Pillar Extraction, being applied in an actual mine. Even though the mining method design attempts to build on a base of standard mining practice and to extrapolate from standard practice as much as possible, there are new elements in the mining operation which must be approached cautiously until sufficient experience demonstrates the concept by practice.

4. Remaining safety details applicable to this concept, which are not included or forseen, would be handled by the mine safety organization which should be complete with authority to enforce appropriate safety measures.

One last comment on mine safety is appropriate. The continuous miner helper is exposed to the same degree of hazard by this concept as he suffers in other continuous mining operations. This person is subject to roof falls, to injuries from moving machines and machine parts. The degree of risk encountered by this person is judged by the writers to be unacceptable. Although contractual obligations may preclude it, reassignment of the continuous miner helper, or elimination of this position from the section crew would improve safety and productivity significantly.

## MINE ENVIRONMENT

### Ventilation

As mentioned previously, the mine would be provided with a blowing ventilation system with a capacity to provide the equivalent of 100,000 CFM of fresh air for each mining section. Regardless of how well a mine ventilating system is maintained, there are always many losses (fugitive air) which diminish the amount of fresh air at the point of mining. In the writer's experience, fugitive air may account for 50 per cent of the volume produced by the fan. Allowing for the above loss, it is expected that a minimum of 50,000 CFM flow can be maintained through the Sublevel Caving by Pillar Extraction section. In other words, a minimum volume of 50,000 CFM will flow into and sweep the dust and gasses out of the working places of the three rooms into the return along the cave line.

This mining method has a great advantage in ventilation compared with standard room and pillar methods. This is due to the small number of openings both in the development and in the retreat mining sections. Fewer ventilation structures are needed and less ventilation maintenance is required. The larger volumes of air flowing into the working areas of each section will maintain a fresh air atmosphere almost free of dust and gasses. In the wintertime the places may be cold, but nevertheless healthy. In the event that a tight cave chokes off the air returning along the void between the pillar and the cave, the return airway can readily be re-established with line brattice.

### Float Dust

The greatest disadvantage in the application of the continuous miner to this concept is the generation of float dust. The operation of the continuous miner, (during both advance mining and loading) and the shooting of the top coal are expected to be the two main sources of float dust with this mining method.

The problem of float dust generation with the continuous miner is well recognized and many practices are utilized to combat it. The writer has had experience in the method of providing water on the miner cutter bits. This method effectively reduces dust, but suffers from the difficulty of maintaining the rotating seals. The most practical method of reducing float dust is to mount at least 40 spray nozzles on the front of the cutter head and to maintain a pressure of at least 300 psi on these sprays. The sprays will require approximately 32 gpm of water, and will increase the moisture content of the coal about 2 per cent. The added moisture will also assist in reducing generation of dust during subsequent handling of the coal.

A greater concentration of float dust can be expected to be introduced into the return when the continuous miner is driving the advance openings than during loading of top coal. It will sometimes be necessary for the miner operator to reduce large pieces of fallen top coal with the miner cutter wheels. However, it is believed that the relatively large amount of atomized water from the cutter head will virtually eliminate the escape of float dust into the return during loading or combined cutting and loading of top coal.

The shooting of the top coal will generate dust and smoke which will be carried into the return. The shots for all the falls of top coal are open-ended into the return. The large volume of air available in the return is expected to move the smoke and dust rapidly away from the area in which men are working. No one would be allowed to be in the return airway when powder smoke and dust from the shot would be present. The use of very light explosive charges is also expected to reduce the amount of dust produced by the explosion.

Beltway / Return Airway

Automatic sprays would be mounted on both the input and discharge ends of the belt feeder-breaker to reduce float dust in this area and along the beltway. The belt conveyor is on return air which is high in humidity and above freezing temperature. Some float dust in the return air will adhere to the moist surfaces in the beltway. Sprays would also play on the moving coal at belt transfer points. Good belt cleaners, either scrapers or power brushes, will help to minimize float dust along the beltway and reduce cleanup work. Directing a jet of water onto the returning load side of the belt and introducing a small amount of water onto the returning non-load side of the belt could be considered if necessary.

The Sublevel Caving concept uses only two section entries in order to maintain effective ground control. This means the beltway must return the entire volume of air leaving the section. The velocity of this volume of air will range between 350 and 700 feet per minute. The belt speed would be at least 400 feet per minute to handle the coal during surge periods. Given these relative speeds of coal and return air traveling in the same direction, very little dust would be blown into the air from coal on the belt.

Despite measures to reduce float dust along the beltway, it will continue to be present. It will also be necessary that mine personnel enter the beltway periodically during the shift. It is expected, for example, that the timber and utility man, the mechanic and the section foreman would need to enter the beltway to perform cleanup, repairs, etc. This work should be performed during those periods when the continuous miner is inactive and when smoke and dust from blasting are not exhausting through the return airway.

The proposed hypothetical mine would be large enough to have a crew specialized in belt maintenance and in moving the belt, both extending and retracting it. These tasks would be accomplished on the third shift to provide a dust-free atmosphere for these personnel to work in. This crew would be responsible for maintaining and extending and retracting power feeder cables and water lines. They would also perform the primary rock dusting chores and be responsible for a thorough cleanup of the distances traversed by each belt move. The third shift would also be utilized for the main materials handling work.

A trickle rock duster located just outby the feeder-breaker would maintain dilution of the float dust in the beltway and keep the inert percentage of the dust accumulations in the beltway within accepted standards.

A problem area as regards dust is that portion of the shuttle car roadway within the beltway/return. This is the portion from the belt feeder breaker back to the door or travel curtain into the rooms. In this area the workmen, primarily the shuttle car operators, will be downwind from the continuous miner and subject to air containing that amount of float dust which escapes the cutter head sprays. The large volume of air will have a significant diluting effect on whatever dust does escape. If these dust concentrations exceed allowable limits, the shuttle car operator can be required to wear a respirator during the 1-1/2 minutes he would be exposed to this atmosphere during discharge of the shuttle car load into the feeder breaker. It would be necessary to apply rock dust once or possibly twice during the shift to comply with MESA standards. It would also be necessary to clean and sprinkle the area around the feeder-breaker and the roadways to avoid the generation of float dust due to the grinding action of the shuttle car tires on coal rubble.

## Cleanup and Rockdusting

Cleanup is a necessity in any underground coal mine and is a proper concern for mine inspectors. As well as reducing danger to lungs and danger of explosion due to dust generated by machine travel, good housekeeping is indicative of overall attitudes toward safety. As mentioned previously, a substantial amount of cleanup would be done on the third, or maintenance shift. This is also when the primary rock dusting would occur. As shown in the Mining Cycle Charts, however, this work could also be performed during the two production shifts as needed.

When driving the section openings, a thorough cleaning job would be done when the continuous miner leaves the face being advanced to go to a top coal fall. The front-end loader would clean the center portion of the room or crosscut. The tight places behind the jacks and against the ribs would be cleaned by hand shoveling the loose coal into windrows where the front-end loader could pick it up and dump it against the face. Later it would be loaded out by the miner. This would leave the place clean, orderly, free of fine coal and ready for rock dusting. Only minor amounts of rock dusting should be required on the production shift.

Despite the foregoing, rib sloughage subsequent to the initial thorough cleaning would not be cleaned up because the sloughage lying against the rib helps to stabilize it. This accumulation of sloughage would contain a low percentage of fines and would be easily rendered harmless by the rockdust.

The best method of reducing rib sloughage is to install wooden dowel pins, which hold a 2-inch by 6-inch rough plank against the center of the rib. These wooden rib pins are roughly 1-3/4 inches in diameter, 5 feet long, with a slit sawed in each end so they will anchor by means of a wedge both in the coal and a hole drilled in the plank. Holes for the rib pins can be drilled with the top coal drill or with a hand-held electric coal drill. Permissable power take-offs can be specified on any of the face machinery for the hand-held drills. Rib pinning can follow the continuous miner, and if necessary, can be done on the 3rd shift. Rib pinning would be used mainly in the entries where the ribs have to stand for long periods of time. However, if rib sloughage were too rapid in the rooms, it could be applied there. In driving the pillar pockets the continuous miner would merely tear out the rib pins.

Cleanup and rock dusting of the retreat mining areas would be accomplished as necessary during the production shifts when the continuous miner and shuttle cars are engaged in room advance, when the continuous miner is inactive for mechanical delay or enroute from one place to another.

Noise Pollution

The most serious noise source associated with the concept is the exhaust fan used during entry driving. This fan produces approximately 115 dB of noise. No satisfactory method of attenuating this noise is believed to exist at this time. The best method for controlling this hearing danger is to isolate the fan from the workmen. This isolation is also necessary because of the dust introduced by the trickle duster into the air stream of the fan. Therefore, the fan would be located one crosscut downwind from the feeder-breaker. The exhaust would be pointed away from the feeder breaker and the whole installation shrouded with a brattice curtain to help reduce the noise and to reflect it away from the location of the personnel working around the feeder-breaker. The person assigned to keep the trickle duster full would have to wear ear protection or would turn the fan off while in its vicinity. Other machinery proposed for this mining method does not subject personnel to excessive noise levels.

## SECTION 3

## PERFORMANCE CHARACTERISTICS

ASSUMPTIONS:

The Sublevel Caving by Pillar Extraction method is evaluated for performance characteristics within a hypothetical mining operation which produces approximately 1 million tons per year from a property consisting of four sections of land (2560 acres) underlain by a 20-foot thick coal seam with cover ranging from 0 to 2,000 feet in depth. (See Drawing No. 2, Hypothetical Mine General Layout.)

Coal density is .04 tons per cubic foot.

Four mining units are on retreat, 2 are under "adverse conditions" and 2 are under "favorable conditions".

One mining unit is driving main and section entries; 50% of the time under "favorable conditions", and 50% of the time under "adverse conditions".

One standby section is fully equipped.

One spare mining unit is in the shop.

Work 2 production shifts per day.

Work 230 days per year.

Productivity

Estimate of average production per day

| | | |
|---|---|---|
| Favorable retreat | = (4) (593) | = 2372 |
| Adverse retreat | = (4) (475) | = 1900 |
| Favorable advance | = (1) (400) | = 400 |
| Adverse advance | = (1) (270) | = 270 |
| Total | | 4942 tons |

Estimated production per year = (230) (4942) = 1,136,660, say 1,140,000 tons.

## PRODUCTIVITY

In analyzing productivity for the mining method, top priority has been on the process of retreat mining or top coal and pillar recovery. Estimated shift cycles have been developed for three retreat mining production shifts for both "adverse" and "favorable" conditions. Attention is called to Tables 3-1 through 3-4 which present these cycles. Tables 3-1 and 3-2 present the estimated shift cycles for one production shift for advance mining under this method. Tables 3-3 and 3-4 present the shift cycles for retreat mining. Care has been exercised in estimating productivity to allow sufficient time for each task to be performed and to provide for mechanical and unforseen delays. Time elements and total productivity have been reviewed at length and the consensus is that the times allotted to each task are reasonable based on experience in mining thick coal.

Although the limited number of work places impairs production by causing crowding, especially for shuttle car traffic, and by reducing flexibility of the mining process, there are some advantages due to the smaller number of places which are worth noting. Productivity and safety should be improved due to closer supervision. Provision of mine utilities and ventilation is simplified.

### Advance Mining

Productivity is enhanced in room and entry advance mining, by providing roof support without taking the miner out of the work place. This advantage is relatively greater under "adverse" conditions. By leaving the miner at the face during placement of support, tramming delays and time needed to restart mining are both reduced. Also the roof support method recommended with the continuous miner has a distinct advantage over conventional equipment where the number of work places is limited to 2 or 3 as this method dictates. For example, the cutter, the drill, and the loader would be forced to wait on each other and would thereby impair production.

Table 3-1 shows the main elements of the entry driving cycle under "adverse" conditions which are repeated 9 times during the shift. Extra time is allowed for starting (Step 1) and to clean up and change miner bits (Steps 9, 13, 14, 20). Time is allowed for mechanical delays (Step 21) and a miscellaneous delay factor is also included (Step 22). The main repeating phases of the cycle are:

1. Mine 30 tons of coal at a rate of 2.5 tons per minute or 12 minutes for the 30 tons which will advance the place 5 feet to provide room for a crossbar set.

2. Place crossbar set with the aid of the miner. Extend the ventilation tube and test for gas. Time allowed for these tasks--20 minutes.

Production rate for the shift is approximately 270 tons. This production rate is believed attainable as an average in driving the section entries under "adverse" top coal roof conditions.

In driving section entries under "favorable" conditions with a continuous miner unit, the cycle is repeated eight times during a production shift.

The main repeating phases of the cycle are:

1. Mine 50 tons of coal at a rate of 2.5 tons per minute--time 20 minutes.

2. Extend ventilation tube and test for gas.

3. Install 2 timber props.

Time required for these tasks--15 minutes.

4. This step is similar to step 3 above except 3 props are installed. Time required--20 minutes.

Production rate for the shift is approximately 400 tons.

The section entry driving crew for both "adverse" and "favorable" top coal roof conditions are as follows:

| | |
|---|---|
| 1 | Continuous miner operator |
| 1 | Continuous miner helper |
| 1 | Timber and utility man |
| 2 | Shuttle car operators |
| 1 | Mechanic |
| 1 | Foreman |
| 7 | Total |

TABLE 3-1
Section Entry Driving
Adverse Roof Conditions
Estimated Continuous Miner Shift Analysis

Leave miner in face to assist in setting timber using miner head to lift and hold crossbar.

| Work Phase | Time Min. | Time Cum. | Tons |
|---|---|---|---|
| 1. Start, check places, turn on power tram miner into face. | 10 | 10 | |
| 2. Mine 30 tons. | 12 | 22 | 30 |
| 3. Place crossbar set, gas test, and vent. tube extension. | 20 | 42 | |
| 4. Mine 30 tons. | 12 | 54 | 30 |
| 5. Place crossbar set, gas test, and vent. tube extension. | 20 | 74 | |
| 6. Mine 30 tons. | 12 | 86 | 30 |
| 7. Place crossbar set, gas test, add length to vent. tube. | 20 | 106 | |
| 8. Mine 30 tons. | 12 | 118 | 30 |
| 9. Place crossbar, gas test, and extend vent. tube, clean up behind miner with front end loader. | 20 | 138 | |
| 10. Mine 30 tons. | 12 | 150 | 30 |
| 11. Place crossbar, gas test. | 20 | 170 | |
| 12. Mine 30 tons. | 12 | 182 | 30 |
| 13. Place crossbar, back miner up, clean place with front end loader, replace dull cutter bits. | 30 | 212 | |
| 14. Test gas and tram back into face. | 5 | 217 | |
| 15. Mine 30 tons. | 12 | 229 | 30 |
| 16. Place crossbar set, test gas, extend vent. tube. | 20 | 249 | |
| 17. Mine 30 tons. | 12 | 261 | 30 |
| 18. Place crossbar set, extend tube, test gas. | 20 | 281 | |
| 19. Mine 30 tons. | 12 | 293 | 30 |
| 20. Place crossbar set, back miner up and clean up. | 20 | 313 | |
| 21. Allow for mechanical delays. (50% of cont. miner op. time) | 48 | 361 | |
| 22. Miscellaneous delays. | 29 | 390 | |
| TOTAL | 390 | xxx | 270 |

Tons per labor manshift = $\frac{270}{6}$ = 45

TABLE 3-2
Section Entry Driving
Favorable Roof Conditions
Estimated Continuous Miner Shift Cycle Analysis

Leave miner in entry face including crosscut until break through is made.

| | Work Phase | Time Min. | Time Cum. | Tons |
|---|---|---|---|---|
| 1. | Start, check places, turn on power tram miner to face. | 10 | 10 | |
| 2. | Mine 50 tons. | 20 | 30 | 50 |
| 3. | Test for gas and extend vent. | 10 | 40 | |
| 4. | Mine 50 tons. | 20 | 60 | 50 |
| 5. | Install 2 props, test gas, extend vent. tube. | 15 | 75 | |
| 6. | Mine 50 tons. | 20 | 95 | 50 |
| 7. | Test for gas, extend vent. tube. | 10 | 105 | |
| 8. | Mine 50 tons. | 20 | 125 | 50 |
| 9. | Install 3 props, test gas, extend vent. tube, clean up. | 20 | 145 | |
| 10. | Mine 50 tons. | 20 | 165 | 50 |
| 11. | Test for gas, extend vent. | 10 | 175 | |
| 12. | Mine 50 tons. | 20 | 195 | 50 |
| 13. | Install 2 props, extend vent. tube, check miner, i.e. bits, etc., test gas. | 30 | 225 | |
| 14. | Mine 50 tons. | 20 | 245 | 50 |
| 15. | Test gas, extend vent. tube. | 10 | 255 | |
| 16. | Mine 50 tons. | 20 | 275 | 50 |
| 17. | Mechanical delays. (50% miner op. time) | 80 | 355 | |
| 18. | Miscellaneous delays. | 35 | 390 | |
| | TOTAL | 390 | xxx | 400 |

Average tons per manshift labor = $\frac{400}{6}$ = 67

All tasks such as timbering and extension of ventilation, which interrupt continuous miner production, would be designated as crew tasks. For example, the entire crew would pitch in to set timber as rapidly as possible in order to return the miner to operation.

Retreat Mining

Retreat mining is the unique and critical phase of the Sublevel Caving by Pillar Extraction method. All other mining operations should work toward its successful accomplishment.

An important factor in the productivity of the retreat will be the degree of success which can be attained in shooting down the top coal. Too much explosive may damage a weak roof stratum above the top coal and cause it to fall before the top coal can be loaded. Too little explosive in the top coal holes will cause some coal to be left against the roof affecting recovery and aggravating spontaneous combustion. The number of rows of holes which can be shot at one time affects the productivity directly and also the safety of the concept. If only one or two rows can be shot at once, there will be stops and starts, ensuing delays and loss of productivity. If as many as four can be shot, the brow will be advanced in the safest manner and productivity will be not only competitive but quite attractive.

The speed with which the top coal holes can be safely and effectively charged is also an important determinant of productivity. Much consideration has been given this topic and it is believed that each hole can be charged in approximately two minutes using a wooden charging stick, clay dummies and a plastic retainer to hold the charge until the dummies are placed.

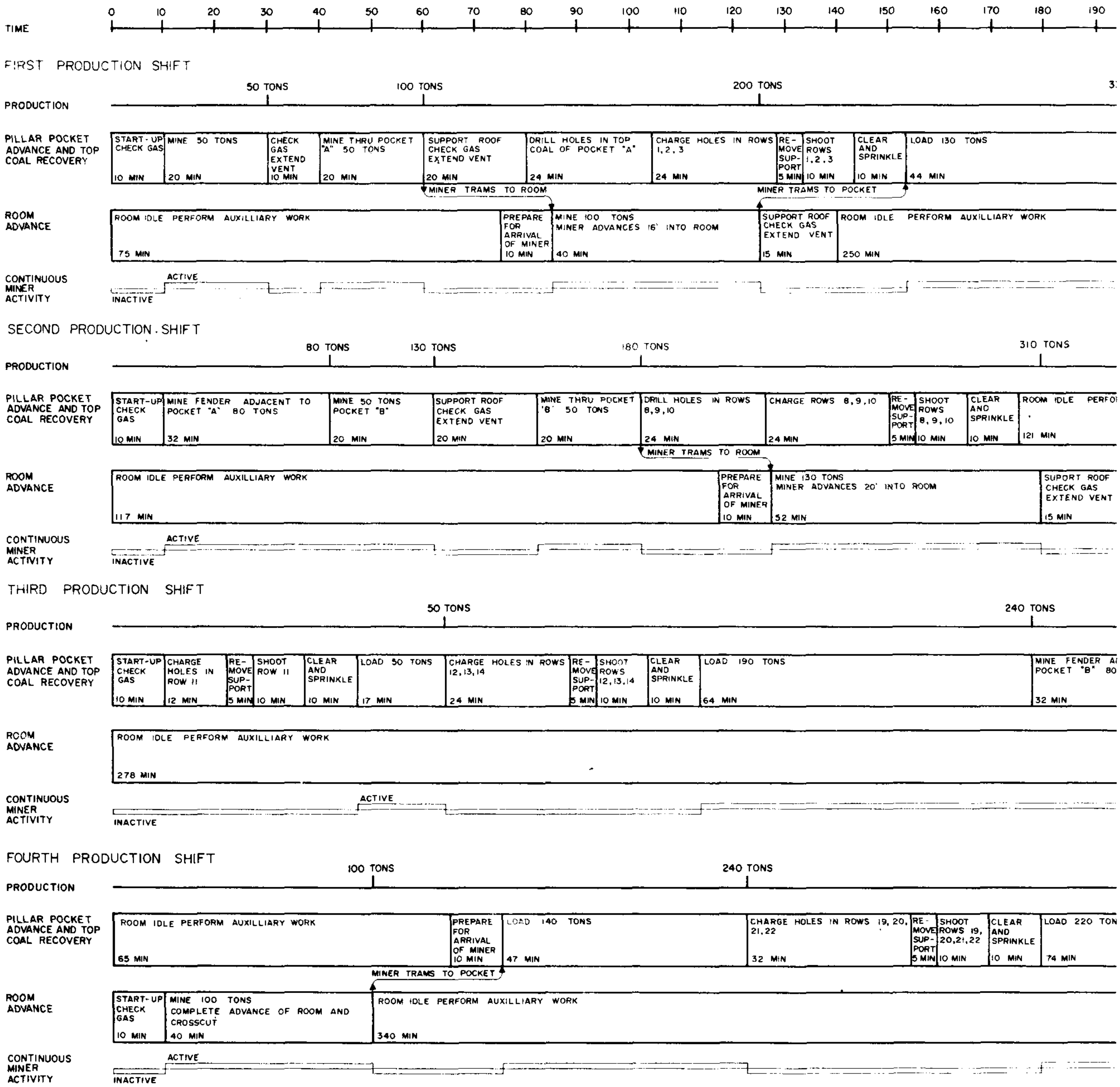
The time the roof rock will stand before caving determines both the amount of coal which can be loaded from the fall and also the degree of success in winning the pillar pocket fenders. Experienced miners can often judge the time before a cave quite well by watching rock trickle and listening to roof sounds.

An important consideration in productivity is achieving a good balance between miner time allocated to advance of the room and recovery of the fallen top coal. Tramming the miner back and forth eats into production time. Therefore, the miner must be left at the room being advanced long enough to get something accomplished before being moved to the room and pillar being retreated. Likewise, the miner must be left in the retreat area long enough to recover the entire fall of top coal and as much of the fenders as possible before returning to the advancing room. However, a balance would be maintained, i.e. one pillar would be created in the time one pillar would be extracted.

Drilling of the top coal was originally believed to be a critical determinant of productivity. This is in fact the case as regards the pillar pockets in which the drill holes must be placed quickly to avoid production delays. In the rooms, however, there occur substantial amounts of time in which drilling can be accomplished well in advance of shooting. Therefore, the amount of time spent to accomplish this does not materially affect productivity.

## RETREAT MINING CYCLE – FAVORABLE CONDITIONS

ROOM, CROSSCUT AND PILLAR POCKET ADVANCE
ROOM AND PILLAR TOP COAL RETREAT




NOTES:

1. UNDER "FAVORABLE" CONDITIONS THE ASSUMPTIONS ARE MADE THAT THE TOP COAL ROOF IS RELIABLY SELF SUPPORTING AND THAT THE STRATA ABOVE THE TOP COAL ARE ALSO STRONG AND RELIABLE. THIS PEMITS RECOVERY OF APPROXIMATELY 50 % OF THE COAL IN EACH FENDER OF THE PILLAR.
2. ESTIMATED PRODUCTION RATE UNDER "FAVORABLE" CONDITIONS IS 593 TONS PER SHIFT.
3. TOTAL PRODUCTION FROM THE BALANCED ADVANCE OF ONE ROOM AND CROSSCUT AND THE RETREAT OF ONE ROOM PILLAR AND CROSSCUT IS 2170 TONS.
4. ESTIMATED RECOVERY UNDER "FAVORABLE" CONDITIONS IS 76% FOR THE SECTION
5. "AUXILIARY WORK" INCLUDES
   a. PRE-DRILLING HOLES IN ROOM
   b. CLEAN UP LOOSE COAL.
   c. ROCK DUSTING
   d. UTILITIES ADVANCE
   e. SPRINKLE
   f. CAREFUL SCALING OR TRIMMING OF THE TOP COAL ROOF TO IMPROVE SAFETY AND TO HELP AVOID MISFIRES DUE TO SHOT WIRES BROKEN BY FALLING TOP COAL.
   g. PLACEMENT OF SAFETY PROPS AND "BREAKER ROWS" OF TIMBERS AS NEEDED
   h. MATERIALS HANDLING
6. TRAMMING TIME IS ESTIMATED AT 25 MIN
7. LOADING AND MINING PERIODS SHOWN ON THE CYCLE CHART TO BE LONGER THAN 20 MINUTES. TAKE INTO ACCOUNT ROUTINE STOPS TO CHECK FOR GAS.

Table 3 - 3

## RETREAT MINING CYCLE – FAVORABLE CONDITIONS (CONTINUED)

ROOM, CROSSCUT AND PILLAR POCKET ADVANCE
ROOM AND PILLAR TOP COAL RETREAT

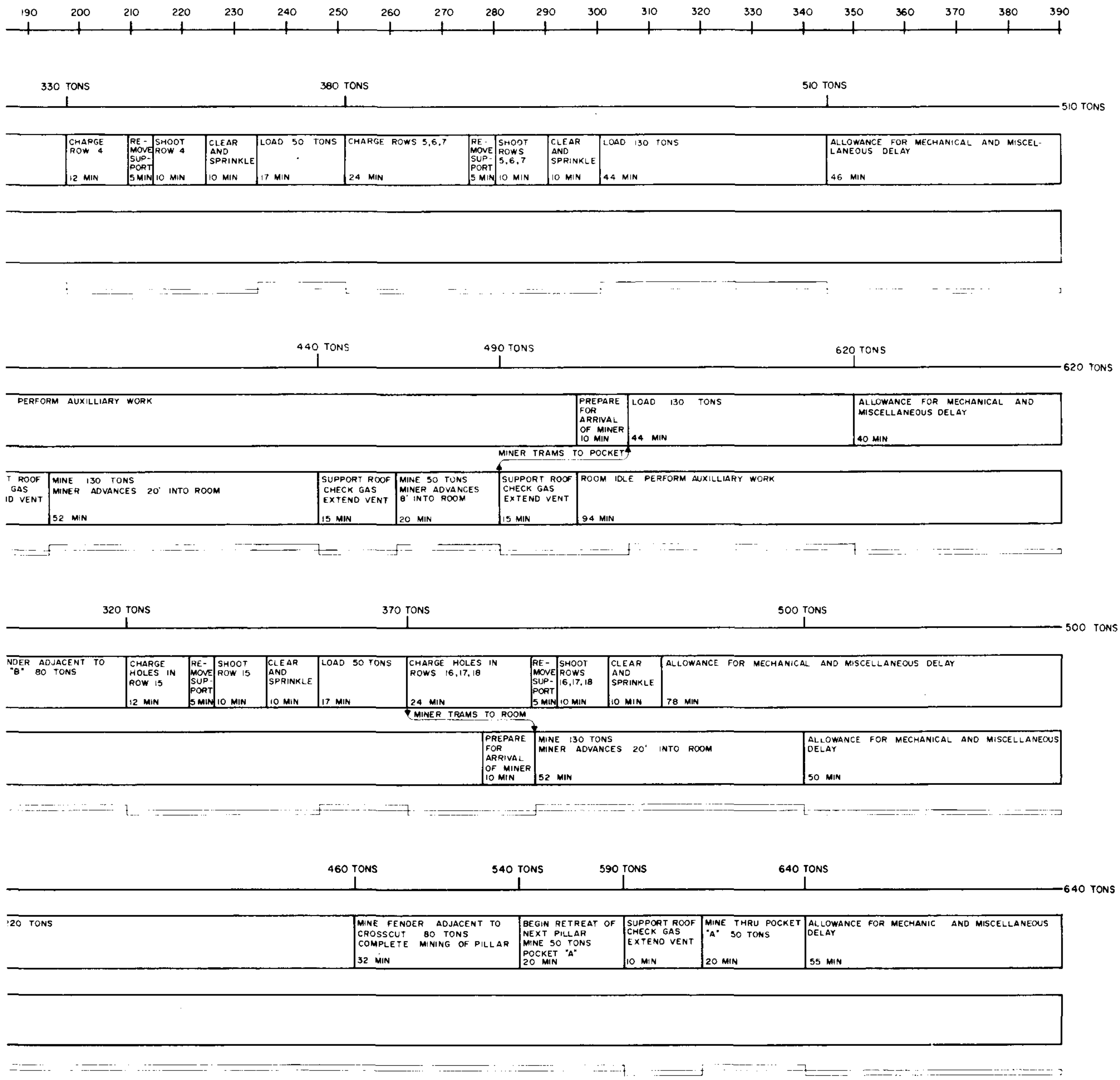
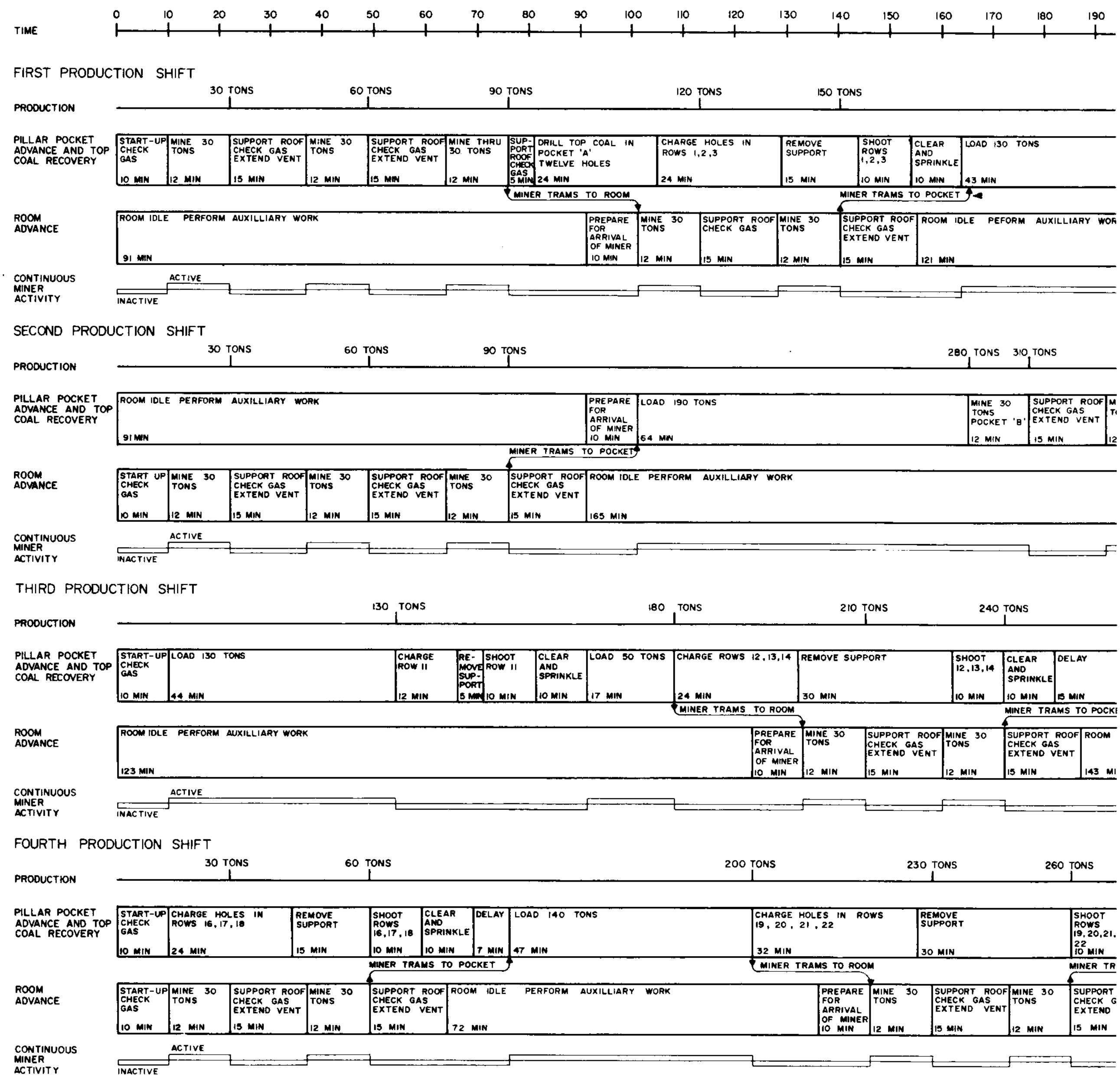



Table 3 - 3 (cont.)

## RETREAT MINING CYCLE - ADVERSE CONDITIONS

ROOM, CROSSCUT, AND PILLAR POCKET ADVANCE
ROOM AND PILLAR TOP COAL RETREAT




NOTES:

1. UNDER ADVERSE CONDITIONS THE ASSUMPTION IS MADE THAT NO RECOVERY OF THE FENDERS WILL BE POSSIBLE. UNDER THESE CONDITIONS THE FENDERS MUST BE LEFT INTACT TO PRESERVE THE ROOF SO THAT TOP COAL LOADING CAN PROCEED.
2. TOTAL PRODUCTION FOR BALANCED MINING OF ONE ROOM AND ONE PILLAR IS ESTIMATED AT 1900 TONS
3. AVERAGE PRODUCTION PER SHIFT IS ESTIMATED AT 475 TONS
4. ESTIMATED SECTION RECOVERY IS 67% UNDER "ADVERSE" CONDITIONS
5. "AUXILLIARY WORK" INCLUDES
   a. PRE-DRILLING HOLES IN ROOM.
   b. CLEAN UP LOOSE COAL
   c. ROCK DUSTING
   d. UTILITIES ADVANCE.
   e. SPRINKLE.
   f. CAREFUL SCALING OR TRIMMING OF THE TOP COAL ROOF TO IMPROVE SAFETY AND TO HELP AVOID MISFIRERS DUE TO SHOT WIRES BROKEN BY FALLING TOP COAL.
   g. PLACEMENT OF SAFETY PROPS AND "BREAKER ROWS" OF TIMBERS AS NEEDED.
   h. MATERIALS HANDLING.
6. TRAMMING TIME IS ESTIMATED AT 25 MIN.
7. LOADING AND MINING PERIODS SHOWN ON THE CYCLE CHART TO BE LONGER THAN 20 MINUTES. TAKE INTO ACCOUNT ROUTINE STOPS TO CHECK FOR GAS.

Table 3 - 4

## RETREAT MINING CYCLE - ADVERSE CONDITIONS (CONTINUED)

ROOM, CROSSCUT, AND PILLAR POCKET ADVANCE
ROOM AND PILLAR TOP COAL RETREAT

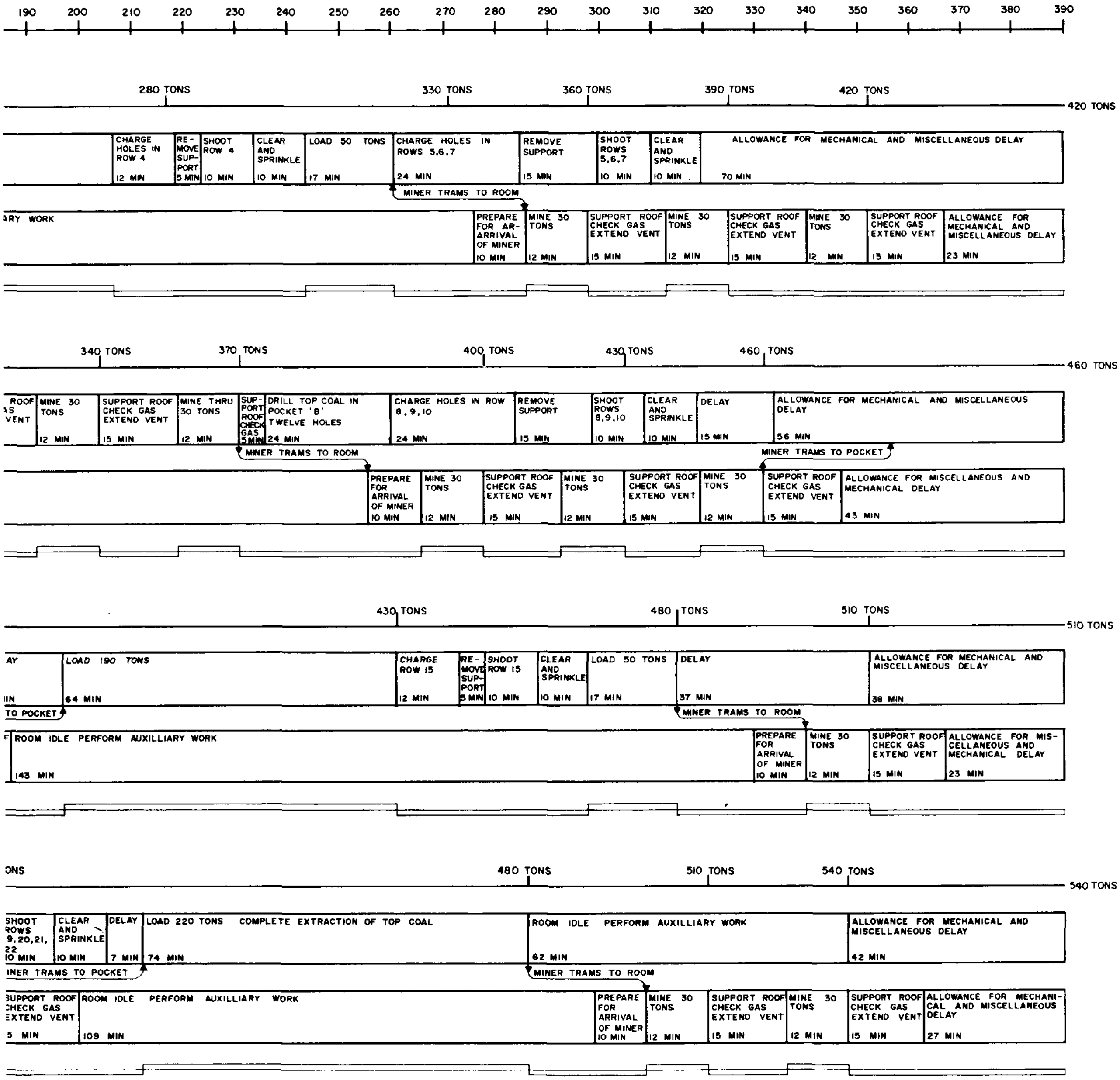



Table 3 - 4 (cont.)

TABLE 3-5

ESTIMATED PRODUCTIVITY

Section

NOTE: Section crews include foremen.

| | | |
|---|---|---|
| Favorable retreat = | $\frac{593 \text{ tons}}{9 \text{ men}}$ = | 65.9 tons per section manshift |
| Adverse retreat = | $\frac{475 \text{ tons}}{9 \text{ men}}$ = | 52.8 tons per section manshift |
| Favorable advance = | $\frac{400 \text{ tons}}{7 \text{ men}}$ = | 57.1 tons per section manshift |
| Adverse advance = | $\frac{270 \text{ tons}}{7 \text{ men}}$ = | 38.6 tons per manshift |
| Estimated average productivity per = manshift | $\frac{4{,}942 \text{ tons}}{86 \text{ men}}$ = | 57.4 tons per manshift |

Total Payroll

| | | |
|---|---|---|
| Estimated average productivity = per manshift | $\frac{4{,}942 \text{ tons}}{178 \text{ men}}$ = | 27.8 tons per manshift |
| Estimated average productivity = per man hour | $\frac{27.8 \text{ tons/manshift}}{8 \text{ hours/manshift}}$ = | $\frac{3.48 \text{ tons}}{\text{man hour}}$ |

NOTE: See Tables 4-7 through 4-9 for detailed listing of manpower requirements.

## RECOVERY

Within the outcrop, the 4-section property would contain approximately 89,000,000 gross tons of coal in place. The area available for mining is surrounded by a property barrier 150 feet in thickness which extends to the boundary of the property. Coal within this property barrier is equal to 4,750,000 tons, or 5.3% of the total coal reserve. The mining sections occupy an area 8,860 feet X 10,560 feet in size containing approximately 72,700,000 tons of coal in place, or about 81.5% of the coal on the property. The sections south of the main entries are 4,130 feet in length. Those north of the main entries are 4,730 feet in length. An average section, then, is 4,430 feet in length and 450 feet in width including the fire barrier. The 40 mining sections in the mine would each contain an average of 1,595,000 tons of coal in place. The remainder of the coal on the property is within the area occupied by the main entries and the main entry barriers (each 400 feet wide). This area is approximately 1,400 feet in width and 10,500 feet in length and contains approximately 11,000,000 tons, or 12.4%, of the coal reserve.

### Section Recovery "Adverse Conditions"

The total coal available for recovery in the mining section is the total coal in place minus the coal left as fire barriers to isolate the mined out section from the area still being mined. The fire barriers are 50 feet in width and extend the 4,430-foot length of the section. They contain approximately 177,000 tons of coal, or 11.1% of the coal in place in the section. Approximately 1,418,000 tons of coal are available for recovery from each section.

Under "adverse conditions" the total coal available in the area encompassed by one 80-foot X 20-foot room, one 18-foot X 20-foot crosscut and one 62-foot X 20-foot pillar is 2,560 tons. It is estimated (See Mining Cycle Chart Adverse Conditions) that 1,900 tons of this amount will be recovered, or 76% of the coal available from this area.

The estimate of recovery from the section then is 76% of the coal available for recovery (.76) (1,418,000) or 1,078,000 tons. The percent recovery for the section is then: 1,078,000/1,595,000 = 67%.

#### Section Percentage Recovery "Adverse Conditions"

| | |
|---|---|
| Coal Recovered | 67.0% |
| Loss to Fire Barriers | 11.1% |
| Loss in Mining Process | 21.9% |
| | 100.0% |

Section Recovery "Favorable Conditions"

The total coal available for recovery from the section under "favorable conditions" is 1,418,000 tons.

The estimate of the coal recovered from one 80-foot X 20-foot room, one 18-foot X 20-foot crosscut and one 62-foot X 20-foot pillar is 2,170 tons of the 2,560 tons available in this area. The percentage recovery from the area is 85%. (See Mining Cycle Charts - Favorable Conditions.)

The estimate of recovery from the section is then 85% of the coal available for recovery (.85)(1,418,000 tons), or 1,205,000 tons. The percent recovery is 1,205,000/1,595,000 = 76%.

Section Percentage Recovery "Favorable Conditions"

| | |
|---|---|
| Coal Recovered | 76.0% |
| Loss to Fire Barriers | 11.1% |
| Loss in Mining Process | 12.9% |
| | 100.0% |

Section Recovery - Overall Average

| | |
|---|---|
| "Adverse Conditions" | (.67) (36,350,000 tons) = 24,355,000 tons |
| "Favorable Conditions" | (.76) (36,350,000 tons) = 27,626,000 tons |
| Total | 51,981,000 tons |

Percent Section Recovery 51,981,000/72,700,000 = 71%

Main Entries and Main Entry Barriers Recovery

The main entry pillars, main entry barriers and associated top coal can be expected to be under very adverse conditions. The cave would be encroaching from three sides. The roof and pillars may have stood for a very long time and deteriorated in strength. Although the same method of recovery is used, its success should be lessened. There is no way to determine before hand how much it should be lessened to account for the factors mentioned above. For purposes of this study, we will use as a rough estimate that 50% of the coal available from these areas will be recovered.

The coal in place is 11,000,000 tons. Fire barriers 50 feet wide and 10,000 feet long would be left to seal the final retreat from the gasses from the cave area of the mined out sections. These barriers would contain 845,000 tons of coal. The coal available for recovery is approximately 10,155,000 tons. At 50% recovery, 5,077,000 tons would be won during the final retreat. The overall percentage recovery during final retreat is 46%.

Final Retreat Percentage Recovery

| | |
|---|---|
| Main Entries and Main Entry Barriers | 46% |
| Loss to Fire Barriers | 7% |
| Loss in Mining Process | 47% |
| | 100% |

Overall Mine Recovery

| | |
|---|---|
| Mining Sections | 51,996,000 tons = 71% |
| Main Entries and Barriers | 5,077,000 tons = 46% |
| Total | 57,151,000 tons = 64% |

Mine Percentage Recovery

| | |
|---|---|
| Coal Recovered | 64.0% |
| Loss to Fire Barriers | 9.0% |
| Loss to Property Barriers | 5.3% |
| Loss in Mining Process | 21.7% |
| | 100.0% |

TABLE 3-6

ESTIMATE OF POWER CONSUMPTION

| Number | Machine | Total HP | Hrs./Day | KW Load | KW-Hour Req |
|---|---|---|---|---|---|
| 5 | Cont. Miner | 2,250 | 8 | 1,680 | 13,440 |
| 10 | Shuttle Car | 800 | 12 | 596 | 7,152 |
| 5 | Top Coal Drill | 250 | 8 | 187 | 1,496 |
| 5 | Feeder-Breaker | 625 | 10 | 466 | 4,660 |
| 5 | Aux. Fan | 250 | 9 | 187 | 1,683 |
| 2 | Mine Fan | 1,000 | 24 | 746 | 17,904 |
| 5 | Section Conveyor | 1,500 | 15 | 1,119 | 16,785 |
| 1 | Main Conveyor | 750 | 15 | 560 | 8,393 |
| 1 | Outside Coal | 500 | 15 | 373 | 5,595 |
| | Misc. Including Losses | | | | 6,885 |
| TOTAL | | | | | 84,000 |

NOTE:

1. Power consumed per ton of coal mined = 84,000/4,942 = 17 KW-hour per ton.

2. Cost of power is approximately 2¢ per KW-hour at 84,000 KW-hour per day rate of consumption. Cost of power per ton = (84,000) (.02)/4,942 = 34¢ per ton.

TABLE 3-7

MINE AND PLANT OPERATING SUPPLIES

| Item | Sublevel Caving Estimate | IC 8682A Estimate |
|---|---|---|
| Mining machine parts | .63 | .63 |
| Lubrication and hydraulic oil | .25 | .25 |
| Timber | .02 | .31 |
| Rock dust | .12 | .13 |
| Ventilation materials | .10 | .19 |
| Bits | .02 | .12 |
| Cables | .06 | .06 |
| Explosives and blasting supplies | .05 | --- |
| Power | .34 | .38 |
| Misc. | .15 | .15 |
| Total | $1.74 | $2.22 |

NOTES:

1. Bureau of Mines Information Circular/1976 8682A - Basic Estimated Capital Investment and Operating Costs for Underground Bituminous Coal Mines - was used as a guide for estimating the cost of operating supplies.

2. Estimates shown for mining machine parts, cables, lubrication and hydraulic oil are taken directly from this source. It is expected that Sublevel Caving by Pillar Extraction will not differ from standard room and pillar mining in these categories.

NOTES TO TABLE 3-7 Cont'd

3. Timber "Unfavorable Conditions"

Expendable roof support consists of 3-piece crossbars comprising 32 linear feet of round timber set on 4-foot centers in section entries and crosscuts. Section entries average 4,430 feet in length. There are two entries with crosscuts on 80-foot centers. There are 5 crosscuts 20 feet long which add 1,100 feet of openings.

Total length of opened area in section entries = (2) (4,430) + 1,100 = 9,960 feet.

Total number of wooden crossbar sets = 2,490 per section.

Linear feet of timber = 32 X 2,490 = 79,680 + 20% for replacement and waste = 96,000 linear feet. Coal mined per section under "unfavorable conditions" = 1,078,000 tons.

Timber per ton under "adverse conditions" = 96,000/1,078,000 = .09 linear feet per ton.

4. Timber "Favorable Conditions"

Expendable roof support consists of 8-foot round wooden posts set on 6-foot centers.

The length of the openings in the section entries is 9,960 feet.

There are 111 rooms on 40-foot centers in the average section. Each room is 400 feet long and has five 20-foot crosscuts. The total length of the opened area is 500 feet per room and the total length of the rooms comprising the section is 55,500 feet.

The total length of openings for the section is then 55,500 +9,960 = 65,460 feet.

The total number of wooden supports is 65,460/6 = 10,910 and the total number of linear feet of wooden support timber is 8 X 10,910 = 87,280 +20% for replacement, waste and extra support as needed = 87,000 + 17,000 = 104,000 linear feet.

Linear feet of expendable wooden support per ton of coal mined under "favorable conditions" = 104,000/1,205,000 = .086 linear foot per ton.

NOTES TO TABLE 3-7 Cont'd

5. Overall Timber Usage

Overall timber requirements per ton of coal mined are 104,000 + 96,000/1,078,000 + 1,205,000 = .087 linear foot per ton.

At the rate of 26¢ per linear foot for timber, the expense for timber is 26¢ X .087 = 2¢ per ton. The reuseable crossbar and jack support sets would be a capital item.

6. Rock Dust

The design of Sublevel Caving by Pillar Extraction specifies pressurized, bulk rock dust systems which are of a slight cost advantage.

7. Ventilation Materials

Sublevel Caving utilizes a smaller amount of ventilation material due to the two section entry system, the limitation on rooms open at one time to three and the practice of returning air along the cave. For example, the two section entry system would use only one row of stoppings to isolate the return from the intake. A five section entry system with a double split of air would use 3 rows of stoppings to isolate the intakes from the returns and beltway. These factors should reduce the ventilation materials in the section entries by two thirds. In the rooms, similar savings would be available when compared to many room and pillar operations. A conservative estimate is that savings of 50% in ventilation materials would be achieved. The IC 8682A estimate is accordingly reduced from 19¢ per ton to 10¢ per ton for ventilation materials for Sublevel Caving by Pillar Extraction.

8. Bits

A relatively small proportion of coal is mined using the continuous miner cutting bits. The great majority of drilling to be done is through coal. Therefore, the expense of replacing drill and cutter bits is reduced substantially below the expense for these items in a mine utilizing a continuous miner and roof bolting for roof support.

During an average production day, it is estimated that 830 tons of coal would be mined on advance and 4,112 tons would be shot down from the top coal roof.

NOTES TO TABLE 3-7 Cont'd

Cutter bit usage would be reduced by the ratio 830/4,942 = .17 below that in a mine using only continuous miners to mine the same amount of coal. The IC 8682A estimate is 12¢ per ton for bits in a mine utilizing continuous miners and roof bolting. Using this figure as a basis, the expense for bits in the Sublevel Caving method is 12¢ X .17 = 2¢ per ton.

9. Explosives and Blasting Supplies

Each drill hole is to be loaded with 1.12 pounds of explosive. There are 4 holes per row and 22 rows of holes in the area associated with one room, two pillar pockets and one crosscut. These explosives produce 1,900 tons of coal, or (4) (22) X 1.12/1,900 = .05 pounds of explosive per ton of top coal.

During an average production day, 4,112 tons of coal would be shot and loaded and 213 pounds of explosives used. Explosive use per ton of coal mined for the whole mine is then 213/4,942 = .04 pound per ton.

Each hole would require an expense of approximately:

| | | |
|---|---|---|
| Clay stemming | .15 | @ 5¢ each |
| Blasting cap | .66 | each |
| Plastic retainer | .10 | each |
| Explosives | .49 | @ 44¢ per pound |
| Total | $1.40 | |

To shoot the 88 top coal holes associated with one room, two pillar pockets and one crosscut, requires an expense for blasting materials of $123.00 or $123/1,900 = 6.5¢ per ton of top coal.

Based on estimated production for one day, expense of blasting materials is (.065) (4,112) = $267.00 and cost per ton of coal is $267/4,942 = 5¢ per ton for the whole mine.

## SECTION 4

## ECONOMIC FEASIBILITY ANALYSIS

### Assumptions and Summary

The discounted cash flow analysis of the Sublevel Caving by Pillar Extraction mining method indicates that the method would be comparable in economic feasibility when compared to standard room and pillar mining operations. The calculated selling price necessary to sustain a 15% return on invested capital is $13.22 per ton. Bureau of Mines Information Circular IC8682A/1976 was used as a guide for the economic analysis of the hypothetical mining venture.

Assumptions:

Twenty year economic life for the mining venture.

Land leased from the federal government with an imposed royalty of 8% of the selling price of the coal.

No costs, other than exploration, for land acquisition.

1976 labor costs and tax rates are used for calculation of labor costs and payroll taxes.

An annual capital expense of $160,000 for unforseeable equipment acquisitions and replacements is included in the Summary of Discounted Cash Flow, Table 4-14.

TABLE 4-1

SUMMARY OF ESTIMATED CAPITAL INVESTMENT, OPERATING COSTS, AND SELLING PRICE AT 1,140,000 TONS MINED PER YEAR

| | Total | Per Ton |
|---|---|---|
| Capital Investment | | |
| Initial Investment (Table 4-2) | $20,730,000 | $18.18 |
| Deferred Investment (Table 4-3) | 18,273,000 | 16.03 |
| Total Investment | $39,003,000 | $34.21 |
| Production Cost (Table 4-6) | $11,797,700 | $10.35 |
| Selling Price per ton at 15% DCF (Table 4-15) | | $13.22 |

TABLE 4 - 2

ESTIMATED INITIAL CAPITAL INVESTMENT (1)

| Number | Item | Estimated Cost (2) |
| --- | --- | --- |
| 1 | Mining Machinery | $6,405,000 |
| 2 | Underground Transportation and Support Equipment | 707,000 |
| 3 | Bulk Rock Dust Storage and Distribution | 320,000 |
| 4 | Underground Electrical and Communication System | 520,000 |
| 5 | Mine Industrial Water System | 465,000 |
| 6 | Ventilation Fans | 387,000 |
| 7 | Section Roof Support Equipment | 1,056,000 |
| 8 | First Aid, Safety, Rescue, Testing Instruments | 73,000 |
| 9 | Concrete Underground Roadways (3) | 500,000 |
| 10 | Mine Portals | 72,000 |
| 11 | Industrial Buildings | 800,000 |
| 12 | Office Equipment, Warehouse Equipment, Shop Equipment, Lamps, Charging Racks | 220,000 |
| 13 | Surface Electrical System (4) | 400,000 |
| 14 | Coal Preparation Plant | 1,305,000 |
| 15 | Surface Vehicles | 42,000 |
| 16 | Bulldozer, Front End Loader | 250.000 |
| 17 | Potable Water System | 83,000 |
| 18 | Sewage Treatment System | 66,000 |
| 19 | Roads (5) | 100,000 |
| 20 | Site Investigation and Preparation (including environmental repair) (6) | 335,000 |
| | Subtotal | $14,106,000 |

TABLE 4 - 2 (cont.)

ESTIMATED INITIAL CAPITAL INVESTMENT

| Number | Item | Estimated Cost |
|---|---|---|
| 21 | Engineering and Fees (3.4% of subtotal) | 477,000 |
| 22 | Overhead and Administration (7% of subtotal) | 987,000 |
| | Subtotal | $15,570,000 |
| 23 | Contingency (15% of above) | 2,336,000 |
| | Subtotal | $17,906,000 |
| 24 | Development and Exploration Costs in Excess of Value of Coal Recovered During Development | 1,000,000 |
| | Total Mine Cost (tax and insurance base) | $18,906,000 |
| 25 | Working Capital (including spare parts inventory) | 1,824,000 |
| | Total Initial Investment | $20,730,000 |

NOTES:

1. Most thick coal seams are in the Western U. S. and are on Federally owned land. Therefore no capital investment for land acquisition is shown. A Federal royalty of 8% of selling price, f.o.b. mine, is assumed.
2. Equipment costs shown are complete with freight estimated at $6.50/cwt, sales tax at 5% and necessary options.
3. Concrete underground roadways both in the main slope and the section entries are included to facilitate materials handling and personnel transport. Section entries are expected to stand open approximately 4 years each.

TABLE 4 - 2 (cont.)

ESTIMATED INITIAL CAPITAL INVESTMENT

NOTES (cont.):

4. Surface electrical estimate does not include transmission lines to mine site.
5. No roads to the mine site are included.
6. Site preparation includes an estimate of $200,000 for initial environmental repair and protection such as landscaping, drainage around the mine site, selective storage of soils, etc.

TABLE 4 - 3

SUMMARY OF DEFERRED CAPITAL INVESTMENT
AT 1,140,000 TONS MINED PER YEAR

| | Replacement at 5 years | Replacement at 10 years | Replacement at 15 years |
|---|---|---|---|
| Mining machinery | | $6,405,000 | |
| Underground transportation and support equipment | | 707,000 | |
| Bulk rock dust storage and distribution equipment | | 320,000 | |
| Underground electrical and communication equipment | | 520,000 | |
| Mine indicator water supply | | 465,000 | |
| Mine ventilating air locks | | 57,000 | |
| Exhaust fans | | 57,000 | |
| Section roof support equipment | $1,056,000 | 1,056,000 | $1,056,000 |
| First aid, safety rescue, and testing | 73,000 | 73,000 | 73,000 |
| Concrete underground roadways | 440,000 | 440,000 | 440,000 |
| Office, warehouse and shop equipment, lamps and charging racks | 220,000 | 220,000 | 220,000 |
| Coal preparation plant machinery | | 650,000 | |

TABLE 4 - 3 (Cont.)

SUMMARY OF DEFERRED CAPITAL INVESTMENT
AT 1,140,000 TONS MINED PER YEAR

| | Replacement at 5 years | Replacement at 10 years | Replacement at 15 years |
|---|---|---|---|
| Surface vehicles | 42,000 | 42,000 | 42,000 |
| Bulldozer, front end loader | | 250,000 | |
| Potable water system | | 83,000 | |
| Sewage treatment system | | 66,000 | |
| Totals | 1,831,000 | 11,411,000 | 1,831,000 |
| Contingency | 160,000 | 160,000 | 160,000 |
| Total | $1,991,000 | $11,571,000 | $1,991,000 |

| | |
|---|---|
| Summary: | |
| 5 years | $ 1,831,000 |
| 10 years | 11,411,000 |
| 15 years | 1,831,000 |
| Add: annual contingency outlay for equipment, drilling, etc. (20 yrs. x $160,000 per year) | 3,200,000 |
| Total deferred capital | $18,273,000 |

TABLE 4 - 4

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 1 | Continuous Miner | 7 | Lee Norse HH-486 | $285,000 | |
| | | | Cable | 5,000 | |
| | | | Ground Fault Monitor | 3,200 | |
| | | | Cab | 1,400 | |
| | | | Remote Control | 14,500 | |
| | | | | 309,100 | |
| | | | Sales Tax | 15,455 | |
| | | | Freight | 6,500 | |
| | | | Misc. | 945 | |
| | | | | $332,000 | $2,324,000 |
| 2 | Coal Drill | 7 | Eimco-Secoma 350B-2G | | |
| | | | (Cable included) | $95,000 | |
| | | | Cab | 3,000 | |
| | | | Sales Tax | 4,900 | |
| | | | Duty and Freight | 15,012 | |
| | | | Misc. | 2,088 | |
| | | | | $120,000 | 840,000 |
| 3 | Shuttle Car | 14 | Joy 10SC (Complete) | $90,000 | |
| | | | Sales Tax | 4,380 | |
| | | | Freight | 2,210 | |
| | | | Misc. | 410 | |
| | | | | $97,000 | 1,358,000 |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 4 | Clean-up Machine | 7 | Eimco 911-26 (Cable Incl.) | $45,000 | |
| | | | Fire Suppression | 1,200 | |
| | | | Sales Tax | 2,310 | |
| | | | Misc. | 490 | |
| | | | | $49,000 | $343,000 |
| 5 | Power Center | 7 | 750 KVA (Complete), (Delivered) | $30,000 | 210,000 |
| 6 | Feeder-Breaker | 6 | Long Airdox Rosco 2 | $40,000 | |
| | | | Sales Tax | 2,000 | |
| | | | Freight | 2,177 | |
| | | | Misc. | 823 | |
| | | | | $45,000 | 270,000 |
| 7 | Section Belt Conveyor Drive | 6 | Long Airdox 42" 150-HP (Complete with tail pulley) (Delivered) | $26,000 | 156,000 |
| 8 | Section Belt Conveyor Hardware | 6 | Hardward, Complete $14.00 per foot. Amount required 6 x 4,430 feet x $14.00 | $62,000 | 372,000 |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 9 | 42" Section Conveyor Belting | 1 | $9.50 per foot. Amount required 6 x 2 x 4,430 feet x $9.50 | | $505,000 |
| 10 | Belt Fire Detection | 1 | Master Monitor | $1,000 | |
| | | 4 | Receivers | 400 | |
| | | 4 | Belt Monitors (Sensors at 50 foot intervals.) | 2,800 | |
| | | | 37¢/foot x 6 x 4,430 feet | 10,600 | |
| | | | Misc. Controls | 6,000 | |
| | | | | $20,800 | 20,800 |
| 11 | Main Belt Conveyor | 1 (initial) | Long Airdox (Complete with tail pulley and fire deluge) (Delivered) | | 63,000 |
| 12 | Main Belt Conveyor Hardware | | 48" Hardware $27.00/foot installed. Amount required 1,500 feet x $27.00 | | 42,000 |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 13 | Main Belt Conveyor Belting | | 48" Belting $13.00/foot<br>Amount required<br>3,000 feet x $13.00 | | $39,000 |
| 14 | Utility Tractor | 4 | Battery or diesel powered<br>(Estimate)<br>Sales Tax<br>Freight<br>Misc. | <br>$50,000<br>2,500<br>2,000<br>500<br>$55,000 | 220,000 |
| 15 | Material Supply Trailer | 18 | Kersey 7-1/2 Ton<br>(Delivered) | $5,000 | 90,000 |
| 16 | Heavy-Duty Tractor for Continuous Miner | 1 | Kersey 24 (Complete with spare battery and charger)<br>(Delivered) | | 96,000 |
| 17 | Heavy-Duty Trailer for Continuous Miner | 1 | Estimate (Delivered) | | 15,000 |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 18 | Personnel Carriers | 7 | Long Airdox (Complete with batteries and chargers) | $14,500 | |
| | | | Freight | 200 | |
| | | | Sales Tax | 225 | |
| | | | Misc. | 1,075 | |
| | | | | $16,000 | $112,000 |
| 19 | Portable Mechanic's Station | 6 | Estimate (Complete) | $3,000 | 18,000 |
| 20 | Portable Bulk Hydraulic Oil Tank | 6 | 300 gallon capacity. Estimate (Complete) | $2,000 | 12,000 |
| 21 | Bulk Rock Dust System | | MSA | | |
| a | | 1 | 142-ton Bin (Surface) | $25,000 | |
| b | | 6 | 6-ton Pressure Tanks | 150,000 | |
| c | | 2 | 6-ton Bulk Rock Dust Carriers | 30,000 | |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 21 | Bulk Rock Dust System (Cont.) | | | | |
| d | | | 3" thin wall MSA Pressure Pipe @$3.50/foot including fittings (Installed) Amount required 6 x 4,500 x $3.50 | 94,500 | |
| e | | | Control Hoses and Valves @ $350 per section | 2,100 | |
| f | | 2 | Entry Rock Dust Machines, Long Airdox BVD (Complete) (Delivered) | 18,400 | |
| | | | Rock Dust System Subtotal | | $320,000 |
| 22 | Section Entry Exhaust Fan | 3 | Joy 29 1/4 - 17 Fan Including Trickle Duster (Complete) (Delivered) | $14,000 | 42,000 |
| a | Fiberglass Rigid Tubing | | 18" Tubing Including Fittings ABC Corp. | 15,000 | 15,000 |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 23 | Section Roof Support Equipment | 6 | | | |
| a | | | 16-foot Aluminum Crossbars 5083 Al 1/2" x 4" x 6" Box Section @ $200. 300 per section | $60,000 | |
| b | | | 10-ton Commercial Shearing Hyd. Jacks @ $170. 560 per section. | 95,200 | |
| c | | | 18-foot Steel Bridge Bars 1/4" x 4" x 6" Box Section @ $70. 22 per section. | 1,500 | |
| d | | | 20-ton Commercial Shearing Hyd. Jacks @ $235. 80 per section. | 18,800 | |
| e | | | Misc. | 500 | |
| | | | Section Roof Support Equipment Subtotal | $176,000 | $1,056,000 |

TABLE 4 - 4 (Con't)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 24 | Underground Electrical and Communications System | | | | |
| a | | | Main Entry 250 MCM 8,000 Volt Armored Cable Including PLM Couplers and Messenger Wire. $15/foot x 1,500 feet. | $23,000 | |
| b | | | Section Entry No. 1, 8,000 Volt Armored Cable Including PLM Couplers and Messenger Wire. $10/foot x 26,000 feet | 260,000 | |
| c | | 8 | Main Sectionalizing Circuit Breakers | 96,000 | |
| d | | 6 | Section Belt Power Center | 72,000 | |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 24 | Underground Electrical and Communications System (cont.) | | | | |
| e | | | Main Belt Power Center | $19,000 | |
| f | | | Dial Telephone System (Installed) | 50,000 | |
| | | | Underground and Electrical Communications System Subtotal | $520,000 | $520,000 |
| 25 | Industrial Mine Water System | | | | |
| a | | | Main Entry 6" Line @ $12/foot x 1,500 feet | $18,000 | |
| b | | | Section Entry 4" Line @ $5.80/foot Amount required $5.80 x 27,000 feet. | 156,600 | |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 25 | Industrial Mine Water System (cont.) | | | | |
| c | | | 400,000 Gallon Tank | $75,000 | |
| d | | | Water Supply to Tank | 100,000 | |
| e | | | Mine Drainage System | 100,000 | |
| f | | | Section Small Water Lines, Hoses, Valves | 12,000 | |
| g | | | Misc. | 3,400 | |
| | | | Industrial Mine Water System Subtotal | $465,000 | $465,000 |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 26 | Concreted Mine Roadways | | 12' wide, average 4.5" thick, wire netting reinforced. One cubic yard per 6' of entry. Concrete 2 roadways in main entryway. Concrete 1 roadway in each section. 28,800 feet divided by 6 feet per cubic yard = 4,800 cubic yards. Estimate $104 per cubic yard installed | | $500,000 |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 27 | Surface Electrical Installation | | One 5,000 KVA substation at mine surface plant location, 7.2 KV secondary switches, lightning protection, fencing, etc. Misc. 7.2 KV distribution transmission lines on surface. One 1,000 KVA substation, 7.2 KV to 480 volts, to serve the coal preparation plant, mine fans, shop, office and bathhouse. | | $ 400,000 |
| 28 | Coal Preparation Plant | | A plant to crush the coal to 2-inch nominal size and deliver it to a 1,000 ton bin for rail car loading. The facility will consist of: | | |
| a | | 1 | 48" conveyor to R.O.M. transfer conveyor. | | |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 28 | Coal Preparation Plant (cont.) | | | | |
| b | | 1 | 80' long, pivoted, moveable stacker conveyor, 48", to place coal in a surge pile in the event of rail shut-down. | | |
| c | | 1 | 48" R.O.M. transfer conveyor to feed R.O.M. bin. | | |
| d | | 1 | Suspended tramp iron magnet and cleaning facility at R.O.M. bin. | | |
| e | | 1 | Steel 1,000 ton R.O.M. coal storage bin. | | |
| f | | 1 | 60" x 80" Syntron H.D. vibrating feeder. | | |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 28 | Coal Preparation Plant (cont.) | | | | |
| g | | 1 | 42" conveyor, to screening and crushing plant. | | |
| h | | 1 | 8' x 20' vibrating screen. | | |
| i | | 1 | Penna. ring crusher | | |
| j | | 1 | 48" conveyor, to loading bin. | | |
| k | | 1 | Steel 1,000 ton loading bin. | | |
| l | | 1 | Motor driven bin gain, remote operated. | | |
| m | | 1 | Belt scale | | |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 28 | Coal Preparation Plant (cont.) | | | | |
| n | | | Misc. structural steel, foundations, platework, wire, conduit, lighting, and electrical heating. | | $1,305,000 |
| 29 | Industrial Buildings | | One mine industrial building, 150' x 80', insulated and divided for lamp room, safety engineer's office, combination dressing and training room, shower room for 100 miners per shift, toilet and lavatory room, lady's toilet, superintendent's office, foreman's office, secretary and clerk's office, waiting room, sprinkler system, | | |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 29 | Industrial Buildings (co) | | electric heat, plumbing, lighting, concrete foundation and floor, including also the mine shop, insulated, with lighting and heating, concrete foundation and floor, and the mine warehouse, insulated, no heat and electrical, concrete foundation and floor, erected. (Note lighting provided in electrical estimate.) One oil building 24' x 36'. One roof support storage shed. One explosives storage facility. | | $ 800,000 |
| 30 | Shop, Warehouse and Office Equipment | | One 400 amp. electric welder. One lot tools, including jacks, hoists. One lot warehouse racks, bins. One lot office equipment, including desks, chairs, tables, file cabinets. | | |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 30 | Shop, Warehouse and Office Equipment (cont.) | | First aid room table, cabinets. 200 locker baskets with hangers and locks. Change room benches for 200 men. Lamps and charging racks. | | $ 220,000 |
| 31 | Mine Rescue, Safety, First Aid Equipment | | Rescue equipment - includes rescue stations underground with chemical breathing apparatus, with replacement chemicals and canisters, cap lights, oxygen administering equipment, oxygen, carbon monoxide and methane measuring devices, stretchers, blankets and first aid material. Self rescuers, goggles and miners' hats, special clothing, first aid and dispensary equipment, continuous methane monitoring devices. | | |

TABLE 4 - 4 (Cont.)

EQUIPMENT AND FACILITIES COST ESTIMATES

| Number | Item | No. Req'd | Description | Cost | Total Cost |
|---|---|---|---|---|---|
| 31 | Mine Rescue, Safety, First Aid Equipment (cont.) | | Portable instruments for measuring oxygen, methane, carbon monoxide concentration, air flow and dust concentration. | | 73,000 |
| 32 | Miscellaneous Surface Vehicles | | | | |
| a | | 1 | Dump truck - 5 ton | | |
| b | | 1 | Utility truck - 5 ton | | |
| c | | 1 | Pick-up truck - 3/4 ton | | |
| d | | 1 | Company car | | 42,000 |

TABLE 4 - 5

WORKING CAPITAL DETERMINATION

Assume three months cost of the following items:

| | Item | Annual Cost | Three Months' Cost |
|---|---|---|---|
| 1. | Direct Labor | $2,770,000 | $693,000 |
| 2. | Mine and Plant Supplies | 1,984,000 | 496,000 |
| 3. | Payroll Overheads | 448,000 | 112,000 |
| 4. | Indirect Costs | 713,000 | 178,000 |
| 5. | Taxes, Insurance (2% of mine cost) | 378,000 | 95,000 |
| 6. | Spare parts Inventory | ------- | 250,000 |
| | Total | | $1,824,000 |

TABLE 4 - 6

## ESTIMATED OPERATING COSTS

| Item | Annual Cost | Cost per Ton |
|---|---|---|
| Labor and Payroll Overhead (1) | $3,218,000 | $2.82 |
| Mine and Plant Supplies (2) | 1,984,000 | 1.74 |
| UMWA Welfare (3) | 1,083,000 | .95 |
| Taxes, Insurance | 378,000 | .33 |
| Black Lung Insurance (35% of labor) | 722,000 | .63 |
| Depreciation | 1,696,000 | 1.49 |
| Depletion of Reserves (5) | 285,000 | .25 |
| Unexpected Mine Problems (6) | 228,000 | .20 |
| Mine Extension Costs | 285,000 | .25 |
| Indirect Costs, Administration (15% of labor supervision and supplies) | 713,000 | .63 |
| Cost before royalty | $10,592,000 | $9.29 |
| Federal Royalty on Mining Lease (assume 8% of annual sales of $15,071,100) | 1,205,700 | 1.06 |
| Total Operating Costs | $11,797,700 | $10.35 |

NOTES:
1. See supporting Tables 4-7 thru 4-11
2. See Table 4-12
3. See Table 4-10
4. Depletion of reserves represents the declining resale value of the coal property and should not be confused with the Depletion Allowance granted for tax purposes.
5. Represents an operating contingency.
6. Mine extension costs represents primarily the cost of extending mine utilities and underground support facilities not otherwise accounted for.

TABLE 4 - 7

LABOR COST SUMMARY

| Labor | No. Men | Cost per Day | Annual Rate | $/Ton |
|---|---|---|---|---|
| Mine Sections | 76 | $4,972 | $1,143,560 | $1.00 |
| General Mine | 20 | 1,253 | 288,190 | .25 |
| 3rd Shift | 14 | 980 | 225,400 | .20 |
| Outside | 30 | 2,026 | 465,980 | .41 |
| **SUBTOTAL** | 140 | $9,231 | $2,123,130 | $1.86 |
| Salary | 38 | 2,812 | 646,750 | .57 |
| TOTAL PAYROLL | 178 | $12,043 | $2,769,880 | $2.43 |

TABLE 4 - 8

LABOR COSTS - HOURLY PERSONNEL (1)

Mine Sections

Retreat (4 sections, 8 shifts)

| Men per Section Shift | Men per Day | Occupation | Base Rate per Shift | Base + Fringe + Overtime | Total per Day | Total per Year |
|---|---|---|---|---|---|---|
| 1 | 8 | Continuous Miner Operator | $55.00 | $68.75 | $550.00 | $126,500.00 |
| 1 | 8 | Continuous Miner Helper | 51.98 | 64.98 | 520.00 | 119,600.00 |
| 2 | 16 | Shuttle Car Operator | 49.23 | 61.54 | 985.00 | 226,550.00 |
| 2 | 16 | Coal Drill Operator and Shot Firer | 55.00 | 68.75 | 1,100.00 | 253,000.00 |
| 1 | 8 | Timber and Utility Man | 49.23 | 61.54 | 492.00 | 113,160.00 |
| 1 | 8 | Mechanic | 55.00 | 68.75 | 550.00 | 126,500.00 |
| 8 | 64 | Total | $315.44 | $394.31 | $4,197.00 | $965,310.00 |

NOTE: 1. 1976 labor rates are used throughout the cost estimates.

TABLE 4 - 8 (cont.)

LABOR COSTS - HOURLY PERSONNEL

Advance (1 section, 2 shifts)

| Men per Section Shift | Men per Day | Occupation | Base Rate per Shift | Base + Fringe + Overtime | Total per Day | Total per Year |
|---|---|---|---|---|---|---|
| 1 | 2 | Continuous Miner Operator | $55.00 | $68.75 | $138.00 | $31,740.00 |
| 1 | 2 | Continuous Miner Helper | 51.00 | 64.98 | 130.00 | 29,900.00 |
| 2 | 4 | Shuttle Car Operator | 49.23 | 61.54 | 246.00 | 56,580.00 |
| 1 | 2 | Timber and Utility Man | 49.23 | 61.54 | 123.00 | 28,290.00 |
| 1 | 2 | Mechanic | 55.00 | 68.75 | 138.00 | 31,740.00 |
| 6 | 12 | Total | $259.46 | $325.56 | $775.00 | $178,250.00 |

TABLE 4 - 8 (cont.)

LABOR COSTS - HOURLY PERSONNEL

General Mine

| Men/Shift | Men/Day | Occupation | Base Rate per Shift | Base + Fringe + Overtime | Total per Day | Total per Year |
|---|---|---|---|---|---|---|
| 2 | 4 | Pipemen | $49.23 | $61.54 | $246.00 | $56,580.00 |
| 1 | 2 | Wiremen | 49.23 | 61.54 | 123.00 | 28,290.00 |
| 3 | 6 | Masons | 47.03 | 58.79 | 353.00 | 81,190.00 |
| 1 | 2 | Roving Mechanic | 55.00 | 68.75 | 138.00 | 31,740.00 |
| 2 | 4 | Material Supply Men | 55.00 | 68.75 | 275.00 | 63,250.00 |
| 1 | 2 | Belt Men | 47.03 | 58.79 | 118.00 | 27,140.00 |
| 10 | 20 | Total | $302.52 | $378.16 | $1,253.00 | $288,190.00 |

TABLE 4 - 8 (cont.)

LABOR COSTS - HOURLY PERSONNEL

General Mine - 3rd Shift

| Men/Shift | Occupation | Base Rate per Shift | Base + Fringe + Overtime | Total per Day | Total per Year |
|---|---|---|---|---|---|
| 2 | Fire Boss | $55.00 | $69.75 | $140.00 | $32,200.00 |
| 2 | Mechanics | 55.00 | 69.75 | 140.00 | 32,200.00 |
| 4 | Material Supply Men* | 55.00 | 69.75 | 280.00 | 64,400.00 |
| 4 | Rock Dusters* | 55.00 | 69.75 | 280.00 | 64,400.00 |
| 2 | Belt Movers* | 55.00 | 69.75 | 140.00 | 32,200.00 |
| 14 | Total | $275.00 | $348.75 | $980.00 | $225,400.00 |

*Belt Moving Crew

TABLE 4 - 8 (cont.)

LABOR COSTS - HOURLY PERSONNEL

Outside

| Men/Shift | Men/Day | Occupation | Base Rate per Shift | Base + Fringe + Overtime | Total per Day | Total per Year |
|---|---|---|---|---|---|---|
| 1 | 2 | Lamp and Bathhouse Attnd. | $49.23 | $61.54 | $123.00 | $28,290.00 |
| 3 | 6 | Material Supply and Front End Load Operator | 51.98 | 64.98 | 390.00 | 89,700.00 |
| 8 | 16 | Mechanic-Shop | 55.00 | 68.75 | 1,100.00 | 253,000.00 |
| 3 | 6 | Mechanic-Tipple Operator | 55.00 | 68.75 | 413.00 | 94,990.00 |
| 15 | 30 | Total | $211.21 | $264.02 | $2,026.00 | $465,980.00 |
| | | Total Hourly Labor Cost | ------ | ------ | $9,231.00 | $2,123,130.00 |

TABLE 4 - 9

LABOR COSTS - SALARIED PERSONNEL

Salary

| Men | Occupation | Base rate per Month | Total per Day | Total per Year |
|---|---|---|---|---|
| 1 | Superintendent | $2,400.00 | $125.21 | $28,800.00 |
| 1 | Master Mechanic | 1,850.00 | 96.52 | 22,200.00 |
| 1 | Mine Foreman | 1,800.00 | 93.91 | 21,600.00 |
| 1 | Assistant Mine Foreman | 1,700.00 | 88.70 | 20,400.00 |
| 1 | Foreman General Mine and Outside | 1,680.00 | 87.65 | 20,160.00 |
| 10 | Section Unit Foremen | 1,680.00 | 876.52 | 201,600.00 |
| 1 | Office Manager | 1,580.00 | 82.43 | 18,960.00 |
| 1 | Chief Engineer | 1,875.00 | 97.82 | 22,500.00 |
| 1 | Mine Engineer | 1,700.00 | 88.70 | 20,400.00 |
| 1 | Design Engineer | 1,680.00 | 87.65 | 20,160.00 |
| 1 | Draftsman | 1,325.00 | 69.13 | 15,900.00 |
| 2 | Mine Surveyors | 1,325.00 | 138.26 | 31,800.00 |
| 1 | Industrial Engineer | 1,680.00 | 87.65 | 20,160.00 |
| 1 | Safety Director | 1,680.00 | 87.65 | 20,160.00 |
| 2 | Safety Inspectors | 1,208.00 | 126.05 | 28,992.00 |
| 2 | Dust and Noise Qualified Men | 875.00 | 91.30 | 21,000.00 |
| 3 | Stenographers | 660.00 | 103.30 | 23,760.00 |
| 1 | Personnel Man | 1,325.00 | 69.13 | 15,900.00 |
| 1 | Time Keeper | 917.00 | 47.83 | 11,000.00 |
| 2 | Accountants | 1,317.00 | 137.40 | 31,600.00 |
| 3 | Warehouse Men | 825.00 | 129.13 | 29,700.00 |
| 38 | Total Salary Costs | ------- | $2,811.94 | $646,750.00 |
| 178 | Total Payroll (Hourly and Salary) | | $12,043.00 | $2,769,880.00 |

TABLE 4 - 10

MINE LABOR FRINGE BENEFITS

| | | |
|---|---|---|
| Base Rate - Continuous Miner Operator - Per Shift | | $55.00 |
| Add | | |
| Regular Vacation -Per Shift | $2.75 | |
| Birthday, Extra Vacation | 0.72 | |
| Cost of Living Increase - 1st Year | 1.20 | |
| Second Shift Differ. - Avg of 1st and 2nd Shifts | 0.60 | |
| Paid Holidays | 2.39 | |
| Soap | 0.09 | |
| Safety Clothes | 0.33 | |
| Sick Leave | 1.20 | 9.28 |
| Overtime - 5% at Time and One-Half | | 4.13 |
| TOTAL: | | $68.41 |

$\frac{68.41}{55.00}$ = 1.244 - Use 25% Increase

For 3rd Shift, Add $1.00 Per Shift to Above

## TABLE 4 - 10 (cont.)

## MINE LABOR WELFARE COSTS PER TON

| | | |
|---|---|---|
| $\dfrac{\$0.90 \times 140 \text{ men per day} \times 8 \text{ hrs.}}{4942 \text{ tons}} =$ | | \$0.20 |
| Add 5% for overtime | | 0.01 |
| Welfare charge - per ton | | 0.74 |
| | TOTAL | \$0.95 |

TABLE 4 - 11

PAYROLL OVERHEADS - ANNUAL COST (1)

Hourly Personnel

| | |
|---|---|
| Social Security - 5.85% on a $15,300 base<br>140 men x $15,300 x .0585 | $125,307 |
| Workmen's Compensation - 4% of payroll<br>(2,035,400) x .04 | 81,416 |
| Unemployment Compensation - Utah - 2.7% on a $4,200 base<br>140 men x 4,200 x .027 | 15,876 |
| Unemployment Compensation - Federal - .7% on a $4,200 base<br>140 men x 4,200 x .007 | 4,116 |
| Liability and Property Insurance - 1% of payroll<br>2,123,130 x .01 | 21,231 |
| TOTAL | $247,946 |

NOTE: 1. 1976 rate and base wage used.

TABLE 4 - 11 (cont.)

PAYROLL OVERHEADS - ANNUAL COST

| | | |
|---|---|---|
| Salary Personnel: | | |
| Social Security<br>38 x $15,300 x .0585 | | $34,012 |
| Worksman's Compensation<br>627,852 x .04 | | 25,114 |
| Unemployment Compensation - Utah<br>38 x 4,200 x .027 | | 4,309 |
| Unemployment Compensation - Federal<br>38 x 4,200 x .007 | | 1,117 |
| Liability and Property Insurance<br>646,752 x .01 | | 6,468 |
| | Subtotal | $71,020 |
| Fringe Benefits - Salaried Personnel (Assume 20% of payroll)<br>646,752 x .20 | | 129,350 |
| | TOTAL | $200,370 |
| TOTAL - HOURLY & SALARY PAYROLL OVERHEAD | | $448,316 |

TABLE 4 - 12

MINE AND PLANT SUPPLIES

| | Cost per Ton of Coal Mined | Annual Cost |
|---|---|---|
| Machine Parts | $0.63 | $718,200 |
| Lube & Hydraulic Oil | 0.25 | 285,000 |
| Timber | 0.02 | 22,800 |
| Rock Dust | 0.12 | 136,800 |
| Ventilation Materials | 0.10 | 114,000 |
| Bits | 0.02 | 22,800 |
| Trailing Cables, Connectors | 0.06 | 68,400 |
| Power | 0.34 | 387,600 |
| Explosives, Blasting Supplies | 0.05 | 57,000 |
| Miscellaneous | 0.15 | 171,000 |
| TOTAL | $1.74 | $1,983,600 |

TABLE 4 - 13

DEPRECIATION SCHEDULE

| Number | Item | Straightline Depreciation - Years | Yearly Charge |
|---|---|---|---|
| 1 | Section Roof Support Equipment | 5 | $211,000 |
| 2 | First Aid, Safety, Rescue, Testing Instruments | 5 | 15,000 |
| 3 | Office, Warehouse, Shop Equipment Lamps, and Charging Racks | 5 | 41,000 |
| 4 | Surface Vehicles | 5 | 8,400 |
| 5 | Underground Roadways (Section) | 5 | 88,000 |
| 6 | Mining Machinery | 10 | 640,500 |
| 7 | Underground Transportation and Support Equipment | 10 | 70,600 |
| 8 | Bulk Rock Dust System | 10 | 32,000 |
| 9 | Industrial Mine Water System | 10 | 46,500 |
| 10 | Bulldozer, Front-end Loader | 10 | 25,000 |

TABLE 4 - 13 (Cont.)

DEPRECIATION SCHEDULE

| Number | Item | Straightline Depreciation - Years | Yearly Charge |
|---|---|---|---|
| 11 | Potable Water System | 10 | $8,300 |
| 12 | Sewage Treatment System | 10 | 6,600 |
| 13 | Exhaust Fans | 10 | 5,700 |
| 14 | Mine Portal Air Locks | 10 | 5,700 |
| 15 | Underground Electrical and Communication System | 10 | 52,000 |
| 16 | Coal Preparation Plant Machinery | 10 | 65,000 |
| 17 | Portals | 20 | 3,600 |
| 18 | Surface Electrical System | 20 | 20,000 |
| 19 | Ventilation Fans (less air locks) | 20 | 13,700 |
| 20 | Coal Preparation Plant (less machinery) | 20 | 32,800 |

TABLE 4 - 13 (Cont.)

DEPRECIATION SCHEDULE

| Number | Item | Straightline Depreciation - Years | Yearly Charge |
| --- | --- | --- | --- |
| 21 | Industrial Buildings | 20 | $40,000 |
| 22 | Roads | 20 | 5,000 |
| 23 | Underground Concrete Roadways (Main Entries) | 20 | 3,100 |
| 24 | Site Investigation and Preparation | 20 | 16,800 |
| 25 | Engineering and Fees | 20 | 23,900 |
| 26 | Overhead and Administration | 20 | 49,400 |
| 27 | Contingency | 20 | 116,800 |
| 28 | Development and Exploration Costs in excess of Value of Coal Recovered During Exploration. | 20 | 50,000 |
| | Total | | $1,696,400 |

TABLE 4 - 14

SUMMARY OF DISCOUNTED CASH FLOW
1,140,000 TONS MINED PER YEAR

| Year | Capital Investment | Cash Flow | Present worth factor at 15% | Present worth capital investment at 15% | Present worth cash flow value at 15% |
|---|---|---|---|---|---|
| 0 | $20,730,000 | ($20,730,000) | 1.0000 | $20,730,000 | ($20,730,000) |
| 1 | 160,000 | 3,926,200 | .8696 | 139,100 | 3,414,200 |
| 2 | 160,000 | 3,926,200 | .7561 | 121,000 | 2,968,600 |
| 3 | 160,000 | 3,926,200 | .6575 | 105,200 | 2,581,500 |
| 4 | 160,000 | 3,926,200 | .5718 | 91,500 | 2,245,000 |
| 5 | 1,991,000 | 2,095,200 | .4972 | 990,000 | 1,041,700 |
| 6 | 160,000 | 3,926,200 | .4323 | 69,200 | 1,697,300 |
| 7 | 160,000 | 3,926,200 | .3759 | 60,100 | 1,475,800 |
| 8 | 160,000 | 3,926,200 | .3269 | 52,300 | 1,283,400 |
| 9 | 160,000 | 3,926,200 | .2843 | 45,500 | 1,116,200 |
| 10 | 11,571,000 | (7,484,800) | .2472 | 2,860,400 | (1,850,300) |
| 11 | 160,000 | 3,926,200 | .2149 | 34,400 | 843,700 |
| 12 | 160,000 | 3,926,200 | .1869 | 29,900 | 733,800 |
| 13 | 160,000 | 3,926,200 | .1625 | 26,000 | 638,000 |
| 14 | 160,000 | 3,926,200 | .1413 | 22,600 | 554,700 |
| 15 | 1,991,000 | 2,095,200 | .1229 | 244,700 | 257,500 |

TABLE 4 - 14 (Cont.)

SUMMARY OF DISCOUNTED CASH FLOW
1,140,000 TONS MINED PER YEAR

| Year | Capital Investment | Cash Flow | Present worth factor at 15% | Present worth capital investment at 15% | Present worth cash flow value at 15% |
|---|---|---|---|---|---|
| 16 | 160,000 | 3,926,200 | .1069 | 17,100 | 419,700 |
| 17 | 160,000 | 3,926,200 | .0929 | 14,900 | 364,700 |
| 18 | 160,000 | 3,926,200 | .0808 | 13,000 | 317,200 |
| 19 | 160,000 | 3,926,200 | .0703 | 11,200 | 276,000 |
| *20 | 1,664,000 | 5,750,200 | .0611 | (101,700) | 351,300 |
| | | | | 25,576,400 | -0- |

| | |
|---|---|
| * Year 20 | |
| Recovery of Working Capital | $1,824,000 |
| Less annual Deferred Cost | 160,000 |
| | $1,664,000 |

NOTE 1. Annual allocation for unforseeable equipment replacements and acquisitions.

## TABLE 4 - 15

## DISCOUNTED CASH FLOW

15 percent - 20 years at 1,140,000 tons mined per year

R = 25,576,400 divided by 6.259 = \$4,086,300
      less depreciation                          1,696,000
      depletion + net profit         = \$2,390,300

Depletion = 10% of Sales
Federal income tax = 50% of net profit
Depletion + net profit = cash flow - depreciation

Sales = 1/0.55 (½ operating cost + cash flow - depreciation)
      = 1/0.55 (5,898,800 + 2,390,300)
      = \$15,071,100

| | |
|---|---|
| Sales | \$15,071,000 |
| Operating cost | 11,797,700 |
| Gross profit | \$ 3,273,300 |
| Depletion (10% of Sales) | 1,507,100 |
| Taxable income | \$1,766,200 |
| Federal income tax | 883,100 |
| Net Profit | \$883,100 |

Annual Cash Flow = net profit + depreciation + depletion
      = 883,100 + 1,696,000 + 1,507,100
      = \$4,086,200

Selling price per ton = 15,071,000 divided by 1,140,000
      = \$13.22

Computation of Royalty:

$$R = .08 \times \text{Sales}$$
$$S = 1/0.55\ (\tfrac{1}{2}\ (10{,}592{,}000 + R) + 2{,}390{,}200$$
$$S = 1/0.55\ (\tfrac{1}{2}\ (10{,}592{,}000 + (.08S) + 2{,}390{,}200)$$
$$S = 1/0.55\ (5{,}296{,}000 + .08S/2 + 2{,}390{,}200)$$
$$S = 5{,}296{,}000/0.55 + (.08S/2 \times 1/0.55) + 2{,}390{,}200/0.55$$
$$S = 9{,}629{,}000 + (0.08S/1.10) + 4{,}346{,}000$$
$$S - .07272S = 13{,}975{,}100$$
$$.9273S = 13{,}975{,}100$$
$$S = \underline{\underline{\$15{,}071{,}100}}$$

$$R = .08 \times 15{,}071{,}100 = \underline{\underline{\$1{,}205{,}700}}$$

## SECTION 5

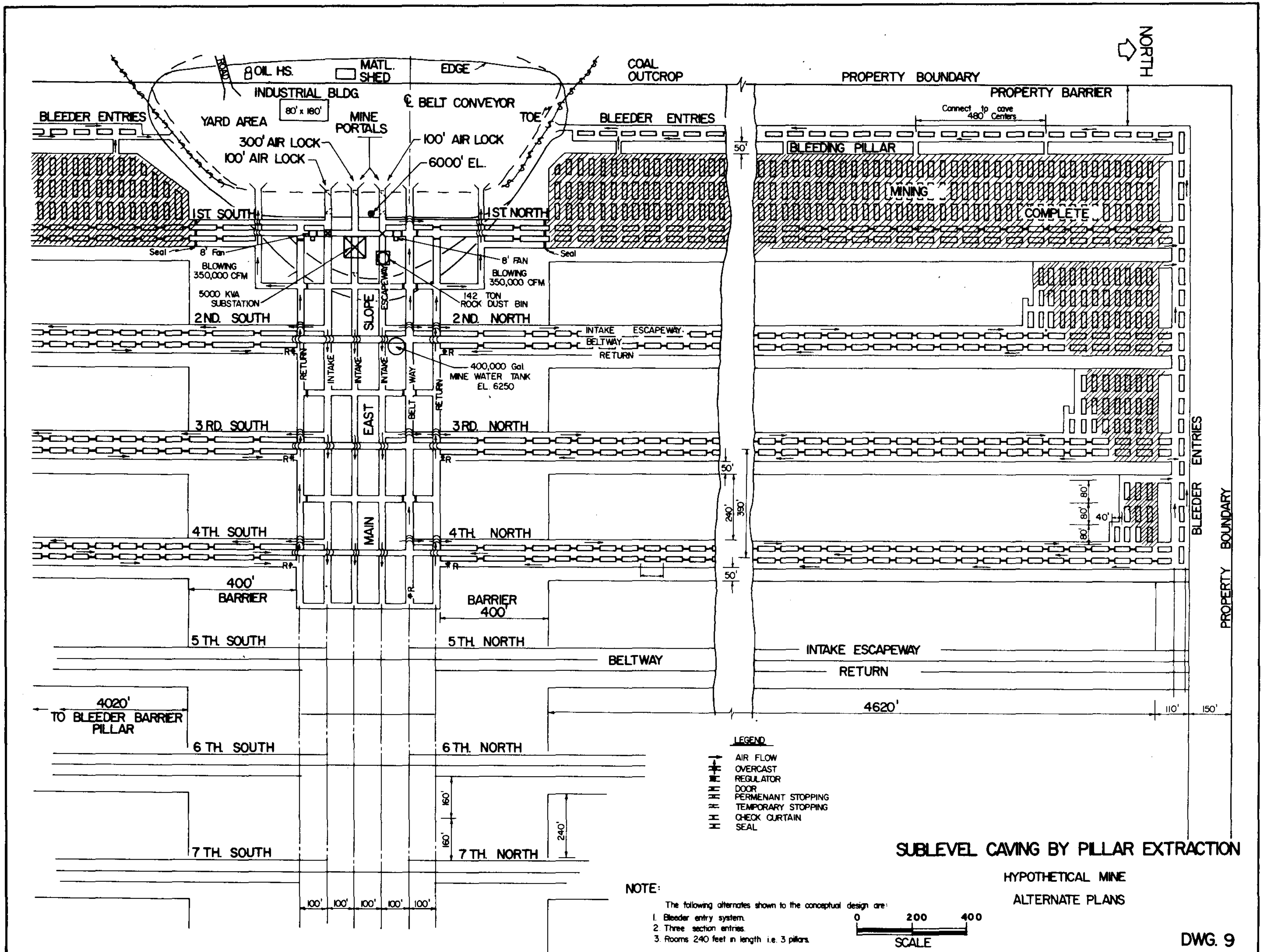
## ALTERNATIVE METHODS AND PROCEDURES

Alternatives to the methods, equipment and procedures shown for the Sublevel Caving by Pillar Extraction mining method are numerous due to its many similarities with standard room and pillar mining. There are numerous combinations of face and support machinery which would suit the concept with the provision that the coal loading machine be one which places the operator a safe distance from the top coal brow. As is the case with any mining venture, the selection of dimensions for openings and pillars is largely determined by local and unique geological conditions. It is essential, however, that the fundamental ground control principles recommended be adhered to for this method to succeed. Drilling and shooting, one of the most critical phases of operation, offers a wide range of alternative possibilities to those recommended.

### GENERAL MINE DESIGN ALTERNATIVES

Drawing No. 9, Alternate Mining Plans, depicts the Sublevel Caving by Pillar Extraction method with bleeder entries incorporated into the mine layout in the manner described under the General Mine Design section of this report. The alternative mining plans show three section entries which afford a separate return airway and isolation of the belt conveyor by a separate split of air. Given three section entries, there would be no necessity to apply to MESA for an exception on the number of entries. Three section entries should improve productivity on advance and offer more operating room on retreat. More operating room should diminish safety hazards associated with crowded work places. The primary disadvantages which argue against the three section entry plan are that ground control would be substantially less certain and roof support more difficult and complicated.

The number of pillars have been reduced from five to three. This reduction shortens the rooms from 400 feet to 240 feet. These modifications offer a number of advantages. As the rooms are much shorter, they would not have to stand open as long--an important consideration in preserving roof integrity. This alternative also reduces shuttle car travel times which aids productivity. Due to relatively more coal lost to fire barriers, percentage recovery



NORTH
COAL OUTCROP
PROPERTY BOUNDARY
PROPERTY BARRIER
OIL HS.
MATL. SHED
EDGE
INDUSTRIAL BLDG.
80' x 180'
℄ BELT CONVEYOR
YARD AREA
MINE PORTALS
TOE
BLEEDER ENTRIES
BLEEDER ENTRIES
Connect to cave 480' Centers
300' AIR LOCK
100' AIR LOCK
100' AIR LOCK
6000' EL.
BLEEDING PILLAR
MINING
COMPLETE
1ST SOUTH
1ST NORTH
Seal
8' Fan
BLOWING 350,000 CFM
8' FAN
BLOWING 350,000 CFM
5000 KVA SUBSTATION
142 TON ROCK DUST BIN
2ND. SOUTH
2ND. NORTH
SLOPE
ESCAPEWAY
INTAKE ESCAPEWAY
BELTWAY
RETURN
400,000 Gal. MINE WATER TANK EL. 6250
RETURN
INTAKE
INTAKE
INTAKE
BELT WAY
RETURN
EAST
3RD. SOUTH
3RD. NORTH
MAIN
4TH. SOUTH
4TH. NORTH
50'
240'
390'
80'
40'
400' BARRIER
BARRIER 400'
5TH. SOUTH
5TH. NORTH
BELTWAY
INTAKE ESCAPEWAY
RETURN
BLEEDER ENTRIES
PROPERTY BOUNDARY
4020' TO BLEEDER BARRIER PILLAR
4620'
110'
150'
6 TH. SOUTH
6 TH. NORTH
160'
240'
7 TH. SOUTH
7 TH. NORTH
100' 100' 100' 100' 100'
LEGEND
AIR FLOW
OVERCAST
REGULATOR
DOOR
PERMENANT STOPPING
TEMPORARY STOPPING
CHECK CURTAIN
SEAL
NOTE:
The following alternates shown to the conceptual design are:
1. Bleeder entry system.
2. Three section entries.
3. Rooms 240 feet in length i.e. 3 pillars.
0 200 400
SCALE
SUBLEVEL CAVING BY PILLAR EXTRACTION
HYPOTHETICAL MINE
ALTERNATE PLANS
DWG. 9

would be diminished. More frequently occurring fire barriers in the mined-out sections of the mine would create adverse subsidence effects. The fire barriers with 240-foot rooms may also cause weight from the cave to shift into the mine openings thus negating the ground control method. In the event 240-foot rooms are used, it may be advisable to eliminate the fire barriers completely and revert to the practice of breaking into the caved area of the previously mined-out section. This procedure would increase hazards from gas and would make isolation of a section in the event of an emergency much more difficult. Elimination of the fire barriers would improve recovery and diminish adverse subsidence effects. The elimination of the fire barriers from the plan and the creation of a "clean cave" may assist in creating the desired caving action by maintaining a long break line along the cave. If the caving action were enhanced weight accumulations over the working areas may be diminished. Elimination of the fire barriers should not be attempted where the coal is prone to spontaneous combustion.

## ROOF SUPPORT ALTERNATIVES

The method recommended for roof support in the original concept proposal was bolting of the top coal with pumpable roof bolts or with resin anchor bolts. These methods are deemed sufficiently attractive for this method to be considered as alternatives for roof support. Pumpable, or resin grouted bolts, supplemented by crossbars and props as needed would offer the advantage of making the top coal itself into a more competent beam. The top coal drill selected would be able to perform both the duty of drilling the holes for the bolts and for blasting. The operating costs would probably increase using pumpable or resin bolts. Adding the bolting machine and the bolting function to the work cycle and removing the continuous miner from the face to install bolts would be serious detriments to productivity. Also bolting of top coal does not provide support in the case of weak or fractured coal as well as do the crossbar sets. The best application of the bolts may be in coal seams thick enough to justify use of this method but thin enough for the bolts to be drilled through the top coal into a rock anchor above the seam.

## MINING MACHINERY ALTERNATIVES

An alternative complement of section equipment with which the Sublevel Caving by Pillar Extraction method would be successful is as follows:

1 - Universal cutting machine equipped with a face coal drill

1 - Flexible shaft drill for drilling of top coal blast holes and holes for pumpable or resin grouted bolts.

1 - Roof bolting machine for installing pumpable or resin grouted bolts.

2 - 9.5 yard diesel LHD's with operator's cab relocated to the rear of the machine.

1 - Belt feeder-breaker

1 - Section 42-inch belt conveyor

1 - Power load center.

This is essentially the section complement originally recommended and it is still considered to be a viable choice. The LHD machines are much more mobile as loading machines and are capable of more rapid loading of the fallen top coal than the continuous miners. The LHD's are capable of functioning as clean-up machines and they perform extremely well in this service, leaving a work place almost devoid of rubble. The modification which moves the operator's station to the rear of the Eimco 915E LHD places the operator 25 feet away from the loading function compared to 20 feet with the continuous miner. The LHD is also well suited to materials handling tasks for the mine. The alternative section complement of machinery recommended above should cost approximately $100,000 less than the recommended continuous miner unit. Problems with float dust generation would be much less severe with the alternative unit than with continuous miners. This complement of machinery, however suffers in comparison to the continuous miner unit in the following areas. The conventional unit has the functions of cutting, drilling and shooting on the advance which the continuous miner does not. The continuous miner can reduce very large lumps, or blocks of top coal which may be present. Both advance and retreat mining can be performed with essentially one production machine. When equipped with remote control the continuous miner would be able to venture farther beyond the brow than would the LHD. The most serious disadvantages to the LHD complement are National Institute of Occupation, Health and Safety and UMWA objections to diesel equipment and the fact that the LHD needs a break step, or something to push against to load effectively. Requiring something to push against could reduce the effectiveness of the LHD in loading top coal beyond the brow and make this machine an unacceptable alternative. The continuous miner, by contrast, uses the gathering arm principle of loading which should be effective beyond the top coal brow.

Electric drive LHD's applied with conventional equipment in the same manner as was proposed for the diesel drive LHD's could be successful. This alternative presents somewhat greater problems in modifying the machine than was the case with the diesel machines because the rear of the machine, which is the only logical location for the operator's station, is also the location of the cable reel assembly. The electric cable-trailing LHD also loses a good portion of the mobility which made the diesel LHD an attractive selection. Approximately 18 months of lead time would be required to re-equip existing large-capacity LHD's with electric drive. Cable-trailing LHD's now on the market do not have the bucket capacity needed to make the method succeed. Battery powered LHD's are not believed to have adequate power or duration between recharges to perform the primary loading function.

A further all-electric option which could be successfully applied to the Sublevel Caving by Pillar Extraction method would be to add remote control to the gathering arm loader used with a conventional complement of face equipment. Joy Mfg. Co. has confirmed that the remote control unit utilized for their continuous miners could be applied to a Joy 14BU Loader for approximately $10,000 per machine.

The alternative machinery complements listed below were rejected for the reasons indicated:

| Machinery Selection | Disqualified Because: |
|---|---|
| Continuous miner advance with diesel LHD's | Combines the worst of both systems. Too expensive. Too many machines in section. |
| Battery-equipped LHD's | Insufficient duration of power. |

| Machinery Selection | Disqualified Because: |
| --- | --- |
| Continuous miner advance, retreat with cable-equipped LHD's | Combines the worst of both systems. Too expensive. Too many machines in section. |
| Continuous miner advance. Retreat with remote equipped loader. | Too expensive. Too many machines in section. Second loading machine deemed redundant. |

The use of diesel-driven support equipment for transportation of men and material offers very decided advantages for a mine in which belt conveyors are selected for moving the coal out of the mine. Battery tractors and trailers have been specified for this service but only because of the objections mentioned above to diesel underground equipment in coal mines. Should widespread acceptance of diesels in coal mines occur, diesel support equipment would be a far better choice due to their greater mobility and sustained power supply.

Underground track haulage is not recommended for this method unless track haulage is present in a candidate mine. Track haulage would require at least a third entry in the mining sections. The advantages of adhering to the 2 entry plan are deemed sufficient to recommend against track haulage.

## ALTERNATIVE PROCEDURES FOR DRILLING AND SHOOTING

Early in our study, the machine loading of charges from a remote position was investigated. Techniques and equipment to accomplish this task with permissable explosives and within the bounds of coal blasting regulations do not exist at present. The objective of this inquiry was to try to arrive at a method which could keep the men at a safer distance from the brow during these operations.

Hand loading of vertical shot holes on 4-foot centers was considered and determined to be feasible. This method suffers from the disadvantages of requiring that shot firers be near the brow, probable loss of coal due to cascading into the void upon shooting, and the time required to drill and charge the number of holes determined by the 4-foot spacing.

It may be possible to pre-drill all shot holes, including those required above the pillar pockets. This procedure would anticipate that all shot holes in the rooms and crosscuts would be drilled at 60-degree angles as presently recommended. Shot holes for top coal above the pillar pocket would be drilled from the room in a vertical fan pattern before the pillar pocket is driven by the continuous miner. This method would have the advantage that the lowest hole above the pillar pocket, which would be the first to be shot, would be almost horizontal and the coal would be propelled toward the mine floor. Charging of these holes would be accomplished from a very protected position as regards the brow and the encroaching cave.

This concept could be extended even further. All shot holes required to shoot down the top coal above the room and the pillar pocket could be drilled from the room adjacent. As the shot holes would be long and at a low angle, the coal would be propelled toward the mine floor as mentioned above. All drilling and charging would be accomplished from protected positions as regards the brow and shot wires (which would extend into the adjacent room) would be protected from breakage due to top coal sloughage. The disadvantages of this alternative are that the long, almost horizontal holes might be hard to drill accurately and charging of the shot holes would be more time consuming.

Cutting of the top coal with a cutting machine and shooting the charges most distant from the brow was also looked into. This alternative would be most effective in confining the fall of coal but would add another mobile machine to the already crowded work space. Further disadvantage would accrue from adding an extra step to the production cycle and extra capital and operation expense.

Two methods of drilling and placing 60-degree angled holes were investigated. The first of these methods was to drill top coal holes to the rock on 5, 6, and 7 foot centers. Shorter angled holes would be drilled approximately 4 feet into the top coal between the rows of long holes, where the maximum burden would be expected to occur after firing of the charges near the top of the seam. This method is believed to be feasible and will insure breakage of the entire top coal thickness. If loading were to be accomplished with a machine other than the continuous miner, this method would be best employed.

The feasibility of shooting down the top coal with high pressure compressed air (Airdox) was investigated. The advantages this method offers are that breakage of the coal causes no smoke and the work can be accomplished in the immediate vicinity of other work being performed with electrical equipment. The investigators do not have direct operating experience with this method of blasting and are therefore unable either to recommend with confidence or to rule it out entirely based upon first hand knowledge. The advantages to the Airdox method for Sublevel Caving by Pillar Extraction, which preclude its endorsement, are as follows:

1. As stated above, the lack of direct experience of the project affiliates with this method.

2. The design philosophy of the project which is to restrict as many of the operational elements to proven and widespread practice wherever possible. Thereby feasibility is enhanced and focus is restricted to the unique aspects of the method for those who might possibly study or use it.

3. The cylinders would be difficult to retain in the up holes.

4. As is the case with the shot wires, the small compressed air lines leading to the cylinders would be vulnerable to top coal sloughage.

5. Lower production would be experienced because of a greater time required to break an equivalent amount of coal.

6. The cylinders would be difficult to handle during the loading operation.

7. The equipment is difficult to maintain and cumbersome to move.

8. The additional machinery creates additional capital and operating cost and aggravates crowding of the work space.

9. It would not be possible to shoot multiple rows of holes.

10. Drilling costs and drilling time would be increased.

## OPERATIONS WITH DIESEL LHD'S AND CONVENTIONAL EQUIPMENT

The originally proposed method of utilizing diesel LHD's along with a complement of conventional face equipment is considered by the authors to be of sufficient promise to describe briefly the operations with this equipment and to include in this report some of the preliminary conclusions as to performance with this selection of machinery.

General mine design, ground control, roof support, drilling and shooting and other overall considerations would not change from the recommended procedures or from the alternatives already discussed when applying the LHD and conventional unit to the operation.

The required alteration of the LHD to fit this concept is relocating the operator cab to the rear end of the machine. This revision from standard can be included in specifications for ordering. Eimco engineers have indicated that this alteration can be made without encumbering basic functions of the machine. The alteration is the key to productive safe loading of a pile of coal beyond the top coal brow. Without the alteration the operator is about 10 feet from the bucket. With the alteration the operator is about 25 feet from the bucket. The reaching capacity is therefore increased 15 feet.

The LHD's specified would also be used for many of the materials handling and transport functions in the mine. They would be utilized for cleanup duty as well.

A standard universal coal cutting machine equipped with a bugduster, and an 11-foot cutter bar fixed for water would be required to meet the cutting needs of the project. A face drill will be mounted on the cutter and drill the blast holes after the cut is finished. Provision of the drill in this manner reduces the number of mobile machines traveling in the small operating area and allows efficient utilization of manpower. The drill receives its hydraulic power from the cutting machine and costs less than a mobile coal drill. The mounting can be done by the manufacturer using assemblies of standard parts. Both Joy and Jeffrey Companies have supplied combination machines. The 8 feet of mining height provides ample clearance for tramming the combination cutter drill.

It may be advisable to make roof cuts instead of floor cuts in order to have a smooth unfractured roof and to maintain the 8 foot room height accurately. Also, the smooth roof would expedite installing and removing of crossbar sets and props.

The authors are confident that the pumpable roof bolter will be eventually developed into a practical machine capable of drilling the hole and placing the bolt without the operator being near the bolter head. Pumpable bolts have been placed by the prototype machine in a mine roof in Utah. The machine is capable of placing the 6-foot bolts, but cannot at present keep up with the operating cycle. The pumpable bolt is comparable to the resin grouted steel bolt in roof supporting ability and will replace it when the cost becomes competitive.

The flexible shaft roof drill being developed by Eimco Corp. for the U.S.B.M. has the potential of drilling both blast and pumpable bolt holes in the top coal roof with the operator remaining in his safety cab under primary supported roof.

The application of conventional equipment reduces the float dust problem as compared with a continuous miner. Water on the cutter bar and sprays on the coal drills reduce the generation of float dust almost completely from the operation of these machines. Each fall of coal would be sprinkled as it is loaded. Generation of dust would be reduced due to fewer shuttle haulways.

The concept described confines five mobile machines into a small area of operations. Rigid safety rules controlling the movement of these machines are, therefore, an important requirement.

TABLE 5 - 1

ALTERNATIVE SECTION CREW

The section crew is estimated as follows:

2 - Cutting machine operators.

2 - Mobile coal drill operators. (These men would also operate the pumpable roof bolter.)

2 - LHD operators.

1 - Shot firer and brattice man.

1 - Belt attendant.

1 - Mechanic

1 - Forman

10 Total

## Productivity - LHD Alternative

The mining cycle for this conventional unit in advance and retreat room and pillar work is restricted by the small number of work places and the limited travel space determined by this concept. The productivity, however, is enhanced by the large amount of top coal available per fall to the loading position. For example, 64 tons are available for each face cut compared to as much as an estimated 150 tons of top coal in the pillar split and the adjacent room. The large amounts of coal per fall would enable high utilization of the LHD's despite restricted space and fewer work places per section.

### TABLE 5 - 2

### MINING CYCLES - LHD ALTERNATIVE

#### Estimated Section Cycle 1st Shift - Room

| Room ready to load fall | Minutes | Cumulative Minutes |
|---|---|---|
| Load 70 tons | 35 | 35 |
| Install 15 pumpable roof bolts | 75 | 110 |
| Cut and drill | 40 | 150 |
| Charge holes and shoot | 30 | 180 |
| Clear smoke and sprinkle | 10 | 190 |
| Load 70 tons | 35 | 225 |
| Install 15 pumpable bolts | 75 | 300 |
| Cut and drill | 40 | 340 |
| Stop advance to balance retreat | | |
| Delay - perform dead work | 50 | 390 |

Production 140 Tons

#### 2nd Shift - Room

Work stopped in room to maintain balance of room progress and pillaring.

## TABLE 5 - 2 (Cont.)

## MINING CYCLES - LHD ALTERNATIVE

### Estimated Section Cycle 1st Shift - Pocket

| Pocket - ready to start | Minutes | Cumulative Minutes |
|---|---|---|
| Cut and drill | 40 | 40 |
| Charge holes and shoot | 30 | 70 |
| Clear smoke and sprinkle | 10 | 80 |
| Load 46 tons | 25 | 105 |
| Place 3 crossbar sets | 30 | 135 |
| Delay - perform dead work | 25 | 160 |
| Cut and drill | 40 | 200 |
| Charge holes and shoot | 30 | 230 |
| Clear smoke and sprinkle | 10 | 240 |
| Load 46 tons | 25 | 265 |
| Place 2 crossbar sets | 20 | 285 |
| Drill fenders and top coal | 40 | 325 |
| Charge holes | 40 | 365 |
| Remove 2 x-bars | 10 | 375 |
| Shoot (1,2,3 Dwg. No. 4) | 5 | 380 |
| Clear and sprinkle | 10 | 390 |

Production 92 Tons

### 2nd Shift - Pocket Ready to Load

| | | |
|---|---|---|
| Load 148 tons | 75 | 75 |
| Drill and charge holes | 35 | 110 |
| Remove 3 x-bar sets | 15 | 125 |
| Shoot (4,5 Dwg. 4) | 5 | 130 |
| Clear and sprinkle | 10 | 140 |
| Load 88 tons | 45 | 185 |
| Drill and charge holes | 55 | 240 |
| Remove 5 x-bars and bridge bar | 40 | 280 |
| Shoot (6,7,8,9 Dwg. 4) | 10 | 290 |
| Clear and sprinkle | 10 | 300 |
| Load 140 tons | 70 | 370 |
| Delay - perform dead work | 20 | 390 |

Production 376 Tons

TABLE 5 - 2 (Cont.)

MINING CYCLES - LHD ALTERNATIVE

Estimated Section Cycle 1st Shift.

| Crosscut and chain (entries) | Minutes | Cumulative Minutes |
|---|---|---|
| Fall of coal ready to load | | |
| Delay - wait for LHD | 40 | 40 |
| Load 70 tons | 35 | 75 |
| Delay - wait for bolter | 45 | 120 |
| Install 15 bolts | 75 | 195 |
| Delay - wait on cutter | 15 | 210 |
| Cut and drill | 40 | 250 |
| Charge and shoot | 30 | 280 |
| Clear smoke and sprinkle | 10 | 290 |
| Load 70 tons | 35 | 325 |
| Install 13 of 15 pumpable bolts | 65 | 390 |

Production 140 Tons

2nd Shift

| | | |
|---|---|---|
| Install 2 pumpable bolts | 10 | 10 |
| X-cut finished start chain pillar pocket | | |
| Cut and drill | 40 | 50 |
| Charge and shoot | 30 | 80 |
| Clear smoke and sprinkle | 10 | 90 |
| Load 46 tons | 25 | 115 |
| Place 3 x-bar sets | 30 | 145 |
| Cut and drill | 40 | 185 |
| Charge and shoot | 30 | 215 |
| Clear smoke and sprinkle | 10 | 225 |
| Load 46 tons | 25 | 250 |
| Place 2 x-bars | 20 | 270 |
| Drill fenders and top coal | 40 | 310 |
| Charge holes | 40 | 350 |
| Remove 2 x-bars | 20 | 370 |
| Shoot | 5 | 375 |
| Clear smoke and sprinkle | 15 | 390 |

Production 92 Tons

Expected Average Cycle Production - 420 tons

An estimate of the average mining unit rate of production is 362 tons per unit shift, or 36.2 tons per section man shift including the foremen. This is the rate expected under "adverse conditions". Under "favorable conditions" it is reasonable to average at least 400 tons per unit shift. Production would also be improved and loss of coal reduced if a washing plant were available for removal of refuse from the run-of-mine coal.

A production rate of 362 tons per mining unit is considered obtainable. This rate is less than the expected productivity arrived at in the cycle of unit operations shown in Table 5 - 2 and is used because there are always many unpredictible factors and incidentals which reduce rates of production of mining unit. Delays of many kinds creep into the operation.

Preliminary estimates of recovery, utilizing the LHD alternative, for a 20-foot seam indicate that about 75 percent can be expected from the section which accounts for loss of the ventilation barrier, pillar stumps, 25 percent of the top coal (due only to dilution) and other losses of about 5 percent.

## CONCLUDING STATEMENT

It is widely acknowledged that coal must assume a much more important place in our national energy economy. The challenges which must be overcome for this to occur are formidable. Past methods of coal production and use are no longer acceptable in many instances. Previously undreamed of environmental constraints are now codified in law. Law also governs safety and health practices in coal mining more rigidly (sometimes in a counterproductive fashion) than ever before. The coal industry has been severely undercapitalized for the past quarter century. Little capital has accumulated to finance expansion needed now, and only miniscule funding has been available for research into new and better equipment and techniques. Coal is perceived by a large percentage of the public as a "dirty" and therefore unacceptable alternative to oil or natural gas. Many would argue that the industry is ill-prepared to provide, and the nation ill-prepared to accept a large scale swing to coal as a national priority fuel.

Against this background, the authors find pleasure in submitting this report on the Sublevel Caving by Pillar Extraction mining method which shows promise of better health and safety and reduced resource waste for thick seam underground coal mining.