DOE/ET/14246-T1
(DE83001401)
Distribution Category UC-88

ASSESSMENT OF THE SINGLE PASS THICK SEAM LONGWALL MINING METHOD

FINAL REPORT

by

Roger F. J. Adam, P.E.
William J. Douglas, Ph.D.

KETRON, INC.
Hickory Hill Plaza
151 S. Warner Road
Wayne, PA 19087

PREPARED FOR

U. S. Department of Energy
Pittsburgh Mining Technology Center

Under Contract No. U. S. DOE DE-AC01-79ET14246
April 1982

REPORT DOCUMENTATION PAGE

1. Report No.	2.	3. Recipient's Accession No.
4. Title and Subtitle Assessment of the Single Pass Thick Seam Longwall Mining Method		5. Report Date April 1982
		6.
7. Author(s) Roger F. J. Adam, P.E. William J. Douglas, Ph.D.		8. Performing Organization Report No. 1822
9. Performing Organization Name and Address KETRON, INC. Hickory Hill Plaza 151 South Warner Road Wayne, Pa. 19087		10. Project/Task/Work Unit No.
		11. Contract or Grant No. DE-AC01-79ET14246
12. Sponsoring Organization Name and Address U.S. Department of Energy Pittsburgh Mining Technology Center Post Office Box 10940 Pittsburgh, PA 15236		13. Type of Report Final
		14.
15. Supplementary Notes		

16. Abstract

The objectives of the project are a review of the foreign experiences in longwall mining of thick seams, an evaluation of the U. S. thick seam reserves, a mine design, equipment specifications, and an economic study for longwall mining in a single pass up to 16 feet in U. S. conditions. The review of foreign experience shows a steady increase in the maximum height of extraction with several examples in the range of 13 to 15 feet. Longwall face equipment is available up to an 18 feet height of extraction, based upon shields support and shearer loader. There are important reserves of thick coal seams which can be mined by a longwall in a single pass (125 billion tons). In U. S. conditions, a retreating face with a 16 foot height of extraction and a double entry system, driven 10 feet high, are proposed. The face stability can be improved by using a two bench face cut in good geological conditions. The economic study compares longwall mining with the room-and-pillar method. Longwall mining 16 feet high coal in a single pass can compete advantageously with room-and-pillar mining.

17. Originator's Key Words Thick Seam Longwall Mining	18. Availability Statement		
19. U. S. Security Classif. of the Report Unclassified	20. U. S. Security Classif. of This Page Unclassified	21. No. of Pages	22. Price

FOREWORD

This report was prepared by KETRON, Inc., Mining Engineering Division, Pittsburgh, Pennsylvania, under DOE Contract Number DE-AC01-79ET14246. This contract was administered under the technical direction of Pittsburgh Research Center with Jasinder S. Jaspal acting as Technical Project Officer. Stephen Butterfield, and later Philip Cooper, was the contract administrator for the Department of Energy.

ACKNOWLEDGEMENT

KETRON would like to acknowledge the cooperation received from the following professionals whose knowledge, advice, and assistance contributed immensely to this report.

The author is indebted to the management of the Westfalen Mine, Warndt Mine, Sufco Mine and Colorado Westmoreland, and especially Messrs. P. Goettig, W. Dietrich, A. Weaver, and R. Hochfner, who took the time to review the objectives of this study and to share their experience in this area.

Of the many experts who helped us, we would like to mention Dr. Hahn, Dr. Irresberger, Professor Raffoux, Dr. Dejean, Mr. Bottle and Mr. Haarmann who were quite helpful in providing us with valuable information and the support of their expertise.

TABLE OF CONTENTS

TABLE OF CONTENTS (continued)

TABLE OF CONTENTS (continued)

LIST OF FIGURES

LIST OF FIGURES (continued)

LIST OF FIGURES (continued)

LIST OF FIGURES (continued)

LIST OF FIGURES (continued)

LIST OF TABLES

EXECUTIVE SUMMARY

Large reserves of thick coal seams are located in the western United States. These seams are generally only partially extracted when mined by underground mining methods. Current methods employed in thick seams are not appropriate to recovering the remaining coal in subsequent mining activities.

Longwall mining represents one approach for improving the recovery of coal reserves. It is more appropriate to thick seams than the room-and-pillar method when the seam thickness excludes mining the seam in one pass. The longwall system also adapts more easily to changes in seam thickness.

Longwall mining methods for thick seam can be classified according to seam thickness as follows:

- Single pass mining is currently applied to seams with thickness ranging to 4.5 m (14.7 ft.). Six meters (20 ft.) can be the upper limit using the present technology.

- Successive slices are recommended if the seam is regular and the thickness exceeds 6 m (20 ft.).

- Sublevel caving is more appropriate to seams with rapid changes of thickness or with a bad roof and applies to a 6-15 m (20-50 ft.) thickness range.

With the introduction of shield supports, there has been a dramatic improvement in the ability to mine thick seams by longwall mining systems. This trend is evident at Daw Mill Colliery (Great Britain) as well as Warndt Mine (Saar, West Germany) or in Poland (Zabrze Mine). Westfalen Mine (West Germany), in 1975 was the first to apply single pass technology in a height of extraction exceeding 4 m (13 ft.). This application is a good example of the single pass technique in conditions where the coal is soft. Recently, face equipment with shields extendable to 6 m (20 ft.) with telescopic legs (three extensions) have been ordered by Westfalen Mine.

Entry development represents an important aspect of the longwall mining system. Longwall mining in the United States is different from the European method because the Federal mining regulations require three (or two) entries for the working of a longwall panel. In Europe, the longwalls are worked with a single entry development system.

Some European mine layouts suitable for thick seam mining are very appealing. Some mines in West Germany successfully use advancing longwall faces which, at the panel end, are turned 180 degrees to mine a second panel without having to move the equipment. Meyreuil Mine (France), where face-to-face moves are efficiently organized, uses the layout shown in Figure 4-5, which complies with Federal regulations during the panel extraction phase but requires a single entry development.

Application of these methods would necessitate variances from the Code of Federal Regulations, and are not likely to be approved. The suggested development method is a two-entry plan, or if necessary, a three-entry plan. The resultant entry height will be limited to 9 or 10 feet to improve safety conditions. Driving multiple entries in a 16 foot height is not recommended.

A consequence of the height requirement is that the size of equipment, especially for roof support, must match the entry size. The best approach for entry configurations at the face end is to develop entries with top coal, when possible.

Longwall mining is a high capital cost mining method. An acceptable return on this investment can only be obtained through very high production. The factors limiting the production of longwall faces in thick seams are:

- dust level
- down time caused by large lumps of coal unable to pass under the shearer or to turn at the delivery point
- roof control and sloughing of the face

To reduce dust generation, a number of techniques should be applied:

- Reduction of the coal breakage through deeper pick penetration by increasing the haulage force, and reducing the number of picks.
- Reduction of pick speed by lower revolutions per minute and smaller drum diameter.
- Utilization of a dust collector.
- Delivery of water in accordance with the amount of coal produced.

To reduce the downtime caused by coal lumps produced above the shearer, it is recommended that the top coal be cut with the shearer advancing in the direction of coal haulage. In this case, the wave of pressure is moving ahead of the shearer on the desired side. Large blocks of coal falling on the headgate side of the shearer can be efficiently hauled by side delivery and roller curve conveyors.

Roof control and sloughing of the face are sensitive to face operations. With present longwall mining equipment, a 5 m (16 ft.) height of extraction can be worked in two ways, (1) full face cut, or (2) two bench face cuts. Full face cut is commonly used with double ended drum shearers. In the United States, the shearers generally cut in one direction only, to keep the miners upwind and reduce dust exposure. The full face cut can be applied in seams 5 meters thick; however, in thick seams, face sloughing frequently occurs, causing delays and requiring face nailing.

By first cutting the top bench and advancing the roof support when the bottom bench is still present, and by cutting the bottom bench when anti-spalling plates are set against the top coal, sloughing of the face can be reduced significantly. In addition, when using a double drum shearer to mine two successive benches, the smaller drum diameter is beneficial to dust control.

A two bench mining method is therefore recommended for single pass thick seam mining. The face should be equipped with:

1. A double drum shearer loader with drums not exceeding 2 m (6.6 ft.) diameter, 1 m "wide" with less than 30 rpm and a limited number of picks. The shearer would use both drums to first cut the top bench and then both drums for the bottom bench.

2. One web back, two leg shield supports which will be advanced as soon as the top bench is won. Face sprags will be set before mining the bottom bench.

3. Conveyor carrying underframe which allows the use of shields with long base members.

Although there are advantages relating to dust control, there is no difference in productivity when mining in both directions or mining unidirectionally, if the top bench is cut first and the roof supports are advanced before cutting the bottom bench.

Since the current mining operations in the selected mines utilize room-and-pillar methods in a 9 foot height of extraction, it is reasonable to apply the current results of this practice as a basis for a comparison with the expected results of longwall mining. The current productivity is 850 tons per producing shift, with two producing shifts and one idle shift per day.

In longwall mining, the time required for face end operations has an important impact on the optimum face length. Therefore, two values are considered, the minimum time required and the minimum time plus 20 minutes of face end delay. When sumping time is increased by 20 minutes, the machine time is reduced to about 50% of the total available shift time. This is considered as the base case of the economic analysis.

When studying the replacement of continuous miner operations by longwall mining 16 feet in a single pass, two questions arise:

- What is the effect of different face organizations (full face cut, two bench method) and panel dimensions?

- What would be the results of extracting only 11 feet?

Comparison of Plan 1 (16 feet full face cut), Plan 2 (16 feet two bench method), and Plan 4 (extracting 11 feet only) shows that Plan 2 is the most favorable. Therefore, the economic study has concentrated on varying panel dimensions (Table 6-4), using Plans 1 and 2 and comparing to Plan 3. The sensitivity analysis has been made around the base case of Plan 2 (500 feet x 3000 feet panel).

Comparison between continuous miner and longwall is illustrated by three graphs which describe capital expenditures, production levels and operating costs. In the case of a 16 foot seam and 500 foot panel width, it is necessary to produce 6200 tons per day in the longwall to compete with a room-and-pillar method producing an average of 850 tons/shift. This is feasible (with effective dust control) if sloughing of the face does not cause important delays. At the present time, it is preferable to cut the top coal first and then the bottom.

The extraction of a 16-17 feet high coal seam can be achieved through single pass longwall mining methods with the currently available hydraulic powered roof supports and high capacity coal mining machines if the coal strength is sufficient. The results of this study indicate that the production and economics of such longwall mining operations would be better than room-and-pillar or multi-slice longwall mining methods in the extraction of such thick seams.

1.0 STATE-OF-THE-ART REPORT OF SINGLE PASS THICK SEAM LONGWALL MINING IN THE UNITED STATES

1.1 Background

The definition of thick seam varies according to country and mining condition, from 2.2 to 4 m. For the purpose of this report, we consider 3 m as a practical limit because available longwall equipment is well adapted and commonly used up to this thickness.

The study of longwall mining a thick seam in a single pass was limited to cases where the entire thickness of the seam is mined at the face and the roof is caved. Hydraulic or pneumatic stowing is not a part of this study. Also, sublevel caving methods, which may be considered as a single pass technique, are not considered.

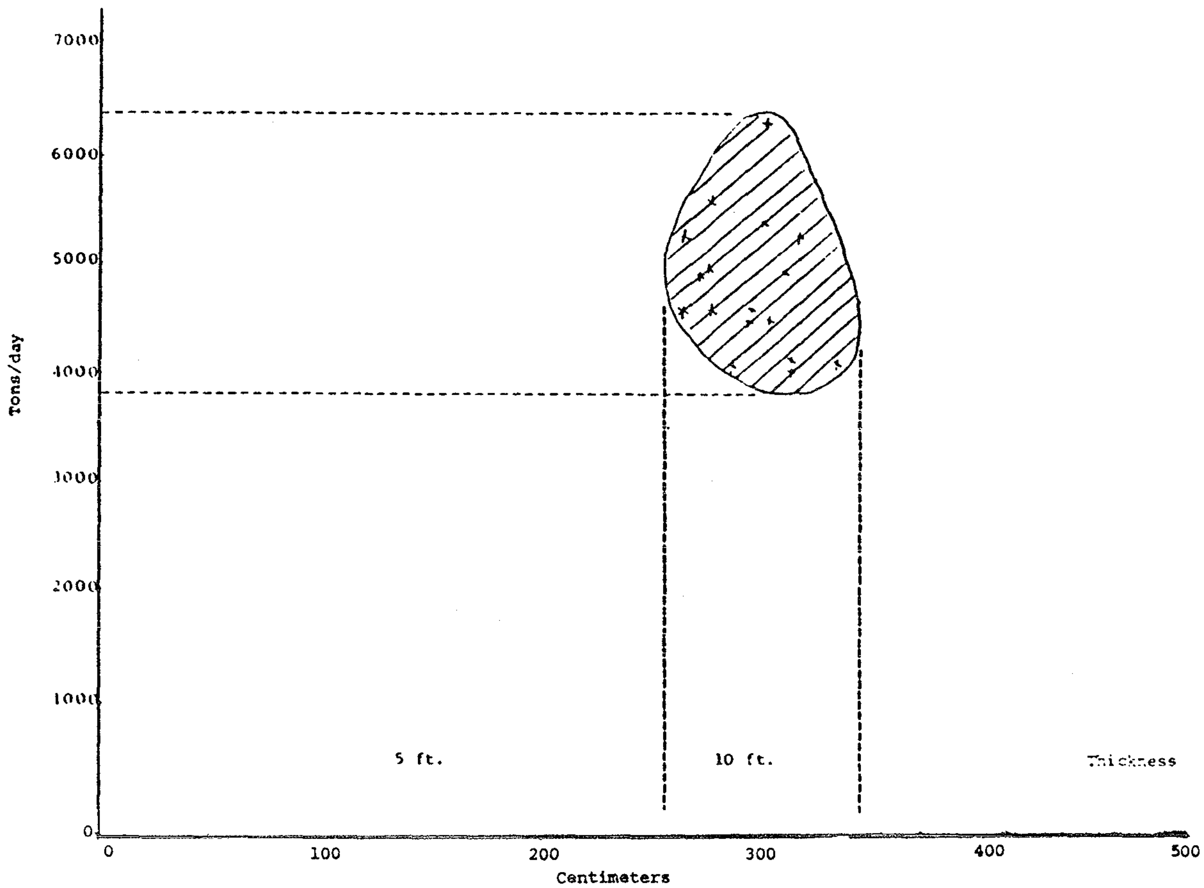
1.1.1 Underground Coal Reserves in Thick Seams

In addition to the United States, several countries have important coal reserves in thick seams. Among the countries which are mining these reserves are the USSR, India, Poland, Australia, United Kingdom, Hungary, and Japan. Several methods were developed to mine these seams, ranging from room-and-pillar methods, with or without pillar extraction, to longwall mining methods in successive slices or in a single pass, with or without sublevel caving. Hydraulic or pneumatic stowing is often used to alleviate problems of strata control because rock movements increase with the mined thickness.

In the past, the high performance faces were always located in seams of 2.1 to 2.4 m (7 to 8 ft.), but following the introduction and development of shield supports, the average thickness of the best faces increased steadily. For instance, in 1978, the 18 best faces in Germany were mining from 2.7 to 3.7 m (9 to 12 ft.) although trials are being pursued in thicker seams (see Figure 1-1).

1.1.2 Problems in Mining Thick Seams

In a longwall operation, the solid coal ahead of the face line clearly plays a vital role in supporting the face. This emerges from the simple comparison of the pressures exerted by the roof on the solid coal and by the face supports. The results depend mainly on the cohesion of the solid coal and the behavior of the coal face. If the coal is soft, the coal face will tend to spall, increasing the surface area of roof to be supported. This phenomenon inevitably becomes more marked as the working thickness increases (Figures 1-2 and 1-3). Figure 1-3 illustrates the magnitude of strata movement in a very thick seam mined by sublevel caving in France. A rising face is disadvantageous from this



Note: The best faces were in the range of 2.7 to 3.4 m (8.86 to 11.16 ft.)

FIGURE 1-1

High Performance Longwall Faces in West Germany
Source: Gluckauf June 21, 1979

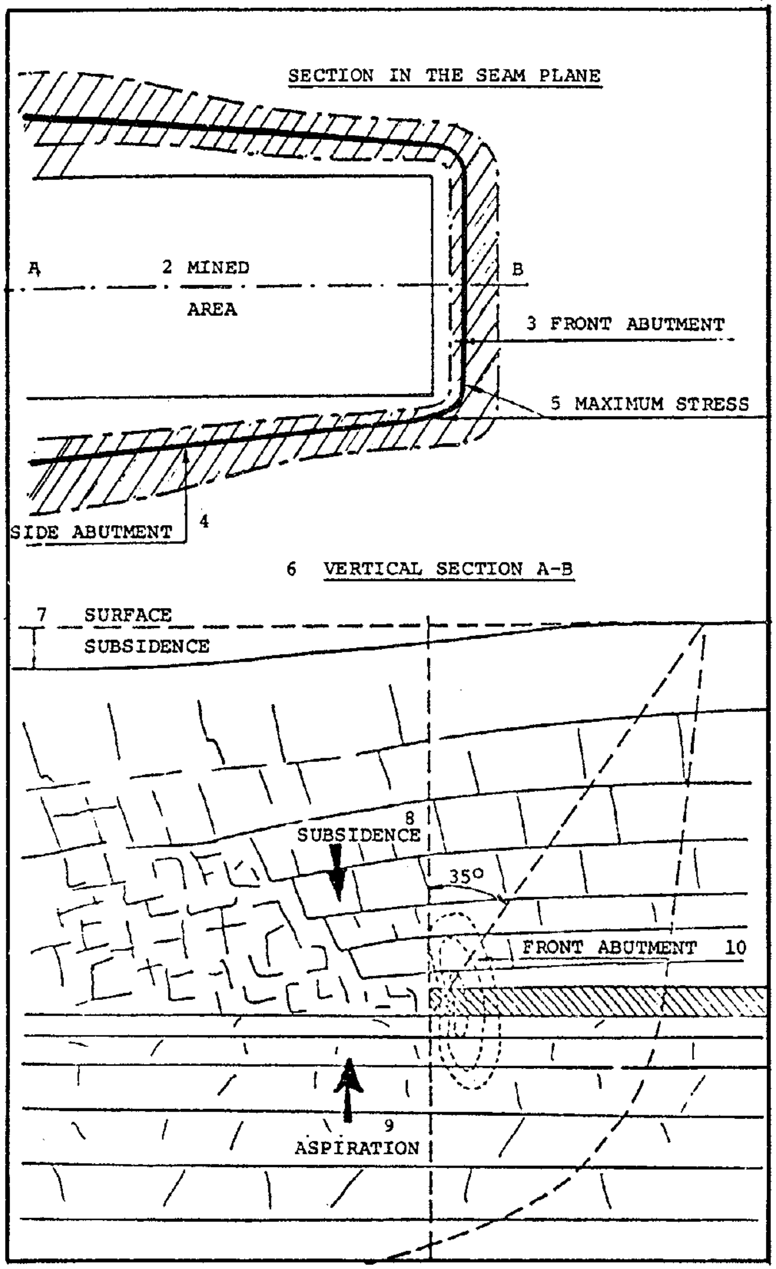


FIGURE 1-2

Influence of Mining on Surrounding Strata

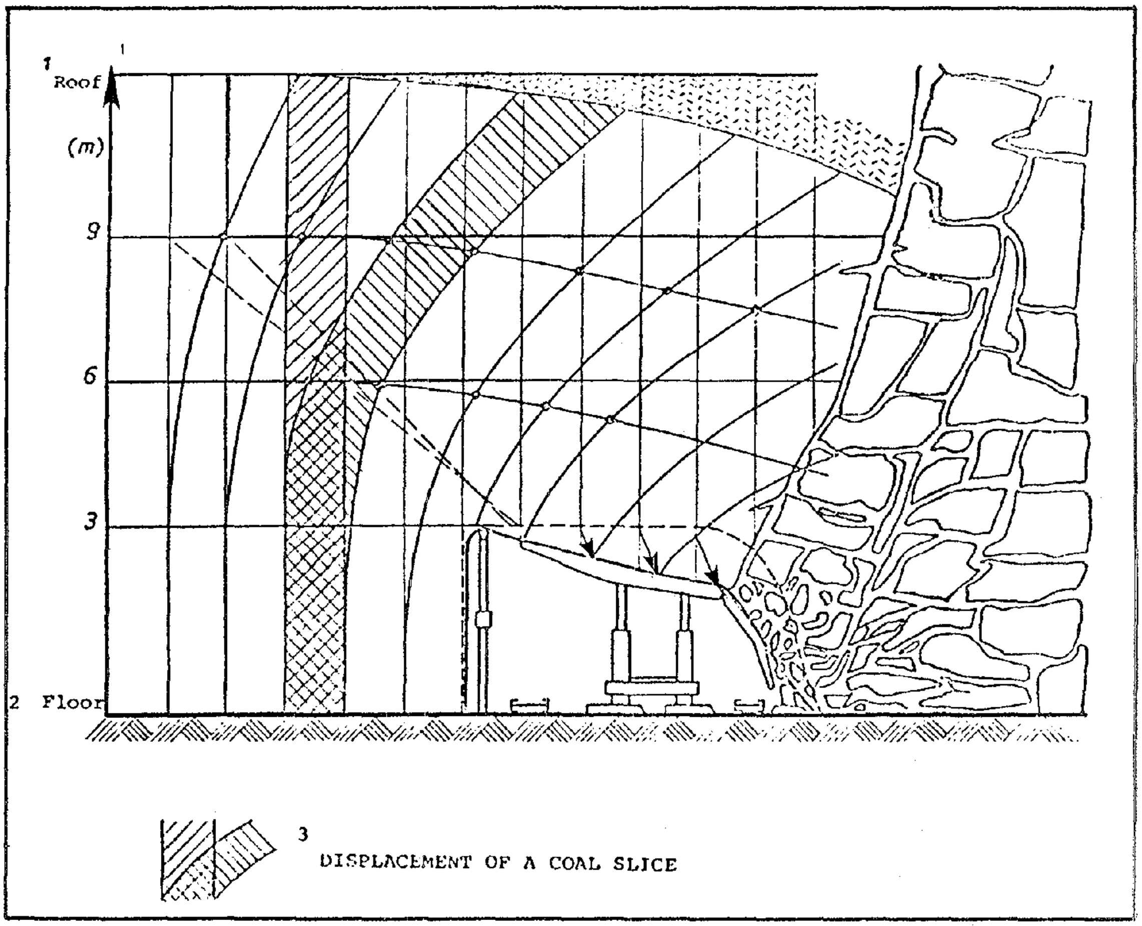


FIGURE 1-3

Scheme of a Longwall Face with Sublevel Caving - Deformation of the Solid point of view, especially where the seam is steep and the coal is soft or is crushed by the pressures and ground movements of the operation. On the other hand, a dipping face will be advantageous.

The roof control and, consequently, the productivity of the longwall face, depends upon several main parameters, such as:

- coal strength
- roof quality
- roof abutment pressure
- gradient of the seam and orientation of the face
- height of extraction
- control of the gob
- face equipment

The feasibility of applying the single pass method to a selected site must take into account all of these parameters.

The bottleneck of most longwall faces lies in the outby haulage or in the supply and maintenance of the entries. In mining a thick seam, roof pressure and strata movements may generate quick closure of the entries when the depth exceeds a limit depending on the strata compressive strength.

The study of longwall faces currently operating in thick seams will allow us to evaluate the upper technical limit of seam thickness compatible with the single pass system. Also, we will consider the economic limits which are much more important in the U. S. market than they are in Europe, where coal mines are generally state-owned, and where the amount of coal reserves is limited.

1.1.3 State-of-the-Art Report

This study of present thick seam longwall operations is concerned with evaluating the present limits of safely and economically mining a thick seam by a longwall in a single pass. The progress achieved during the last five years has slowly raised the upper limit of seam thickness, but this depends on geological conditions. Conditions under which such an operation may be successfully achieved must be carefully determined to assess the applicability of the method to U. S. conditions.

Successful experience in longwall mining 4.9 m (16 ft.) thick coal seams in a single pass is very limited. The United Kingdom has only a few thick seams and is mining them by slices, with the thickness of the slices gradually increasing (up to 4 m (13 ft.) at Daw Mill Colliery). France has a large part of its reserves in very thick seams, but uses sublevel caving or successive slices, and the mined thickness in one pass is generally less than 3.7 m (12 ft.). Germany had mined its few thick seams by taking slices or by leaving some coal. Presently, the seams mined by longwall in a single pass are up to 4.5 m (15 ft.) thick. This is limited to a few faces.

In the Eastern European countries and the USSR (which was the first country to mechanize sublevel caving), there are some examples of thick seams up to 5.5 m (18 ft.) which are mined in a single pass with caving. Czechoslovakia and the USSR (Karaganda) recorded some good results with OMKT or KM130 shield supports. In Poland, where 25% of the reserves are in thick seams, extensive use is made of hydraulic stowing. Yugoslavia is mining thick seams by sublevel caving. One longwall face has been recently equipped with Alpine shields to mine 4.2 m (13.8 ft.) in a single pass. In India, longwall mining by successive slices or with sub-level caving is used. Australia's thick seam reserves are generally mined by room-and-pillar. (A symposium on thick seam mining was held at Rockhampton in 1976.) Hungary and Japan are longwall mining thick seams by successive slices.

To obtain accurate information on most of these countries, it is necessary to get direct information from persons who have had the opportunity to visit some of these faces. To obtain detailed and up-to-date reports on current operations in thick seam mining, as well as practical recommendations from experienced coal operators, KETRON used the services of NAMCO (North American Mining Consultants, Inc.) and of several mining equipment manufacturers. The collected information is presented in the following pages.

1.2 USSR Practices

The Soviet coal industry has developed roof supports for longwall mining thick seams in a single pass up to 5 m (16.4 ft.). One type of roof support used in thick seams is the KM81. Its main characteristic is the fact that the advancing system is at the roof level, and the elements are moving, suspended between their neighbors. A KM81E support with extended legs is used in seams up to 4 m (13.1 ft.) thick (Mikhalvolskaia Mine).

Another type, the KM120, is an extension of the OMKTM shield support. It is combined with a four drum shearer loader for mining thick seams up to 5 m (16.4 ft.). A short face of 60 m (197 ft.) was visited by a foreign group in October 1970 at the Iaroslavskaia 3 mine, of the Lenin-Kousnetz Group. This face was 5 m (16.4 ft.) high. At this time, the setting load was only 20 t/m^2 and the yielding load 45 t/m^2.

In 1975, according to a paper in Ugol (October 1976) from Professor Korin, Director of the Guiprouglemash, two sets of equipment were used in the Raspadskaia Mine and at the aforementioned Iaroslavskaia Mine in the Kousnetz. The maximum daily production of one 60 m (197 ft.) face was 2500 tons, with an output per man shift of 80 tons.

To update this information, a letter was sent in December 1979 to the Coal Ministry asking for more information. We received, with a letter on February 8, 1980, a trilingual booklet with the following English text and accompanying photograph (Figure 1-4).

KM-120 COAL WINNING MECHANISED SYSTEM

For the first time in world practice, the KM-120 coal winning system solves the problem of full mechanization of all face operations when extracting the full seam section of flat seams 3.5-5.0 m high in a single pass. The system provides mechanized coal cutting and loading, roof supporting and controlled waste caving.

The system includes the following unique equipment:

- the M 120 powered support;
- the K 120 narrow-web shearer loader;
- the CHM 120 armoured face conveyor;
- a stage loader;
- hydraulic and electric equipment.

The main design features of the M 120 support are:

- o full coverage of working face area from gob side is ensured;
- o a well protected special traveling way through the chocks is available that guarantees safety for faceworkers;
- o special active devices used increase longitudinal stability of the support;
- o good side stability is ensured by means of the increased width of the chocks (2.2 m) and with help of special active devices;
- o hydraulically operated and power set extension bars offer immediate support to the newly cut roof when the shearer is cutting at its maximum speed - 10 m/min.;
- o a large effective section of the support ensures high output achieved even in the conditions of gaseous-and-dusty mines. The M 120 is the only support to be used used when upper seam section is inclined.

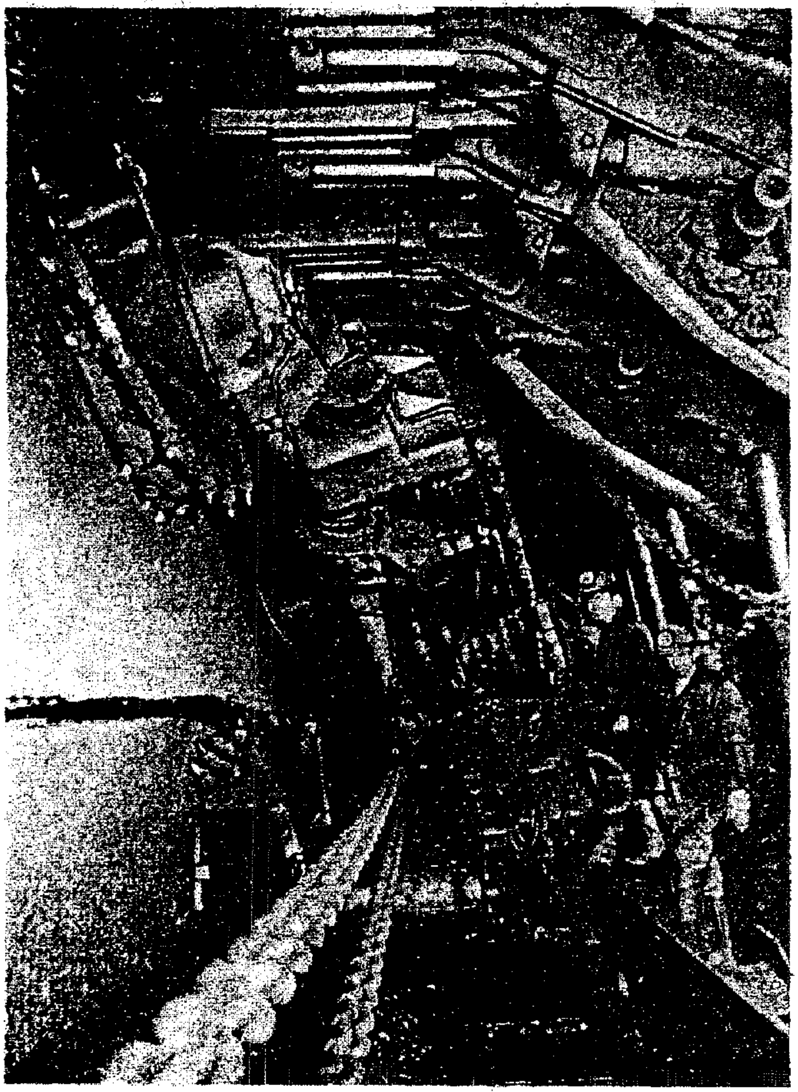


FIGURE 1-4
KM-120 Coal Winning Mechanised System

The conveyor mounted shearer is designed for cutting in both directions. Its four ranging drums cut the full seam section in one pass, ensure frontal sumping and, as a result, allow for stable hole elimination at both ends of the face; the drums' spiral vanes ensure effective loading of the broken coal onto the A.F.C. The shearer is fitted with dust collectors located near the cutting drums and a dust suppression system that feeds water directly into the cut. The controls are positioned on the shearer or the machine is controlled remotely.

The inclined upper seam section with controlled inclination formed only by the KM 120 shearer allows face stability to be maintained when the coal has the intensive tendency to break out of the face.

The conveyor is flexible and its clamping joints have a very high safety factor.

Active horizon control of A.F.C. ensures perfect clearing of broken coal left in the cutting track and prevents leaving a coal band on the floor.

Full roof coverage by powered support shield chocks, the inclined upper seam section formed by the power loader, and the elimination of hand clearing - all these factors ensure safe working conditions in the face where the KM-120 system is operated.

The high power of the coal winning system allows output up to 15 t/min and o.m.s. up to 80 t to be achieved.

Due to cutting the full seam section in one pass, the KM-120 mechanized system ensures the following main advantages:

- o coal losses are reduced by 10-15%;
- o coal self-ignition fire risk is considerably reduced;
- o development and installation work is reduced twice.

The hydraulic relief valve to be used in KM-120 powered support units has been patented in the USA.

The firm may be offered technical documentation numbering 1000 sheets of the 24 size."

The following comments on KM 120 may be added:

- This support retains the features of the original shields, keeping the area of the roof to be supported at a minimum and reducing accordingly the required load density. The yielding load is only 114 to 140 t/m, about 60 to 70 t/m2.

- The width of a shield is up to 2.2 m (7'4"), double the early OMKT width.

- It is the only example we know of an inclined upper cut.

- The conveyor is frame-mounted to allow the use of a roof support with a long base.

Problems encountered during the trials limited the research to the two aforementioned sets of face equipment. The KM 130 roof support, limited to a thickness of 4.2 m (14 ft.), seems to be more successful.

Table 1-1 is a list of collieries in the Soviet Union in which seams with a thickness of 3 to 5 m (10 to 16.4 ft.) are worked by fully mechanized longwall operations (as of September, 1976).

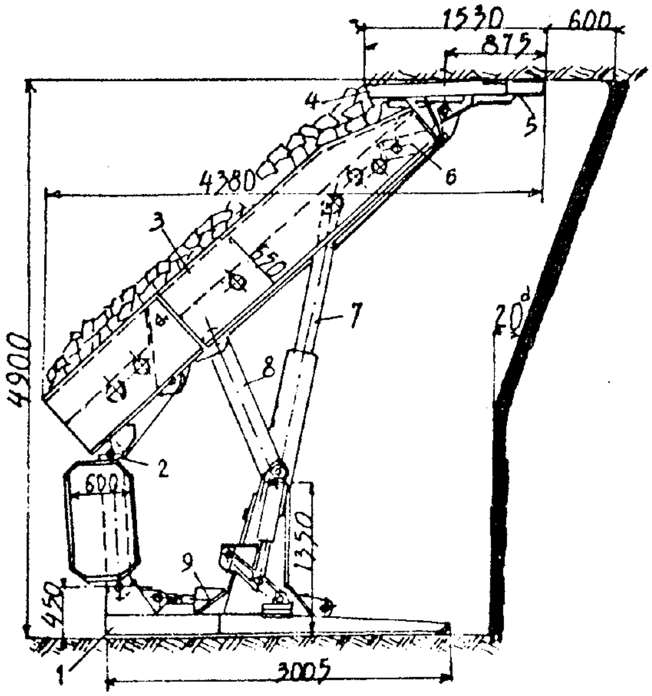


FIGURE 1-5

KM 120 Roof Support

KM 120 System Technical Data

Extracted seam ranges, m (ft)	3,7 - 5,0 (12 - 16.9)
Face Length, m (ft)	120 (394)
Minimum face output, t/day	2500
Total electric motor power, kW	1000
Voltage, V	660
KM 120 Powered Support (See Figure 1-5)	
Unit yield load, t	250-310
Unit center, mm (ft)	2200 (7.2)
Advancing ram effective stroke, mm (ft)	up to 700 (up to 2.3)
Unit weight, kg (t)	14500 (14.5)
KM 120 Shearer	
Effective web width, mm (ft)	500 (1.64)
Range of drum adjustment, mm (ft)	3200-5100 (10.5 - 16.7)
Haulage pull, t	up to 40
Haulage speed, m/min.	up to 5
Total electric motor power, kW	530
Dimensions, mm (ft)	1210x2650x10160 (4x8.7x33.3)
Weight (less haulage chain), kg (t)	43600 (43.6)
CHM 120 Conveyor	
Chain center, mm (ft)	600 (2)
Chain	0 23 x 86 mm pitch
Chain speed, m/sec (ft/sec)	1.3 (4.28)
Total electric motor power, kW	220
Weight, kg (t)	145000 (145)

TABLE 1-1

USSR Collieries Mining Thick Seams in a Single Pass

Designation of the Coal Districts and Collieries	Thickness (m)	Mined Seam Thickness (Thickness of the Slice) (m)	Angle of Dip of Seam (°)	Designation of the Face Equipment	Average Output per Working Face (t_{clean}/d)	Mining System
1	2	3	4	5	6	7
<u>Moscow District</u>						
Zapadnaja Nr. 1	3,0	2,8	8	OMKTM-10	735	Longwall Mining
Maevskaja	3,0	2,7	0	OMKTM-10	815	"
Scekinskaja	3,0	3,0	0	OKP	920	"
Pokrovskaja	3,1	3,1	6	OKP	1 135	"
Plechanovskaja	3,3	2,9	12	OMKTM-10	910	"
Lomincevakaja	3,0	3,0	0	OMKTM-6	435	"
Podlesnaja	3,2	3,0	0	OKP	1 355	"
Mostovakaja	3,09	2,7	0	OMKTM-10	715	"
Begicevskaja	3,1	2,7	0	MK	465	"
Skolniceskaja	3,7	3,7	1	OMKTM-10	1 455	"
Crankovskaja	3,0	2,7	0 - 1	OMKTM-10	530	"
Krasnoarmejskaja	3,0	2,8	8	OKP	1 300	"
Sirinskaja	3,25	2,9	3	OKP	960	"
Progress	3,3	3,17	0	OKP	1 620	"
Novomoskovskaja	3,0	2,8	1 - 8	OMKTM-10	1 325	"
Zubovskaja	3,4	2,9	1	OKP	864	"
Pronskaja	4,0	3,2	0	OKP	890	"
Gornjak	3,0	2,9	0	OMKTM-10	1 040	"
Nelicovskaja	3,35	3,2	0	OKP		"
Podmoskovnaja	3,2	3,1	3	OKP	2 385	"
Smorodinskaja	3,15	3,00	8	OMKTM-10	670	"
Sovernaja	3,2	3,2	0	OKP	1 605	"
Sokolovskaja	3,6	3,5	2	OMKTM-10	880	"
Druzba	3,4	3,1	4	OKP	1 270	"
<u>Karaganda District</u>						
Gorbaceva	3,4	3,3	9	KM-81E	1 350	Longwall Mining
Stepnaja	3,1	2,7	15	OKP	1 470	"
50-letija SSSR	4,0	3,2	12	2M-81E	2 070	"
Michajlovskaja	3,3	3,1	11	KM-81E	2 485	"
Ajbajskaja	3,4	3,3	0	30KP	1 920	"
Sachanskaja	3,5	3,0	8	OKP	1 340	"
Majkudukskaja	3,5	3,5	12	OKP	985	"
Curubaj-Nurinskaja	3,5	3,0	19	OKP	1 250	"
Sachtinskaja	4,8	2,4	25	OMTKM-10	1 310	"
Molodeznaja	4,0	3,6	15	30KP	1 480	"
Sokurskaja	3,2	3,0	12	OKP	1 480	"
50-letija Oktjabrskoj revoljucil	3,4	3,0	8	KM-81E	1 520	"
V.I. Lenina	3,0	3,0	11	OKP	1 000	"

TABLE 1-1 (Continued)

Designation of the Coal Districts and Collieries	Thickness (m)	Mined Seam Thickness (Thickness of the Slice) (m)	Angle of Dip of Seam (°)	Designation of the Face Equipment	Average Output per Working Face (^{t}clean/d)	Mining System
1	2	3	4	5	6	7
<u>Pecora District</u>						
Promyslennaja	3,93	3,46	22	KM-81E	2 295	"
Vorgasorskaja	3,2	2,85	6	OKP	1 805	"
Vorkutinskaja	3,0	3,0	15	OKP	1 205	"
Centralnaja	3,8	3,4	19	OMKTM-8	1 635	"
Oktjabrskaja	3,2	2,9	12	OKP	2 295	"
<u>East Siberian District</u>						
Kirova	3,0	3,0	3	OKP	1 420	"
Enisejskaja	3,0	3,0	5	OMKTM-10	910	"
<u>Deposits in the Far East</u>						
Urgalskaja	4,0	3,5	3	KM-81E	1 080	Longwall Mining
Skotovskaja	4,0	3,5	3	OKP	1 080	"
<u>Kuznetzk District</u>						
Pervomajskaja	3,0	3,0	17	OKP	1 240	"
Zapadnaja	3,0	2,9	8	KM-81E	570	"
Volkova	3,6	3,3	8	KM-81E	925	"
Oktjabrskaja	3,2	2,9	10	OKP	1 100	"
Juznaja	4,7	3,5	18	KM-81E	810	"
Kolduginskaja	3,0	3,0	6	OKP	1 290	"
Jaroslsvskogo	4,6	4,2	5	OKP	1 930	"
Polysasvskaja	3,2	3,2	9	KM-81E	1 275	"
Karagajlinskaja	3,1	3,0	15	OKP	1 250	"
Vachruseva	3,5	3,0	35	KM-81E	540	"
Bajdaevskaja	3,8	3,5	10	KM-81E	1 150	"
Lenina	3,0	3,0	9	OKP	640	"
Kapitalnaja	3,8	3,6	8	KM-81E	1 105	"
Novokuzneckaja	3,66	3,4	10	3OKP	1 970	"
Towskaja	3,8	3,3	6	3OKP	620	"
Raspadskaja	3,9	3,5	8	KM-81E	2 200	"
Inskaja	3,3	3,5	20	OKP	1 765	"
<u>Donatzk District</u>						
Vatutinskaja	3,5	3,4	0	OMKTM-10	1 250	"
Korostysovskaja	4,0	3,8	0	OMKTM-10	930	"
Svetlopolskaja	3,1	3,1	2	OKP	820	"
Verbolozovskaja	3,4	3,2	0	OMKTM-10	1 435	"

1.3 Polish Practices

In Poland, with 25% of coal reserves in thick seams, hydraulic stowing is extensively used. The Polish mining practice is interesting from several points of view. The main point is a comparison of Russian, Polish, and Western European roof supports. The second is the high strength of the coal. This latter property especially favors the application of the single pass thick seam longwall mining method.

The transition from top slice mining to the single pass thick seam mining technique has taken place during the past few years, with increasing use of Western technologies in conjunction with the highly efficient Polish mining industry. Because of the particular political situation, there is more information available on longwall faces equipped with West German machines and for this reason accessible to foreign experts. Described briefly below are some case examples from the Upper Silesian coal field.

1.3.1 Selected Examples

Table 1-2 contains a synopsis of longwall faces representative of the mining districts of the Upper Silesian coal field. They are divided according to divisional headquarters, in terms of the geological conditions, except the seam thicknesses. The seam thicknesses of these longwall faces average between 3.6 m (11.8 ft.) and 4.3 m (14.1 ft.). Although seams having substantially greater thicknesses also occur in the Upper Silesian District of Poland, single pass longwall mining is presently limited to a 4.5 m (14.8 ft.) seam thickness, due to the limited cutting height of the winning machine and the restricted capacity of the Polish-manufactured face conveyors.

The seam dip varies according to the mining districts. Whereas the pits in the areas of Katowice, Myslowice and Bytom are working in flat seams, the seams in the areas of Zabrze, and in particular of Rybnik, dip at a high angle.

Coal strength varies, but the coal is generally stronger than in West Germany. Seams in the areas of Katowice, Myslowice and Bytom generally contain very hard coal, which promotes the stability of the coal face. Of these deposits, however, the Bytom seams are most prone to rockburst. Some of the faces in the mines of Katowice have spalling problems even though the face heights of these seams cannot be considered in the thick seam category.

The coal strength of the Zabrze and Rybnik areas is somewhat lower; the tectonic stresses, however, are higher. Disturbed seams are found on the southern periphery of the Rybnik area, as, for example, at the Borynia Colliery. Seam 409/4 has the greatest geological difficulties of the selected longwall faces. The average longwall face length is 155 m (510 ft.).

The winning machines (double-ended ranging drum shearer loaders), developed and manufactured in Poland, are used in all the longwall faces listed in Table 1-2. The face conveyors are derived from both Polish and West German sources. Currently in the Polish bituminous coal mining industry, most longwall faces with working seam thicknesses of up to 3 m (9.8 ft.) are equipped with self-advancing supports developed in Poland or manufactured under license (licensors are predominantly West German support manufacturers). The longwall faces with a seam thickness greater than 3 m (9.8 ft.) are mainly equipped with self-advancing supports of foreign origin. As shown in Table 1-2, only shield supports are used in thick seams.

A feature of the longwall faces operated in the Bytom area is that the seams are prone to rockbursts. In these longwall operations, the coal face is stress-relieved by drillings and also nailed against spalling. Only a few shield legs were damaged or destroyed by severe rockbursts, or by roof falls occurring ahead of the support.

In the Upper Silesian coal field, faulted seams with soft coal and friable immediate roof may occur as in the West German mines. An example of this is the Borynia Colliery. The seam 409/4 (3.8 m (12.5 ft.) thick) was not workable by longwall mining in a single pass until shield supports were used, with satisfactory results. However, additional measures, such as nailing of the coal face, are still required.

The Katowice and Myslowice Districts present the most favorable mining conditions. Seams with a thickness of about 4 m (13 ft.) are worked successfully due to good geological conditions such as hard, stable coal and stable roof. Rockburst hazards are nearly non-existent.

1.3.2 Example of Zabrze Mine

A paper was published by J. Jankowski and St. Gagewski in Wiadomasci Gormicza 1-2/1979 entitled "One Million Tons of Coal from a 4.5 m High Caved Longwall Face of Zabrze Mine." The paper presents this face as the first in Poland to mine a thick seam (3.8 to 4.5 m, 12.5 to 14.8 ft.) in a single pass with caving.

The seam, numbered 507, has a roof of sandy shale or shale turning to sandstone, with a gradient of 3 to 15 degrees. The depth is 780 m (2559 ft.). The length of the face is 157.5 m (516.7 ft.). The following equipment was used:

2 x 200 t two-leg shield roof supports from Klockner Becorit

1800/3850 or 2450/4500 shield, width 1.5 m (4.9 ft.), weight 13 tons

TABLE 1-2

Synopsis of Selected Coal Face Operations

Mine	Divisional Headquarters	Seam	Seam Thickness (m) (ft)	Dip (°)	Face Length (m) (ft)	Direction of Cutting	Type of Stowing	Winning Machine made in .. Type	Face Support made in .. Type	Supportive Resistance (kN/m²) (Mp/t)
Wujek	Katowice	509	3.80 12.5	5-9	110 361	in the strike	caving	Double-ended ranging drum shearer loader, Poland KWB 6	Shield support, W. Germany, Thyssen, KHS 20/40 L	630 64
Nowy Wirek	Bytom	510	3.60 11.8 -3.80 -12.5	7-9	160 525	in the strike	caving	Double-ended ranging drum shearer loader, Poland, KWB 3 - RDU	Shield support, W. Germany, Thyssen, KHS 22/45	650 66
Nowy Wirek	Bytom	510	3.60 11.8 -4.30 -14.1	5-7	160 525	in the strike	caving	Double-ended ranging drum shearer loader, Poland, KWB 3 - RDU	Shield support, W. Germany, Thyssen, KHS 22/45	650 66
Dymitrow	Bytom	510	3,60 11.8 -4.80 -15.8	9	150 492	in the strike	caving	Double-ended ranging drum shearer loader, Poland, KWB 6	Shield support, W. Germany, Thyssen, KHS 22/45 L	650 66
Pokoj	Bytom	507/1	3.60 11.8	13-14	200 656	in the strike	caving	Double-ended ranging drum shearer loader, Poland, KWB 3 - RDU	Shield support, W. Germany, Thyssen, KHS 17/35 L	600 61
Zabrze	Zabrze	507	4.00 13.1	13-14	160 525	in the strike	caving	Double-ended ranging drum shearer loader, Poland, KWB 6	Shield support, W. Germany, Klöckner-Becorit, 2 Leg Shield 24/45 with conveyor carrying frame	870 89
Borynia	Rybnik	409/4	3.80 12.5	-18	150 492	in the strike	caving	Double-ended ranging drum shearer loader, Poland, KWB 3 - RDU	Shield support, W. Germany, Klöckner-Becorit, 2 Leg Shield 24/45 with conveyor carrying frame	870 89
Marcel	Rybnik	708	4.30 14.1	11-15	130 427	in the strike	caving	Double-ended ranging drum shearer loader, Poland, KWB 3 - RDU	Shield support, W. Germany, Klöckner-Becorit, 2 Leg Shield 24/45 with conveyor carrying frame	870 89
Ziemowit	Myslowice	209	4.30 14.1	0-6	160 525	in the strike	caving	Double-ended ranging drum shearer loader, Poland, KWB 6	Shield support, W. Germany, Klöckner-Becorit, 2 Leg Shield 24/45 with conveyor carrying frame	870 89
Staszic	Katowice	405	4.10 13.5	0-5	155 509	in the strike	caving	Double-ended ranging drum shearer loader, Poland, KWB 6	Shield support, W. Germany, Klockner-Becorit, 2 Leg Shield 24/45 with conveyor carrying frame	870 89

in Thick Seams of Poland

Face Conveyor made in .. Type	Production Rate (^{t}clean/d) (metric)	Output per Manshift (^{t}clean/MS) (metric)	Face Conditions	Remarks
AFC Single middle chain strand, Poland, Rybnik 73	2,200	40	Subject to impacts by coal ribs of upper seam. No problems with roof support. Nailing of coal face.	Face equipment is being used since May, 1977, in several panels.
AFC Single middle chain strand, Poland, Rybnik 73	2,300	35	Very difficult geological conditions, rock bursts. Roof cavities following rock bursts; damaging of several legs. Nailing of coal face.	Face equipment is being used in several panels since April, 1978; despite rock bursts face operation successful.
AFC single middle chain strand, Poland, Rybnik 73	2,500	37	Very difficult geological conditions, rock bursts. Roof cavities following rock bursts and spalling of coal face. Nailing of coal face.	Face equipment is being used in several panels since May, 1978; despite rock bursts face operation successful.
AFC Single middle chain strand, Poland, Rybnik 73	2,500	38	Difficult geological conditions, rock bursts. Despite rock bursts occurring supports in good order. No measures, since coal is very hard.	Seam thickness mined 3.80 m (= 12.5 ft); 1.0 m (= 3.3 ft) coal at bottom leaving unmined; face equipment is being used since March, 1979.
AFC Single middle chain strand, Poland, Rybnik 73	4,000	38	Coal face tends to spalling; however no set backs on production. No supporting problems in spite of coal face spalling. No measures, since roof is solid (Sandy shale).	Face equipment is being used since April, 1977, currently in the third panel.
AFC Double chain strand, W. Germany, Klöckner-Becorit, UFV 600	2,800		Endangered by rock bursts. No problem with roof support. No measures.	Smooth face operation; face equipment is being used since August, 1977, in several panels.
AFC Double chain strand, W. Germany, Klöckner-Becorit, UFV 600	2,000		Very difficult geological conditions; spalling of coal face. Due to geological conditions from time to time problems with roof control. Coal face must be nailed by schedule.	Extraordinarily difficult conditions; heavy falls happened in fault zones; precondition for mining the seam is employment of shield support; face equipment is being used since October, 1977.
AFC Double chain strand, W. Germany, Klöckner-Becorit, UFV 600	3,200 -3,800		Good. No measures.	Face equipment is being used since March, 1978, currently in the third panel; seam thickness mined 4.10 m (=13.5 ft), since top coal is leaving standing.
AFC Single middle chain strand, Poland, Rybnik 73	2,500		Good. No measures, since coal is very hard.	Face equipment is being used since March, 1979; top coal is leaving standing.
AFC Single middle chain strand, Poland, Rybnik 73	2,400		Good. No measures.	Face equipment is being used since March, 1979, without major breakdown; coal is very hard.

short canopy of 2.1 m (7 ft.) (see Figure 1-6)

long base advancing under the conveyor (one web back system)

Shearer loader KWB6, 1000 volts - 2 x 250 kw, equipped with 2 drums Ø 2360 mm, web 630 mm, haulage chain 26 x 92 mm, unidirectional operation

2 x 26 mm chains face conveyor with 736 mm wide pans

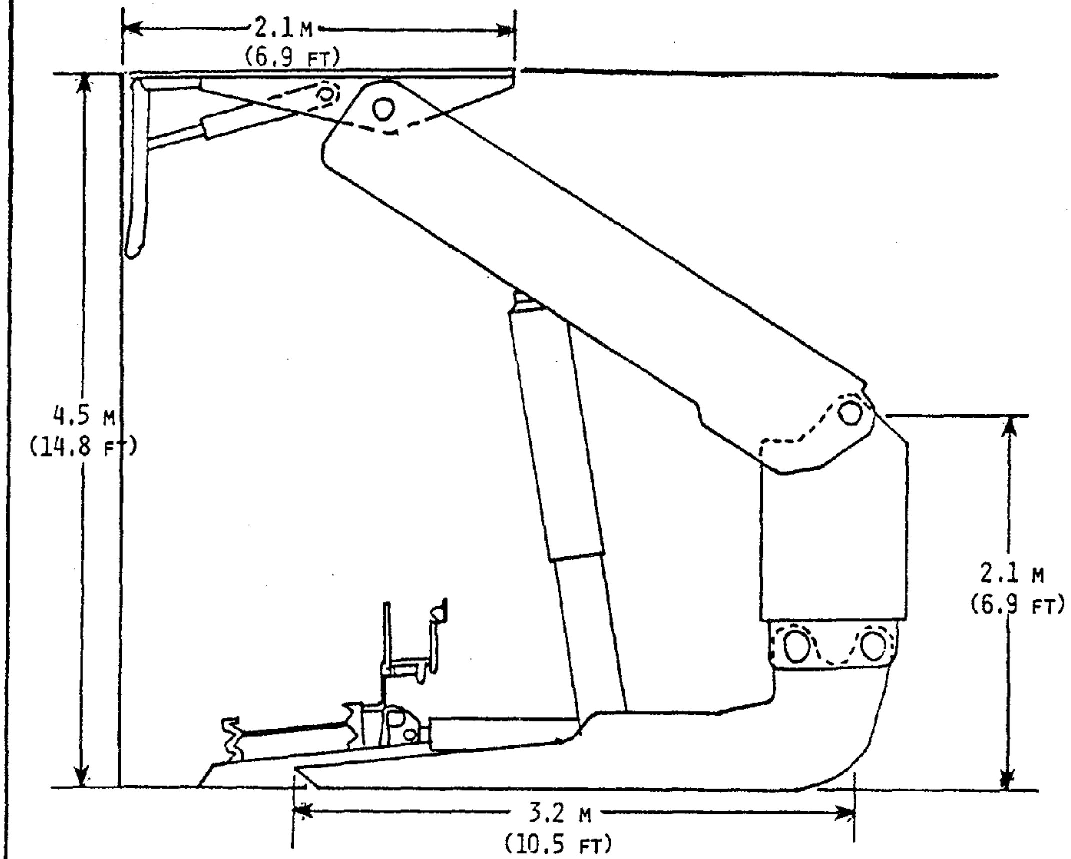
Among the conclusions of the paper are the following:

- o The use of short canopies (2100 mm) had a beneficial effect due to the reduced area of supported roof.
- o The possibility of supporting the roof just behind the winning machine is essential in bad roof conditions. (One web back system.)
- o Shield supports have one articulation and are not lemniscate shields, but no difficulty was encountered. High resistance of the support prevented coal spalling.

1.3.3 Summary

Longwall mining of thick seams in a single pass is increasing in the Upper Silesian District of the Polish coal mining industry due to:

- o the large number of seams in the thickness range of 3.50 to 5.00 m (11.5 to 16.4 ft.)
  - predominantly located in flat to slightly inclined strata,
  - having generally a hard, stable coal,
  - having good roof conditions,
- o the recent progress achieved in winning, support and haulage technology, partly with the aid of West European know-how and licenses.



2 LEGS X 200 TONS = 400 T (360 ST)	EXTENDED - 2.4 M (7.9 FT)
HEIGHT CLOSED - 4.5 M (14.8 FT)	WIDTH - 1.5 M (4.9 FT)

FIGURE 1-6

Polish Thick Seam Shield - Zabrze Mine

1.4 West German Practices

In the bituminous coal mining industry of the Federal Republic of Germany, there are currently only a few mines in which seams having thicknesses of more than 3.5 m (11.5 ft.) are encountered and worked. Such collieries are located in the central and eastern parts of the Ruhr District ("Minister Achenback" in Lunen, "Minister Stein" in Dortmund and "Radbod" in Hamm), which belong to Ruhrkohle AG; the Auguste Victoria Colliery in Marl-Huls of the Badishe Anilinund Sodafabriken AG 9BASF), the Westfalen Colliery in Ahlen (Westphalia) of the Eschweiler Bergwerks-Verein (EBV), as well as in the Saar District (the Ensdorf, Luisenthal and Reden Collieries belonging to the Saarbergwerke AG).

Papers on the collieries "Friederich der Grosse" and "Sachsen" (Ruhrkohle AG) were published some time ago on thick seam mining. These collieries have been closed for several years. In the other bituminous coal mining districts of the Federal Republic of Germany (the Aachen and Ibbenburen Districts) no seams of the above-mentioned thickness are mined.

The thick seams of the Saar District have a greater coal strength than those in the Ruhr District. Moreover, the thick seams worked in the Ruhr District are subject to greater stress and are also weakened by cleats and fissures.

The majority of the thick seam longwall faces operated in West German coal mines are working under difficult conditions, such as:

- o soft coal which, in the Ruhr District, is highly traversed by cleats and fissures;
- o coal faces that tend to spall;
- o relatively soft, frequently friable roof;
- o depth of the seams;
- o high abutment pressures due to unmined pillars in overlying seams,

Earlier attempts failed because winning and support technologies had not been mastered. Mining seams of more than 4 m (13 ft.) in a single pass with caving is now possible with the introduction of efficient cutting machines having a sufficient cutting height and, above all, of shield supports with high load capacity, good protection from the gob, good stability, and equipped with hydraulic alignment jacks and coal face sprags.

Other important measures which contribute to the success of the longwall faces, are the following:

- no unsupported roof due to shield supports (except in case of spalling)
- holding of the coal face by face sprags which are an essential part of the self-advancing supports,
- nailing and bolting of the coal face,
- bonding of the roof strata, when required.

The production results of the investigated longwall faces are satisfactory, in regard to the amount of equipment and productivity, but are not comparable with those from the seams having thicknesses of 2 to 3 m (6.6 to 9.8 ft.). Longwall faces operating in the 2 to 3 m (6.6 to 9.8 ft.) range show the best results in the German coal mines. The reason is that with a decrease in seam thickness, there is a decrease in operational problems, accompanied by a reduction of operational investments. Despite these problems, the trend observed is that thick seams of above 3.5 m (11.5 ft.) are presently worked by the single pass method, when ten years ago the slicing technique was used. The reason may be that the modern mining techniques (e.g., the modern winning and supporting techniques) developed and now employed have made the method of single pass thick seam mining both easier and safer.

1.4.1 Past Examples of Thick Seam Mining

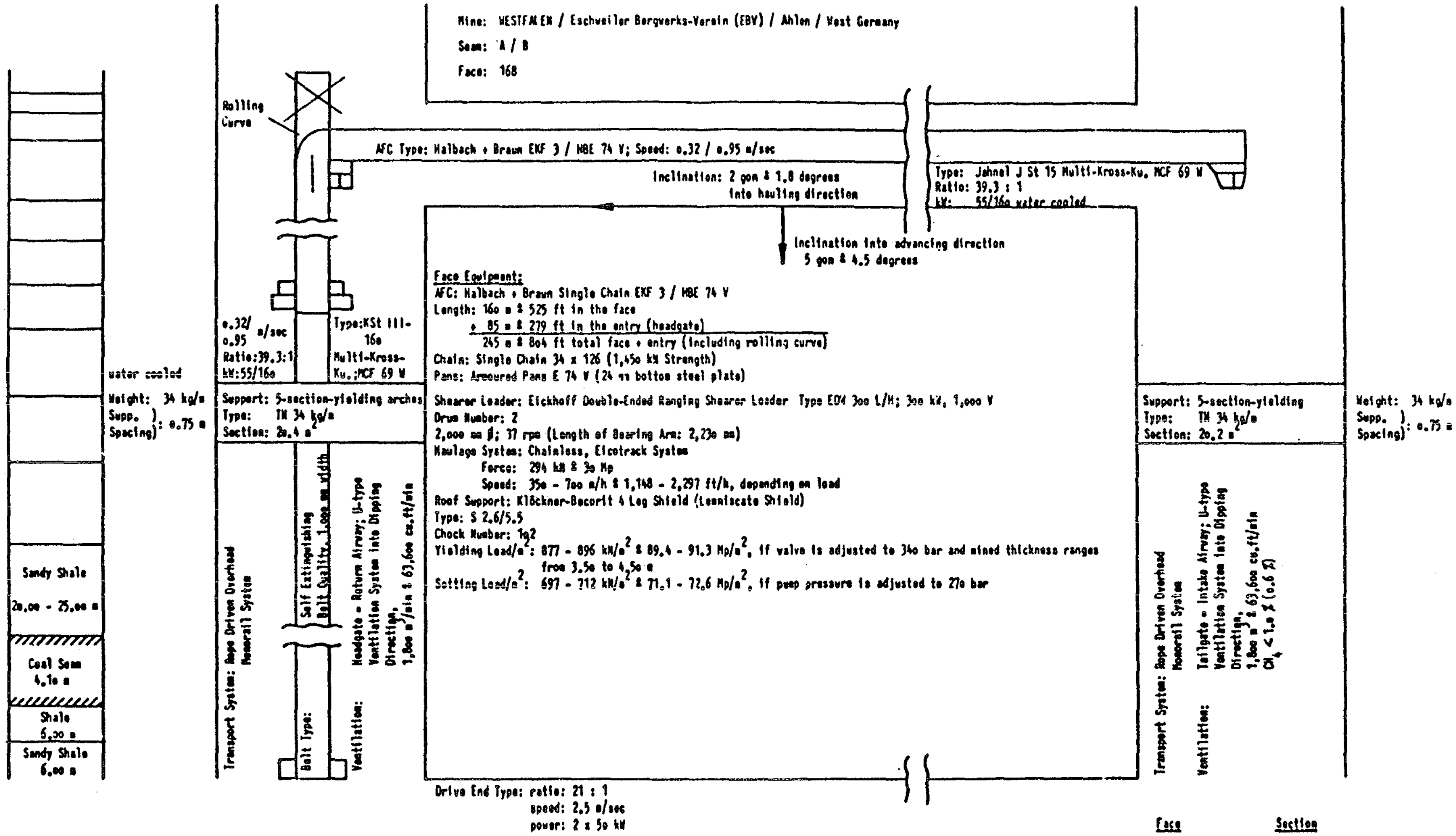
Mining a 4 m (13.1 ft.) seam with a mechanized longwall was first tried at Minister Achenback in 1970 with 5-leg Becorit chocks in the Zollverein VII seam. The next trial was made in 1974 at the Sachsen Mine in the Wilhelm seam. The thickness was limited to 3.8 m (12.5 ft.) leaving some bottom coal. The next attempt was made at the Westfalen Mine in the EFG seam. In November 1975, face equipment designed for a maximum height of 5 m (16.4 ft.) was installed. Since that time, Westfalen has mined several panels. In regard to this 6-year experience, it is interesting to examine in more detail the problems of operating such a face in the A/B seam.

1.4.2 Present Experience

1.4.2.1 Westfalen Mine

Westfalen is mining double or triple seams. The A/B seam is a double seam characterized by a parting of varying thickness.

Total mined thickness is about 4.1 m (13.5 ft.) with an average of about 3.3 m (10.8 ft.) of coal. The net production is 75% of the raw production. The coal is soft and is traversed by cleats and fissures. (See Figure 1.7 for longwall operating data.)



Roof Support:	Klöckner-Becorit 4 Leg (Lemniscate) Shield-Support
Winning Machine:	Eickhoff Double-Ended Ranging Shearer Loader EDW 300 L/H
Face Length:	approx. 154 m ≙ 505 ft
Avg. Seam Thickness:	4,10 m ≙ 13,5 ft, whereof coal 3,40 m ≙ 11,2 ft
Mined Thickness:	4,10 m ≙ 13,5 ft
Advance per day:	Avg. 2,89 ≙ 9,5 ft
Tons per day:	raw coal 3,000 metric tons/d) which was production rate July 79, (4,300 metric tons/d clean coal 2,200 metric tons/d) however, November 79: (3,200 metric tons/d

	Face	Section
Manpower per shift	18 (x 4)	26 (x 4)
per day	72 + 19 = 91	104 + 32 = 136
Time available at the face		280 min
Cutting Time of the shearer loader		168 - 210 min/shift
Utilization rate		60 - 75 %
N_b days for installation		30 working days (WD)
N_b Men/Shifts for installation*		2,500

*Note: This face is a slowing face!

FIGURE 1-7
Longwall Data - Seam A/B, Face: 168

The longwall face unit has been operating in its present panel since May 1979, and was planned as a slewing face. At the time of reporting (Dec. 1979/Jan. 1980), the slewing stage was in operation for the first time. Slewing consists of turning the advancing longwall 90°; further slewing of 180° was planned at a later point in mining.

This mining plan was chosen to increase the life of a panel and to reduce the number of face-to-face moves. Long life is of particular importance for longwall faces in thick seams because large amounts of heavy equipment are installed. The maximum panel length for one face operation achieved by slewing at the Westfalen Colliery exceeds 3 km (approximately 2 miles).

Both gate roads are cut by the face winning machine so that they are kept in line with the advancing face. The result is the that convergence in the two gate roads can be considerably reduced by comparison with the convergence in entries driven in advance. Because of the thickness of the seam, the support of the entry heading, which is advanced together with the face, is difficult; however, it can be successfully achieved.

The criteria required for successful operation in thick seams at the Westfalen Colliery are the following:

- o layout, in view of the long panel life desired, to be based on slewing and, where necessary, on heading through faults and zones of varying thickness;
- o cutting-type winning using shearer loaders which provide better control of the coal face;
- o use of shield support with a maximum extensible height of 5.5 m (18 ft.) for controlling any possible drawslates or cavities; all shields are equipped with at least 5 aligning jacks for stability and alignment corrections;
- o cutting of the entries by the face winning machine for reducing entry convergence;
- o protection of the face ends at the T-junctions with the aid of specially designed shields developed from the standard stowing shield support;
- o use of anhydrite packwalls at least 2 m (6.6 ft.) wide.

A number of problems have been identified which relate to the operation of longwall faces in thick seams. These are summarized below.

The main problem is the instability of the coal face, which is due to both geological and operational reasons. It may give rise to sloughing of up to 5 m (16.4 ft.) measured in the direction of mining advance (as well as to cracks and cavities in the roof ahead of the canopies). The lumps of coal and rock breaking out of the face and roof make haulage operations difficult and endanger the working crew.

Geological causes include:

- pressure wave effects parallel to the stratification in the case of very thick seams;
- occurrence of drawslate between 0.3 m and 1.5 m (1.0 ft. and 4.9 ft.);
- unfavorable direction of the face in relation to the direction of cleats and fissures;
- occasional poor filling of the gob area, hence, the possibility of the roof strata moving parallel to the stratification towards the gob area (the result is that the roof may cave ahead of the support and the support loses its abutment);
- presence of dirt partings of varying thickness in the seam of up to 2.2 m (7.2 ft.);
- soft coal;
- great depth, approximately 1,000 m (3,300 ft.);
- thick sandstone or sandy shale beds in the roof, which create periodic pressures.

Operational causes include:

- unfavorable direction of cutting (to the dip better than to the rise, varying success when cutting in the strike);
- remaining pillars above the mined seam;
- insufficient support resistance;
- excessive delay in supporting;

- o low rate of advance;
- o full face cutting method; this method induces considerable delay in support operations; for this reason, selective face cutting is often more favorable (cutting in two benches).

Measures implemented for achieving successful control of the problem are:

- o use of a 4-leg lemniscate shield with articulated canopy, which is far superior to a circular arc shield and provides high support capacity (up to 750 kN/m^2, 7.8 st/sq. ft.);
- o use of coal face sprags on the canopies, to be set hydraulically against the coal face and kept in this position until shortly before the passage of the shearer loader; this considerably reduces spalling of the coal face;
- o nailing of the coal face using dowels which are firmly clamped with an expanding wedge at the bottom of the borehole; the use of bonded square dowels is too expensive; glass fibre rods were found to have too low a shearing strength, so that quite large coal lumps often break away despite nailing;
- o bonding the coal face and the roof with polyurethane in cases of emergency only, since this method is too expensive for constant use;
- o as a last resort, supporting the roof with steel bars 3 to 4 m (9.8 to 13.1 ft.) long if the coal face sloughs to a considerable extent and major roof areas are exposed.

The longwall face unit has been operated with success. From the viewpoint of roof support, the face is well controlled. Despite very soft coal, little face spalling occurs as a result of the combination of the above procedures. The use of the hydraulically activated coal face sprags is the most important factor.

In the design of the mine plan, consideration was given to orienting the longwall face at a favorable angle to the cleats and fissures. With the slewing operation, difficulties are expected; the longwall face will stand at an unfavorable angle to the cleats and fissures and run for a short while parallel to them.

According to past experience, output losses of 10-15% of the normal production are to be expected during slewing. These disadvantages are acceptable in order to reduce the frequency of face-to-face moves. During this phase, special attention is paid to securing the coal face (e.g., by spragging or nailing).

Table 1-3 presents a synopsis of the trend in accidents in the selected longwall face unit No. 168 in Seam A/B from the time it went into production until the period under review.

TABLE 1-3

Face Accidents in Face No. 168 Seam A/B, Westfalen Colliery, EBV (Period Since Start-Up)

Month	Number of Accidents (Total)	Lost Time Caused by (2) (Shifts)	Number of Accidents as per (2) Attributed to Thick Seam Mining	Lost Time Caused by (4) (Shifts)	Reasons for Accidents as per (4)
(1)	(2)	(3)	(4)	(5)	(6)
May 79	3	41	2	19	rock-fall
Jun 79	6	82	1	17	rock-fall
Jul 79	10	139	3	72	rock- and coal-fall
Aug 79	6	66	1	9	rock-fall
Sep 79	6	81	2	34	rock-fall
Oct 79	5	79	2	28	rock-fall
Nov 79	5	57	1	15	coal-fall
Dec 79	5	93	1	7	face-worker slipped from spill plates
TOTAL	46	638	13	201	

As Table 1-3 illustrates, the number of accidents is not excessive. The number of accidents that could be classed as typical of thick seam operations is quite small. The causes of thickness-related accidents may be said to be due almost entirely to rock and coal falls.

According to the Safety Department of the Westfalen Colliery, the overall number of accidents in all faces, despite mining generally in thick seams, has declined. This may well be attributable to the constant improvements in support technology. Today a wide range of methods are implemented to prevent the spalling of the coal face. The places where accidents mainly occur have shifted to the entries and to the T-junctions (more workers are now being employed in these areas). In absolute terms, however, the number of accidents has also decreased in these locations.

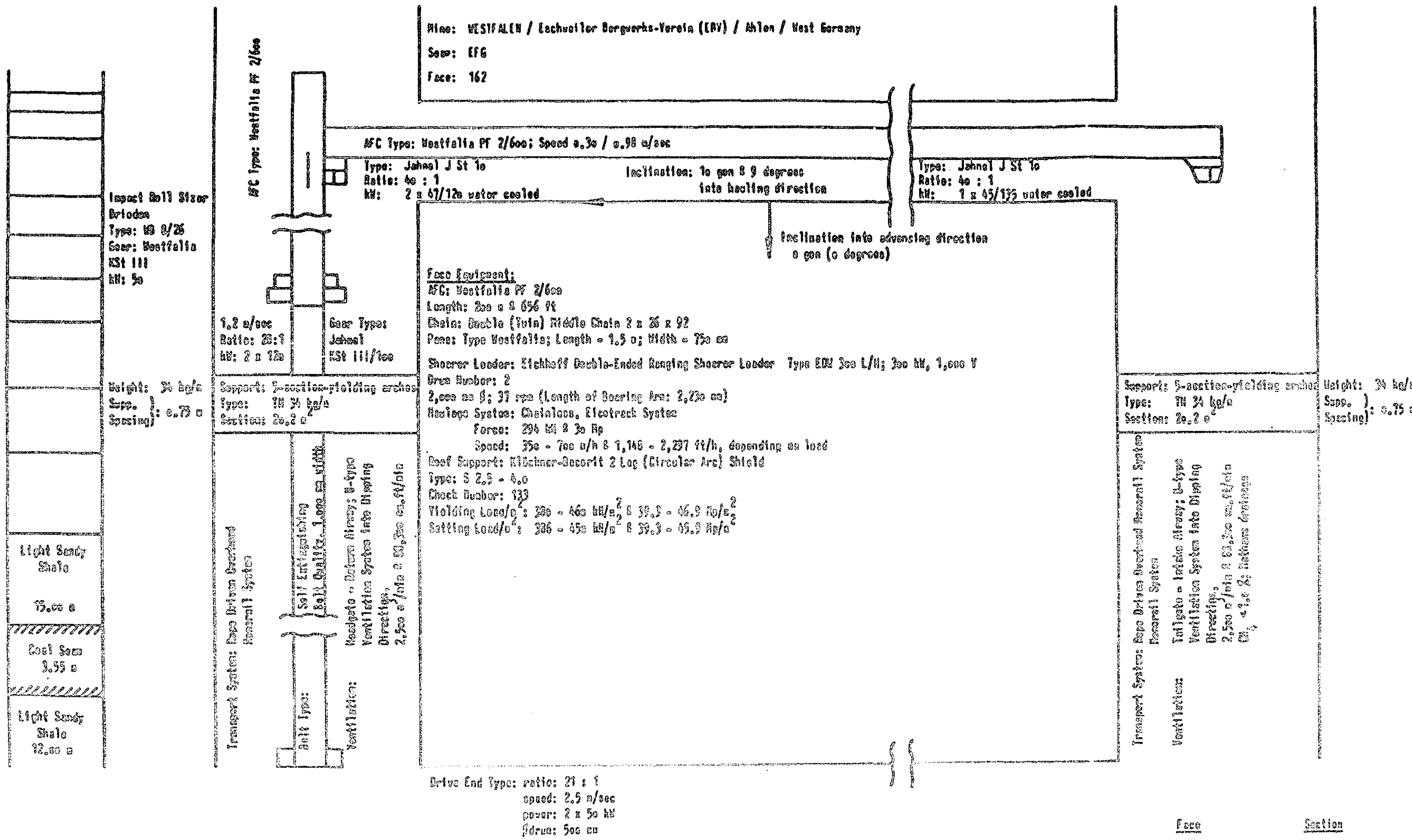
Figures 1-8 and 1-9 contain the data on two other longwall units at the Westfalen Colliery, which are currently being operated in the EFG Seam. These two longwall faces have almost the same layout as that described above, and are also operated as slewing faces. Except for a few technical details (e.g., shield support of earlier design), they are identically equipped.

The difference in output of the two longwall faces in Seam EFG is due to Face No. 164 EFG contending with more difficult conditions, such as:

- o very soft coal,
- o high-degree of spalling of the coal face,
- o and as a result of this - increased tendency for roof-falls.

For these reasons, up to DM 2,100 ($4,200) are spent per day on the use of polyurethane to artificially consolidate the coal face and the roof strata. Furthermore, large coal lumps falling into the conveyor during spalling cause breakdowns of the shearer. Such lumps of coal must be broken with pneumatic picks or by blasting.

Generally speaking, all means of securing and consolidating the coal face have proved their value in these two longwall faces. Without such precautions, coal extraction in these seam thicknesses would be nearly impossible under existing conditions. Two photographs of Westfalen faces illustrate these trials. A Sagem shearer is represented in Figure 1-10. An Eickhoff shearer and Klockner Becorit four-leg shield are shown in Figure 1-11.

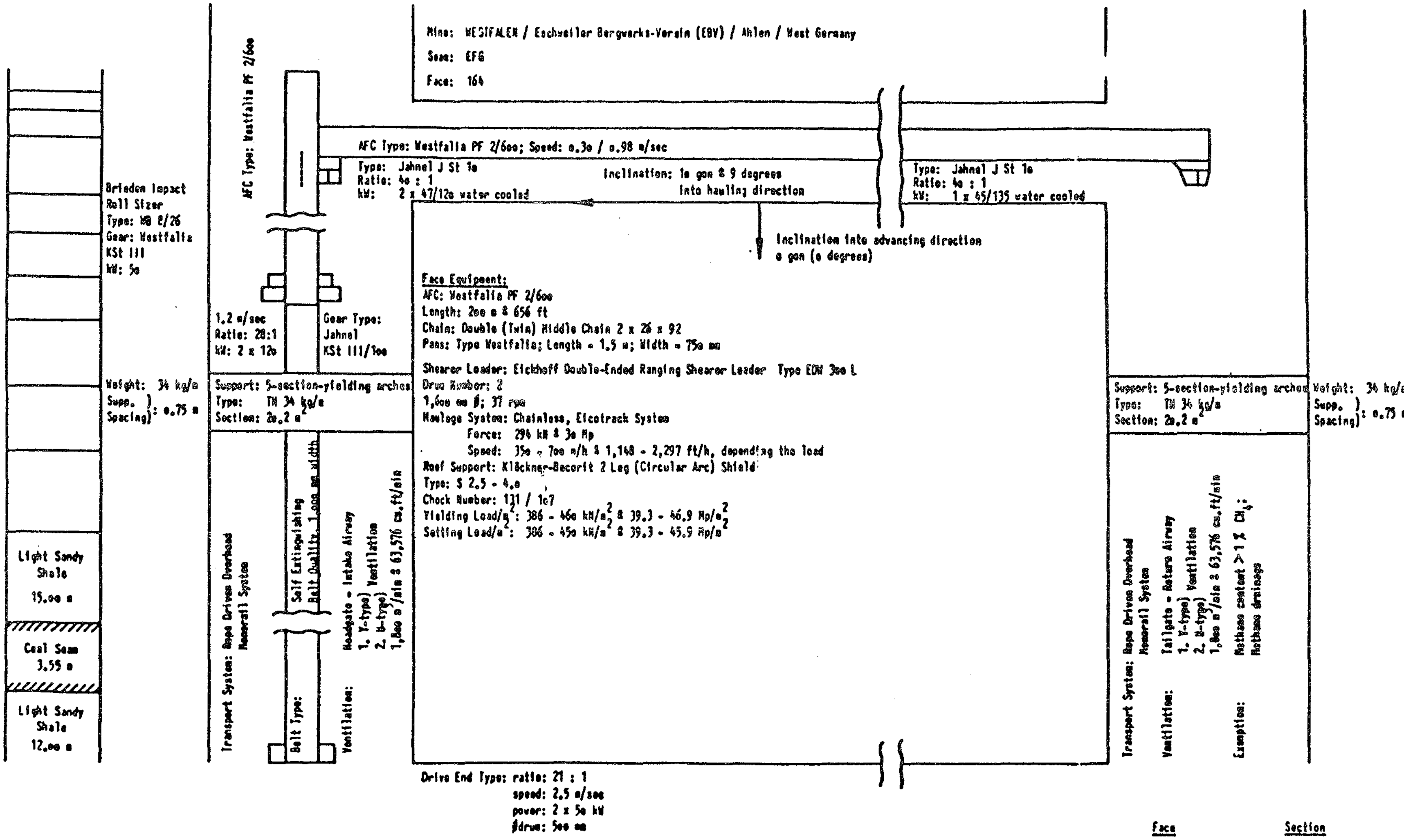


Roof Support:	Klöckner-Becorit 2 Leg (Circular Arc) Shield Support
Winning Machine:	Eickhoff Double-Ended Ranging Shearer Loader EDW 300 L/N
Face Length:	200 m ≙ 656 ft
Seam Thickness:	3.55 m ≙ 11.7 ft
Mined Thickness:	3.55 m ≙ 11.7 ft
Advance per day:	2.5 m ≙ 8.2 ft
Tons per day:	raw coal 3,200 metric tons/d
	clean coal 2,600 metric tons/d

	Face	Section
Manpower per shift	18 (x 4)	26 (x 4)
per day	72 + 19 = 91	104 + 32 = 136
Time available at the face	300 min	
Cutting time of the shearer loader	165 - 180 min/shift	
Utilization rate	55 - 60 %	
N_b days for transfer and installation°	60 working days (WD)	
N_b Men/Shifts for transfer and installation°	~10,000	

° including withdrawal of face equipment and driving of raise

FIGURE 1-8
Longwall Data - Seam EFG, Face: 162



Roof Support: Klöckner-Becorit 2 Leg (Circular Arc) Shield Support
Winning Machine: Eickhoff Double-Ended Ranging Shearer Loader EDW 300 L
Face Length: 200 m ≙ 656 ft
Seam Thickness: 3,60 m ≙ 11,8 ft
Mined Thickness: 3,60 m ≙ 11,8 ft
Advance per day: 2,00 m ≙ 6,6 ft
Tons per day: raw coal 2,000 metric tons/d
clean coal 1,600 metric tons/d

	Face	Section
Manpower per shift	18 (x 4)	26 (x 4)
per day	72 + 19 = 91	104 + 32 = 136
Time available at the face	300 min	
Cutting time of the shearer loader	165 - 180 min/shift	
Utilization rate	55 - 60 %	
N_b days for installation	30 working days (WD)	
N_b Man/Shifts for installation	1,800	

FIGURE 1-9
Longwall Data - Seam EFG, Face: 164

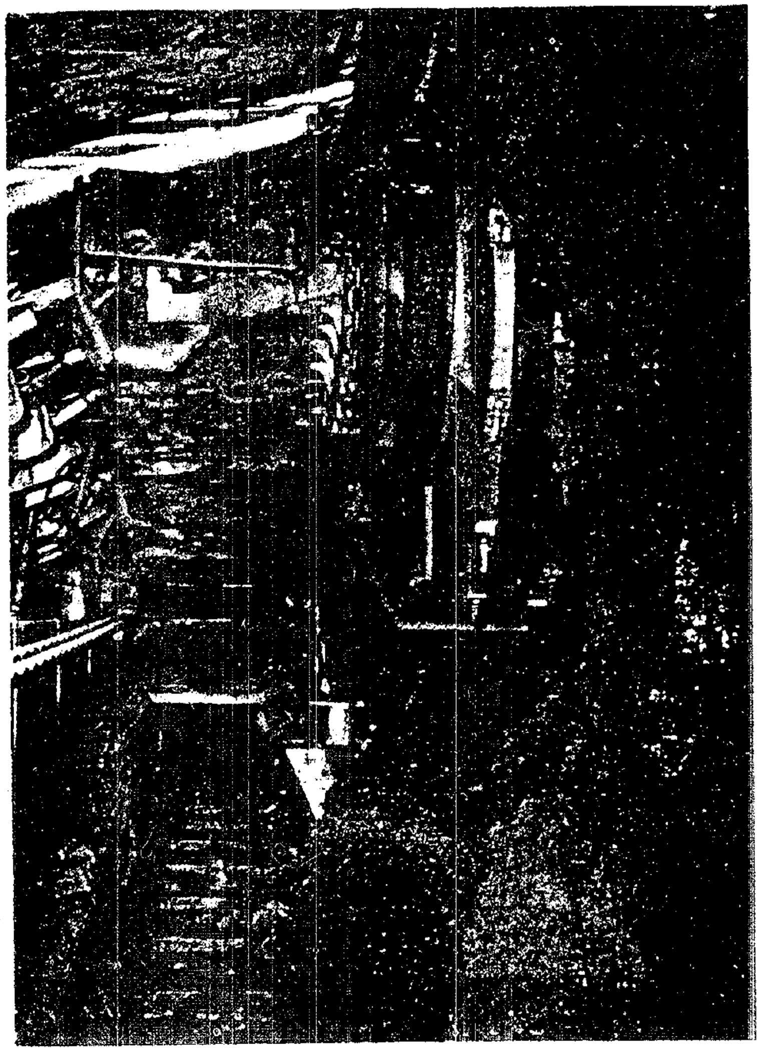


FIGURE 1-10

A Sagem Double Ended Ranging Drum Shearer

FIGURE 1-11
An Eickhoff Shearer and Klockner-Becorit Four-Leg Shields

1.4.2.2 Other Examples in the Federal Republic of Germany

Figures 1-12 and 1-13 present data on two longwall units working in the thick seam Zollverein 2/3 at the Auguste Victoria Colliery in Marl-Huls (Northern edge of the Ruhr District). Both faces yield excellent results, although typical Ruhr District conditions prevail, i.e.:

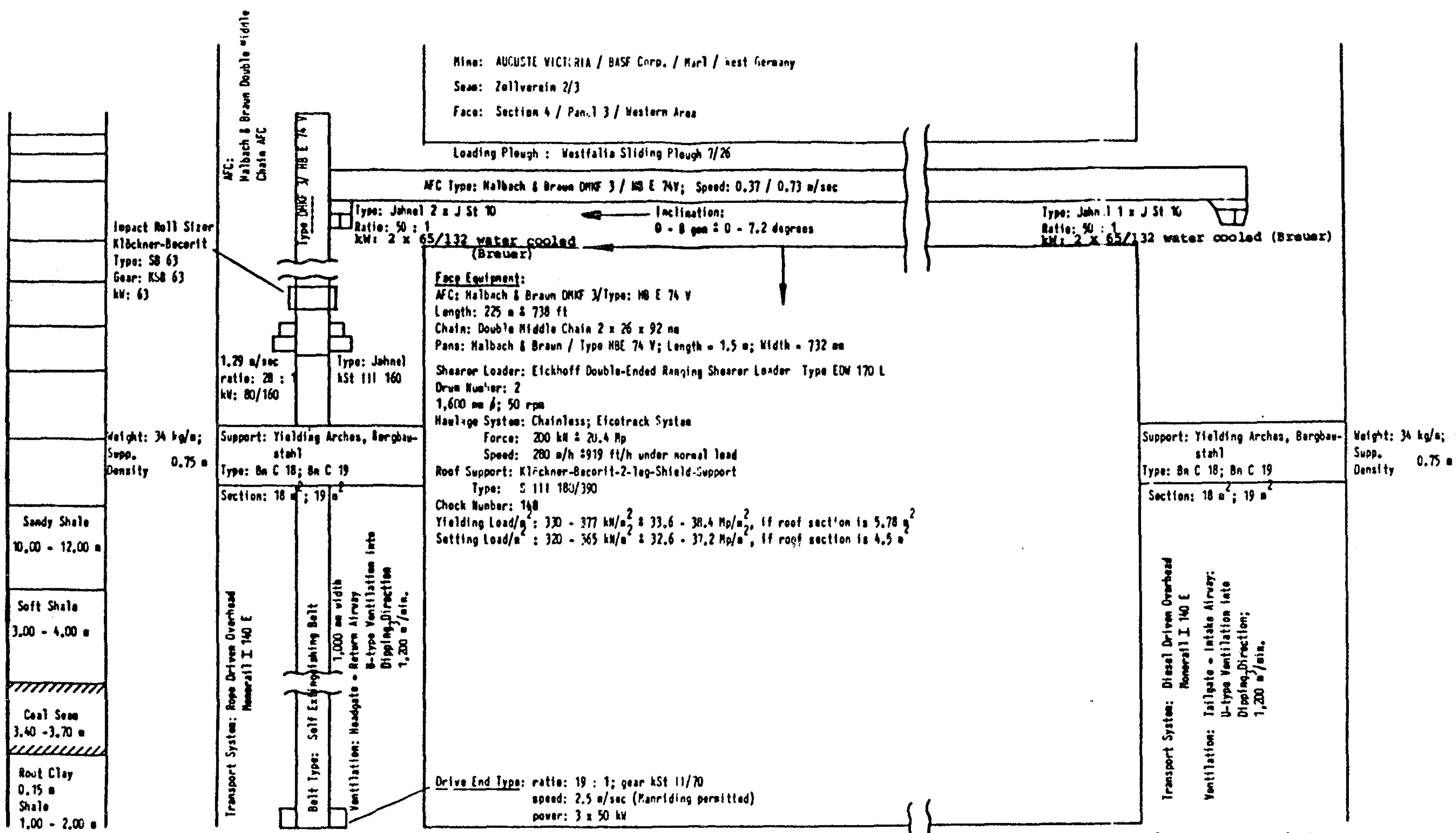
- the coal face tends to spall due to the soft coal,
- the roof is soft and very friable.

For the above reasons, the following measures are implemented:

- In one of the two longwall faces (Figure 1-10) top coal is left standing, despite the use of modern shield support, in order to preserve the roof. In the event of any cavities that may nevertheless occur, either
  - the roof is bonded by polyurethane, or
  - the roof cavities are filled pneumatically with anhydrite.
- Both longwall face units are operated with an exemption of water infusion because it weakens the coal and increases the spalling.
- Hydraulically extensible canopies and coal face sprags are used in both longwall faces in conjunction with shield support. They reduce the need for nailing the face (nailing is an additional measure done under extreme conditions).
- If necessary, the roof at the face ends is consolidated artificially by polyurethane injections.
- Along the entry that has to be kept open for a second use, an anhydrite packwall is constructed at least 2 m (6.6 ft.) wide.

The management of the August Victoria Colliery believes that neither of the two longwall faces described in Figures 1-12 and 1-13 could be worked without the above-listed measures.

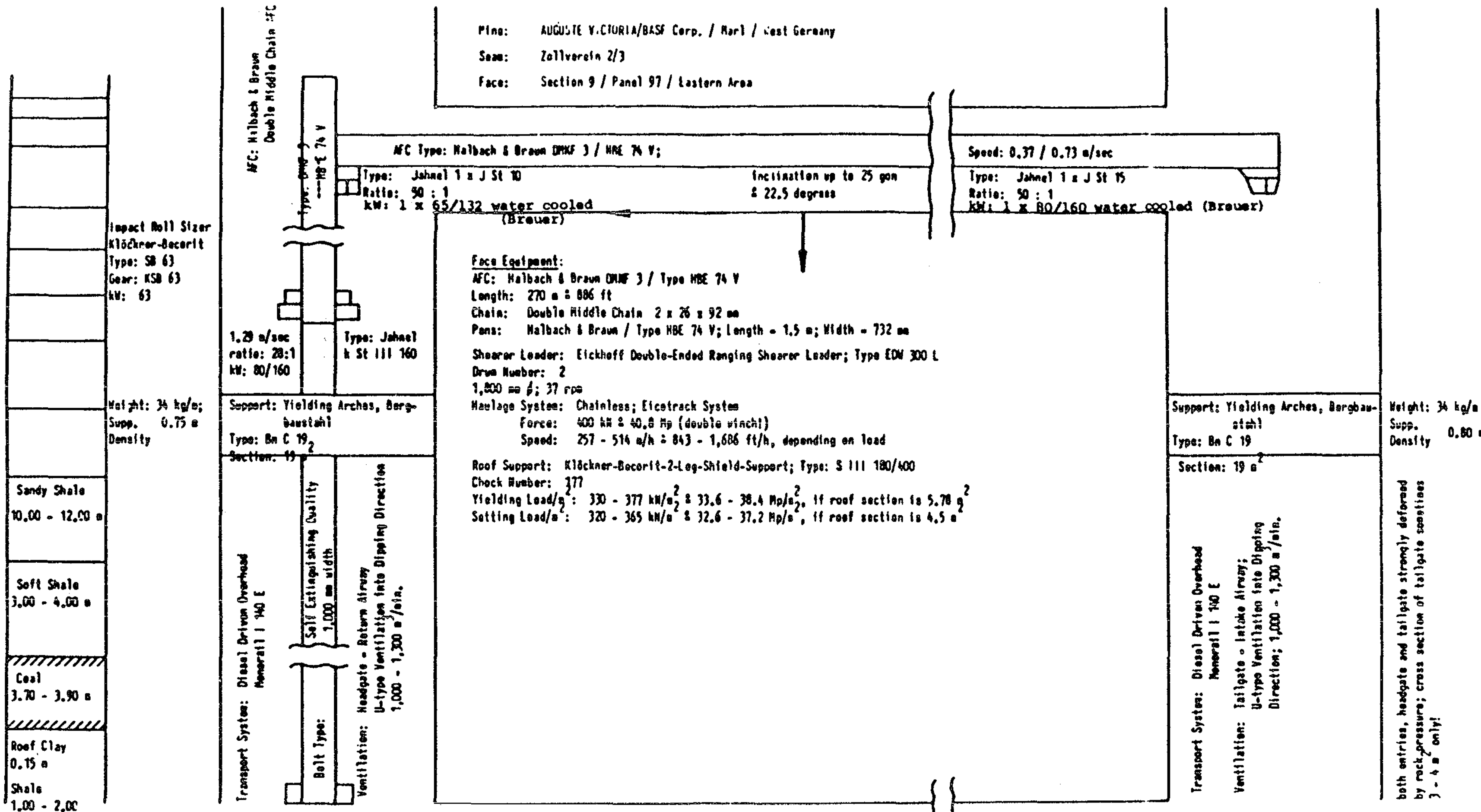
Figure 1-14 provides the operating data for a longwall face at the Reden Colliery belonging to the Saarbergwerke AG. In comparison to the previous case examples from the Ruhr District,



Roof Support:	Klöckner-Becorit-2-Leg-Shield-Support
Winning Machine:	Eickhoff EDW 170 L
Face Length:	225 m ≙ 738 ft
Seam Thickness:	3.40 m - 3.70 m
Mined Thickness:	3.50 m
Advance per day:	3.90 - 4.00 m
Tons per day:	raw coal 6,7ꟷ0 clean coal 3,ꟷ00

	Face	Section
Manpower per shift	12 (x 4)	25 (x 4)
per day	48	100
Time available at the face	300 min.	
Cutting time of the shearer loader	105 - 1ꟷ0 min./shift	
Utilization rate	350 min. ≙ 35 %	
Nb days for transfer and installation	33 - 40 working days	
Nb Men/Shifts for transfer and installtion	3,ꟷ00 - 4,ꟷ00	

FIGURE 1-12
Longwall Data - Seam Zollverein 2/3, Face:
Section 4/Panel 1.3/Western Area



Roof Support:	Klöckner-Becorit-2-Leg-Shield-Support
Winning Machine:	Eickhoff EDW 300 L
Face Length:	270 m ≙ 886 ft
Seam Thickness:	3.70 m - 3.90 m
Mined Thickness:	3.30 m - 3.50 m
Advance per day:	2.50 m
Tons per day:	raw coal 5,600 clean coal 2,500

	Face	Section
Manpower per shift	12 (x4)	25
per day	48	100
Time available at the face	280 min.	
Cutting time of the shearer loader	300 min.	
Utilization rate	300 min. ≙ 30 %	
Nb days for transfer and installation	40	
Nb Men/Shifts for transfer and installation	ca.4,000 for installation only (very difficult conditions!)	

FIGURE 1-13
Longwall Data - Seam Zollverein 2/3, Face:
Section 9/Panel 97/Eastern Area

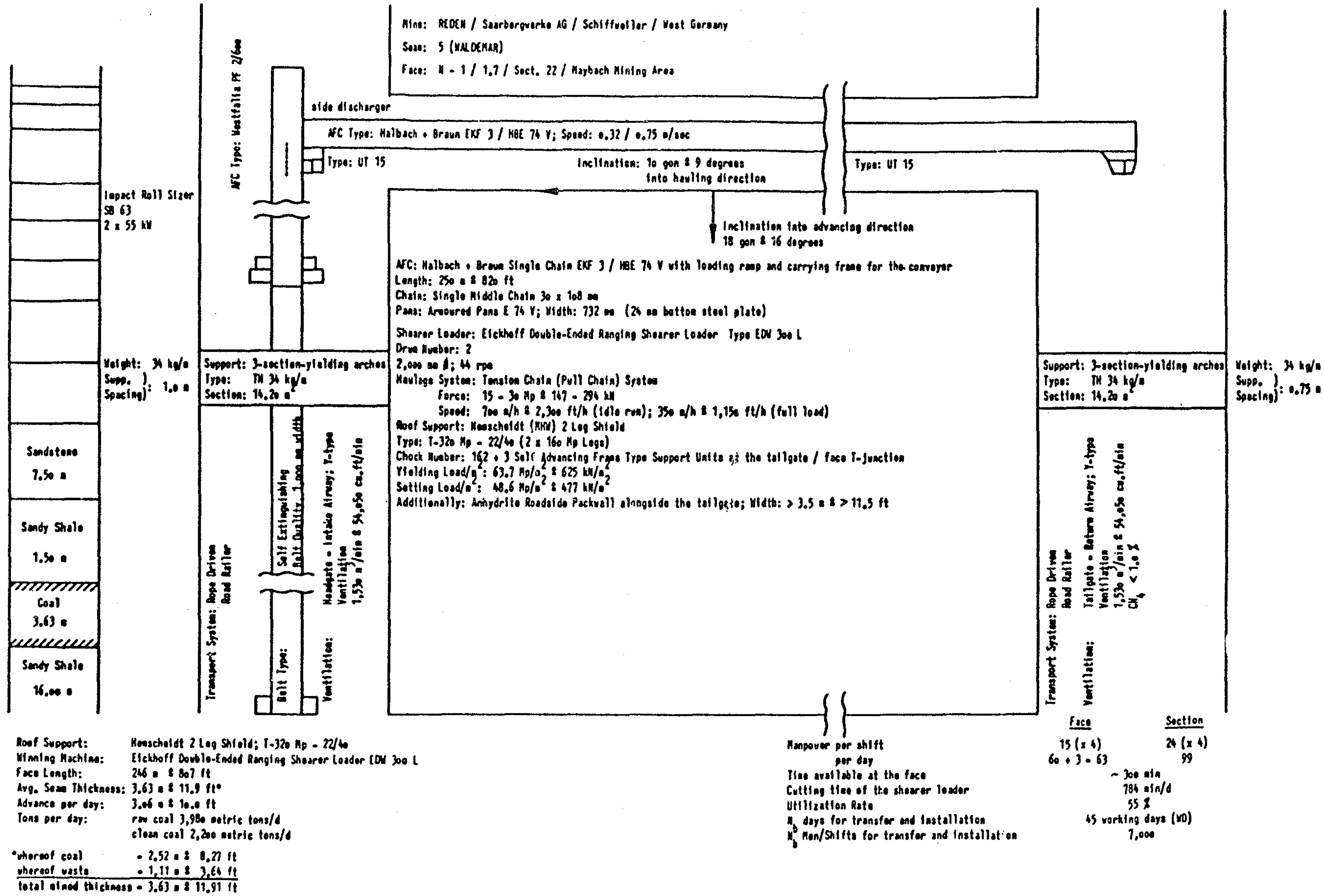


FIGURE 1-14
Longwall Data - Seam 5 (Waldemar), Face:
W-1/1.7/Section 22/Maybach Mining Area

this longwall face operates under substantially more favorable geological conditions.

A fact to be noted is that the longwall face is worked to the dip (inclination 16 degrees), and the coal being somewhat harder, the coal face stands without spalling. For this reason, it is generally not necessary to apply additional equipment such as coal face sprags, or other measures such as nailing or bonding of the coal face. Only the T-junctions are consolidated by resin bolts or polyurethane injections. To improve the operations in the face, the conveyor is equipped with a frame, and the shield support base can be advanced under the conveyor. This improves the stability of the shield support.

1.5 State of the Art Summary

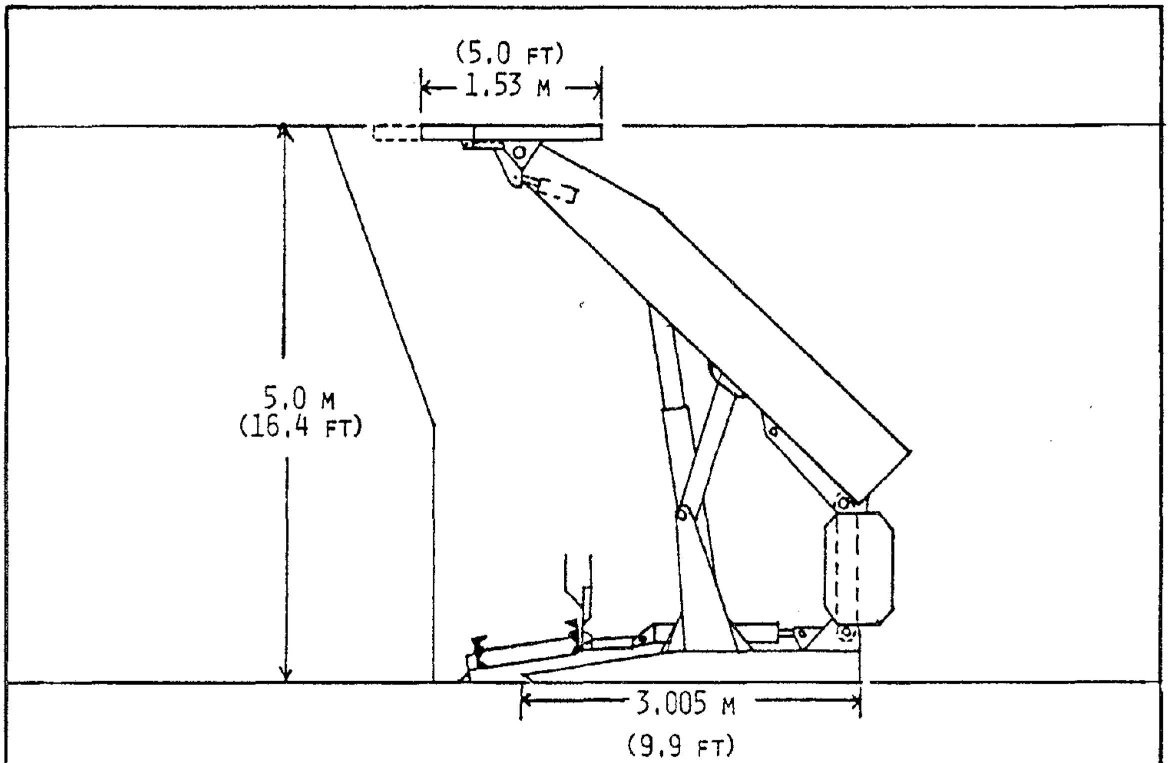
1.5.1 Comparison of Mining Conditions and Face Equipment

Starting with the KTU, which is not a support, but a shield that relies on the solid coal to support the strata, through the OMKT, which was designed to support a narrow band of roof, shield supports have evolved as dinosaurs did. A canopy may reach 3 to 5 m (9.8 to 16.4 ft.), increasing the required load capacity to the square of the length of the canopy. The base, which originally advanced under the conveyor to provide a support close to the face, is now often short. Losing its conceptual advantages, the shield support had to compensate by tremendous increases in load capacity, increases which seem to have no limit. Surprisingly, the width of the shield generally remained unchanged, as did the length of the conveyor pan. It is, therefore, interesting to compare the present face equipment and the related mining conditions. Figures 1-15 to 1-21 represent different shield supports presently used in mining seams thicker than 3.66 m (12 ft.).

The Soviet KM120 (Figure 1-15) shield support kept a short canopy and a long base advancing under the conveyor. The load density is low per unit length but high per unit area. At the other extreme, the Japanese prototype has a canopy of 5 m (16.4 ft.) (see Figure 1-16).

In Germany, there were two different trends: the development of a four-leg shield, with a short base and a long canopy, (Westfalen Mine), (see Figures 1-17 and 1-18); and a parallel development of a two-leg shield, with a long base advancing under the conveyor and a shorter canopy (Saar, Figure 1-19). Westfalen Mine recently ordered face equipment with two-leg shields extendable up to 6 m (20 ft.) with telescopic legs (three extensions) and short base (Figure 1-20 and Figure 1-21).

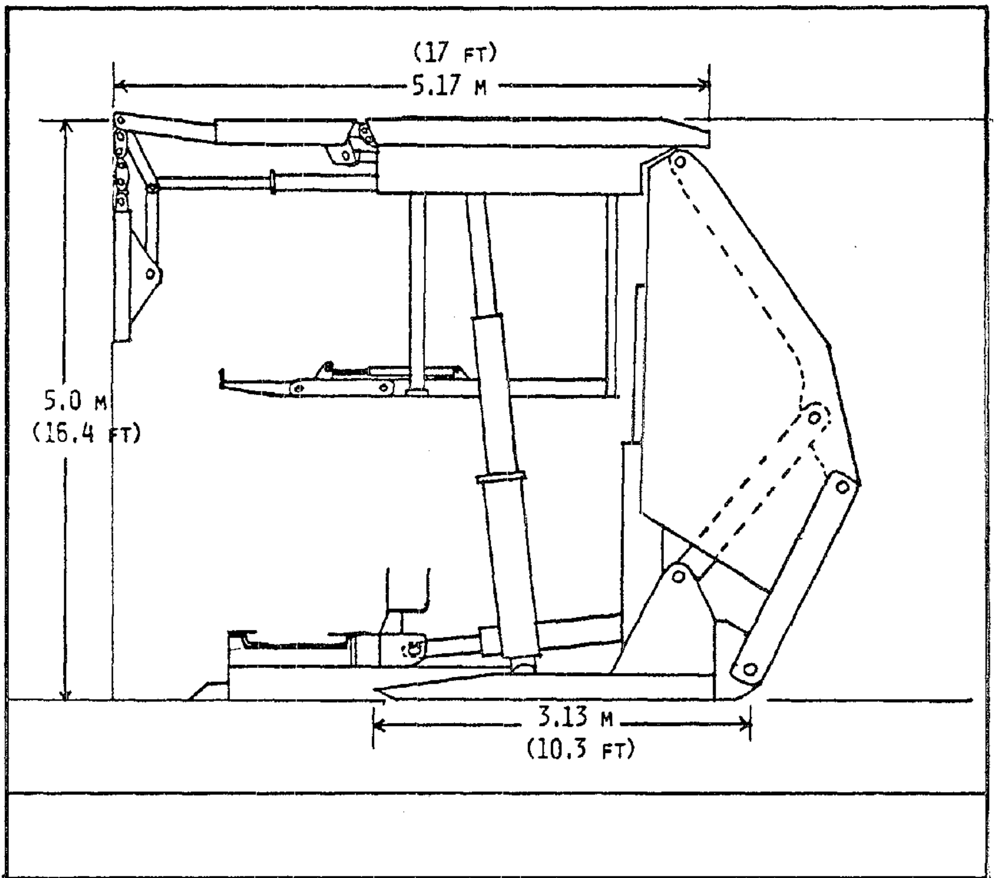
In Poland, where German-manufactured supports are used, a two-leg support is used to get the advantage of a short canopy, and the base advances under the conveyor (Figure 1-6).



2 LEGS X 155 T - 310 T (279 ST)	EXTENDED - 5.0 M (16.4 FT)
HEIGHT CLOSED - 3.5 M (11.5 FT)	WIDTH - 2.2 M (7.2 FT)

FIGURE 1-15

USSR KM 120 Thick Seam Shield

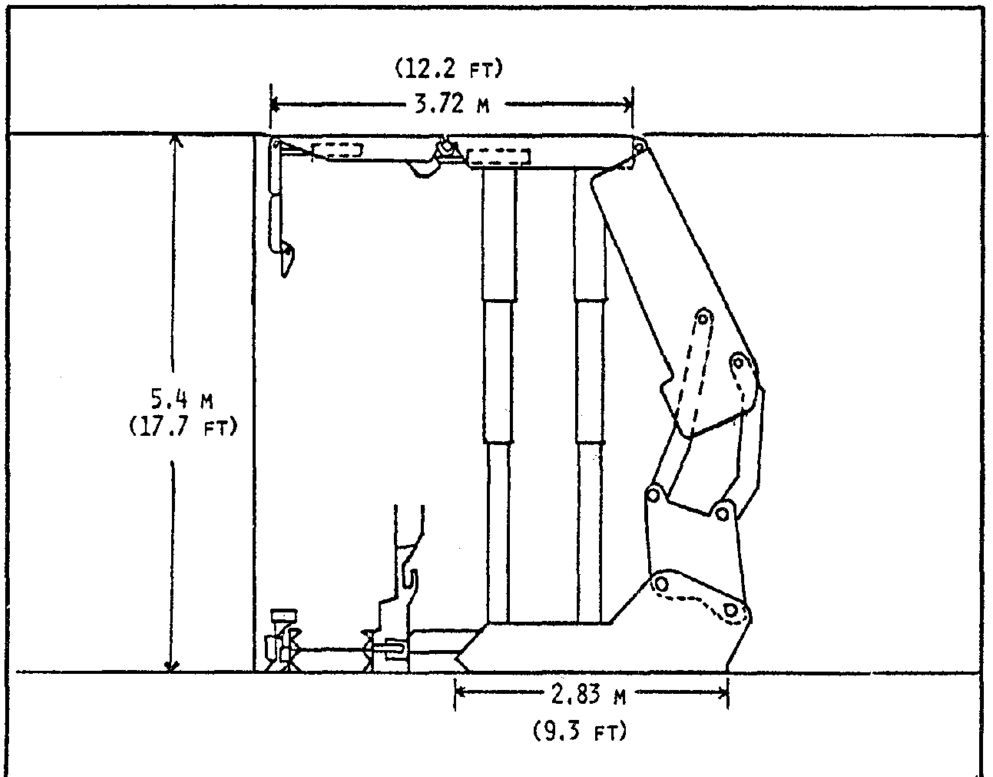


4 LEGS X 180 T = 720 T (648 ST)   EXTENDED - 5 M (16.4 FT)

HEIGHT CLOSED - 2.5 M (8.2 FT)   WIDTH - 1.5 M (4.9 FT)

FIGURE 1-16

Japanese Thick Seam Shield

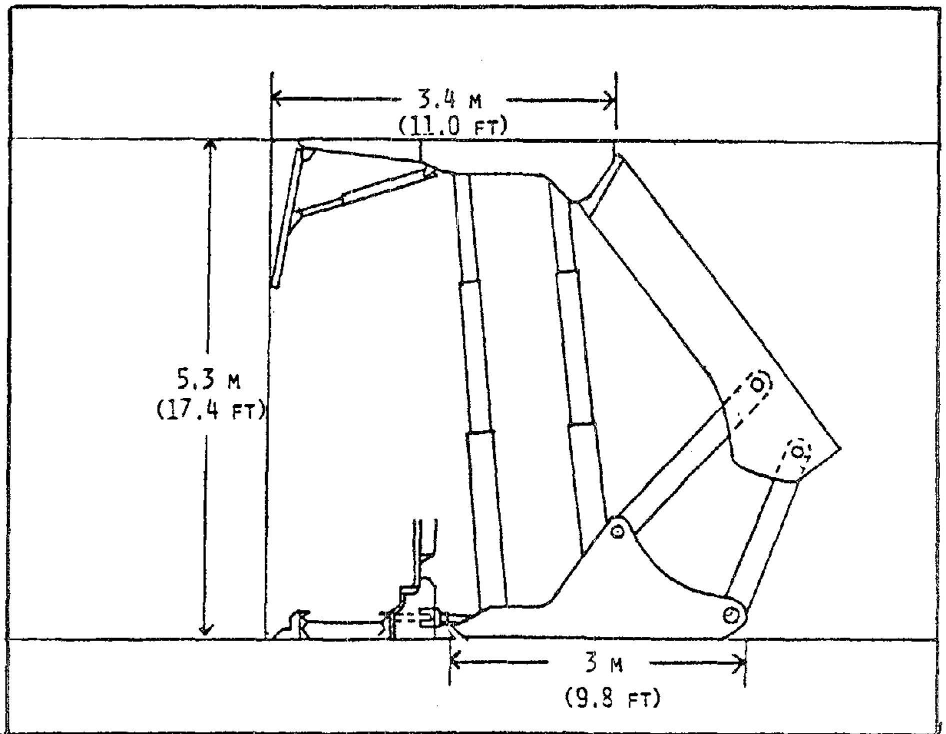


4 LEGS X 165 - 661 ST (727 T) EXTENDED - 5.4 M (17.7 FT)

HEIGHT CLOSED - 3 M (9.3 FT) WIDTH - 1.5 M (4.9 FT)

FIGURE 1-17

W. German - Klockner-Becorit - Thick Seam Shield



4 LEGS X 176 S TONS - 705 ST (775.5 T)

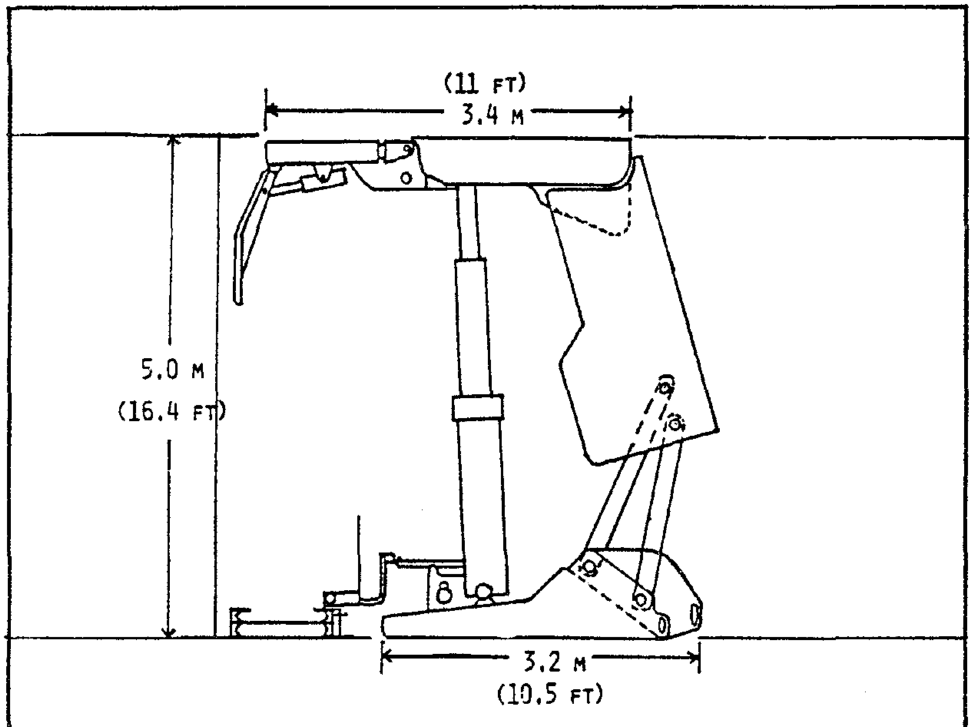
EXTENDED - 5.3 M (17.4 FT)

HEIGHT CLOSED - 2.4 M (7.9 FT)

WIDTH - 1.5 M (4.9 FT)

FIGURE 1-18

W. German - Westfalia Lunen - Thick Seam Shield



2 LEGS X 181 TONS =
362 TONS (326 ST)

EXTENDED - 5.0 M (16.4 FT)

HEIGHT CLOSED - 2.5 M (8.2 FT)

WIDTH - 1.5 M (4.9 FT)

FIGURE 1-19

W. German - Hemscheidt - Thick Seam Shield

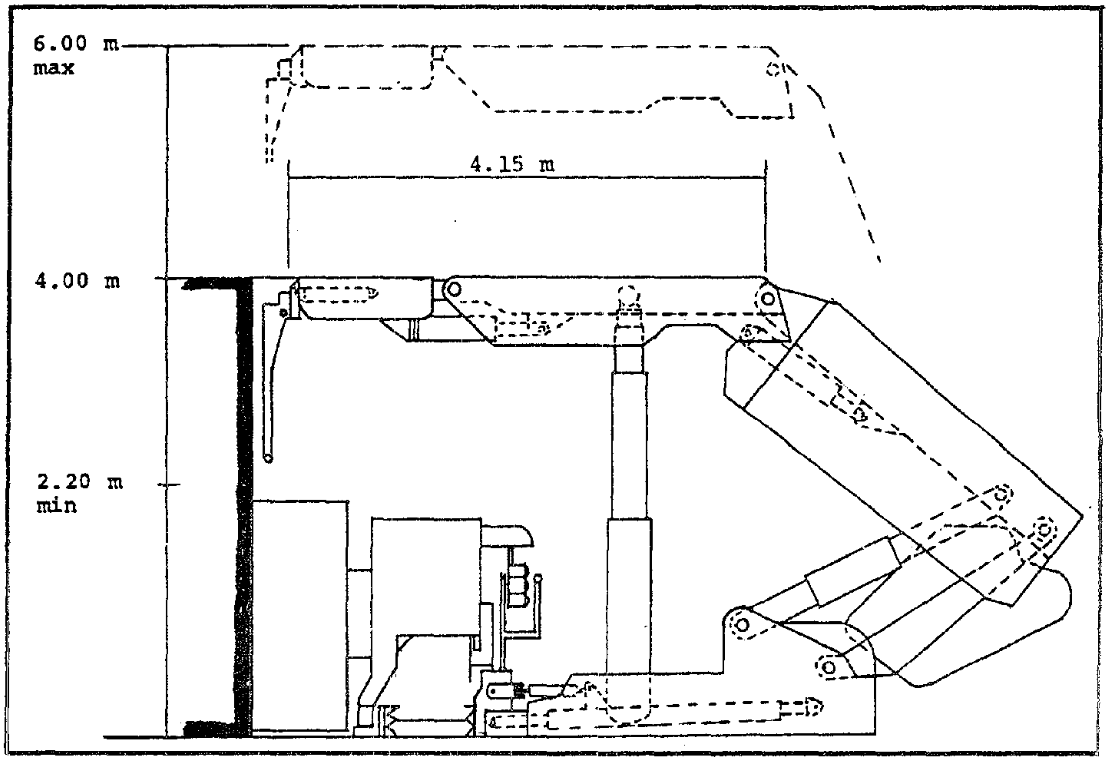


FIGURE 1-20
Westfalen Mine Face Equipment

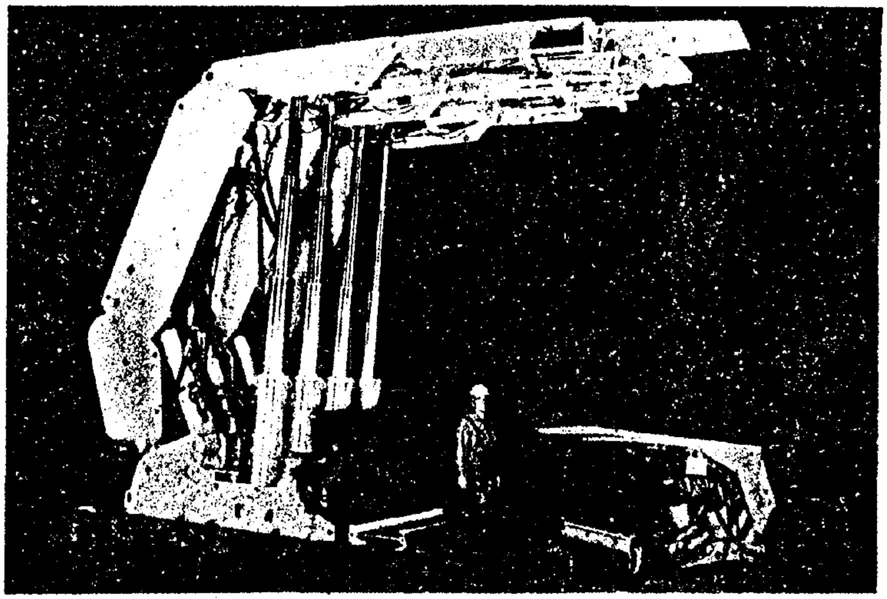


FIGURE 1-21
Photograph of Westfalen Mine Face Equipment

A longer canopy generally means a larger supported roof area. The consequence may be damaging from the point of view of face spalling, for the roof pressure on the solid coal is increased.

Based on the results obtained for the faces mining thick seams, a conveyor carrying frame which permits the roof to be supported closer to the face is favorable. This technique is used in the USSR, Poland and the Saar District of Germany. A two-leg shield is generally favorable, except in the case of a very strong roof at shallow depths.

An increase of the shield width to 2.2 m (7.2 ft.) as it is already done on the Russian KM 120 is recommended. This reduces the manufacturing cost and increases the support stability.

The best results are obviously obtained when the coal is hard enough to have the face standing without using face sprags, nailing or bonding. If the face must be supported, the sprag should be hinged on an extensible part of the canopy to insure its efficiency. If the face sprag is hinged at the end of a fixed length canopy, it acts more as a protection against coal falls than as a face support.

On the KM120 (Figure 1-15), an inclined coal face is represented. The four-drum shearer provides an inclined cut with the two upper drums. In West Germany, a method has been developed where the coal face can be systematically cut at a slope of 65-70 degrees. Following tests at the Bergbau-Forschung GmbH in Essen, an underground trial was conducted at the Rheinland Combined Colliery of Ruhrkohle AG, in West Germany. A specially designed cutting machine, with drums tilted towards the coal face was used (the so-called cutting disc loader), in conjunction with an appropriate shield support system. Development of this system has not yet been concluded.

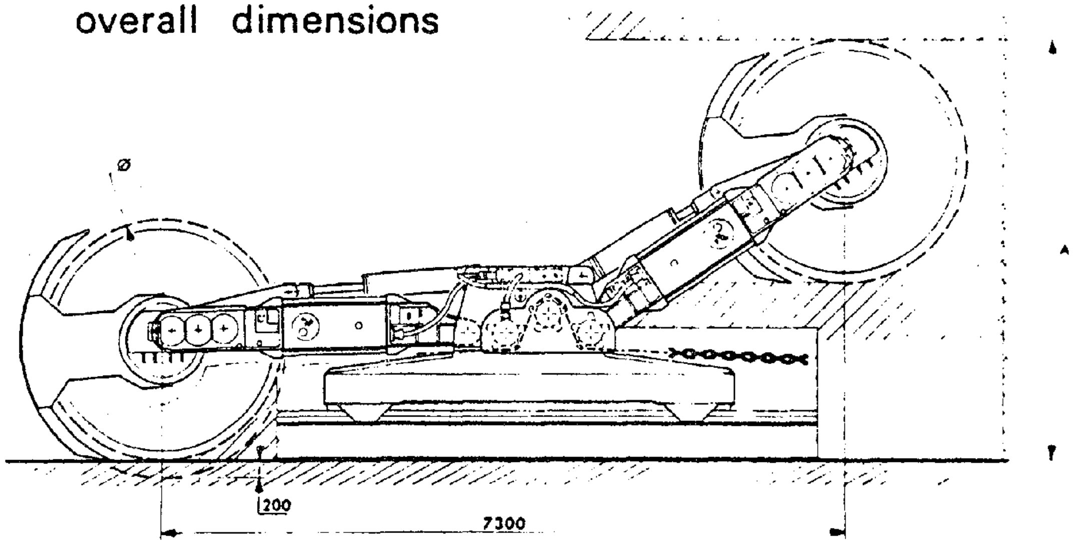
Currently, the upper limit for the shearer loader is 4.5 m (14.8 ft.), except for the SAGEM double drum, 600 kw shearer (Figure 1.22) and the K120 four-drum, 530 kw shearer loader, which cut up to 5 m (16.4 ft.).

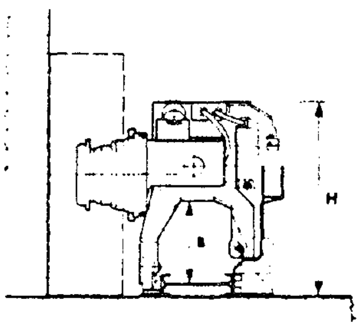
1.5.2 Summary

Of the selected and reviewed coal mining areas of Europe and the Soviet Union, the West German and the Polish coal mining industries presently have the most efficient longwall face units mining thick seams (4 meters) in a single pass.

The majority of the Polish longwall faces are favored by the fact that the thick seams are comprised of hard coal that seldom or never spalls. Additional techniques of coal or roof consolidation are generally not necessary in the Polish mines.

overall dimensions





Ø mm	H mm	A mm	B mm
1600	1500	4100	550
2000	1700	4500	800
2600	1900	5000	1000

general data

– Operating height : from 1,80 m to 5 m
– Power : 600 kW
– Total weight : 30 tons

- **HYDRAULIC WINCH ELECTRONICALLY CONTROLLED**
  . Drawbar pull : 36 tons (40 tons optional)
  . Tramming speed : 0 to 5 m/min
  . Size of haulage chain : 26 x 92 mm (or 30 x 108 mm optional)

- **F.L.P. 300 kW ELECTRIC MOTORS**
  . Stator and rotor : water-cooled
  . Current and temperature electronically controlled

- **CUTTING AND RANGING GEARHEADS**
  . Drum diameter : from 1600 mm to 2600 mm
  . Depht of cut : from 750 to 850 mm
  . Drum rotation speed : 25 rpm
  . Length of the arms : 3250 mm
  . Angle of ranging arm : + 45° / – 10°
  . Dust suppression : high pressure water through drum shafts

FIGURE 1-22

Sagem DTS 600 Shearer

This is not the rule in the West German coal mines, where most of the thick coal seams tend to spall. In addition, roof strata are soft in most West German coal mines.

Satisfactory results in West German longwall faces are attributed to improved technology combining the most up-to-date face equipment with additional means such as nailing, bonding of the coal face, and injections of artificial resin into the coal and the roof. Construction of anhydrite packwalls at the face ends, use of special support structures at the T-junctions, etc., are also important. Modern face equipment allows successful mining of thick seams up to 4.5 m (15 ft.).

In the coal and lignite mines of the Soviet Union, developments in mining thick seams in one single pass are similar to those in West Germany or in Poland. Equipment is slightly different but the problems involved with this method are nearly the same. The productivity rates in Soviet longwall face operations seem somewhat lower than in West Germany or Poland. The absence of economic pressure on the management of Eastern European coal mines allows excessive manpower and lower efficiency.

Geological parameters, such as coal strength, the presence of cleats and fissures, composition and strength of the strata, and type of coal, are important factors. The main factor in mining thick seams is the coal strength, which governs the behavior of the coal face.

The mining layout influences, either in a negative or positive way, the behavior of the coal face, especially where cleats and fissures are present. The behavior or the stability of the coal face is a factor of decisive importance. It determines, above all, the success or failure of such a longwall operation. Furthermore, the selection of the mechanical equipment to be used on longwall faces in thick seams plays an important role. It has been shown that only shearer loaders are suitable for coal winning in thick seams. The application limits of these are currently around a working thickness of 4.5 to 5 m (14.8 to 16.4 ft.). Only shield or chock shield support is used in thick seams. These supports seal off the face from the gob area, and possess high supportive forces and good stability. Hydraulically extensible, hinged forepoling roof-bars and hydraulically activated coal face sprags are essential to improve the efficiency of the shield supports in thick seams. The use of a conveyor carrying frame must be considered. Its use allows for improved stability of the shield and reduced distance between the tip of the shield canopy and the coal face. The face conveyors must be able to take the high weight of the winning machine as well as the pull and thrust forces from the self-advancing support shifting rams. The life of the panel should be as great as possible. For this reason, face conveyors must have a long service life, to avoid time and cost consuming repairs or replacement during panel extraction.

Auxiliary measures for successful operations in thick seams include the construction of anhydrite packwalls and the injection of polyurethane at the T-junctions. In special cases, nailing or bonding the coal face can be helpful, but it is expensive and not an economical way to mine the coal.

In conclusion, after comparing the number of longwall faces in Western and Eastern European countries, including the Soviet Union, it can be stated that:

- o the extraction of thick seams (or slices) up to 5 m (16.4 ft.) is possible from the technical viewpoint and capable of being mastered even in very deep mines,

- o the production results, except in a few cases, are still lower than the results which are obtained in thicknesses between 2.5 m (8.2 ft.) and 3.5 m (11.5 ft.).

Good geo-technical conditions and a proper matching of equipment are required to insure favorable economic results. The first essential condition for economic and technical success is hard coal. The next factor is the selection of the proper equipment. Improper selection will necessitate the use of auxiliary measures such as nailing or bonding to assure face stability.

Technological evolution of face equipment has steadily increased the limit of the seam thickness mineable in one single pass up to 5 m (16.4 ft.); however, careful selection of the site, good planning, and the right equipment selection are still required to insure safe and economically successful operations.

Figure 1-1 shows that the most productive coal faces in West Germany are in the range of 2.4 to 3.6 m (8 to 12 ft.) with soft coal. The slow increase of the mined thickness at Daw Mill (U.K.) is shown in Table 1-4. Two slices are mined from a 9 m (30 ft.) thick seam. The mined thickness has been cautiously increased with the progress of face equipment. Equipment which was delivered at the end of 1981 has a maximum height of 4 m (13 ft.).

TABLE 1-4

Production Results from Advancing Faces at Daw Mill Colliery

District	Year	Face Length (m)	Total Tonnes	Average Weekly Tonnes	Output per m/c shift Tonnes	Heigth of Extrac-tion	Average Weekly C.M.S. Tonnes
1's	1965-70	169	1047422	4206	428	1.91	11.82
6's	1966-80	171	163041	3891	418	1.87	11.59
20's	1969-74	166	1124969	5068	372	2.26	11.36
12's	1972-75	230	674460	4716	399	2.47	14.00
64's	1974-75	230	753457	7245	555	2.58	17.17
71's	1975-76	230	968169	11002	818	3.15	16.90
14's	1975-79	233	664472	8411	612	3.29	18.86
63's	1976-79	230	210403	5260	524	3.22	13.70
75's	1976	232	683288	10512	742	3.29	20.90
72's	1978	226	902732	10211	743	3.20	24.96
15's	1979	231	136246	13668	624	3.41	28.83

2.0 APPLICABILITY OF THE SINGLE PASS SYSTEM TO U. S. MINING CONDITIONS

2.1 Introduction

There are abundant thick seam reserves in the western United States which are recoverable by underground methods. The USBM[1] estimates that at least 45.7 billion tons of sub-bituminous coal are in seams greater than 3.1 m (10 ft.) thick. This represents 56 percent of the sub-bituminous reserve base and 35 percent of the total underground reserve base. The data to estimate bituminous reserves in seams greater than 3.1 m (10 ft.) was unavailable.

Thick coal seams in the western states of the United States are generally partially mined by underground mining methods which are not oriented to save the remaining coal for later mining. If the first slice were completely mined, it should be possible to mine a second slice, when after a few months or years, the caving would be reconsolidated. This is difficult if all pillars are not extracted.

Longwall mining is more appropriate to thick seams than the room-and-pillar method, if two or more slices are possible. It is also easier to adapt to the seam thickness variation.

Longwall mining methods for thick seam can be classified, depending on seam thickness, in single pass mining, slicing and sublevel caving.

- o Single pass mining is currently applied to seams with thickness ranging to 4.5 m (14.7 ft.). Six meters (20 ft.) would be the upper limit using the present technology.
- o Successive slices are recommended if the seam is regular and the thickness exceeds 6 m (20 ft.).
- o Sublevel caving is more appropriate to seams with rapid changes of thickness or with bad roof conditions and applies to a 6-15 m thickness range.

---

1 Matson and White, 1975, p. 13.

In the United States, coal seams of 11-12 feet thickness have been mined successfully by the longwall mining system. Presently, few longwall panels are being planned to mine up to 4.3 m (14 ft.) thickness in a single pass.

While this system could be employed in seams greater than 4.3 m (14 ft.), optimum operation is presently for a mined thickness ranging between 3.1 and 3.7 m (10 and 12 ft.). To determine the applicability of the single pass mining system to U. S. coal resources from 3.1 to 6.1 m (10 to 20 ft.), which is the upper limit technically possible, available resource data has been compiled into a comprehensive thick seam data base.

2.2 Thick Seam Resource Data Base

A resource data base for those states having considerable thick seam coal reserves has been compiled. Alaska, Colorado, Montana, New Mexico, Utah and Wyoming together represent 85 percent of all identified western bituminous coal greater than 1.1 m (3.5 ft.) thick, and nearly 100 percent of all western subbituminous resources greater than 3.1 m (10 ft.) thick. These six states form the basis for the thick seam analysis and are referred to as the thick seam states (see Table 2-1).

TABLE 2-1

Demonstrated Coal Reserve Base of the Thick Seam States, January 1, 1976 by Mining Method

		Millions of Short Tons			
		Potential Mining Method			
Rank Of Coal	State	Under-ground	Surface	Total	% Strippable Reserves
Bituminous	Alaska	617.0	80.5	697.5	11.54
	Colorado	8,467.9	676.2	9,144.1	7.39
	Montana	1,385.4	---	1,385.4	0
	New Mexico	1,258.8	601.1	1,859.9	32.32
	Utah	6,283.8	267.9	6,551.7	4.09
	Wyoming	4,002.5	---	4,002.5	0
Sub-bituminous	Alaska	4,805.9	640.7	5,446.6	11.76
	Colorado	3,972.1	149.2	4,121.3	3.62
	Montana	69,573.5	33,843.2	103,416.7	32.73
	New Mexico	889.0	1,846.8	2,735.8	67.50
	Utah	---	---	---	---
	Wyoming	27,644.8	23,724.7	51,369.5	46.18

Source: Coal Facts 1978-1979, National Coal Association (U.S. Bureau of Mines data).

The data base consists of a series of tables showing resource estimates for thick seams in every county of the states having thick seams. These tables have been used to identify:

- o Beds and counties with abundant thick seam resources.
- o Relative importance of thick seam resources for each state.

2.2.1 Data Source

The National Coal Resources Data System (NCRDS) of the U. S. Geological Survey is the primary source for the resource estimates used in this study. This system is both comprehensive and detailed in its reporting of thick seams. The following is a list of characteristics associated with the NCRDS:

- o Resources in measured, indicated and inferred reliability categories[1]
- o Strippable and underground resources
- o Overburden up to 914.4 m (3,000 ft.)
- o Thickness ranges (Table 2-2)
  - Bituminous coal 1.1 m (3.5 ft.)
  - Subbituminous coal 3.1 m (10 ft.)

TABLE 2-2

Coal Seam Thickness Categories

Seam Category	Bituminous (Inches)	Sub-Bituminous (Inches)
Thin	14-28	30-60
Intermediate	28-42	60-120
Thick	>42	>120

1 See Appendix A for a complete discussion of resource classifications and definitions.

2.2.2 Modifications to USGS Data Base

The use of USGS resource estimates in the analysis of thick seam coal has required adjustments for the following:

- o Strippable resources
- o Unsuitable thickness ranges
- o Uncorrelated resources
- o Resources classed by zone

Strippable Resources

A certain amount of coal recorded in the 0-304.8, 0-609.6 and 0-914.4 m (0-1000, 0-2000, and 0-3000 ft.) overburden categories is strippable under the current economic and political situation. Since only coal which can be mined by underground methods was of interest, it was necessary to subtract a certain percentage from the tonnage estimates. The percentage of strippable reserves was calculated from the information provided in Table 2.1.

Table 2-1 contains information for measured and indicated resources only. It has been assumed that these percentages also apply to inferred resources. Assuming that these percentages are also valid at the bed level, they have been subtracted from 0-304.8, 0-609.6 and 0-914.4 m (0-1000, 0-2000, and 0-3000 ft.) overburden categories. These categories do not provide the detail necessary to analyze seam thicknesses of 3.1 to 6.1 m (10-20 ft.).

Unsuitable Thickness Ranges

KETRON has completed a county-by-county assessment of seam thickness ranges for all coal beds in the selected thick seam states. Those seams with a thickness less than 3.1 m (10 ft.) or greater than 6.1 m (20 ft.) were eliminated from the study. For those seams having a broad thickness range (i.e., 3.1 to 15 m (10 to 15 ft.)) a certain percentage of the resource base was applied. It was assumed that the distribution of resources was uniform over seam thicknesses greater than 1.1 m (3.5 ft.) for bituminous resources, and greater than 3.1 m (10 ft.) for subbituminous resources.

Uncorrelated Resources

Much of the resource base in western states (especially Montana, Wyoming and Alaska) is uncorrelated.[1] For those counties

---

1 The geological evidence necessary to trace a coal bed for any great distance was lacking at the time of the resource estimates. Detailed information (such as seam thickness and overburden) does exist.

having uncorrelated resources, the following method was used to determine thick seam resources:

1. The thickness ranges of all known seams with sub-bituminous resources greater than 3.1 m (10 ft.) or with bituminous resources greater than 1.1 m (3.5 ft.) were recorded.

2. The average of the midpoints of these ranges was calculated.

3. The uncorrelated resources were distributed uniformly around this average using a lower limit of 3.1 m (10 ft.) for subbituminous resources and 1.1 m (3.5 ft.) for bituminous resources. The greatest seam thickness recorded in the county was used as the upper limit.

4. The percentage of uncorrelated resources between 3.1 and 6.1 m (10 and 20 ft.) was then calculated and applied to the USGS data base (Tables 2-3 and 2-4).

Thickness Classed By Zone

A portion of the coal resources of Colorado and New Mexico are classed by zone.[1] Nearly 45 percent (36,151 Mst) of the USGS resource base for Colorado is sorted into this category. Using a method similar to that described for calculating thick seam uncorrelated resources, it was determined that an estimated 12,663 Mst have a thickness range between 3.1 and 6.1 m (10 and 20 ft.).

Over 80 percent (54,498 Mst) of the USGS resource base in New Mexico is classed by zone. Because of this high percentage, and a general lack of seam thickness information, another reference was chosen to supply resource data for part of New Mexico (Fassett and Hinds, 1971, pp. 55-70). This source lists resources greater than those listed on Figure 2-1. A ratio of the area underlain by coal beds 3.1 to 6.1 m (10 to 20 ft.) thick to the area underlain by coal beds greater than 6.1 m (20 ft.) was calculated. This ratio was then applied to Basin in New Mexico to arrive at the total resources between 3.1 and 6.1 m (10 and 20 ft.)

---

[1] A coal zone is composed of several beds. When resources are reported by zone, the detail (such as seam thickness and overburden) necessary to describe individual beds is lost.

TABLE 2-3

Summary of Thick Seam Bituminous Identified Resources for Underground Mining in Millions of Short Tons

	Resources 10-20' Thick (MST)	% of Total Identified Bituminous Resources in Each State	% of USGS Bituminous Resources > 42" in Each State
Colorado	17,438	24	*
New Mexico	540	11	21
Utah	8,573	34	34
Wyoming	2,560	19	26
TOTAL	29,111	26	---

* Due to differences in source consulted, this figure is unavailable.

TABLE 2-4

Summary of Thick Seam Subbituminous Identified Resources for Underground Mining in Millions of Short Tons

	Resources 10-20' Thick (MST)	% of Total Identified Subbituminous Resources in Each State	% of USGS Subbituminous Resources >10' in Each State
Alaska	25,352	23	41
Colorado	2,604	27	35
Montana	37,665	29	49
New Mexico	18,242	25	60
Wyoming	11,035	9	19
TOTAL	94,898	24	---

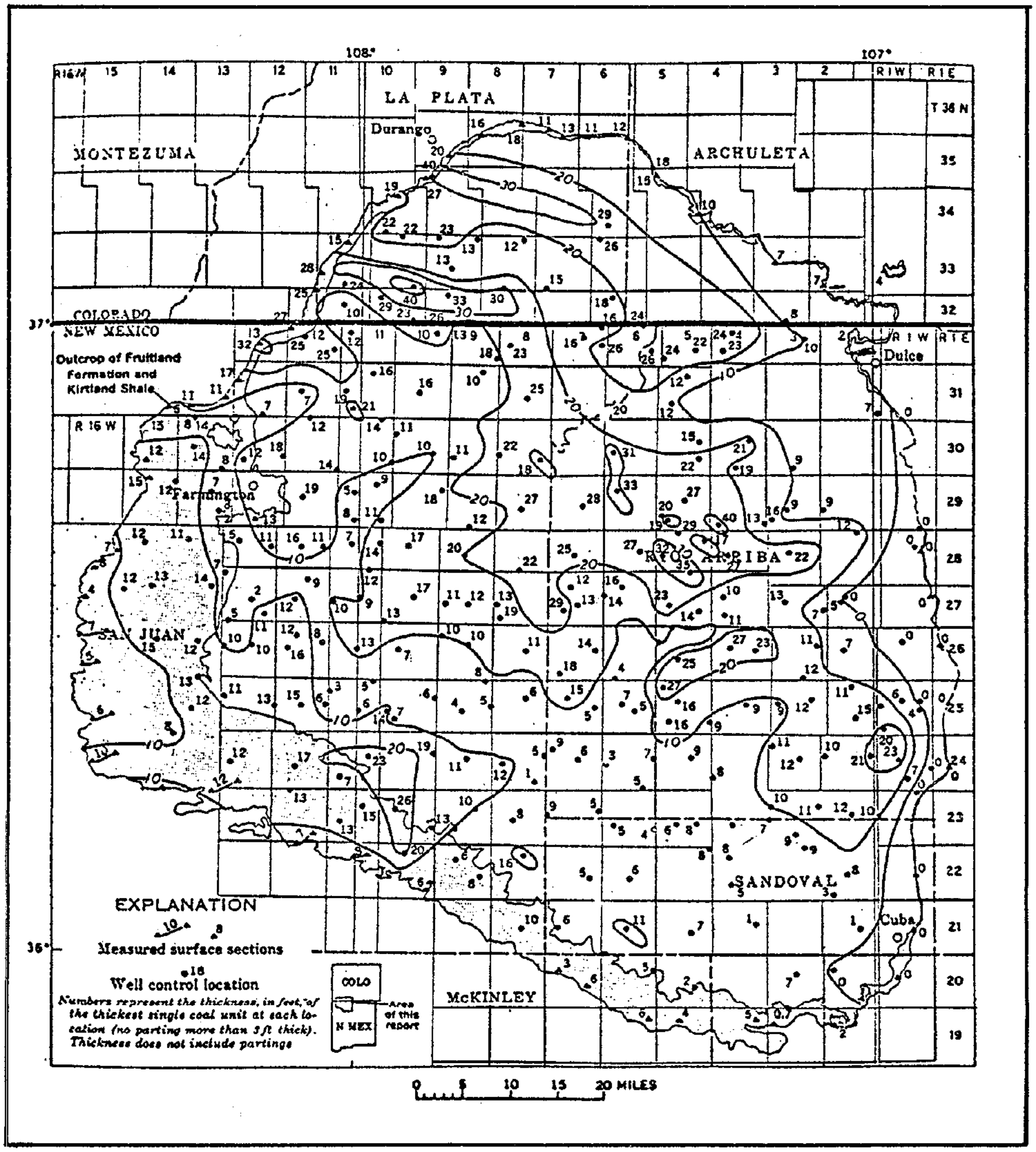


FIGURE 2-1

Isopach Map of the Thickest Individual Coal Units
(No Parting in Excess of 3 ft Thick) in the Fruitland Formation

Source: Fasset and Hinds, 1971, p. 55

thick. The resources in the San Juan Basin for Archuleta, La Plata and Montezuma counties in Colorado were also calculated in this manner.

2.2.3 Resource Data Base

The thick seam resource data base is contained in Appendix B. Listed by county and coal field, the tables contain the following information on known coal beds in the thick seam states:

- Depth
- Rank
- Thickness
- Reliability of data
- Estimated resources between 3.1 and 6.1 m (10 and 20 ft.)
- Dip

Thick seam resources represent 18 percent of the total coal resources of the thick seam states. Tables 2-3 and 2-4 define the contribution of each thick seam state to the entire underground reserves, the importance of each state's thick seam resources relative to its total underground resources, and the percentage of the original USGS resource estimates included.

The resource estimates presented in this study do not reflect the quantity of thick seam coal actually available for mining since factors such as coal quality, multiple beds, gradient of the seams, and poor mining conditions have not been accounted for. Also, only a portion of the coal available for mining will eventually be recovered, the amount depending on the mining system used.

2.3 Applicability of Single Pass Longwall Mining to U. S. Thick Seam Deep Coal Reserves

KETRON contacted several U. S. coal companies owning thick seam reserves and planning to open mines in the near future to mine thick seams. These companies generally plan to mine only part of the thick seams, and 4.3 m (14 ft.) seems to be considered a maximum. However, longwall mining is generally the mining system being selected. It appears that there is a lack of interest in increasing the percent recovery of the in situ coal reserves, since this increase may be offset by uncertainties stemming from the application of an advanced and untested mining system.

The choice will be guided by economic consideration (i.e., the most productively mined thickness), or by institutional considerations (an advanced method will require better trained miners in areas where good miners are not easy to hire). In some cases, the selective mining of part of the seam thickness may give a better roof, and also yield a high quality coal. By mining only the part of the seam that has low sulphur and low ash content, a preparation plant may not be required. A top coal roof will prevent the fall of rocks and insure a high quality R.O.M. coal.

Among the companies owning large coal reserves, Consolidation Coal Company, Rocky Mountain Energy, United States Steel, Coastal States Energy and Utah Power and Light, were contacted. According to collected information, the following companies are planning longwall operations in thick seams:

- o Rocky Mountain Energy
- o Northern Coal Corporation
- o Coastal State Energy Corporation
- o Mid Continent Resources
- o Atlantic Richfield
- o Utah Power & Light
- o United States Steel Corporation
- o Colorado Westmoreland

Rocky Mountain Energy Company, Denver, Colorado, owns large amounts of reserves and is developing a new mine in a joint venture with Dravo Corporation in Hanna, Wyoming. This mine, Carbon County Coal Company, will exploit the No. 80 seam, with thickness varying from 4.3 m (14 ft.) in the southwest portion to 8.9 m (29.30 ft.) in the northeast portion. At the portal, the seam is 5.5 m (18 ft.) thick. The seam is relatively flat (10 percent gradient).

This steam coal has 11,000 BTU, 6 percent ash, .55 percent sulphur, and 12 percent moisture. However, the first foot of coal at the top has a 2 percent sulphur content. Therefore, 1 m (3 ft.) of top coal is left as a roof. The immediate roof is black shale.

The reserves of this mine are 150 million tons. The planned annual production is 1.5 million tons. The longwall operations plan is to initially mine a 3.1 m (10 ft.) thickness, increasing later to a maximum of 4.3 m (14 ft.). The coal will not be cleaned and the main part of the surface facilities is a loading station for unit trains. The management of this mine is not interested in participating in the present DOE study.

Coastal State Energy Corporation, Houston, Texas, operates SUFCo mine and will open a mine at Scofield, Utah, south of Price. They are presently planning to mine a thickness of 3.7 m (12 ft.) at the Skyline Mine. The seams range up to a thickness of 7.3 m (24 ft.).

Mid Continent Resources, Inc., will test longwall multilift mining in its L. S. Wood Number 3 Mine but are presently mining along the roof only 8 1/2 feet out of their 30 feet thick seam.

Atlantic Richfield (ARCO Coal Company), Denver, Colorado, is considering longwall mining of a seam of 4.3 m (14 ft.) at Mount Gunnison, Colorado.

Northern Coal Corporation, Denver, Colorado, is studying longwall mining of a 4.6 to 9.1 m (15 to 30 ft.) seam averaging 6.7 m to 7.3 m (22 to 24 ft.) eventually by sublevel caving.

U. S. Steel Corporation owns reserves in thick seams of 6.1 m (20 ft.) of steam coal but their exploitation first requires a joint venture with a power utility.

Utah Power & Light, Salt Lake City, Utah, is planning to develop thick seam resources in Utah, but not in the range of 16 feet.

Westmoreland Coal Company has a western subsidiary, Colorado Westmoreland, which operates the Orchard Valley Mine in Paonia, Colorado.

Only three coal mines are presently interested in longwall mining a thick seam, up to 5 m (16 ft.) in one single pass, and have appropriate mining conditions. They are:

- SUFCo (Southern Utah Fuel Company)
- Skyline

both subsidiaries of Coastal States Energy Company, and

- Orchard Valley Mine, Colorado Westmoreland Inc.

3.0 SITE SELECTION

The conditions of these mines are very different. Their common features include the variation of seam thickness and the use of diesel vehicles.

3.1 SUFCo Mine

Coastal States Energy Company purchased the Southern Utah Fuel Company (SUFCo) and its rich mountain deposit near Salina, Utah, in 1973. The mine then had an annual production rate of approximately 340,000 tons. Today the mine is producing more than 1,900,000 tons per year. Ultimate capacity is expected to peak at more than 2,000,000 tons annually when expansion plans are completed.

Coastal States' SUFCo mine has in-place reserves of approximately 75 million tons, with an additional 20 million tons of estimated reserves currently under Federal lease application. With normal mining recovery, these reserves are expected to last more than 25 years.

The SUFCo mine is the western-most bituminous coal mine in the United States, and is one of the largest underground mines west of the Mississippi. The product is a high-BTU, low-moisture and low-sulphur steam coal, ideally suited for industrial and utility use.

SUFCo's coal is mined from the Upper Hiawatha seam in the southern Wasatch Plateau. The seam runs as thick as 5.5 m (18 ft.), is free of partings, and it can be shipped without washing.

Much of SUFCo's production is a short haul by truck to a spur of the Denver and Rio Grande Western Railroad in nearby Salina. This rail terminal can load fifty 75-ton hopper cars with 3,750 tons of coal daily.

A second rail terminal at Sharp, Utah, moves SUFCo coal to customers in the Southwestern U. S. The terminal is located 80 miles from the mine on the Provo branch line of the Union Pacific Railroad near Levan, Utah. This rail site can handle 5,000-ton shipments in 100-ton cars, and can be expanded to handle full unit train loading.

The SUFCo mine has unique mining conditions. The coal of the Upper Hiawatha Seam is extremely hard. The shale roof is not of as good a quality as the coal; therefore, a 0.9 m (3 ft.) top coal is left and entries are driven without roof bolting (Figure 3-1). Productivity of the continuous miner sections is excellent due to this advantage (850 tons/shift). The mined thickness is 2.7 m (9 ft.). However, the seam thickness ranges from 3.4 to 5.8 m (11 to 19 ft.) and is in excess of 4.9 m (16 ft.) in some

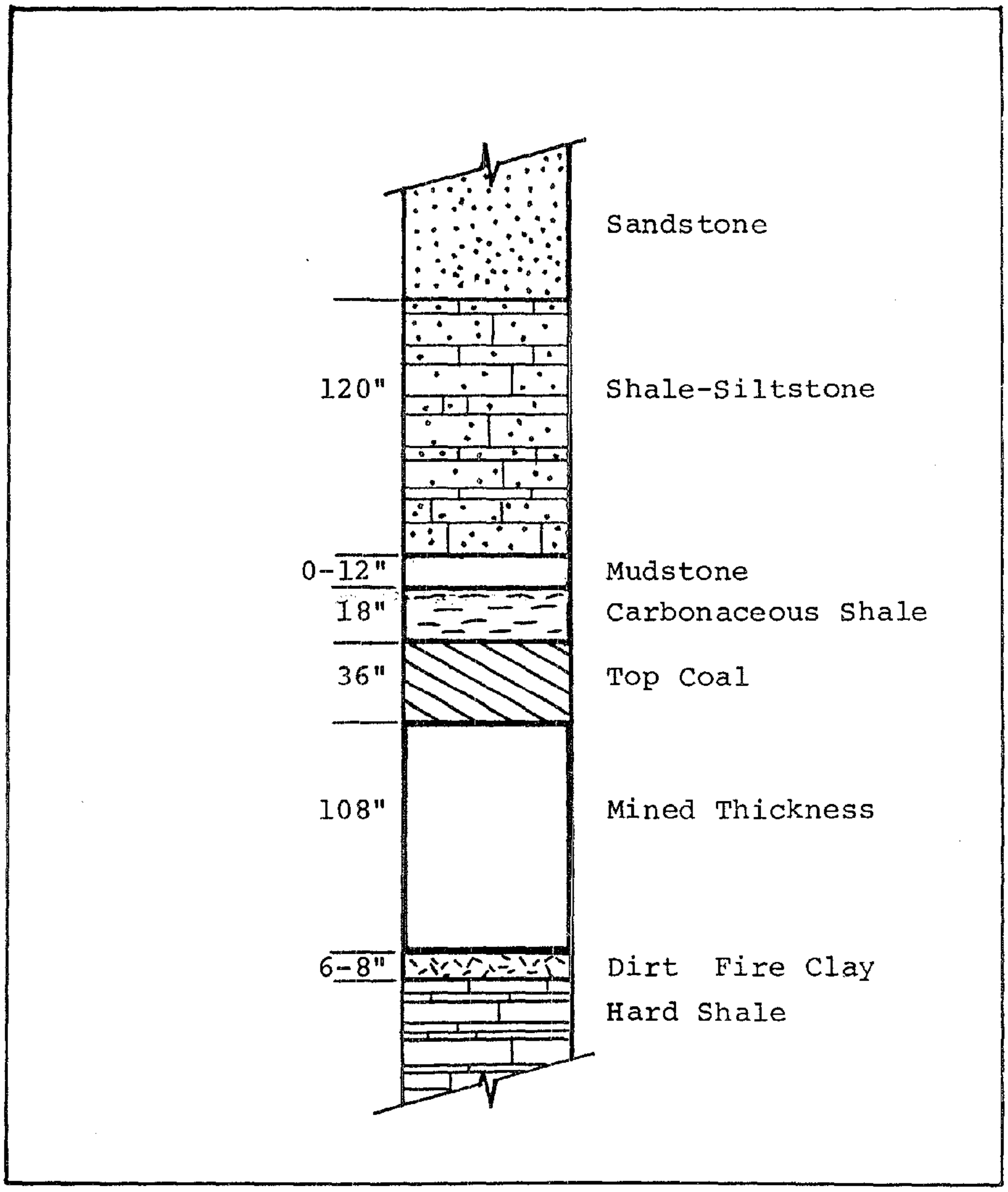


Figure 3-1. Average Cross Section of Top at SUFCo

areas shown on the adjoining map (Figure 3-2). Longwall mining could be utilized easily; however, it will be difficult to increase the productivity of the mine because conditions are favorable for the room-and-pillar mining method.

Longwall mining a thickness up to 4.9 m (16 ft.) under the natural roof should not present any problem. The solid coal will not slough and the roof will cave easily. A study of subsidence has shown that even sandstone beds in the upper roof cave easily. The caving will therefore occur with regularity. Very high productivity will be required from the longwall to compete economically with the continuous miners.

In the case of longwall mining, entries should still be driven leaving 0.9 m (3 ft.) of top coal. This top coal would be mined only in the face. Mining the top coal in the entries would require roof bolting, and the development productivity would be cut in half to achieve this additional recovery.

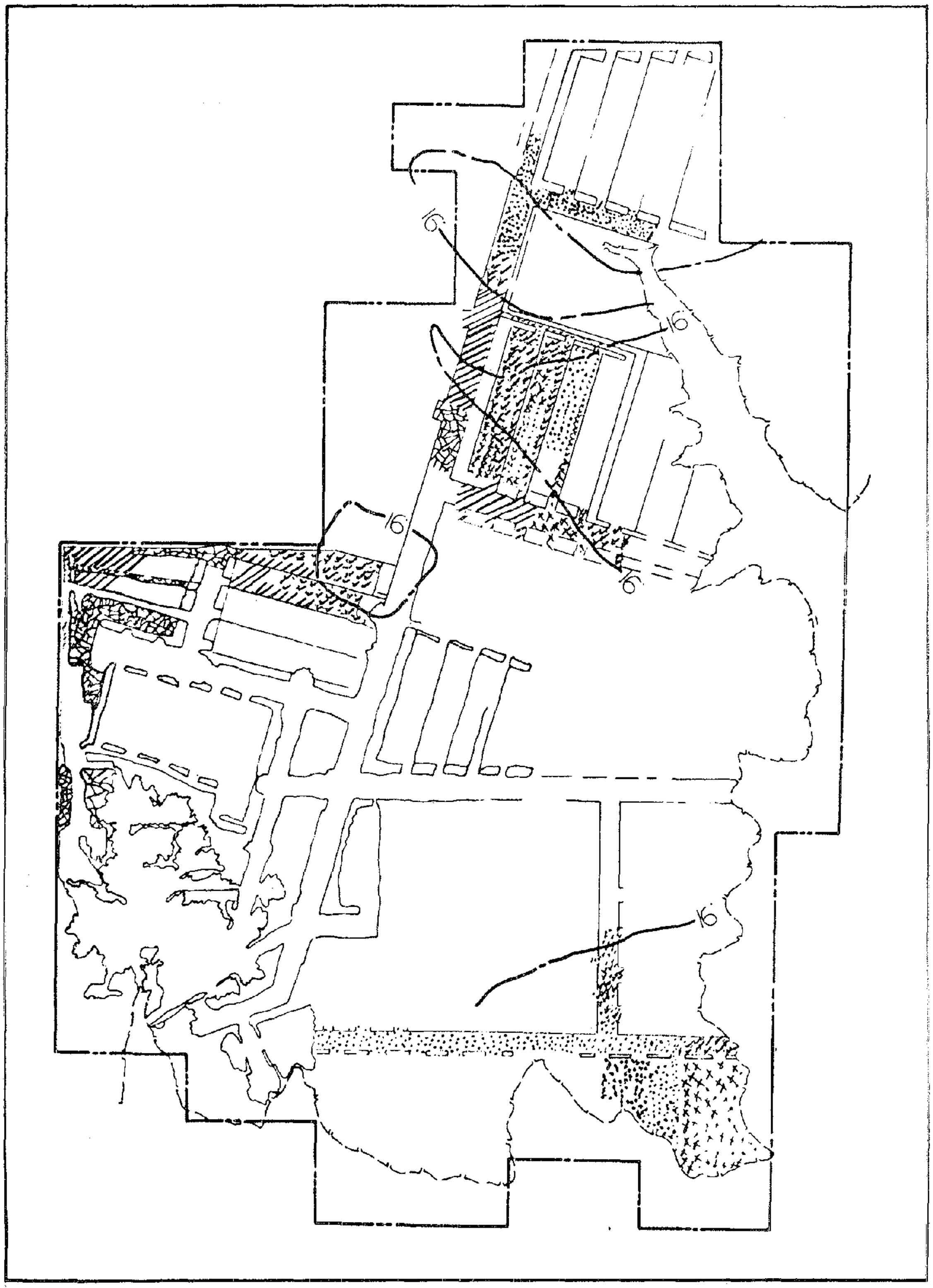
In summary, the strength of the coal and the ability of the roof to cave easily should be helpful in mining the full seam. While technical difficulties are not expected, economics are not as favorable. An investment in longwall face equipment cannot be justified by problems experienced with the room-and-pillar method.

Finally, a trial at SUFCo would assist in the planning and design of the Skyline Mine and the face equipment could later be transferred to the Skyline Mine, thereby gaining additional experience with thick seam longwall mining.

3.2 Skyline Mine

The Skyline Project is a joint venture of Coastal States Energy Company and Getty Mineral Resources Company, with Coastal States Energy Company as operator. Total development cost is currently estimated at $125 million over an eleven-year period. The mining operation is planned on 6,400 acres of federal and county leases which encompass 270 million tons of total reserves in three major seams. The project is located in the Manti-LaSal National Forest on the Wasatch Plateau of central Utah, near the town of Scofield (Figure 3-3).

Three mines, one in each seam, will be developed on the property (Figures 3-4, 3-5, 3-6). The mines will utilize common surface facilities which will include crushing and conveyor systems, storage silos, a unit train loading facility, offices, maintenance facilities and an analytical laboratory. The total combined production from the three mines at full capacity is expected to be about 5 million tons per year, with a work force of 900 employees.



1980
1981
1982
1983
1984

Figure 3-2. Mine Sequence Map for SUFCo

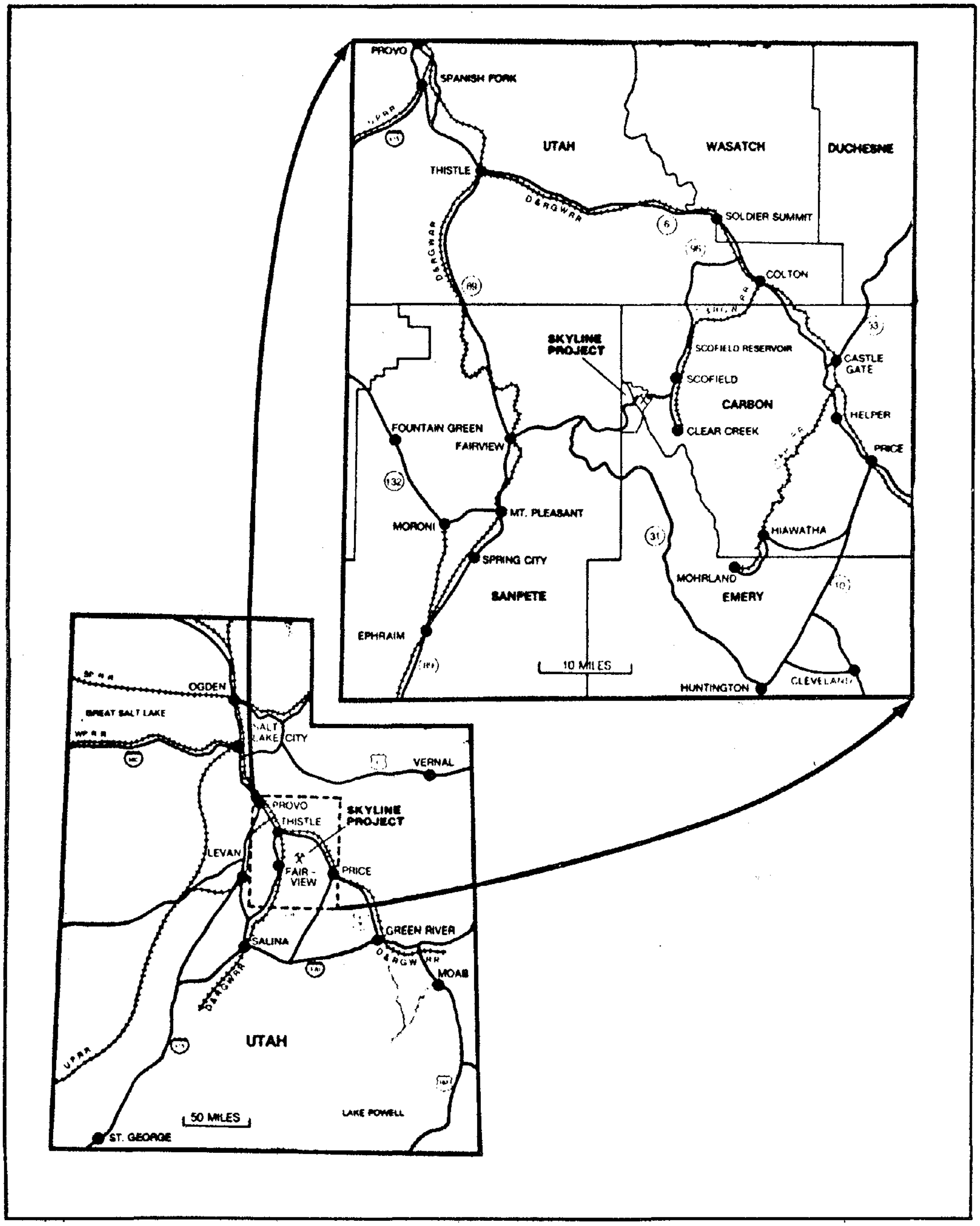


Figure 3-3. Location Map of Skyline Project

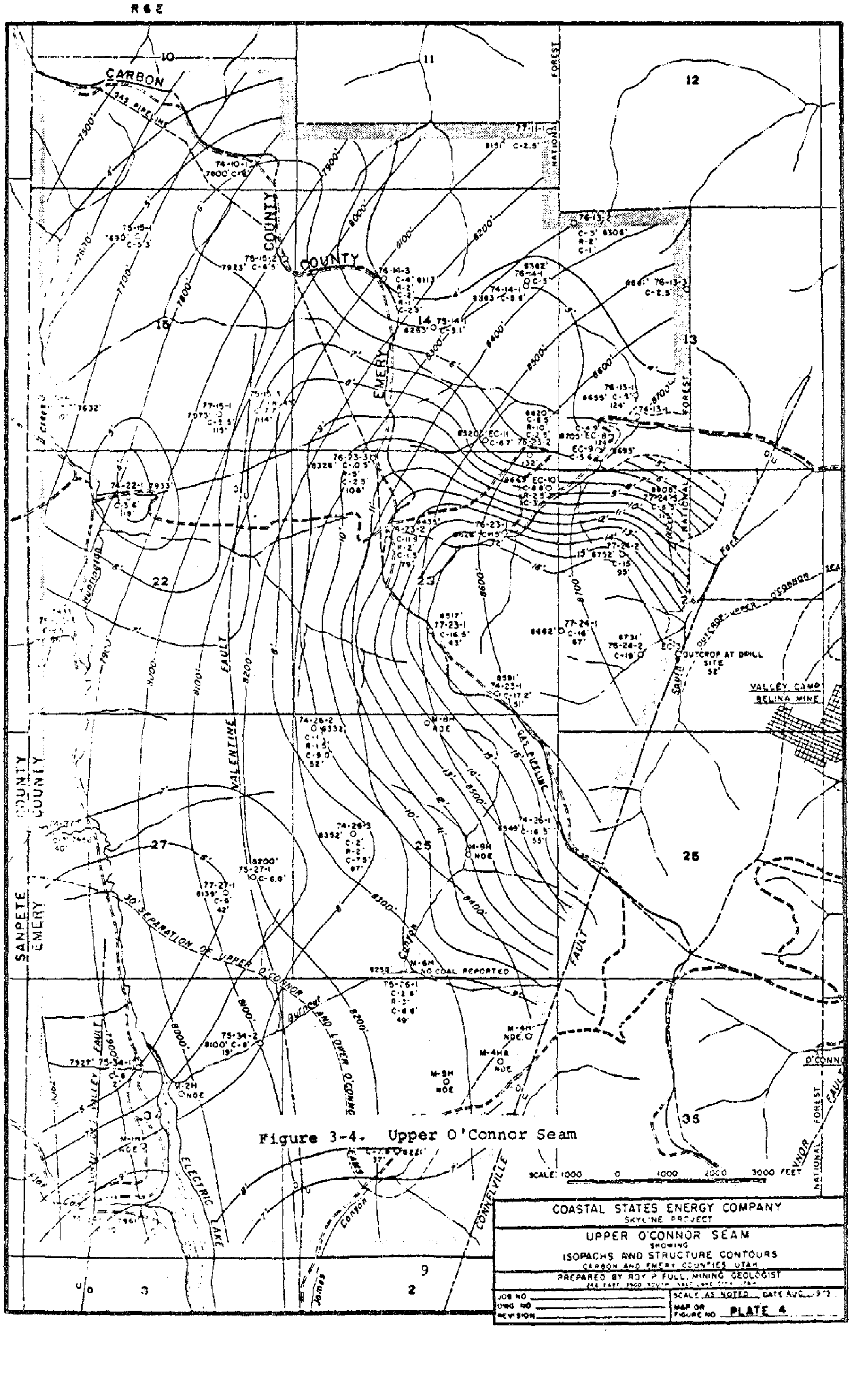


Figure 3-4. Upper O'Connor Seam

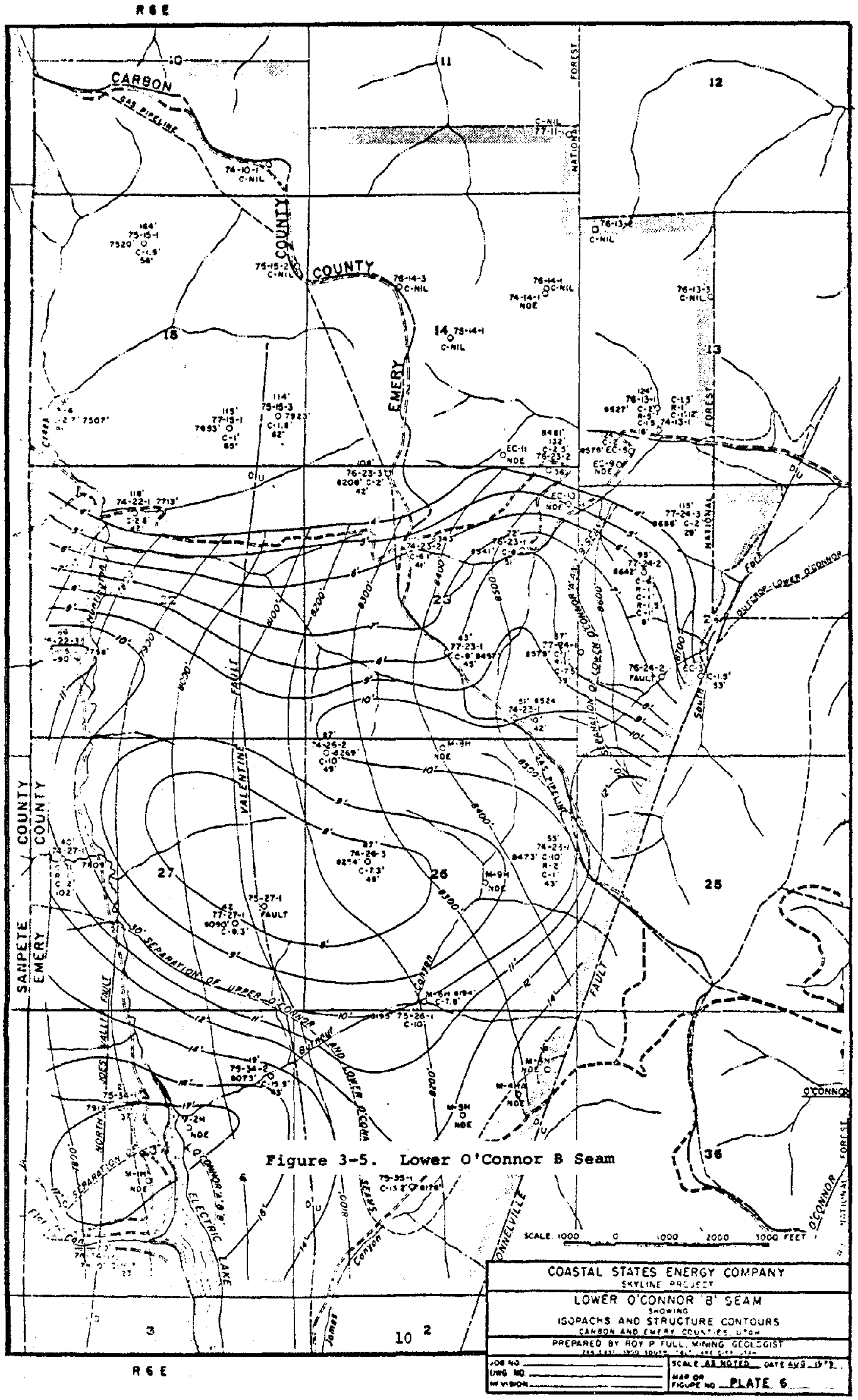


Figure 3-5. Lower O'Connor B Seam

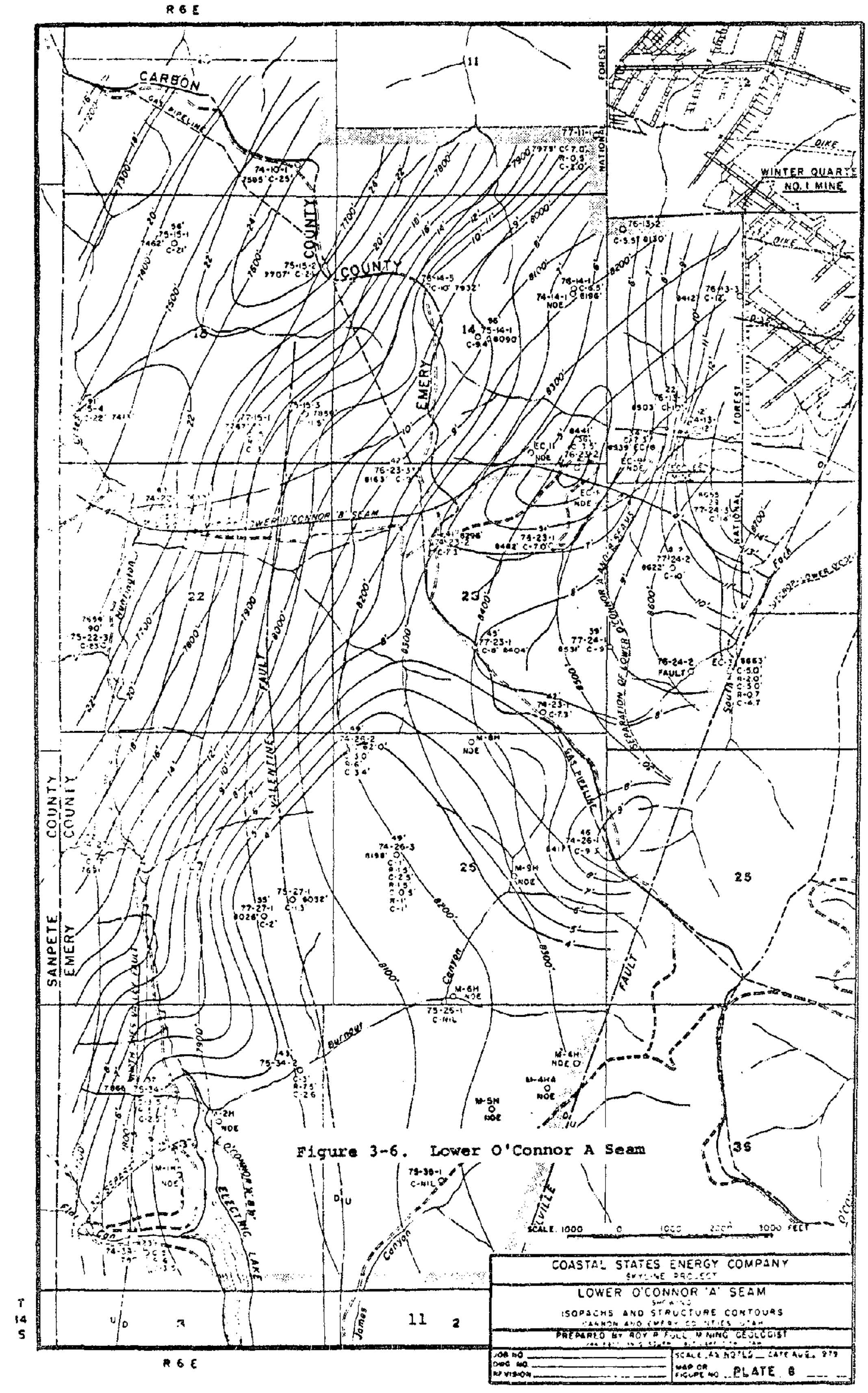


Figure 3-6. Lower O'Connor A Seam

The recoverable reserves are estimated to be 125 million tons which establishes a project operating life of over 30 years (Table 3-1). Marketing of Skyline coal is conducted jointly by the project participants. A high grade, low-sulfur coal will be produced for electric utilities, industrial plants, cement plants and other consumers.

The thickness of the three minable seams varies from 1.2 to 7.3 m (4 to 24 ft.) as shown on the adjoining map. The reserves were estimated by limiting the mining height to 3.7 m (12 ft.). The adjoining table shows that limiting the mining thickness to 3.7 m (12 ft.) will result in a loss of 14% of the reserves.

The Skyline Mine project was initiated in June 1980. Initial production from the first of three mines is expected in 1982. Maximum production will be reached in 1989 from the three mines. While it is difficult at this time to know if a panel will be available at a time suitable for the DOE single pass mining project, Skyline should benefit from a trial at SUFCo.

3.3 Colorado Westmoreland Inc., Orchard Valley Mine

In January 1976, Westmoreland Coal Company purchased the privately held stock of Colorado Consolidated Coal Company, thereby acquiring several tracts of coal-bearing land in Delta County, Colorado. Westmoreland then formed a subsidiary, Colorado Westmoreland Inc., which is responsible for the development of these tracts.

The coal is in the Paonia Coal Field of the Mesa Verde formation, outcropping at 2210 m (7,250 ft.) above sea level on the western slope of the Rocky Mountains. The Orchard Valley Mine began operations in December 1976. The three coal seams to be mined are:

- D seam, the first to be mined, 7.9 m (26 ft.) thick
- C seam, 4 m (13 ft.)
- B seam, 4 m (13 ft.) thick (see Figure 3-7).

Because of the thickness of these seams, large, diesel-powered equipment can be used effectively, as well as electrically-powered continuous and conventional mining equipment.

Mining is currently conducted with the room-and-pillar method using continuous mining techniques. The equipment used in this mine includes diesel-powered, 10-ton shuttle cars, 6-ton front end loaders, face drills and double-boom roof bolting machines and electrically powered continuous miners.

TABLE 3-1

Estimated Coal Reserves on the Skyline Property
(in Thousands of Tons)

Seam	Seam Thickness Feet	Acres	Total Reserve Tons	+12 Foot Thickness	-30 Foot Separation	Electric Lake Exclusion	Minable Tons
Upper O'Connor	4-18	5,646.00	85,908	5,674	13,072		67,162
Lower O'Connor "B"	4-17	3,943.98	74,418	5,966	2,558	2,380	63,514
Lower O'Connor "A"	4-24	4,846.89	107,369	24,375	805		82,189
			267,695	36,015	16,435	2,380	212,865

Minable Tons Between 4- and 5-Foot Thickness

Seam	Acres	Tons
Upper O'Connor	398.99	3,231
Lower O'Connor "B"	108.13	876
Lower O'Connor "A"	224.06	1,815
		5,922

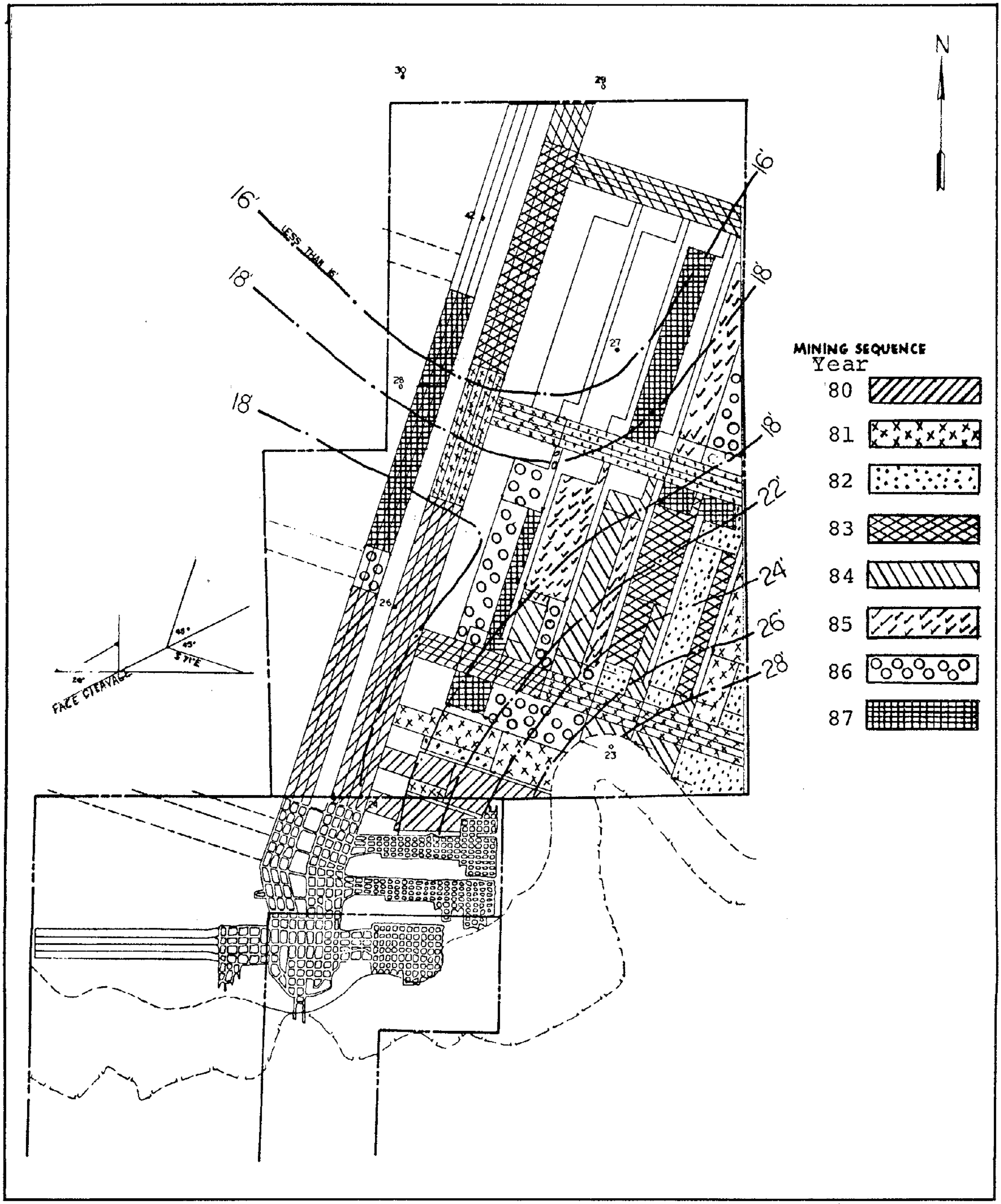


Figure 3-7. Mine Sequence Map for Colorado Westmoreland Inc.

The mine's main ventilation system is comprised of two 9 m (30 ft.) diameter air shafts, 73 m (240 ft.) deep with 800 hp electric motors driving large fans capable of moving 180,000 cfm of air.

Storage silos and train loadout facilities are located between Colorado Highway 133 and the Gunnison River. This facility was designed to receive coal from the crushing operation by both truck and conveyor. The three 7,000 ton silos (21,000 ton capacity) can load the equivalent of two fully loaded unit trains.

The coal at Orchard Valley Mine is hard but is fissured by cleats. The face cleavage of the "D" seam is very pronounced and easily identified. The cleavage coupled with high ribs has created sloughing at the rib, a problem that is difficult to control. Many rib control alternatives have been attempted in the past; e.g., pillar banding, shotcrete, Fibrecrete, pillar bolting, etc., but the most effective alternative appears to be the reorientation of the mine workings to intersect the plane of the face cleavage at 45 degrees and to lower the advance mining height to 3 m (10 ft.). The 45 degree orientation presently practiced should be the same for the longwall. See Figure 3-7.

The roof is easy to cave and no special difficulties should be expected from periodic pressures. As in the other two mines, the use of diesel vehicles will make easier the introduction of the face equipment and the face moves.

Longwall mining should be beneficial to this mine where roof conditions are not too favorable. The increase of the mined thickness from 3 to 4.9 m (10 to 16 ft.) will be an important advantage, in light of the total amount of resources being limited.

3.4 Conclusion

The Orchard Valley Mine of Colorado Westmoreland Inc. seems closer to the average U. S. mining conditions, than the SUFCo Mine. However, for the purpose of "mine design" and "equipment specifications," the Orchard Valley and SUFCo's Mine I will be considered as possible selections.

4.0 MINE DESIGN FOR LONGWALL MINING A THICK SEAM

4.1 European Practices

Mine design in Europe is based almost exclusively on longwall mining and single entry development. The selection of these methods is the consequence of a greater depth and of difficult geological conditions.

In both advancing and retreating longwall mining methods, the headgate and maingate are single roadways, with the majority being steel supported and arch shaped. Roof bolting is used only in favorable conditions and sometimes in conjunction with a support. The belt conveyor entry is used for intake air. Material transport is achieved primarily through the tailgate entry which is on return air.

During retreat mining in Europe, roadways are driven with air ducts and auxiliary fans at the necessary length with the power of the fan and the size of the duct line being the only limitation. When ventilation is interrupted, the fan can be started without endangering risks since the fan motor is on fresh air ventilation.

Roof support must closely follow the face advance, with a distance of 1.8 to 2.4 m (6 to 8 ft.) being the usual limit imposed by roof quality. The safety of this mining system is mainly due to the quality of roadway support.

The organization of all operations and the placement of all equipment in single roadways are difficult; however, blasting does not require a previous cut by a shearer. As a result, only a drilling machine and a loading machine are required for roadway drivage.

It is evident that the foreign method of mine development, or even panel development, will not easily comply with U. S. regulations.

A main concern in European deep mines is the planning of gate road convergence which can be reduced by selecting favorable conditions. The mining of several seams creates low and high pressure zones. It is important to select a favorable position for roadways in a low pressure zone so as to reduce the convergence. For advancing longwalls, the right choice of section and roadside packwall is very important to insure that the final entry cross section will be sufficient after convergence.

Roof bolting is used to a certain extent (30%) in Saar (West Germany) and Lorraine (France), and where geological conditions are favorable. Some rules and precautions limit the use of roof bolting to selected geological conditions. A computer is sometimes used for roof control design. For instance, in Lorraine

(France), roof expansions and/or convergence are measured and data stored in computer memory. When preset limits are exceeded or when there is a sudden change in the speed of the roof bed's expansion, the computer emits support reinforcement orders.

European roof bolting methods are quite different from those used in the U. S. Bolts are generally longer and resin bolts are widely used. Side bolts are inclined to obtain anchorage above the solid coal. Wire mesh is often used in addition to the bolting of roof and walls. Figure 4-1 shows a roof bolting pattern in a thick coal seam.

The difficulty in maintaining entries has an impact on panel design. The optimum longwall face length increases as the cost of entries increases. Face orientation is, if possible, chosen in accordance with the general direction of faults and cleats to keep them perpendicular to the face line.

Another interesting technique in thick seam mining is to turn the face at the end of a panel and proceed with a second panel at right angles to the first. This technique of "face slewing" provides the important advantage of reducing the number of face to face moves. However, it is not compatible with the U. S. multiple entry system.

Ventilation is also very different in European mines. Due to the single entry system and the fact that the bleeding system does not exist, ventilation procedures vary significantly from U. S. methods.

European ventilation practices could be used in U. S. mines because of existing geological conditions but will not comply with present mine regulations. It is doubtful that variances can be obtained in all of the areas. The extent of the differences and the need for variances can best be appreciated by using a European design and evaluating each aspect of the design against U. S. regulations.

Three methods can be selected: 1) retreating longwall, 2) advancing longwall, 3) mixed method. Whatever the method, the entry main gate is single and never divided.

The most productive method is the fully retreating longwall (Figure 4-2) where successive panels are separated by a barrier pillar, the size of which depends on overburden depth and coal strength. This method is sometimes used with partial extraction to prevent surface subsidence.

If the retreating method is to be used without barrier pillars, an entry can be developed along the caving when consolidated. The conditions are very good from the strata control point of view (destressed area), but it requires experienced miners and the use of steel supports (Figure 4-3).

Bolting pattern at
the Aumance Mine (France)

Bolt Density

Figures 4-1a and 4-1b show the standard bolting pattern for a 5 x 5 m (16.4 x 16.4 ft) section of roadway (sectional view and plan view).

- density of 1.1 (11 bolts per 10 m^2): staggered arrangement in 2 lines, the first containing 5 bolts, the second containing 6;
- maximum distance of bolts from the roadway wall = 50 cm (19.7 in);
- minimum inclination of bolts next to the roadway wall = 15°;
- the remaining bolts in the center (3 or 4 bolts per line respectively) are perpendicular to the roof strata;
- bolt length 2.20 m (7.2 ft) fully resin bonded.

Source, "Rock Bolting"
Revue de l'Industrie Minerale

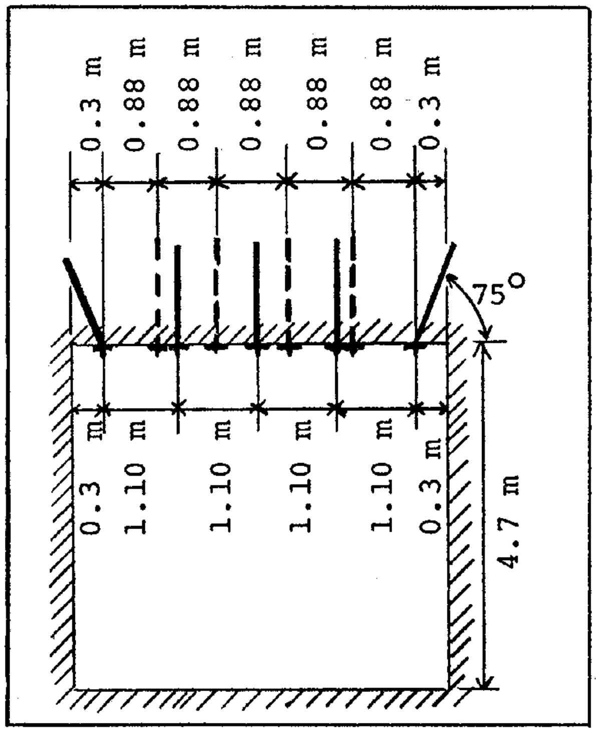


Figure 4-1a. Standard Bolting Pattern

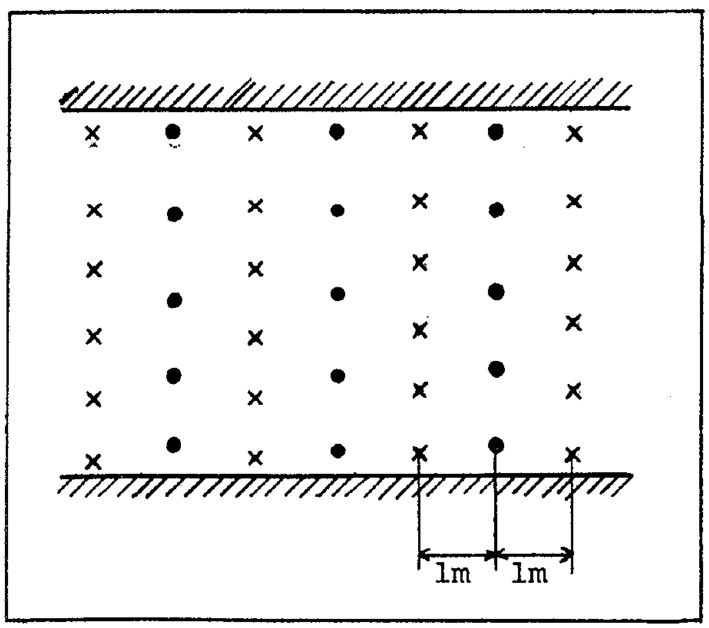


Figure 4-1b. Bolting Density

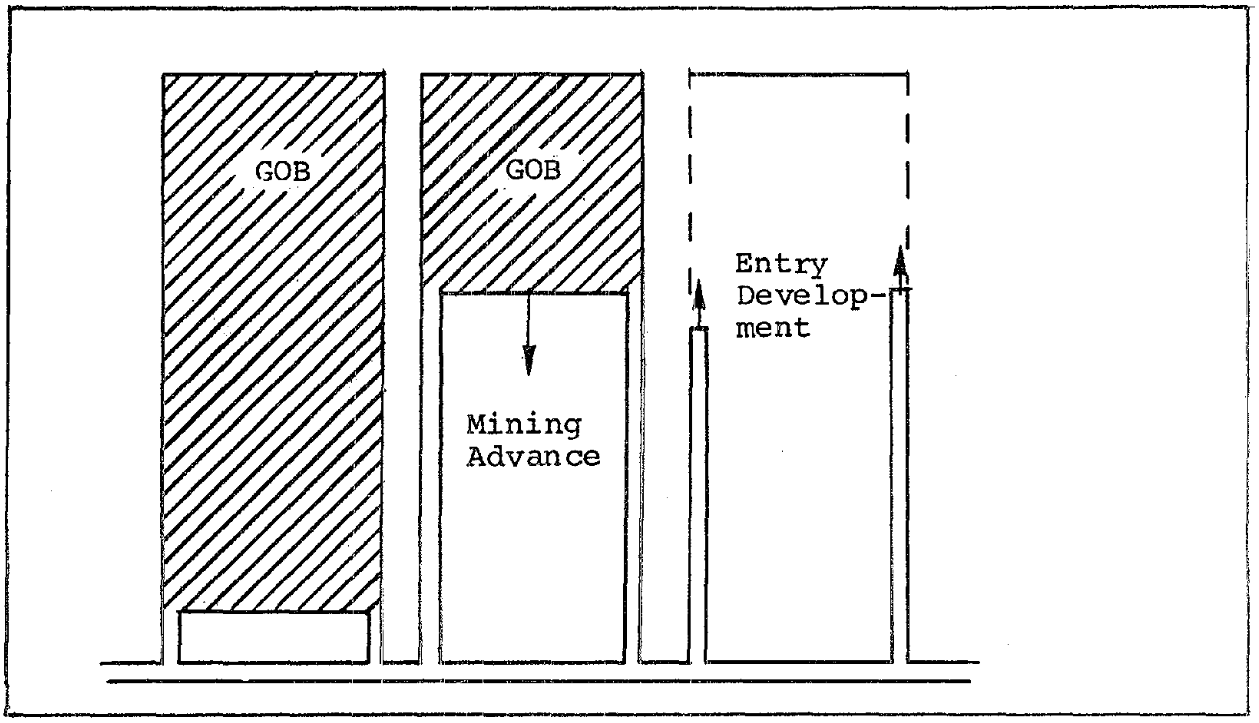


Figure 4-2. Single Entry Retreat Longwall with Barrier Pillars

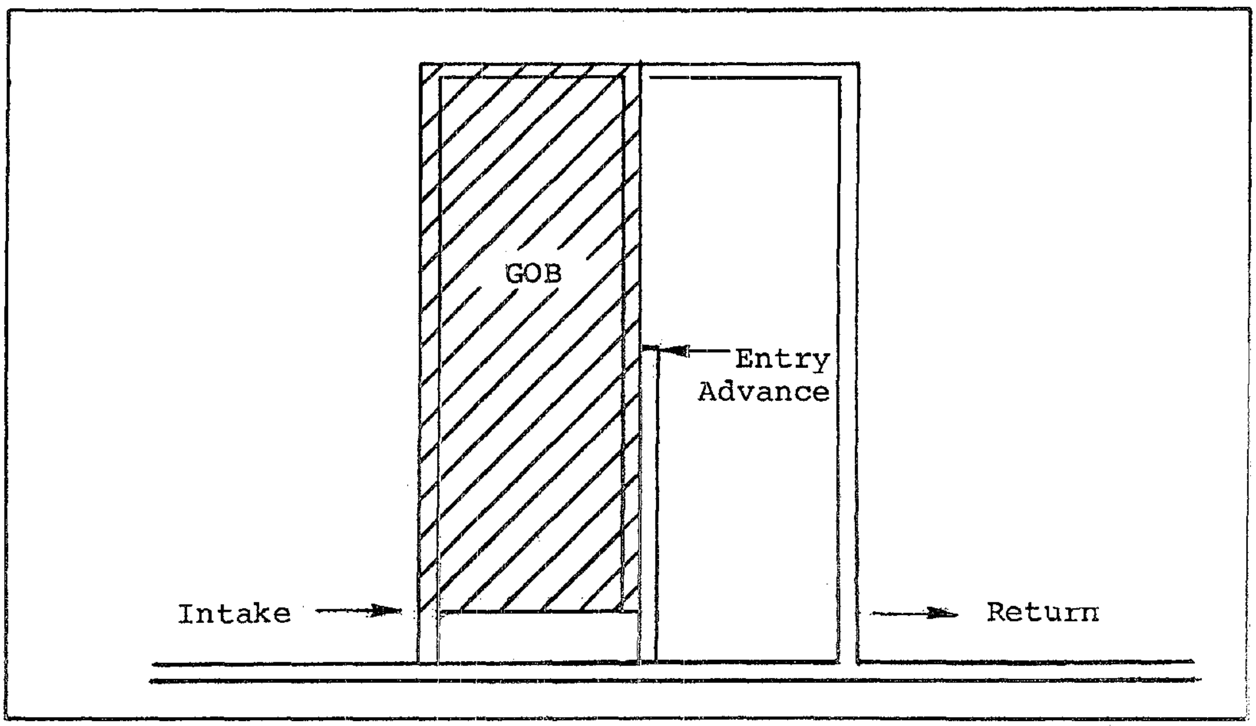


Figure 4-3. Single Entry Development Beside Gob Area After Consolidation

Advancing longwalls are used when the entries are closing ahead of the face. That normally occurs when the depth exceeds a limit of 600 to 900 m (2,000 to 3,000 ft.) (depending on coal and strata strength). With an advancing longwall, the convergence of entries is reduced when they are driven behind the face and when a good packwall is used on the gob side. Driving a gate road 6 m behind the face line instead of 50 m (164 ft.) ahead of the face will reduce the convergence by half. "Where strata conditions are good and the depth is not great, convergence amounts to a few centimeters so that a halving of this rate is not particularly important; however, where the surrounding rock is soft and the depth and corresponding convergence are great, the decision to drive gate roads with or behind the faceline and not in advance can be of decisive importance." (Grotowsky-Banff, 1977). This method has been successfully applied at the Westfalen Mine with steel arches and anhydrite packwall.

A third method which combines advancing and retreating techniques is in compliance with the U. S. regulations during the longwall mining period, but not at the development stage. Figure 4-4 illustrates this method. The first panel is developed by driving two single (non-divided) entries for the tailgate and headgate and making a connection as an installation room for the face. The face is retreating but the belt gate is maintained behind, protected on one side by the solid coal of the next panel and on the other side by a packwall or by a row of cribs as represented in Figure 4-5. The next face uses this belt gate as its tailgate by retreating between this tailgate and a belt gate developed beforehand in solid coal. The belt gate is maintained behind the face to be used as a tailgate for the following panel. The belt conveyor is installed and extended behind the face. The part of the belt gate in advance of the face line is convenient as a third access (a transportation route for men and material). There is only one packwall to be erected, but the problem is to match the face advance with the packwall advance, so that the face advance rate can be maintained. Figure 4-5 shows such a face.

4.2 Adaptability to U. S. Regulations

The mining methods described in the previous section will require several variances from the U. S. coal mining regulations found in CFR 30 Part 75. These regulation variances are necessary for the development phase which uses an undivided single entry with ventilation provided by the fan and the air duct. Among these variances are the following:

Escapeways	75 1704	two separate escapeways-one in intake
	75 1707	intake escapeway separate from belt (and trolley) entries

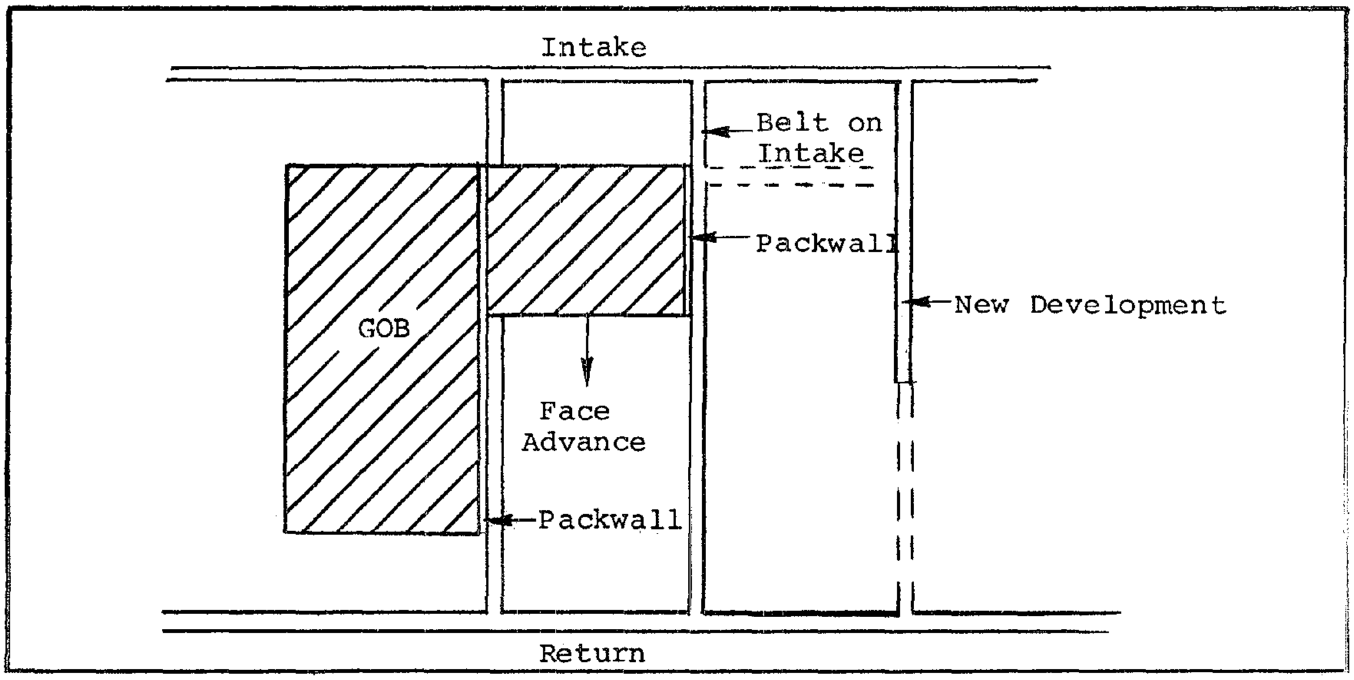


Figure 4-4. Single Entry Longwall Mining with Packwalls

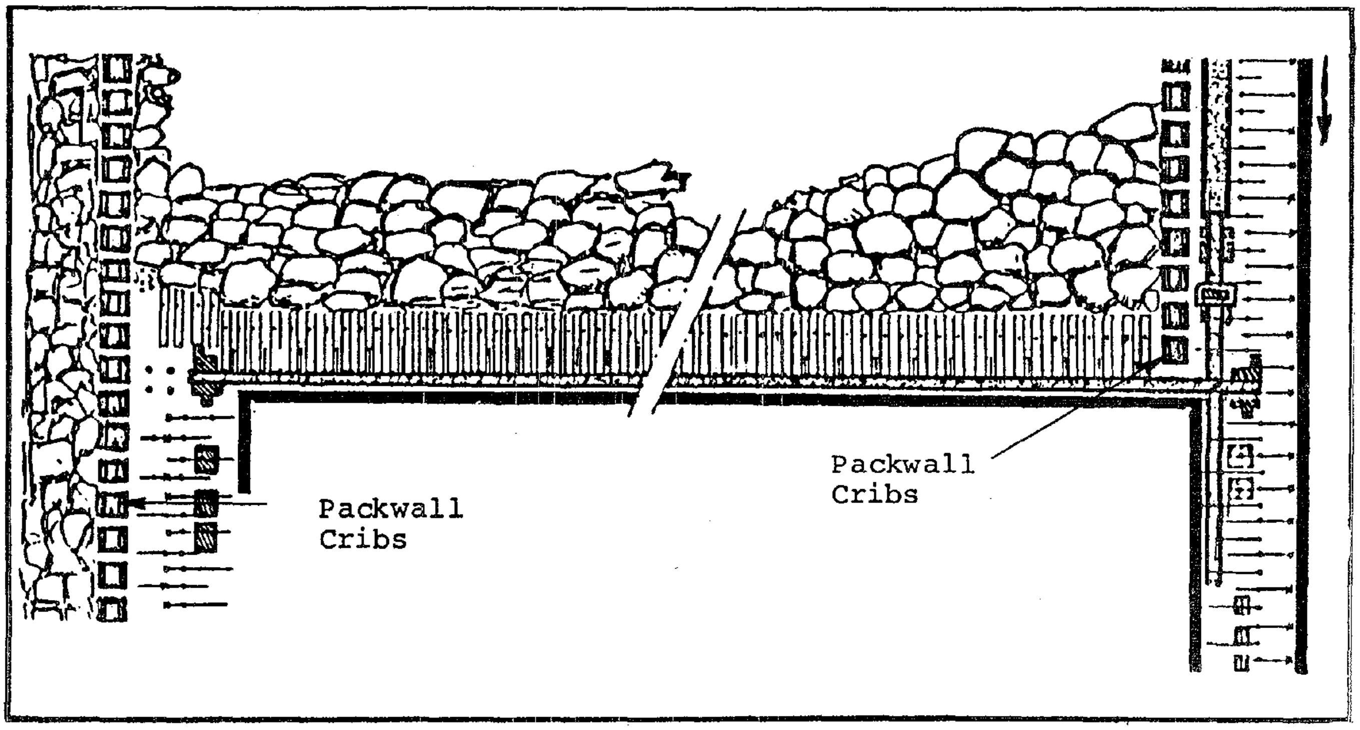


Figure 4-5. Longwall Face Layout Utilizing Packwalls

Ventilation	75 326	intake and return separated from belt entry
Auxiliary Ventilation	75 302.4	restoration of ventilation after stoppage

The safety in a single entry can be higher than in multiple entries for two reasons. One is that roof falls cannot be tolerated when using a single entry. Secondly, in driving only one entry for development of a longwall panel, it is feasible to incur expenses, whatever the cost, for good strata control which is essential for the longwall mining phase.

When planning a U. S. mine with necessary variances for federal regulations, it is important to keep in mind that variances may not be granted, and that an alternative solution requiring no variances be in hand. Therefore, a dual mine design of longwall panels for U. S. operations will be made and the design will be with two or three entries. The single entry European style is premature, and not necessary at the present stage of the project.

4.3 Conceptual Design for SUFCo Mine

SUFCo is presently mining the Upper Hiawatha seam by room and pillars using a continuous miner and three tele-trams per section. Roof bolting is not used because 0.9 m (3 ft.) of top coal is left to support the roof. Driving 2.7 m (9 ft.) high, 6.1 m (20 ft.) wide, a section can produce up to 850 tons/shift. The sections work two shifts, with a third idle shift applied to preventive maintenance of the equipment. Figure 4-6 shows the mining sequence for extracting the pillars.

The size of the panels is 152.4 x 914.4 m (500 x 3000 ft.) and from the strata control point of view, there will be no major difference between longwall mining and the present method. The pillars left between panels are 10 x 30 m (30 x 100 ft.) and are assumed to be yielding pillars. The selection of the size is intended to avoid periodic loading occurring at 150 m (500 ft.) intervals when using larger pillars.

The proposed method will keep the same pattern of development and to retreat 152 m (500 ft.) longwalls (Figure 4-7). It will be necessary to abandon a chain of 9 x 30 m (30 x 100 ft.) pillars between panels. Extracting these pillars with the longwall cannot be recommended due to the danger of roof fall when mining through a crosscut.

On the headgate side, the situation is different because there is not a high pressure abutment. If the development uses a three entry system, the first chain of pillars should be mined with the longwall (Figure 4-8). The double entry system will not require mining a chain of pillars and will be more appropriate for that reason (Figure 4-9).

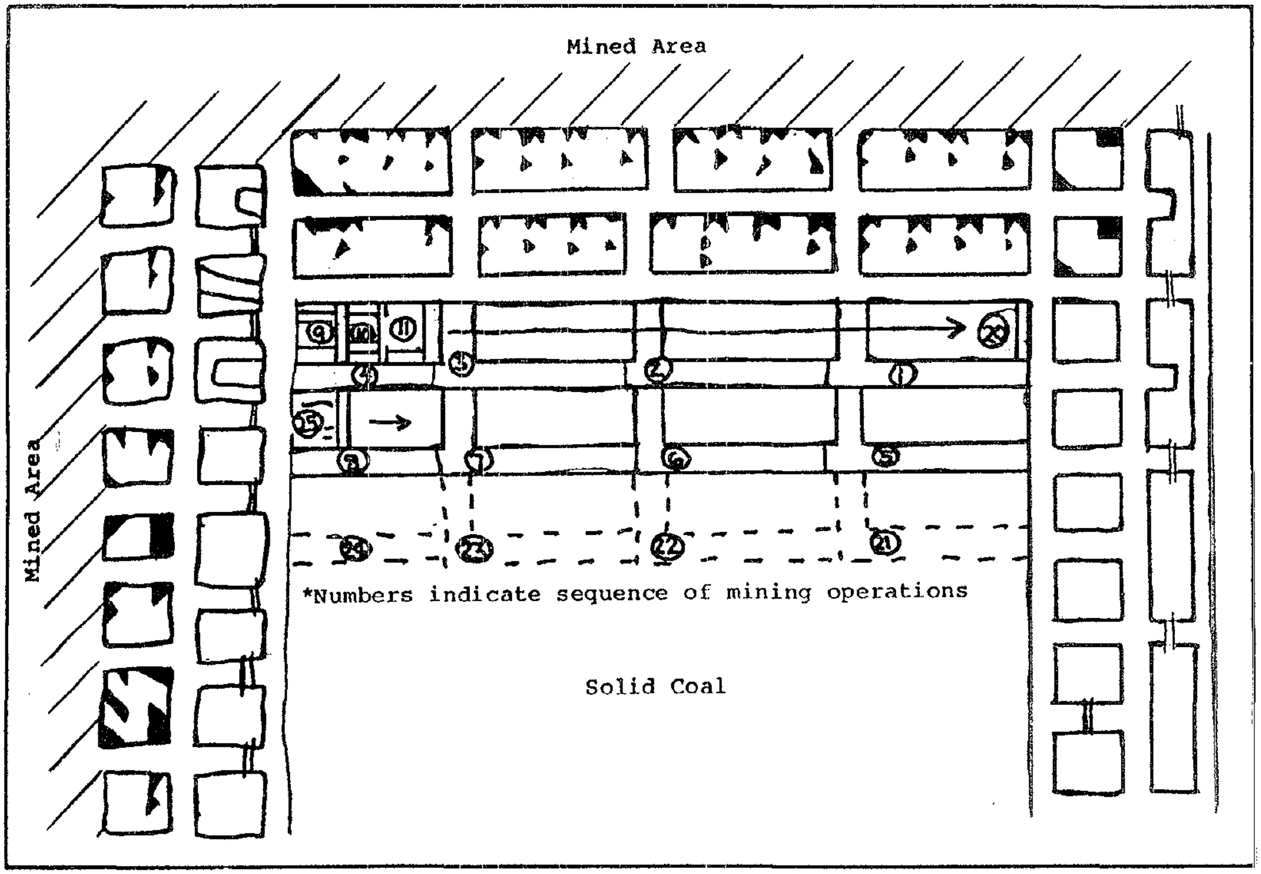


Figure 4-6. Mining Sequence for Pillar Recovery at SUFCo Mine

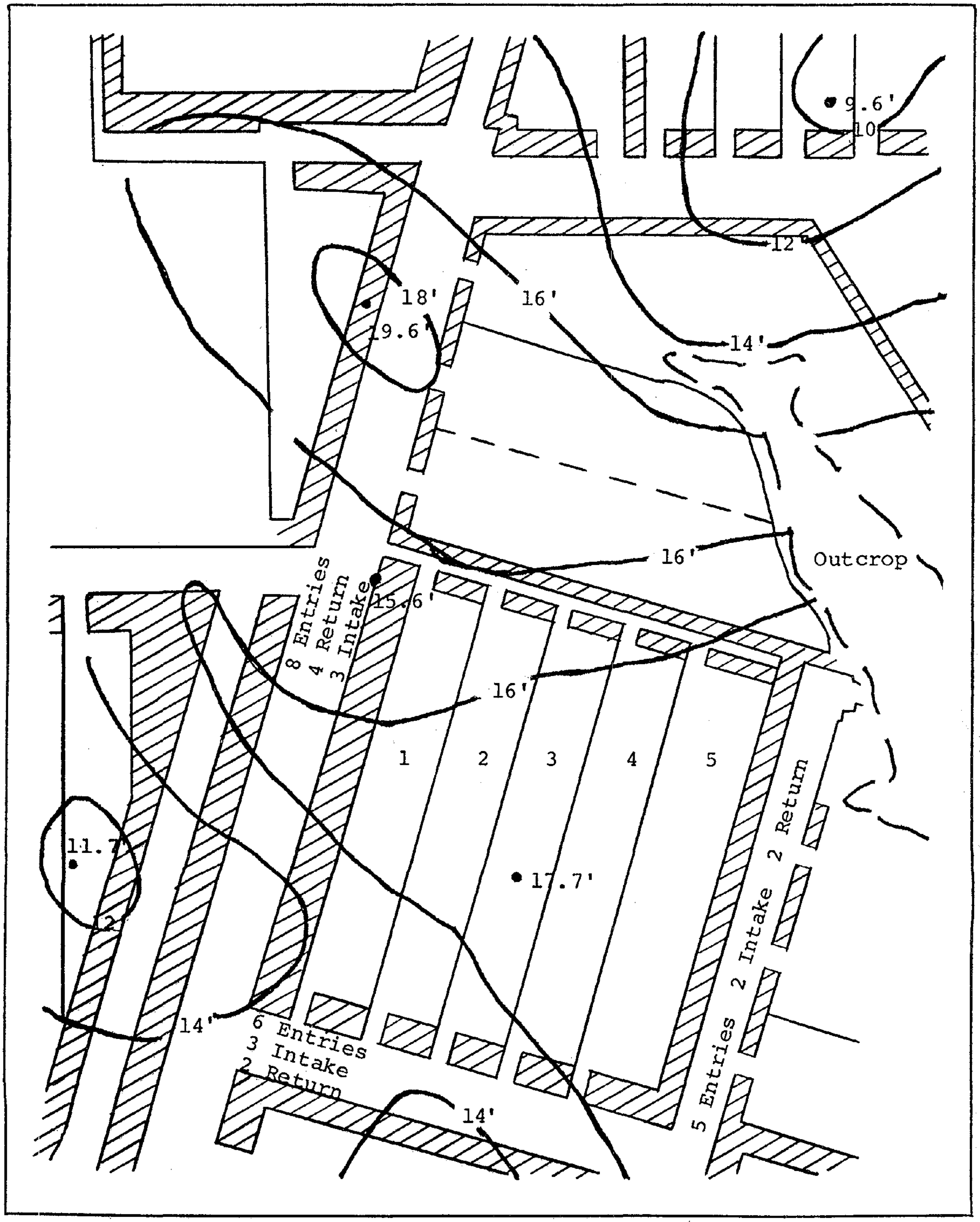


Figure 4-7. Five Panel Longwall Layout Following SUFCo's Future Development with Isothickness Lines

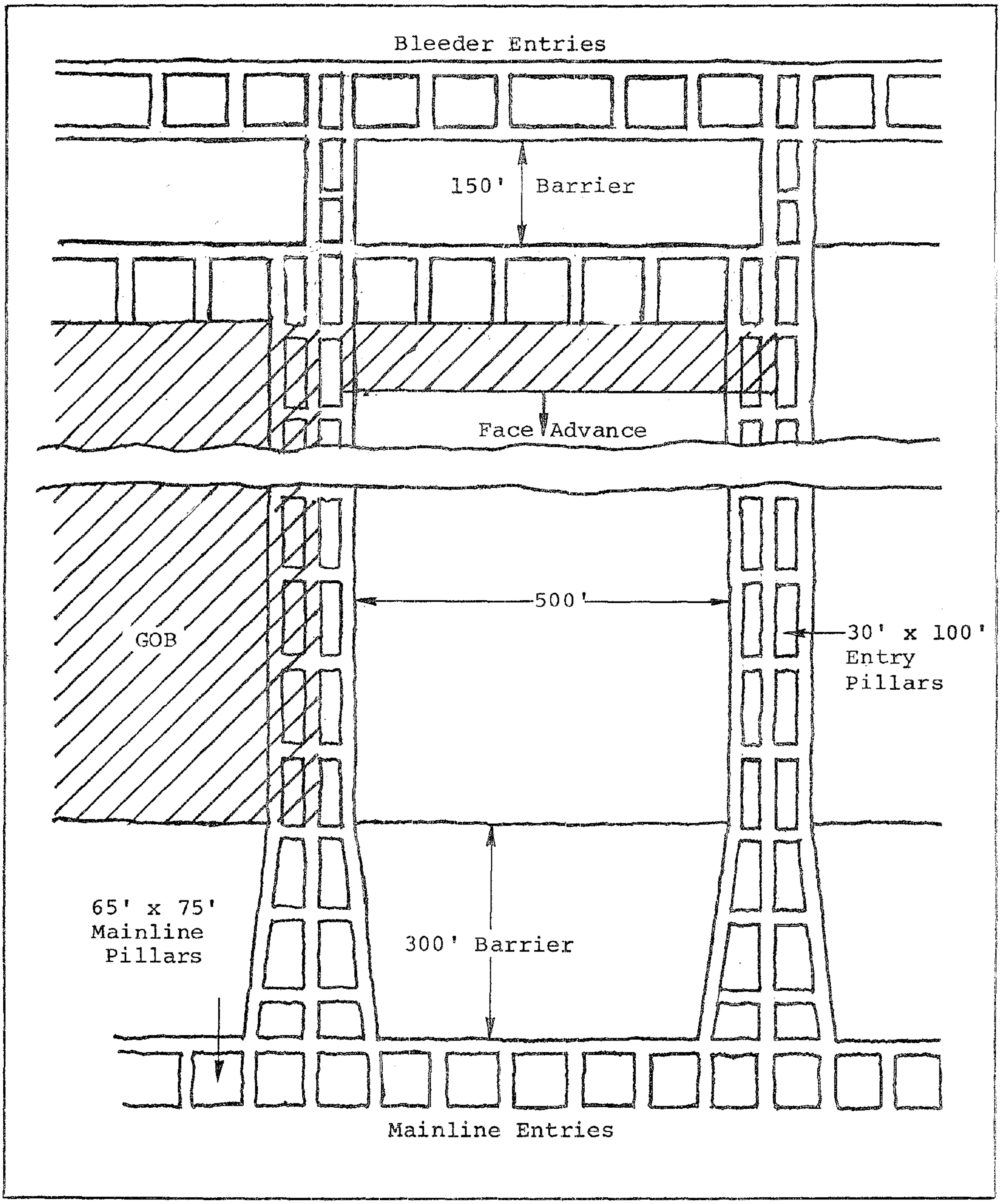


Figure 4-8. Longwall 3-Entry Development Plan for SUFCo Mine

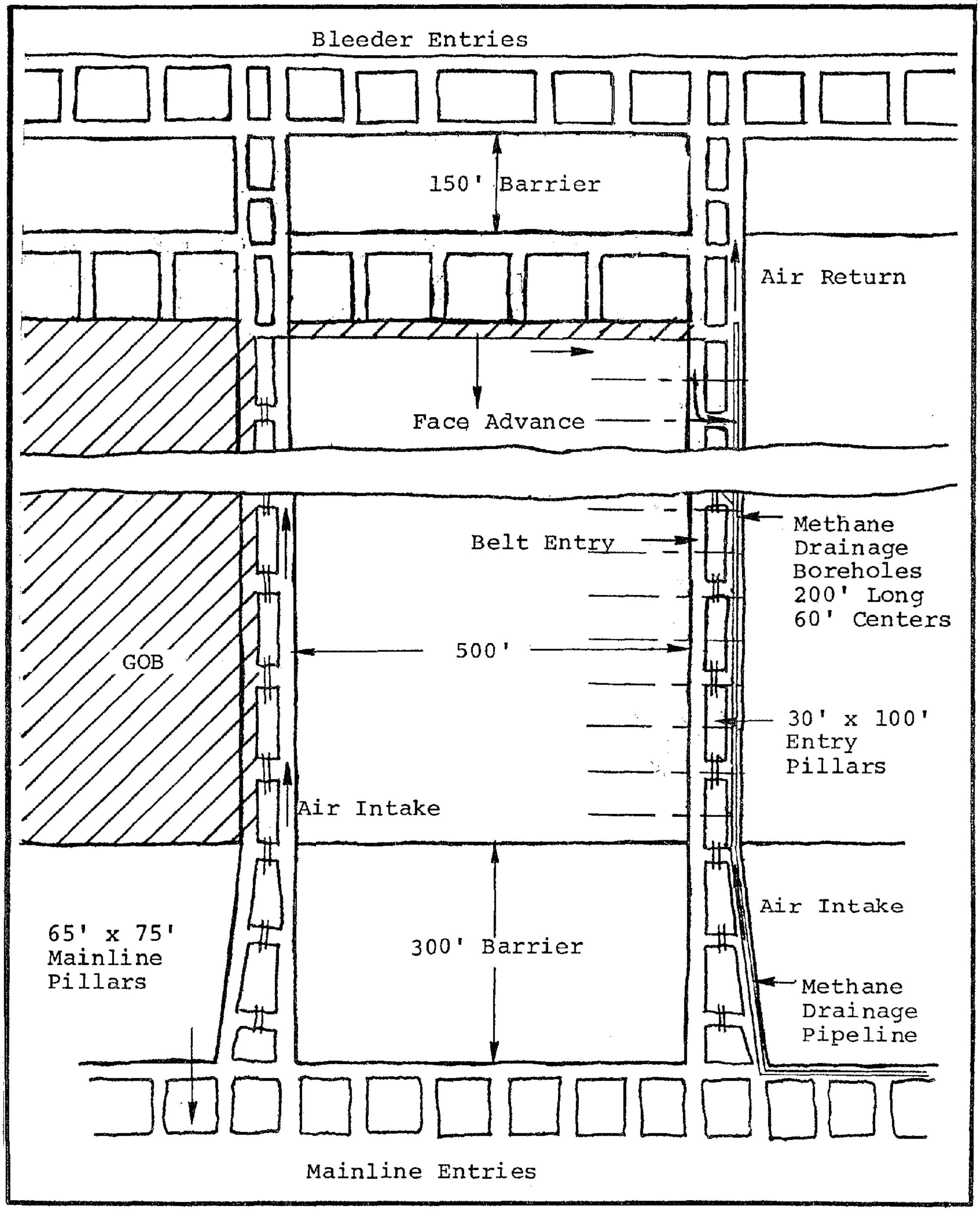


Figure 4-9. Longwall 2-Entry Development Plan for SUFCo Mine

It should be noted that the safest way to mine a chain of pillars is to fill the crosscut with a pump pack installation which will fill the crosscut with coal and cement, thereby replacing the solid coal. This method was used successfully at Coventry Colliery (Great Britain) to cross several roadways with a longwall face.

From the economic point of view, there is an optimum face length, as there is an optimum panel length, but it is advisable to use the present mine layout for a first trial. This layout could be altered following the current studies of the horizontal stress at SUFCo Mine because an excess of horizontal stress has been noted in the direction N 32° West, reaching a value equal to the vertical stress. However, as aforementioned, no additional problems should occur when extracting a panel by longwall instead of using the present method.

Higher productivity from the longwall in relation to the existing method is required to have an economic advantage. The equipment will be selected to meet this challenge.

4.4 Conceptual Design for Orchard Valley Mine

Orchard Valley Mine is in the development stage of operation. Entries are driven 6 m (20 ft.) wide, 3 m (10 ft.) high and roof bolting is on 1.5 m (5 ft.) centers with 1.8 m (6 ft.) long bolts. Five main entries are driven on 30 x 38 m (100 x 125 ft.) centers. Other development entries and panels are driven at 24 x 24 m (80 x 80 ft.) center.

The coal cleavage is very important. Its orientation is S62°W. The mining layout will be set at 45° on the cleavage. The seam dips at 4°.

Mining operations with room-and-pillar methods use the following equipment per section:

1 Continuous Miner -- The mine has 3 Joy 12 CM and 1 Lee Norse 546
1 Roof Bolter -- Twin boom Fletcher
1 LHD
3 Tele-trams -- Wagner 12 tons capacity
1 Feeder Breaker
1 Man Trip
2 Mine Jeeps
1 Utility Vehicle
1 Maintenance Vehicle
5 Face Fans

The R.O.M. productivity reaches 750 tons per shift with an eight man crew.

Figure 3-7 presents an extract of the mine sequence map. Panels for pillar extraction will be 168 m (550 ft.) wide x 610 m (2000 ft.). A 152.4 m (500 ft.) longwall can be substituted for mining the panels. However, 3 m (10 ft.) high entries are driven under the roof. It is preferred that for the longwall operation, the entries be driven with top coal, if this top coal is of adequate consistency. In the equipment specifications, enough flexibility is planned to switch from one solution to the other.

The longwall orientation must be at 45° on the general direction of cleavage, which will fit in the actual plan.

The mining sequence with longwall mining will be identical to the plan for SUFCo with two notable exceptions: 1) the pillar size will be 18.2 x 18.2 m (60 x 60 ft.) (Figure 4-10) instead of 9.1 x 30.5 m (30 x 100 ft.); and 2) a two entry system was determined to be the best way to develop this panel.

Conditions are not as good as at SUFCo and it is not advisable to mine a chain of pillars with the longwall face equipment. If, in the case of three entries, this is necessary, it is strongly recommended to pump pack the crosscut before mining through, even though the mining cost will be much higher.

Figure 4-11 shows the sequence of longwall mining in the case of a two-entry system.

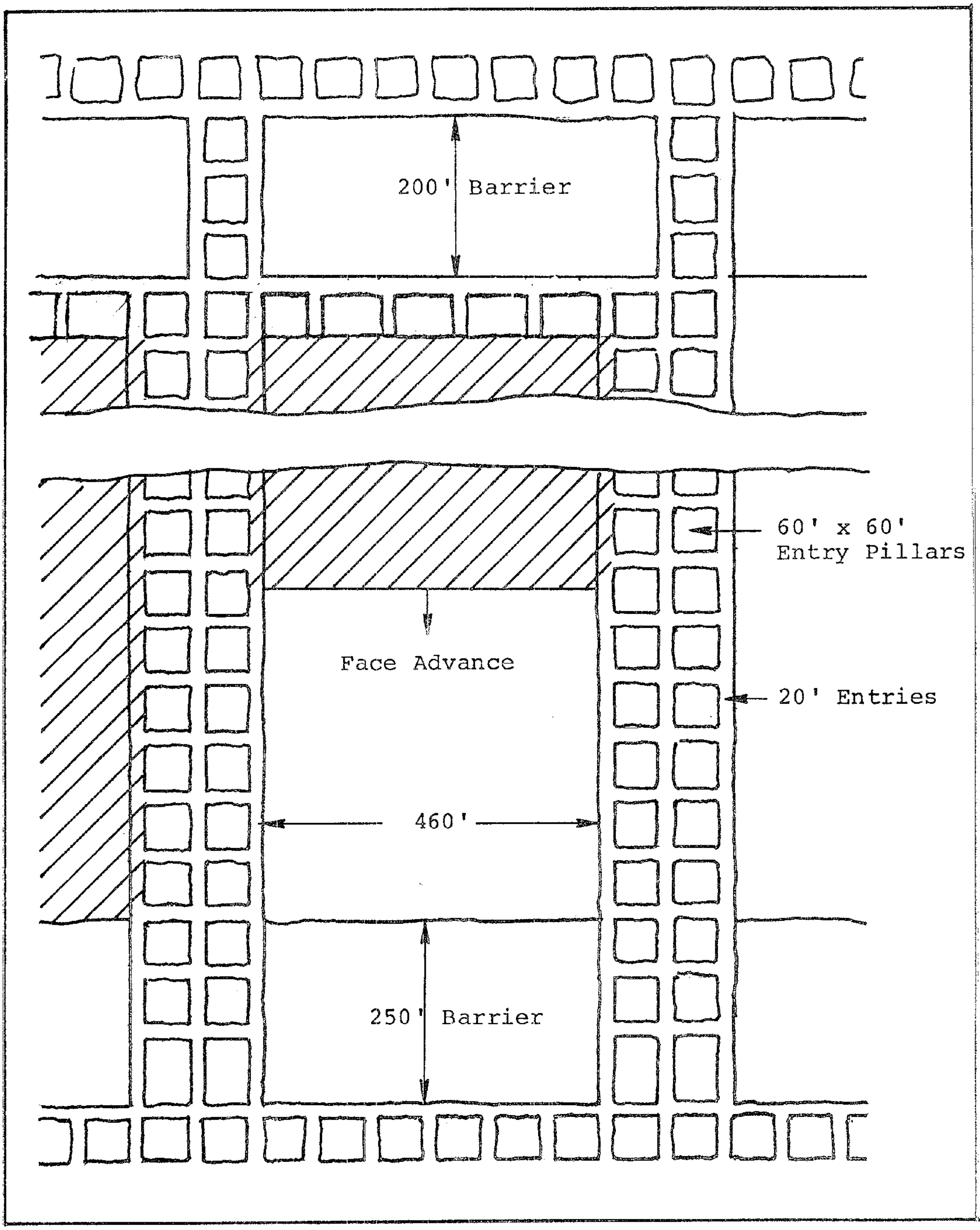


Figure 4-10. Longwall 3-Entry Development for Colorado Westmoreland Inc.

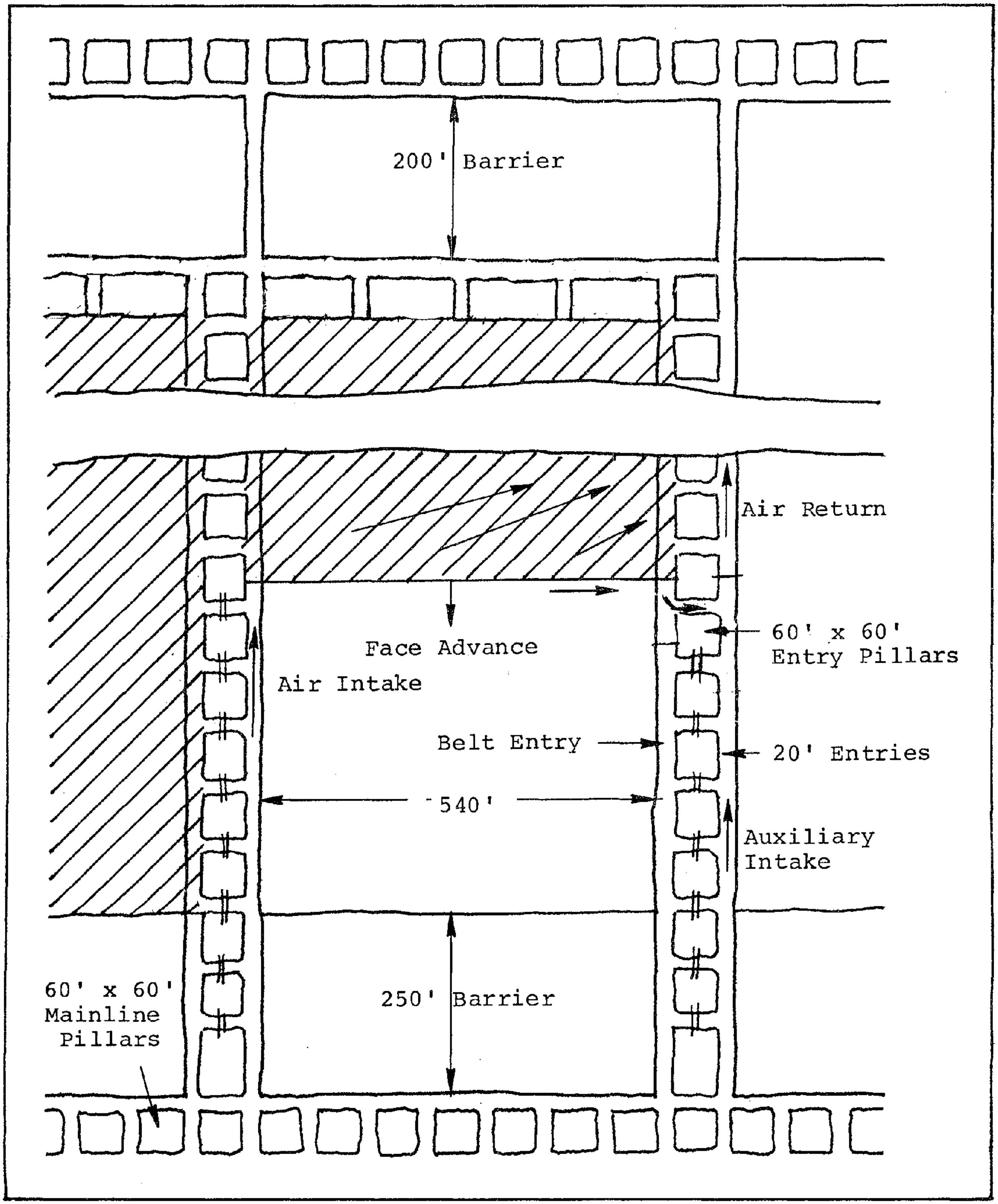


Figure 4-11. Longwall 2-Entry Development for Colorado Westmoreland Inc.

5.0 EQUIPMENT SPECIFICATIONS

5.1 Equipment Selection Criteria

Longwall mining is a high capital cost mining method. The return on this investment can only be obtained by very high productivity. It is not yet accepted to keep the equipment working seven days a week, as is normally done in surface mining with equipment of equivalent price. However, most of the longwalls work three productive shifts, five days a week. Daily production is the key factor in obtaining good operating costs.

The main limiting factors in present U. S. longwall faces are:

1) dust level

2) down time caused by large lumps of coal unable to pass under the shearer or to turn on the delivery point

Other problem areas are:

- o methane
- o roof control and sloughing of the face
- o gate - longwall junction

Each of these is discussed in the following subsections.

5.1.1 Dust Control

The maximum average daily concentration of respirable dust to which coal miners can be exposed is 2.0 mg/m^3 as established by Federal law. Mining companies have experienced great difficulty in complying with this standard for coal mine sections using longwall mining techniques. Over 50 percent of longwall sections are out of compliance with the dust standard at any given time, including about 2/3 of the longwall sections using double drum shearers. When a longwall face broke the production record with 15,000 tons in 24 hours, the dust level reached 10 times the mandatory concentration, even though the water supply was 40 gallons/min. on the shearer and about 20 gallons/min. for additional spraying.

The reduction of dust production can only be obtained by combining all available means.

- o Water injection in the coal: adverse effects of water injection are limited when the injection is conducted at low pressure for a long period of time from drill holes parallel to the face line. Water injection is not always possible but should be very efficient in most of the U. S. coal seams where the permeability is often high.

- Reduction of the coal breakage by reduction of the speed and number of picks and an increase in the pick penetration using the optimum pick lacing: the pick speed can be reduced by decreasing the number of revolutions per minute and also by limiting the drum diameter. The pick penetration will be increased by reducing the number of picks and increasing the haulage force.

- Water spray arrangements, together with a sufficient amount of water and water pressure: water sprays can cause air turbulence or form a water curtain. Spray efficiency is related to the water pressure; however, the amount of water is the main factor. Water spraying on the drums is efficient not only for dust suppression, but also for preventing methane ignition (see Section 5.1.3).

- Dust collectors built in the shearer loader: when mining a thick seam, enough room is available for such a device. Underground tests have already been made with some success.

- Surfactants could also be added to sprayed water.

- Unidirectional cutting to keep the miners upwind.

5.1.2 Reduction of Downtime

Large blocks of coal that fall on the tailgate side of the shearer often cannot pass beneath the shearer, resulting in a pile-up of coal and a halt in the operation until the blocks can be broken. Shearer-mounted lump breakers are not able to handle the problem of coal lumps if their number (or their size) is not limited by other means.

To reduce the risk of coal lump production above the shearer, one possibility is to cut the coal with the shearer advancing in the direction of coal haulage. In this case, the wave of pressure when cutting coal is moving ahead of the shearer on the right side. A better solution is to first cut the top coal and to advance the roof supports before mining the bottom coal. This method should be beneficial to face behavior. Before cutting bottom coal, face sprags should be set.

Large blocks of coal that fall on the headgate side of the shearer tend to pile up at the face conveyor-stage loader transfer point. This is easily solved by using a side delivery or roller curve conveyor.

5.1.3 Methane

- Methane Ignition. The risk of methane ignition is generally due to striking sandstone and/or pyrite with too high of a pick speed for this type of rock. Risk reduction is obtained by choosing a low pick speed and a good water spraying system. A good solution is the injection of water through the picks or behind each pick.

- Methane Drainage. Common methods of methane control in U. S. mines are by means of ventilation of gob areas, and gas drainage through surface boreholes and also through underground boreholes. Surface boreholes can be difficult to drill with the surface topography of selected mines. If methane drainage is required, the recommended method is degasification during longwall operations on the first panel.

  The double entry system allows the methane drainage holes to be drilled from the return entry along the solid coal (see Figure 4-9).

  Borehole diameter should be at least three inches, with the length varying from 200 to 300 feet at a 30 degree inclination to reach the fissured strata of the caved area. Holes will be drilled at 60 foot intervals. Drainage holes will be connected through individual valves to a methane pipeline under negative pressure generated by a pumping station located on the surface.

A handbook entitled "Firedamp Drainage" has recently been published by the Commission of European Communities and gives a detailed description of present methane drainage techniques.

5.1.4 Roof Control and Sloughing of the Face

Mining a thick seam in a single pass cannot be achieved with good results if the coal face is sloughing, which increases the unsupported roof area. Resin injection or dowelling methods are technically possible, but not economical. The foremost method of first cutting the top bench, and advancing the roof support when the bottom bench is still present is better than a full face cut method, but will only be successful if there is sufficient coal strength. Figure 5-1 illustrates the operation's phases.

Another problem is the control of the shearer cut inclination so as to limit the unsupported roof area. A 4.9 m (16 ft.) high shearer with a 0.9 m (3 ft.) wide base is not very stable and not easy to control. A steering underframe will be advantageous, but compatibility with chainless haulage will necessitate having the haulage unit in the underframe and not in the shearer itself (Figure 5-2).

5.1.5 Gate Entry

The retreat method of longwall mining, which is predominant in the U. S., is recommended. Presently, continuous miners are used to develop the longwall panels. Entries developed by continuous miners have their height limited to 3 to 3.7 m (10 to 12 ft.) at the face. A second cut in the floor can be taken. The ribs and, if necessary, the roof should be secured with mesh and bolt. The main problem with driving a full seam entry will not appear before the retreating phase. At this time, support of the entry must be reinforced; cribs are generally used in U. S. coal mines. For a high entry, individual props are preferred, when the floor is hard. At least during the first trial, it would be better to retain the present development method and miners who are accustomed to existing practices. The entry height should be limited to 3 m (10 ft.) (preferably 3.7 m (12 ft.)), even though it will necessitate reducing the minimum size of the face equipment.

Two cases will be considered. If coal is strong enough, a top coal of 1.8 m (6 ft.) will be left in the entries, which will be driven on the floor. If it appears difficult to hold this top coal, the entries would be driven under the roof, a method which will require a down grade in the face (Figure 5-3). The equipment specification makes a provision for this unfavorable case which requires more flexibility in the conveyor and in the support system.

Face Organization

With presently available longwall mining equipment, a 5 m (16 ft.) thick seam (or 5 m high coal) can be worked in two ways, (1) full face cut, or (2) two bench face cuts.

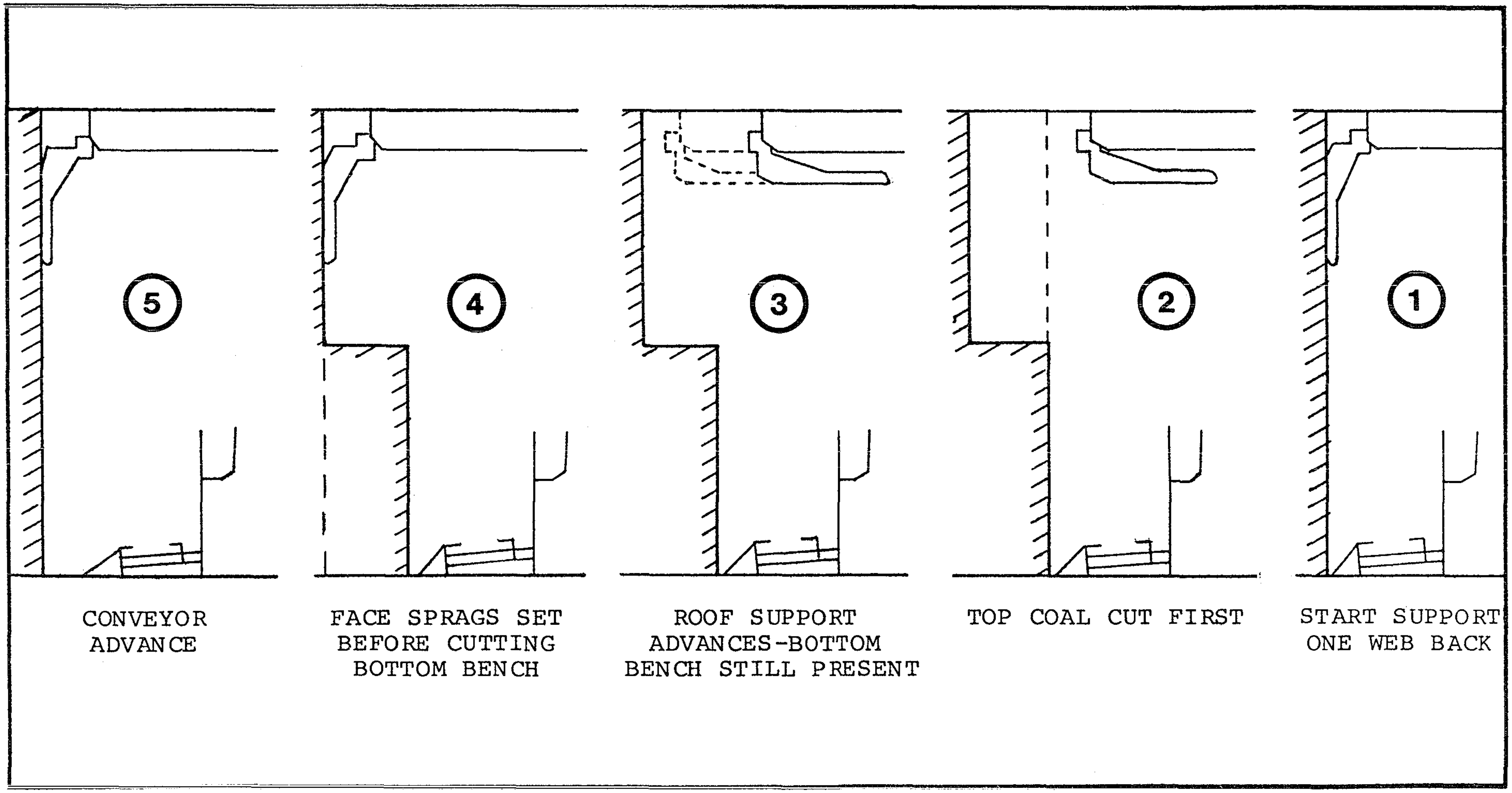


Figure 5-1. Face Advance Phases in a Two Bench Method

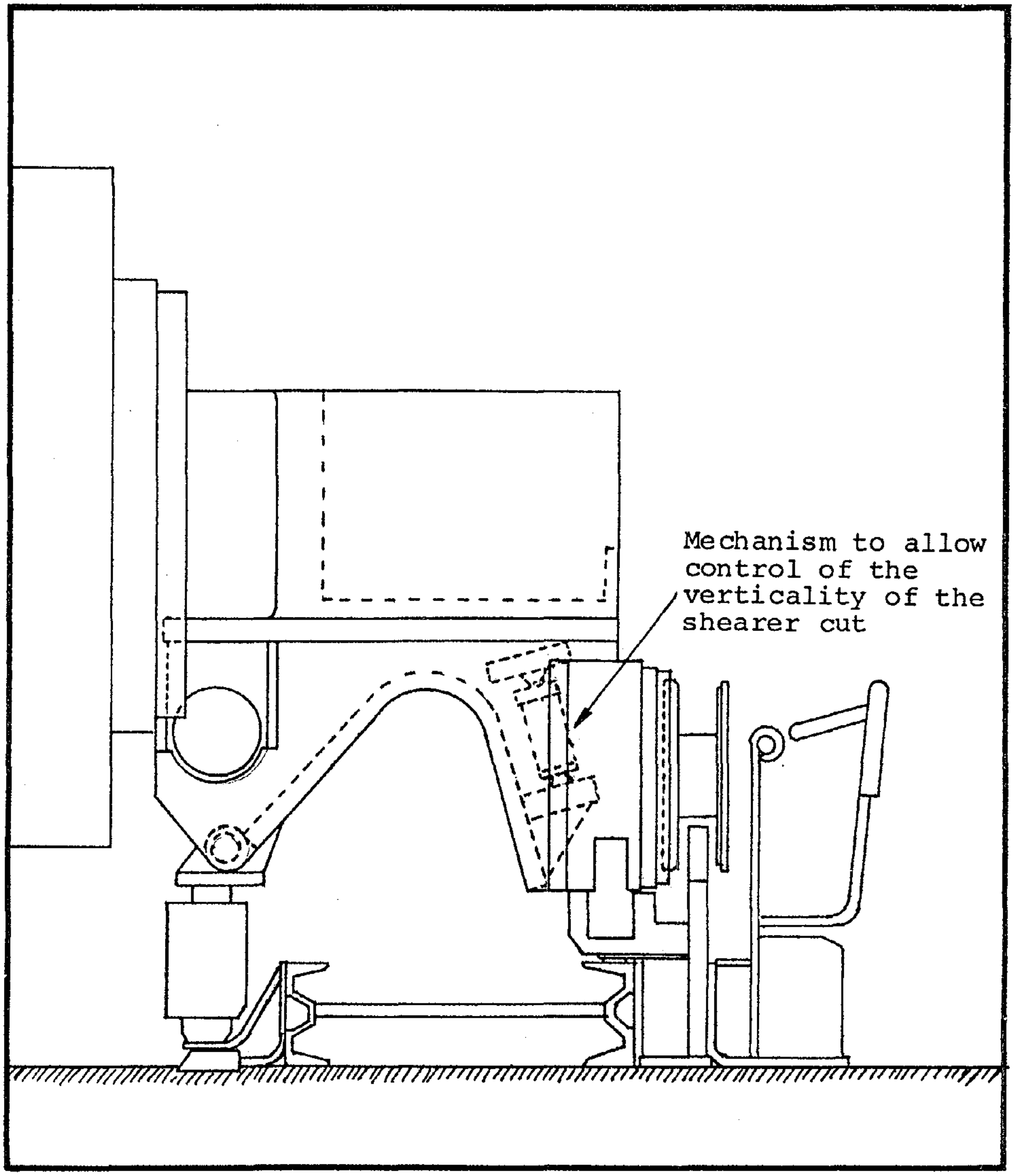


Figure 5-2. AM 500 Mk2 Double Ended Ranging Drum Shearer Fitted with Roll Rack Haulage System

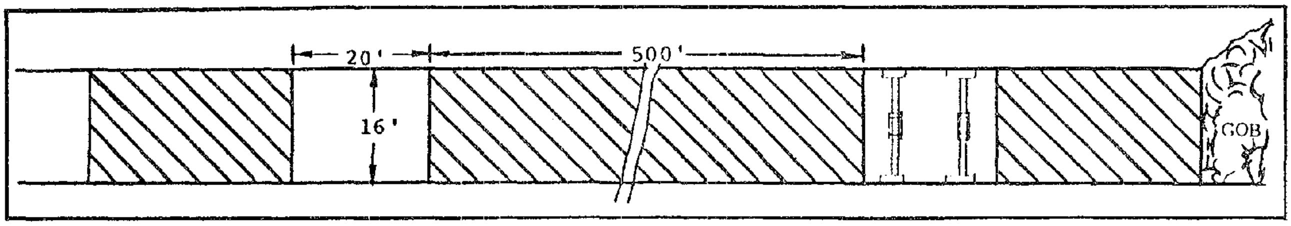


Figure 5-3a. Longwall Mining 16 Feet Seam with 16 Feet High Entries

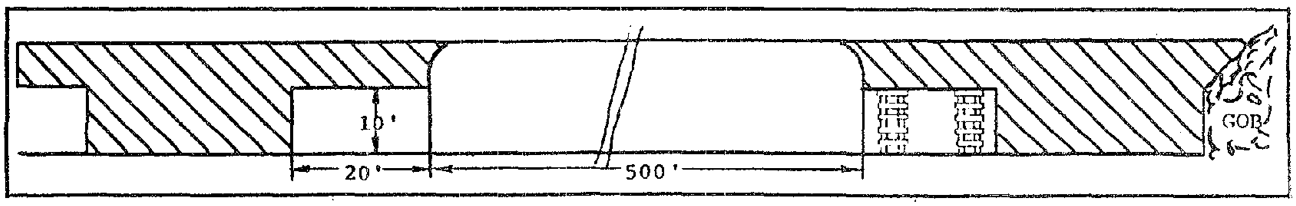


Figure 5-3b. Longwall Mining 16 Feet Seam with 10 Feet High Entries Developed in Lower 10 Feet of Seam

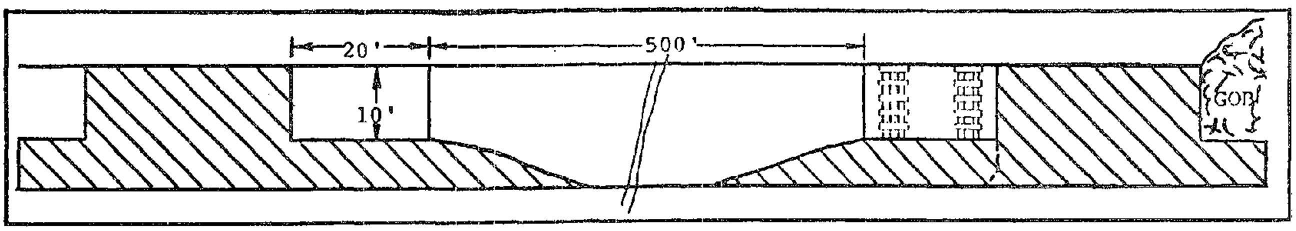


Figure 5-3c. Longwall Mining 16 Feet Seam with 10 Feet High Entries Developed in Upper 10 Feet of Seam

Full Face Cut

Full face cut is commonly used with double ended drum shearers. In the United States, the shearers are generally cutting in one direction only, to keep the miners upwind and reduce dust exposure. The full face cut can be used in up to 5 meters thickness but when the thickness increases, face sloughing occurs, frequently causing delays and requiring face nailing.

Bidirectional cutting of the full face is shown on Figures 5-4a and 5-4b. When coal is hard and dust well controlled, bidirectional full face cutting is feasible.

If necessary, unidirectional cutting may alleviate dust and lumps problems.

Two Bench Face Cut

By first cutting the top bench and advancing the roof support when the bottom bench is still present, and by cutting the bottom bench when anti-spalling plates are set against the top coal, sloughing of the face can be strongly reduced, and the results improved. In addition, when using a double drum shearer to mine two successive benches, the drum diameter can be smaller which is beneficial to dust control.

In most seams, the two bench method (Figures 5-5 and 5-6) is recommended and the face should be equipped with:

- A double drum shearer-loader with drums not exceeding 2 m (6.6 ft.) diameter, 1 m long, with less than 30 rpm and a limited pick number. The shearer would use both drums to first cut the top bench and then both drums for the bottom bench. Figures 5-5a and 5-5b show bi-directional cutting and shearer face travel versus time.
- One web back, two-leg shield supports which will be advanced as soon as the top bench will be won. Face sprags will be set before mining the bottom bench.
- Conveyor carrying underframe which allows the use of shields with long base members.

Suggested face organization is shown on Figures 5-5a and 5-5b. Although there are advantages relating to dust control, there is no difference in productivity when mining in both directions (as shown in Figure 5-5) or mining uni-directionally (Figures 5-6a and 5-6b) as long as the top bench is cut first and the roof supports are advanced before cutting the bottom bench.

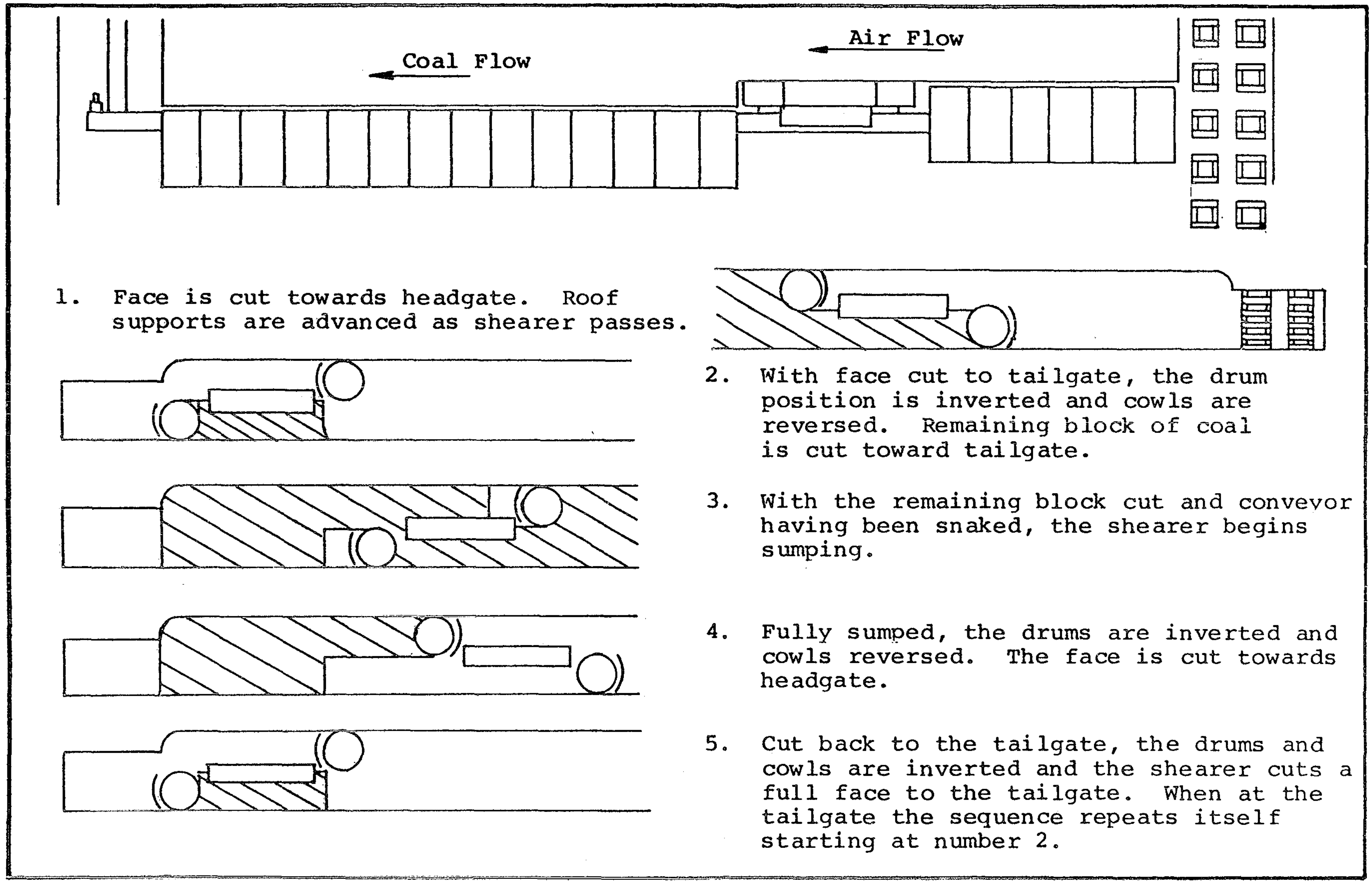


FIGURE 5-4a.

Bidirectional Face Cycle with Large Diameter Drums
("Full Face Cut")

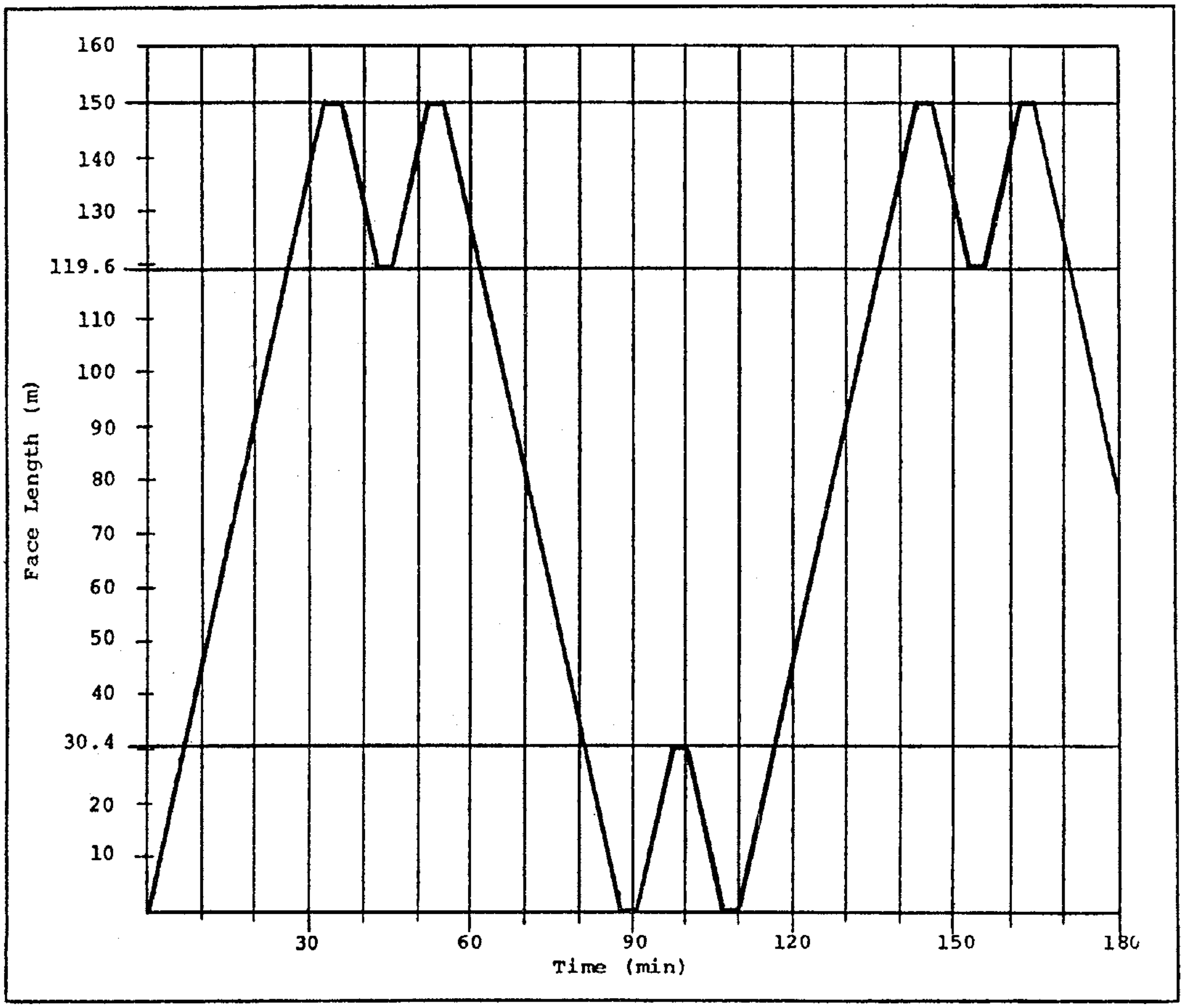


FIGURE 5-4b.
Shearer Face Travel ("Full Face Cut") Versus Time

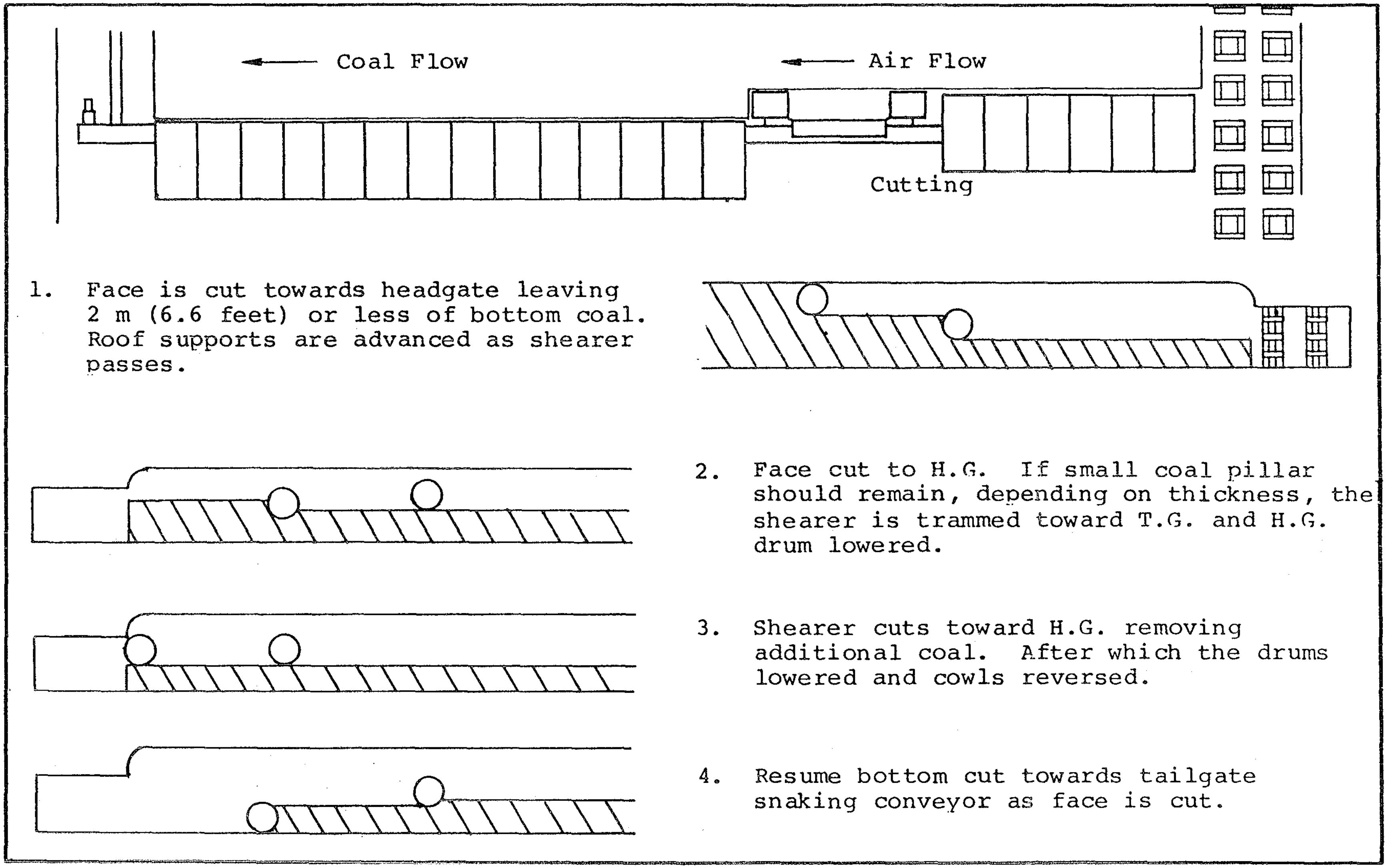
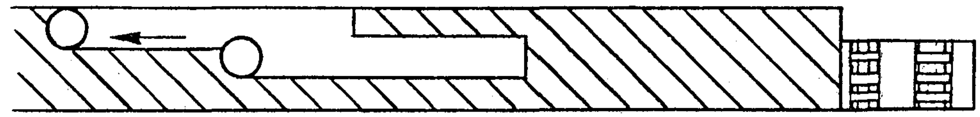


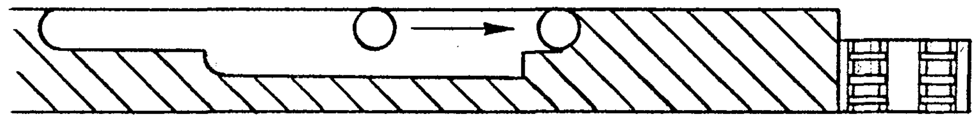
FIGURE 5-5a.

Bidirectional Shearer Face Winning Cycle
(Two Bench Method)

5. After finishing bottom cut at T.G. return drums to top cut position and sump in toward H.G.



6. Once sumping is complete rear drum is raised and top 2 m (6.6 ft) is cut to T.G.



7. Drums are returned top cut position and face is cut to headgate.

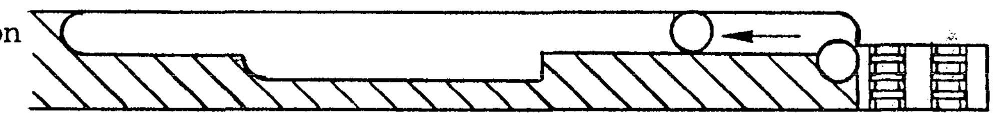


FIGURE 5-5a. (Continued)
Bidirectional Shearer Face Winning Cycle
(Two Bench Method)

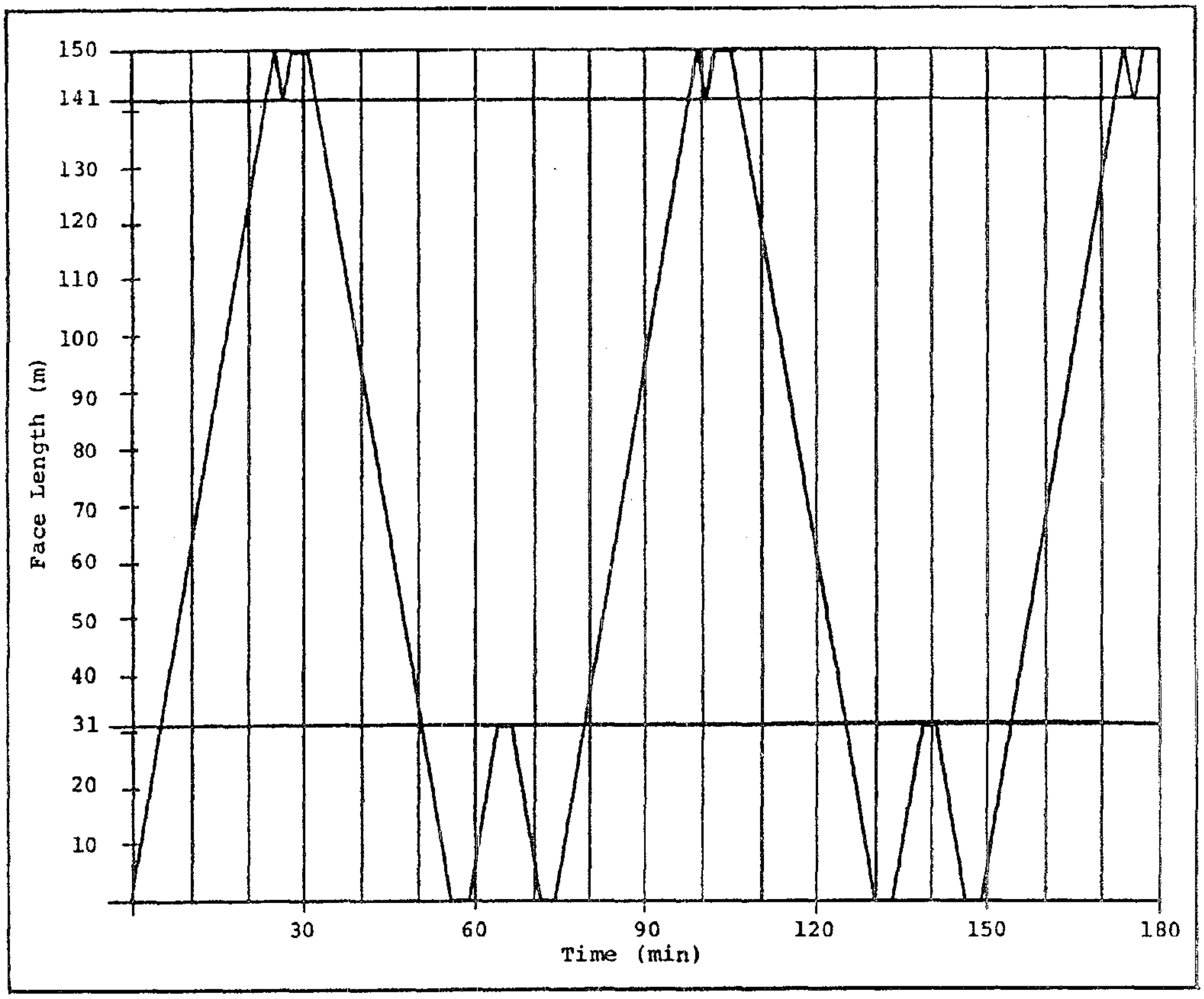
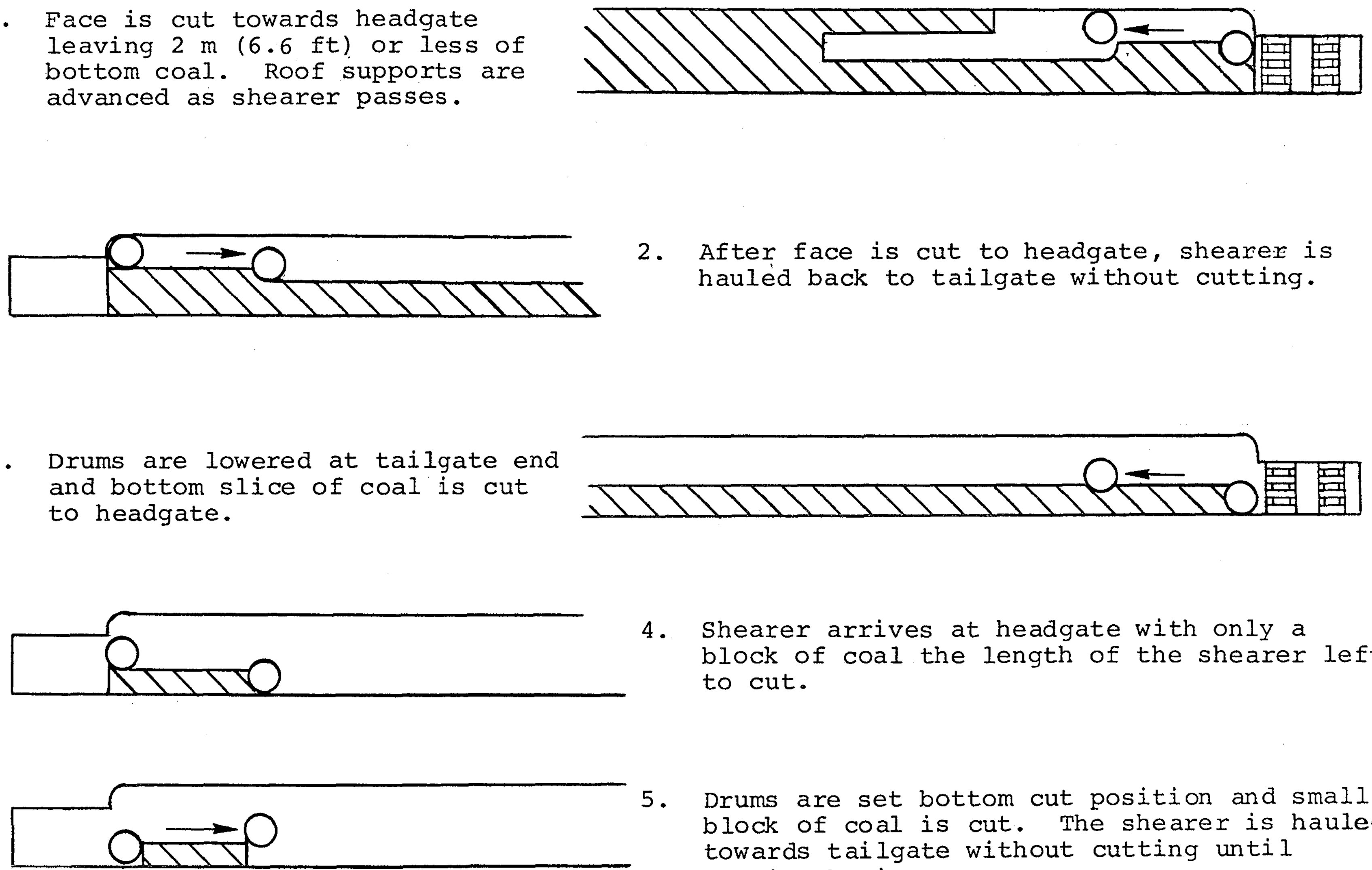


FIGURE 5-5b.

Bidirectional Shearer Travel Versus Time
(Two Bench Method)



1. Face is cut towards headgate leaving 2 m (6.6 ft) or less of bottom coal. Roof supports are advanced as shearer passes.

2. After face is cut to headgate, shearer is hauled back to tailgate without cutting.

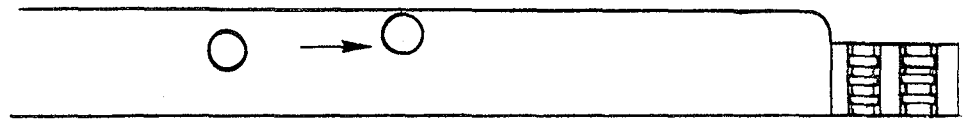
3. Drums are lowered at tailgate end and bottom slice of coal is cut to headgate.

4. Shearer arrives at headgate with only a block of coal the length of the shearer left to cut.

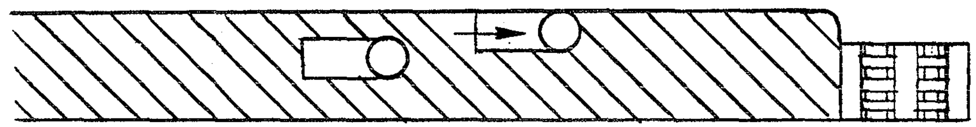
5. Drums are set bottom cut position and small block of coal is cut. The shearer is hauled towards tailgate without cutting until sumping begins.

Figure 5-6a. Unidirectional Face Winning Cycle (Two Bench Method)

6. When at sumping point, the shearer drums are put into top position. At some time between Step 3 and this point, the A.F.C. has been snaked for sumping.



7. Sumping begins on upper section of face and is cut to tailgate. The A.F.C. may be pushed forward for the distance of the face.



8. Shearer drums are set to cut towards headgate.

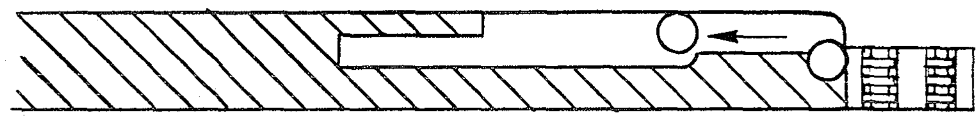


Figure 5-6a. (Continued)

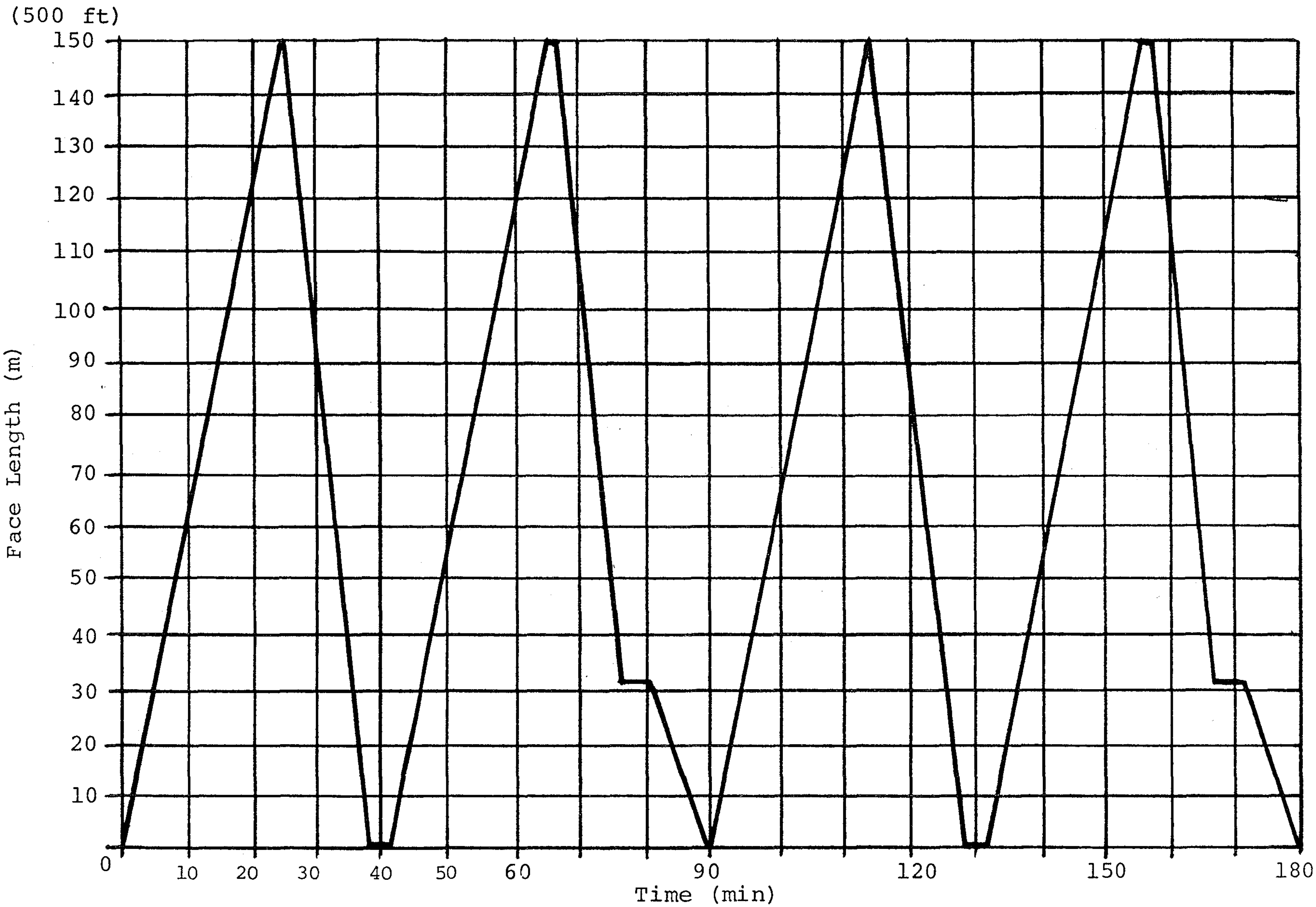


Figure 5-6b. Shearer Face Travel Versus Time for Cutting in One Direction
Time 0 = Beginning of Upper Face Cut

The equipment specifications were written based on this two bench face organization and on calculations presented in the following section.

5.2 Equipment Selection Calculations

5.2.1 Introduction

Equipment specifications are based on U. S. and foreign experience and are oriented to maximize daily production. Because the dust level in longwall faces is a limiting factor, major emphasis is placed on dust reduction calculations.

A second concern is the increase of equipment availability not only through the reduction of down time but also by reducing maintenance during operation.

5.2.2 Dust Control

It is difficult to know during the planning stage if the dust control procedures will permit production of the maximum tonnage which the equipment is technically capable of achieving.

The different factors affecting dust generation are well known. Some are related to natural conditions, such as the rank of coal and of the seam characteristics (moisture content and presence of partings). Sometimes only water injection may improve these natural conditions. Some factors are related to the mining equipment, such as the depth of cut or bit type, the drum speed, the flow of water and its distribution.

Water Infusion

Water infusion of coalbeds for dust control is not commonly used in U. S. coal mines, but has been tried mostly for methane control. Concerning dust, USBM Report of Investigation 8241 has shown a recorded dust reduction of 69 to 79%, using only two gallons of water per ton of coal.

On the other hand, water infusion is mandatory in West Germany. For the mechanized faces in West German coal mines (i.e., more than 98% of the total), water infusion of the face is often an essential means of keeping the concentration of dust down to acceptable levels. With the increase of the average daily output per face and the reduction of down time, it has become increasingly difficult to use water infusion methods from the face working space. The correct use of the infusion technique from the face presently requires planning and organization of the highest order. Continuous infusion from the accompanying gate roads has therefore been used increasingly in recent years. These techniques are as effective as the short or longhole infusion from the face, and should be applied by an infusion team with special experience in drilling and sealing long infusion holes.

As a rule, long-hole infusion parallel or diagonal to the face from gate roads or remote pre-infusion can be better adapted to the layout of a mine and the structure of the deposits than any other infusion technique. The continuous infusion method is now practiced in West Germany on more than 20 percent of the faces on which water infusion is used. Sealing of the infusion boreholes was made much more effective by the use of new additives and grouting equipment. Switchable pumps were able to cope more or less with the different infusion boreholes which had to be dealt with simultaneously. Volume-controlling valves which work independently of the pressure factor enable a larger number of boreholes to be supplied with uniform quantities of water, and new measuring instruments have enabled the infusion process to be monitored in the pit control room.

Water infusion is also considered very efficient in French coal mines. A paper[1] presented at Dusseldorf in May 1976, stated that natural moisture decreases the dust production by half when the content is up to 6 percent. However, an additional 3 percent infused water is very efficient. It is recommended that a slow infusion rate (5 m^3/h max) be used in advance of the face. A system of permanent flow control has been developed to control and record the water flow for each hole.

In Great Britain the coal is difficult to infuse, therefore, their main effort was oriented toward pick lacing, drum design and wet drum. The result was the development of a large pick drum and of phased pick face flushing.

The question is whether water infusion is possible in U. S. coals. Because of the importance of water infusion for dust control, it should be studied using laboratory techniques. Utilizing petrographic analysis, samples are polished, then oxidized and examined with a microscope for microfissuration (possible ways of infusion). A simpler way to measure the permeability of the coal seam is by drilling holes and trying to inject water at different pressures and hole configurations.

For this project, it appears that the D seam of the Orchard Valley Mine is permeable for water infusion, but that the Upper Hiawatha coal seam at SUFCo may be too hard.

Water Spray System

The problem with a water spray system is a dual one: what quantity of water is required and where should it be added? Dust reduction is closely related to the total amount of water, which

---

1 M. Ganier and C. Froger, CERCHAR, presented 9th World Mining Congress, Dusseldorf, May, 1976.

is limited by other considerations such as: coal preparation, belt system, and also discomfort for the miners who are more sensitive to water than to dusty air.

Adding, perhaps, 3 percent of water to the coal produced means huge amounts of water are required (36 m^3/h when production peaks at 1200 t/h, or about 150 gallons/minute or 2.5 gallons/second). However, it is then necessary to reduce this flow when the production is reduced. It is not sufficient to start and stop spraying when coal production starts and stops. It is necessary and should be feasible to regulate the flow of water in relation to the amperage of the shearer by using a variable delivery pump or a valve system. A pump will also help to compensate the pressure drop. However, it is not enough to add large quantities of water, it must be done efficiently. The efficiency of the water spraying is related to the pressure available at the sprays. A pressure of 150 psi will be required and the piping as well as hose size (at least 5 cm (2 in.)) must be chosen in accordance.

On the drum itself, pick face flushing is the best way, according to British experience. The outer sprays should be arranged to confine the dust and not cause turbulence. Tests by the Bureau of Mines were made in thinner seams but the basic idea should be maintained for thick seams: confining the dust along the face by a special arrangement of sprays on the shearer and, in the case of thick seam, by additional sprays on shield canopies (see Figure 5-7). Another possibility to increase the efficiency of water spraying is the use of surfactants.

To summarize, the recommendations are:

1) Design the water supply on the shearer to supply water in quantities equivalent to 3 percent of the coal. Modulate water supply in accordance with production rate.

2) Arrange the spray system to confine the coal dust along the face.

Ventilation

In a 4.9 m (16 ft.) mined thickness, the open section area supported by a two-leg shield support with a 3.5 m (11.5 ft.) canopy should be 140 square feet. When the mined thickness decreases up to 3.7 m (12 ft.), the area is reduced up to 90 square feet.

The selected mines have no recorded methane problems. Ventilation will deal more with dust than with gas. The possibility of dust dilution by increasing the flow of air is limited because excessive speed will have the negative effect of increasing the percentage of airborne dust. The optimum velocity,

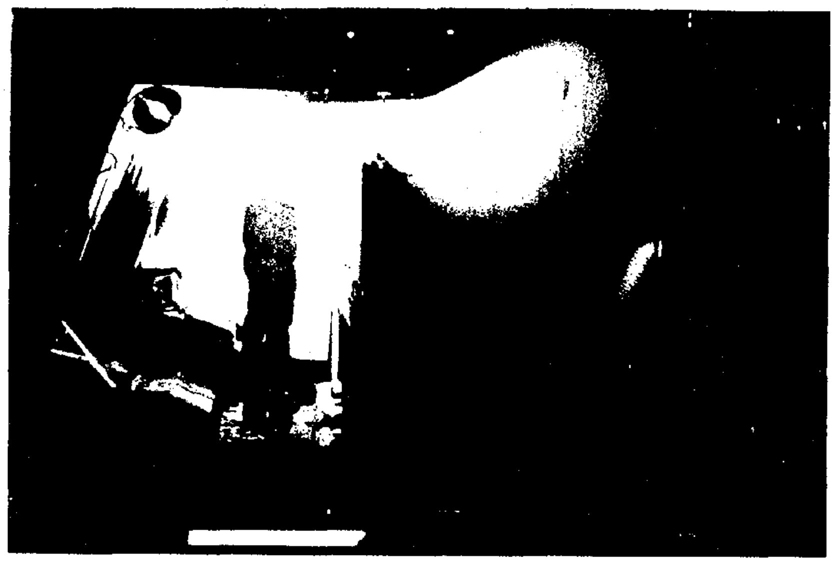


Figure 5-7a. Photograph of Dust Plume by a Conventional Water Spray System

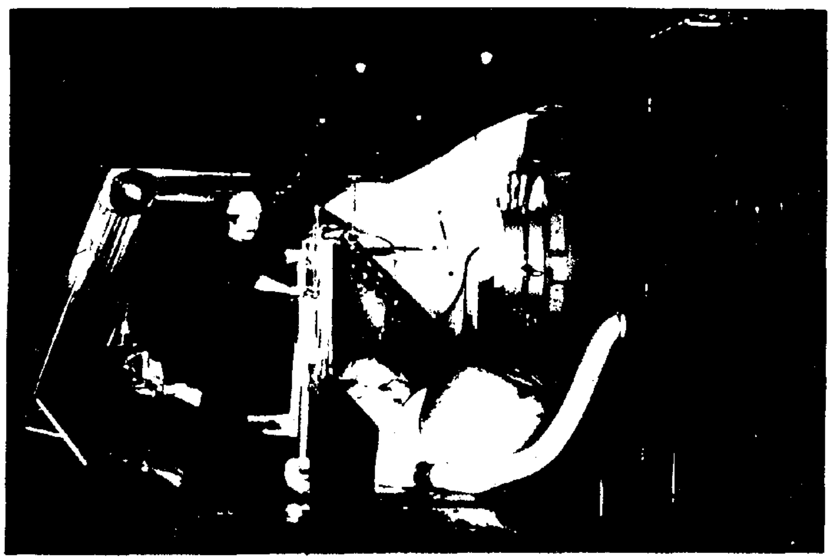


Figure 5-7b. Photograph of the Shearer-Cleaner System Splitting the Primary Airflow and Forcing the Dusty Split Towards the Face

according to current dust studies, is between 120 and 180 m/min. (400 and 600 ft./min.). A velocity of 120 m (400 ft.) per minute through a 90 square feet section gives a 36,000 ft^3/min. quantity of air, which should be satisfactory if the dust control is efficient (Figure 5-8). Taking into account air flowing through the gob, an air flow of 70,000 $ft.^3$/min. should be planned for the air intake.

As mentioned, homotropal ventilation is recommended. Arrows show the proposed air flow on Figures 4-9 and 4-11, which is consistent with the degasification during longwall operations recommended in case of an unexpectedly high methane liberation.

5.2.3 Roof Support

Shield supports are generally preferred for the mining of thick seams from the viewpoint of both safety and economics. Shield supports are the only powered support presently used in thick seams. Moreover, the development of shield supports has allowed successful mechanization of seams thicker than 2.4 m (8 ft.).

Shield support originated in the USSR coal mines. From these origins were developed the two systems: caliper shield and lemniscate shield. At that time, 1960, the purpose of the design was to reduce the roof supported area and to seal the gob area. This shield support was further developed successfully by the German coal industry after its introduction in West Germany from France and Hungary.

During this development period, the length of the canopy was increased considerably as was the yielding load. However, the width of the support elements was stabilized at the pan length of 1.5 m (5 ft.). The yielding load was steadily increased on an empirical rather than theoretical basis.

Yielding and Setting Load

Varied approaches were used to access the minimum values for yielding and setting loads. The approaches, being generally empirical, were influenced by and are representative of natural conditions.

In British coal mines the roof is generally soft and the solid coal is hard. To control the soft roof, it is best to reduce the distance between the solid coal and the first support. A 2 m (6.6 ft.) distance is the rule, which limits the width of the conveyor and the cutting depth of the shearer. Variances had to be granted for increasing the web width and for using shield supports. There is no such rule in France or in West Germany for different natural conditions. There, the emphasis is on the load capacity, especially the setting load. The setting load is as close to the yielding load as is technically feasible (80 percent).

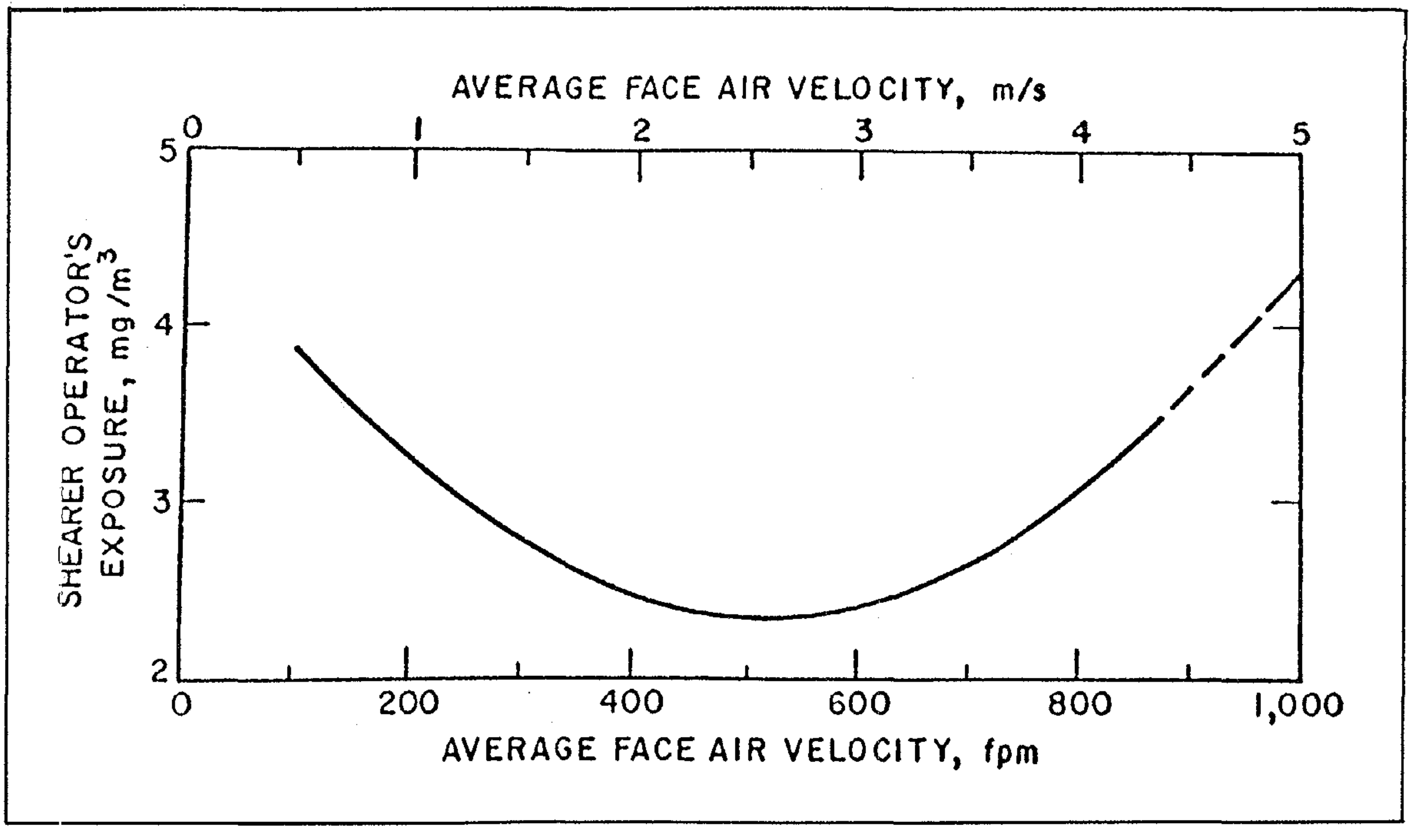


Figure 5-8. Respirable Dust Control in Longwall Mining Operations

In Germany, load capacity is calculated in proportion to the mined thickness and to the supported area. In France, the idea is to reduce the roof convergence with load capacity requirements given in accordance. Figures 5-9 and 5-10 summarize the rules used in these two countries.

The differences of opinion on setting load requirements disappear when steep seam mining is considered, because the setting load must prevent strata beds from sliding down the hill. The German rule is:

$$A = 0.23\ M\ \text{ton/sq ft.}$$

where M is the maximum mined thickness in feet. This rule is recommended for defining the support capacity. This value should be in excess of the strata control requirements taking into account the strength of the coal, which until now has never been considered.

Operational Range

The required thickness range of a roof support is often underestimated and the consequence is a loss of roof control. Coal seams in the U. S. are generally of regular thickness, but in some mines, the mined thickness may vary in a large extent.

A shield support in a level seam is stable. If the shield is too short, the legs will extend at the end of their stroke and the maximum setting pressure will be recorded without having the shield canopy pressed against the roof. This problem is not unique to thick seams, but it will be more difficult in thick seams for the miner to evaluate the thickness of the seam and impossible for him to set wooden blocks on the canopy. Therefore, it is suggested that the roof support range be 0.3 m or 0.6 m (1 or 2 ft.) higher than the maximum cutting range of the shearer.

Canopy Length

Shorter canopies are more desirable, but it is difficult to reduce the length below 2.4 to 2.7 m (8 to 9 ft.). In a thick seam, the provision for men to travel behind the legs, requires an increase in the total length to almost 3.5 m (11.5 ft.). The canopy length should be limited to 3.5 m (11.5 ft.). Furthermore, a two-leg shield is preferable to a four-leg shield or a chock shield (Figure 5-11).

Width of the Shield

The roof support width has always been increased reluctantly, limiting the roof area to release for advancing. With time and technological progress, width has evolved to about 1.5 m (5 ft.). This limit can be justified in European conditions, The statutory minimum support capacity, A, for caving longwall in coal beds under 18° of pitch, is calculated by the following formula:

$$A \geq 1.6 \times 2M \times 2.5 \times 10 \text{ kN/m}^2$$

$$\geq 8M \times 10 \text{ kN/m}^2$$

where

M = maximum mined thickness in meters

The factor of 2 allows for caving of the roof strata to a height of twice the thickness of the coal bed.

2.5 = specific gravity of rocks

1.6 = safety factor

This formula reads in the U. S. system

$$\boxed{A = 0.23\ M \text{ ton/sq. ft.}}$$

where M = maximum mined thickness in feet

When the gradient exceeds 18 degrees, the statutory minimum support capacity

$$A = (5 + \frac{1.5}{9} \times p)M \times 10 \text{ kN/m}^2$$

where p is the pitch in degrees.

This formula reads in the U. S. system

$$\boxed{A = 0.23\ M\ (\frac{45 + 1.5p}{72}) \text{ ton/sq. ft.}}$$

where M = maximum mined thickness in feet

p = pitch of the seam

The roof areas to be considered are the minimum area for calculating the setting load and the maximum roof area for the yielding load.

In both cases, this area is equal to (L + e)c

L length of canopy in ft.

e roof exposed between canopy tips and face

c unit centers in ft.

For the setting load, the value of e_1 of e is measured with the minimum face width. That means after shield advance.

For the yielding load the value of e_2 of e is measured with the maximum face width. That means after winning and before shield advance.

$e_2 - e_1$ = depth of cut

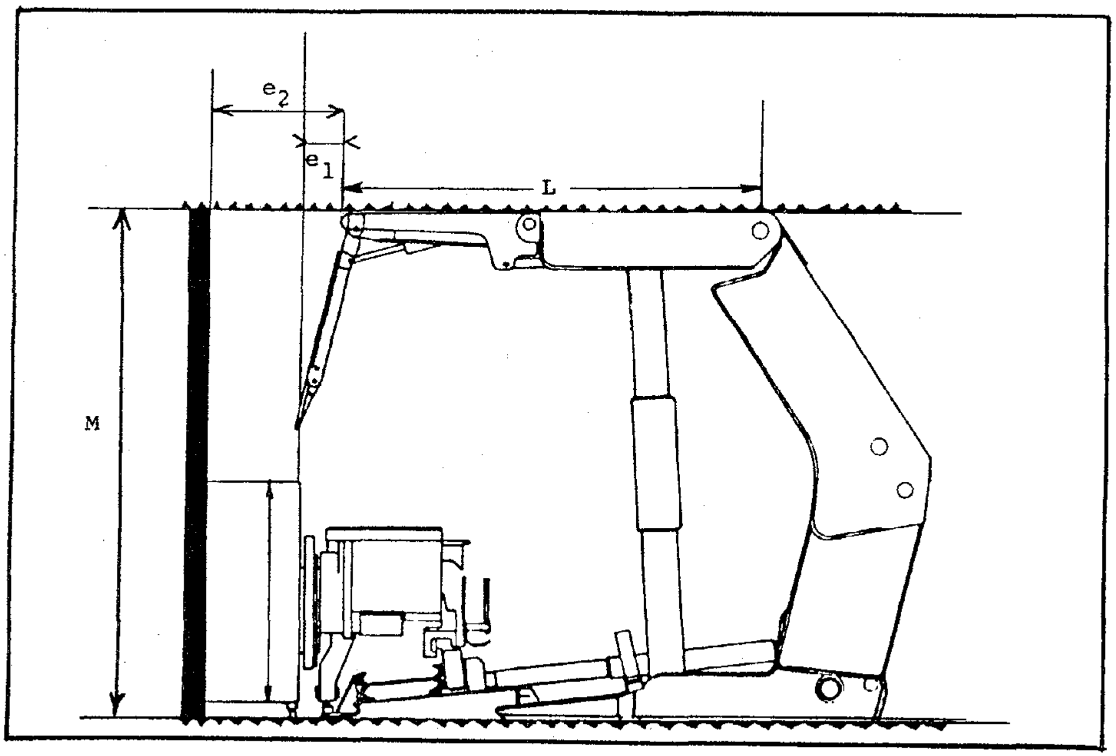


Figure 5-9. Statutory Support Capacity in West Germany

Convergence in a longwall face is a function of the final subsidence and of the support resistance. The following formula was obtained from measures made in hundreds of longwall faces in French coal mines

$$CVT = (qW)^{3/4} H^{-\frac{1}{4}} \left(\frac{6800}{PM} + 66\right)$$

with

CVT = average convergence of roof and floor in mm/m of face advance

w = mined thickness in meter ($0.8 \leq w \leq 3$)

q = a subsidence coefficient (1 caving . 0.6 pneumatic stowing 0.15 hydraulic stowing

H = depth of operations in meters ($100 \leq H \leq 1000$)

PM = support capacity in tons, per linear meter in front face ($20 \leq PM \leq 260$)

Figure B shows the relation between convergence and support load per meter of face for a depth of 500 m.

The roof support should limit the roof convergence at a value for which the roof is not too fractured (about 40 mm/metre). The value varies according to the nature of the roof.

Figure A gives theoretical curves. It is not always possible to obtain such high values if the roof is friable or if the mined thickness is high. The roof support has therefore to deal with a roof damaged by convergence. A shield support is recommended in this case. The same rules apply in steep seam.

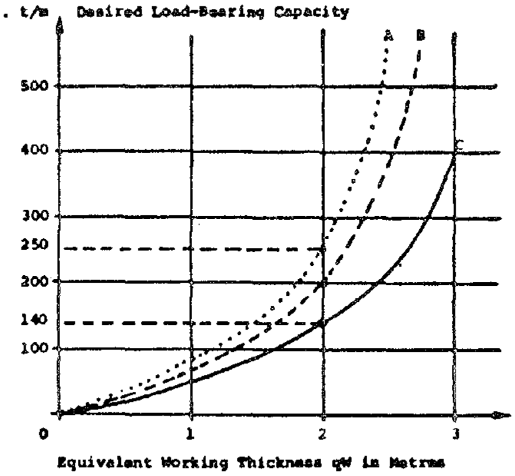


Figure A. Desired Load-Bearing Capacity as a Function of the Working Thickness Depth 500 m (1640 ft)

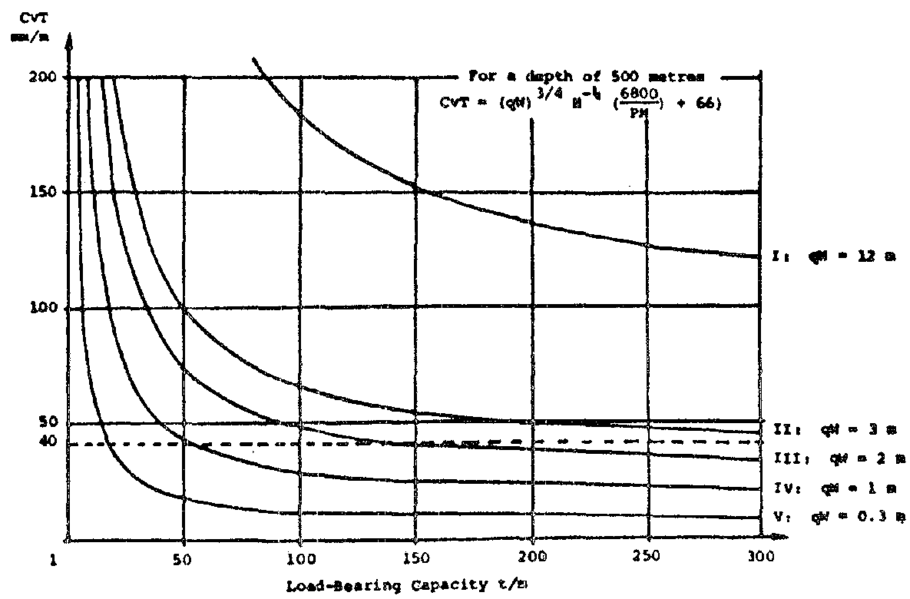


Figure B. Relationships Between Average Convergence and the Load-Bearing Capacity per Linear Metre of the Support

Figure 5-10. Recommendations for Roof Support Selection from Experience in French Coal Mines

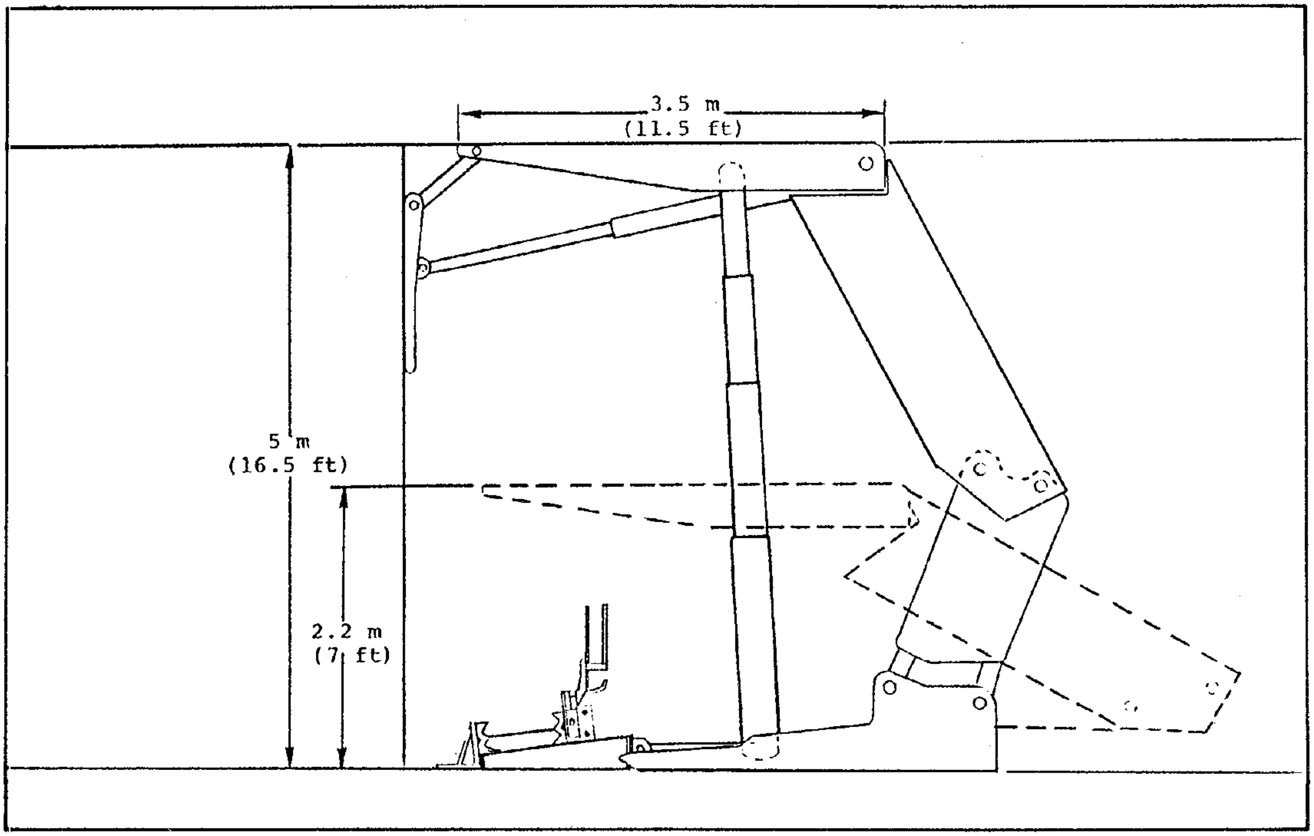


Figure 5-11. Thick Seam Shield Support Conceptual Design

mostly because transportation in narrow entries is quite difficult. In U. S. conditions, there is no reason to abide by this limit, and a width of about 2.1 m (7 ft.) is recommended.

The increase of shield width should be accompanied by an equivalent increase of the pan length and of the rack length which, in chainless haulage, could only be increased by increments in correspondence with the pitch.

Length of the Base

The advantages and disadvantages of the long base advancing under the conveyor supported by a frame, compared to a short base, have been discussed previously. It appears that the best results in thick seam are obtained in Saar and Poland, where long bases are used. While there is probably no definitive reason, at least it can be said that long bases are not an inconvenience. Long bases have presented problems when the floor was too soft and the resultant force is too close to the base tip. Short bases are better in irregular deposits when faults have to be crossed by the face. The flexibility provided by short bases is, in this case, an advantage.

For mining a very thick seam, the long base is recommended for the following reasons:

- When face sloughing is a major problem, an immediate forward support can be of great advantage.
- The load required to pull the increased shield mass can be withstood by the frame, instead of being withstood by the conveyor itself.
- Steps in the floor can be a major cause of down time if the geometrical arrangement of the equipment places the edge of the step under the conveyor. The conveyor will stop when the shearer presses it down (Figure 5-12). This can be avoided by increasing the depth of the web up to 1.2 m (4 ft.) to have the step wide enough to hold conveyor and ramp plates, or by using a conveyor carrying frame.

Adaptability of the Support to Gradient Change Along the Line

Side shields seal the gap between successive shields. The side shields were studied in detail from the dust production point of view and are generally efficient. However, shields were first

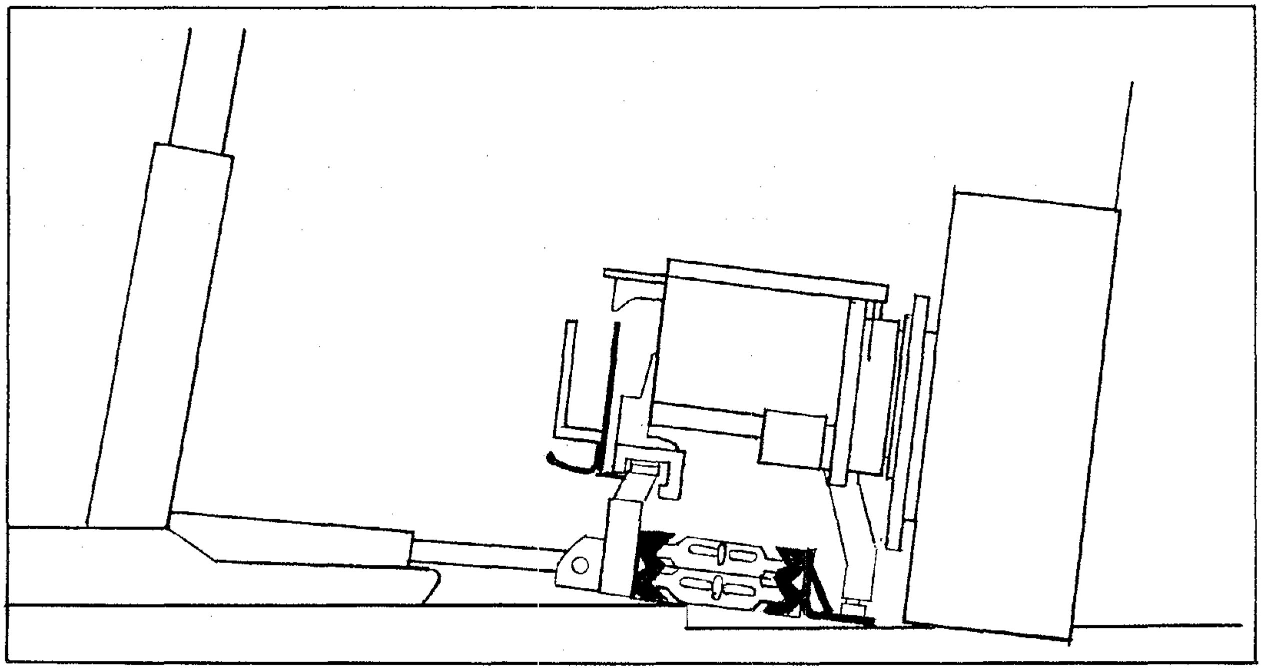
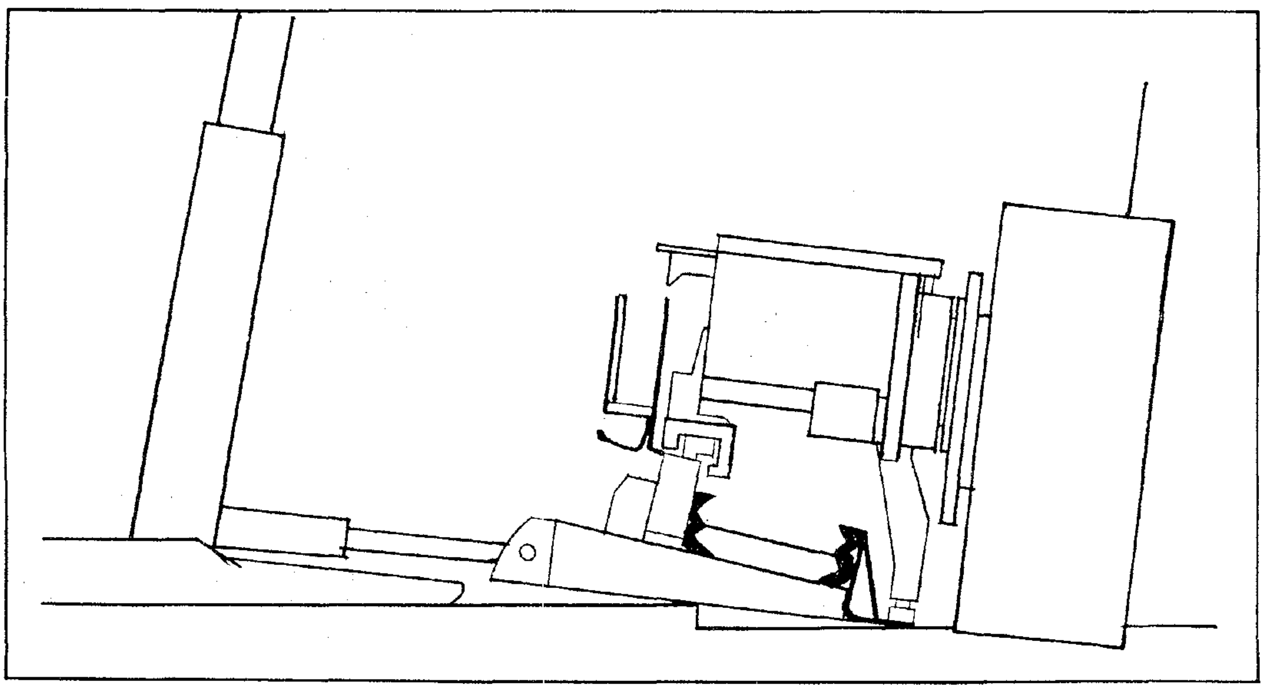


Figure 5-12. A normal conveyor (above) can be jammed when the shearer weight presses the bottom flights against the edge of a step in the floor, whereas, a conveyor carrying frame (below) can straddle the step with no jamming of the chain. Another possible solution could be to match the web dip depth with the overall conveyor width. This may require a 1.2 m (3.9 ft) web for a .732 m (2.4 ft) wide conveyor.



developed for a mined thickness less than 3 m (10 ft.) and the possible lateral displacement is usually limited to 10 cm (4 in.). For a shield support designed for 6.1 m (20 ft.) this lateral displacement of the side shield should be doubled.

For the thick seam single pass system, this requirement is important. A greater flexibility will be necessary if, under difficult mining conditions, the entry is driven with only 3 m (10 ft.) under the roof and if the face must be graded down to reach the mined thickness of 5 m (16 ft.).

Walkway

The man-travelway shall be designed inside the support behind the row of props. An elevated walkway providing a safer position to the miners would be advantageous. Provision will then be made to easily remove it when the mined thickness decreases or when the equipment has to be moved to another face.

5.2.4 Conveyor

There are several formulas for estimating the required conveyor power. Their results are sometimes far from reality. The power varies in accordance with the nature of the product, the design of the pan, the efficiency of the transfer point, etc. The accuracy of several formulas, by comparison of real and calculated data[1] has been studied in French coal mines (Houilleres de Lorraine) and the following formula has been selected.

Power calculation:

$$P_{Empty} = 2L \times M_1 \times V \times \frac{9.8}{1000} \times \cos\Psi$$

$$P_{Loaded} = L \times Q/3600 \times {}_2 \times 9.8 \times \cos\Psi$$

$$P_{For\ Elevation\ change} = L \times Q/3600 \times 9.8 \times \sin\Psi$$

$$P_{Total} = (P_{Empty} + P_{Loaded} + P_{For\ Elevation\ change}) \times \text{Safety Factor}$$

where

P = power in Kilowatts

L = length of conveyor in meters

Q = flow in metric tons/hr

---

1 Charbonnages de France, Publications Techniques No. 1, 1979, p. 53.

V = chain speed in m/sec

M = mass in Kg/m of moving parts

Ψ = slope of conveyor

μ_1 = coefficient of friction of the moving equipment in the pans

μ_2 = coefficient of friction of the product transported in the pans

safety factor = 20% of total power

The following values for μ_1 and μ_2 may be used as estimates:

	Double Center Chain Ø 22 mm Level Conveyor 605 mm Wide	Double Center Chain Ø 22 mm Level Conveyor 500 mm Wide V = 0.88 m/sec	Single Chain in Level Conveyor Ø = 30 mm V = 0.94 m/sec
μ_1	0.55	0.50	0.90
μ_2	0.41	0.54	0.36

Besides the inconvenience of down time, working on the face conveyor is hazardous in a thick seam. When doing so, protective devices must be erected to prevent coal blocks from falling from the face; for instance, by bolting the face with fiberglass bolts. The best method of ensuring worker safety is to keep maintenance and repair at a minimum.

Chain maintenance can be minimized by selecting a greater chain diameter, thus reducing risk of breakage. In the event of chain rupture, a two chain conveyor is preferable to a single chain conveyor. Working on the face side of the conveyor can be reduced by welding the ramp plates to the conveyor instead of bolting them, or by using a conveyor carrying frame.

Pan connection is another weak point. A reinforced type of connection, chainlink or dogbone, should be selected or an additional connection between spill plates should be used.

Finally, the type of delivery should be carefully selected. Normal side delivery or roller curve have both advantages and disadvantages. When thick seam mining is concerned with the problems of large coal blocks, the choice should be limited to side delivery or roller curve. The side delivery system is a well proven system. The roller curve is still a recent development. Equipment specifications are written for a side delivery conveyor, but a roller curve should be an attractive alternative in the future. The power on a roller curve conveyor is limited but the delivery on the belt conveyor prevents carry back.

Martinka Mine, a 152.4 m (500 ft.) face in the Lower Kittanning seam (1.5 m (5 ft.)) is presently equipped with a roller curve conveyor with a single 30 x 108 mm center chain using closed bottom pans (Figure 5-13). This conveyor is driven by two 127 kw headgate motors and one 127 kw tailgate motor. The face length will be increasing up to 213 m (700 ft.). One roller curve conveyor of a larger size (832 mm wide, 34 mm chain) is presently on order for a thick seam in Saar, West Germany.

5.2.5 Shearer Loader

The main concerns are, on one hand, to hold the face to avoid sloughing of the face and production of large blocks of coal; and on the other hand, to reduce dust production. The first problem could be solved by using a single ranging drum shearer with a large diameter drum. The second problem is better solved by using a double drum shearer with smaller drum diameter 1.8 to 2.1 m (6 to 7 ft.) (Figure 5-14). Another advantage of a double drum shearer is the doubling of the maximum power and the subsequent cutting force. In addition, the flexibility in choosing the height of the top bench can compensate for the loss of time at the face end which is greater with a double drum shearer.

The power consumption is related to many factors in addition to the coal nature: pick speed and depth of cut, number and spacing of picks, drum design. Obviously the effort to reduce dust production is beneficial to power consumption. If the face equipment is designed for 12,000 ton daily production in 20 hours available time, the shearer should be able to produce 1200 tons an hour. That will provide a 600 ton/hour average (Figure 5-15). The shearer is therefore assumed to cut the coal with a 6 m/min (20 ft/min) speed with 1 m deep web. If the drum has a 30 rpm speed, the shearer advance per revolution is 600 cm/30 rpm = 20 cm.

By using large bits, a three vane disc should be able to cut with a 6.6 cm (2.6 in.) pick depth of cut in a coal seam without parting. In the D coal seam at Orchard Valley Mine, two vanes could be sufficient. A requirement to obtain this depth of cut is to use a sufficient haulage force. Tangential effort on the picks corresponding at 30 rpm and 250 kw is 80,000 N. It seems reasonable to assume a haulage force of twice this amount to meet the required pick penetration. A pulling force of 30 tons at 6 m/min (19.7 ft/min) seems reasonable for two drums of 2 m (6.6 ft.) diameter.

Face Length	152.4 to 213.3 m (500 to 700 ft)
Conveyor Length in the entry	30.5 m (100 ft)
Pan Width	.732 m (2.4 ft)
One Chain	30 x 108 mm
Power	3 x 170 hp (127 kw)
Chain Speed	67 m/min (220 ft/min)
Capacity	1000+

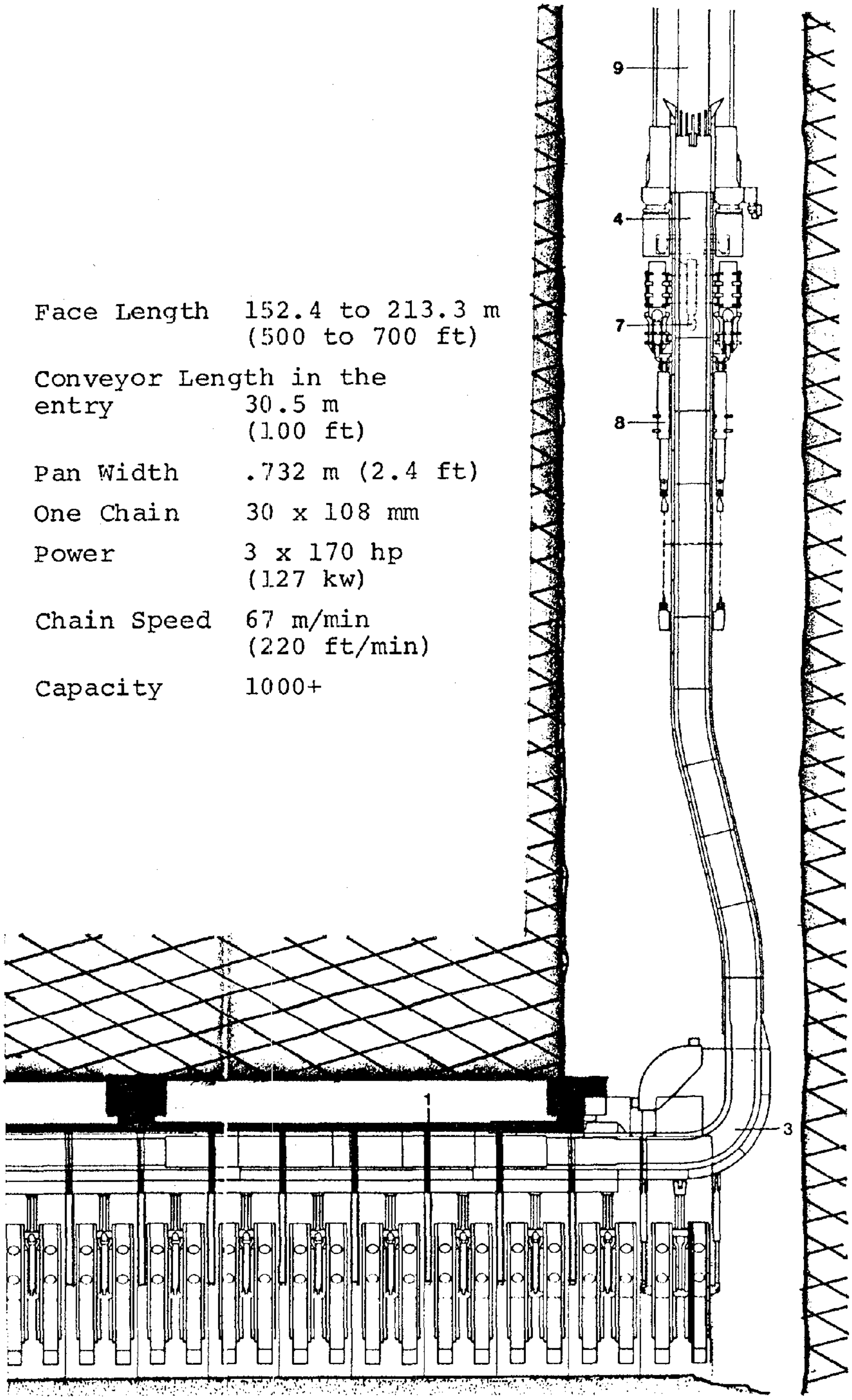


Figure 5-13. Martinka Mine

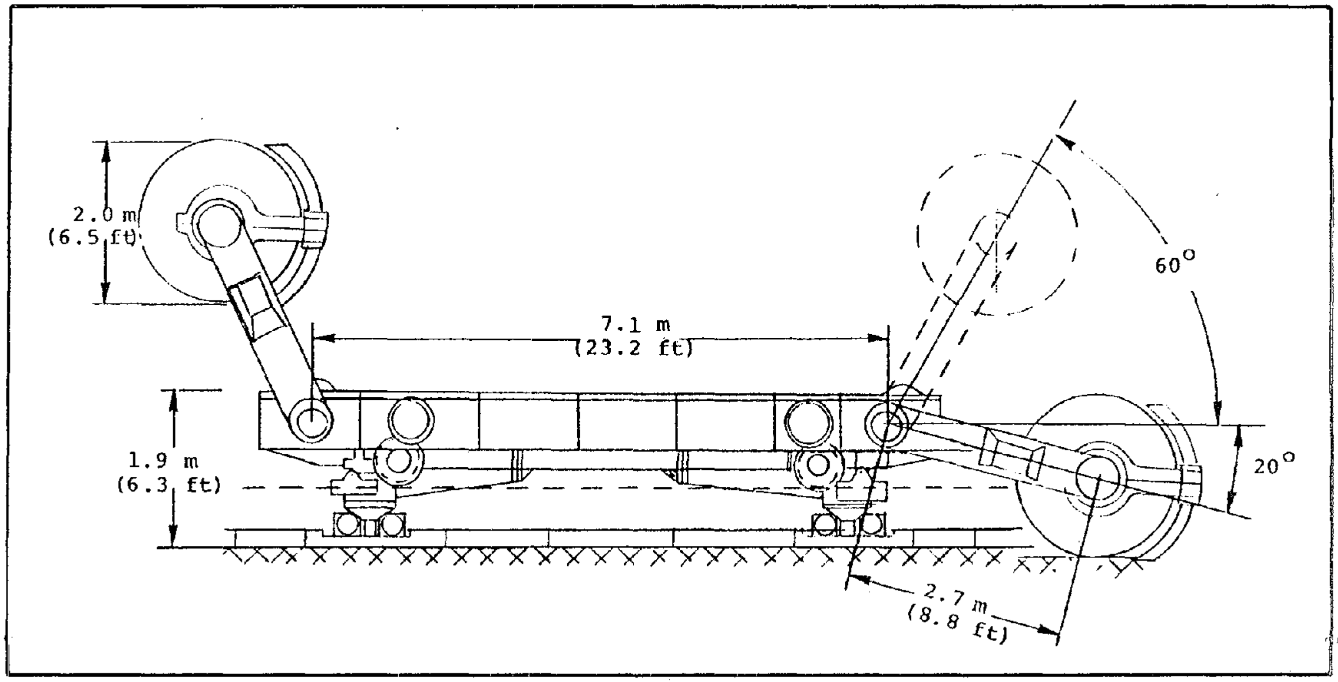


Figure 5-14. Double Ended Ranging Drum Shearer with Chainless Haulage System

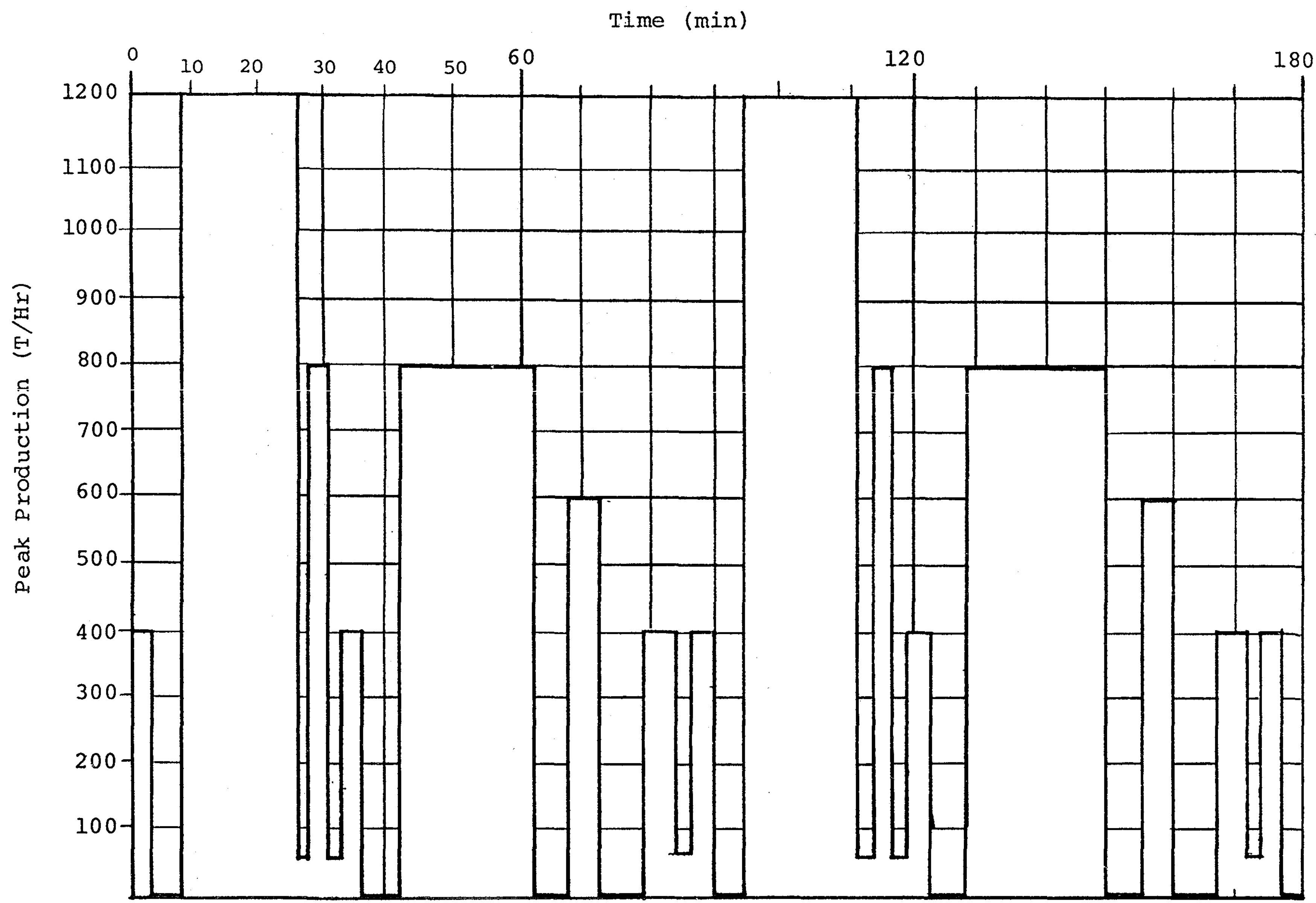


Figure 5.15. Peak Production During Shearer Face Cycle
(Two Bench Method)

Trackless haulage is a must from the safety point of view and also has operational advantages. Two remarks are necessary: the rack limits the conveyor flexibility in a vertical plane and should be kept as low as possible. Steering underframes cannot be used with chainless haulage if the haulage unit is not part of the underframe. The relative arrangement of tooth sprocket and rack is too rigid.

An advantage in thick seams is the fact that ample room is available. The installation of scrubbers on the shearer is therefore possible. Trials were made on an Eickhoff 340L and Sagem DTS 300 longwall shearer (see IG 8753). Figure 5-16 represents a dust collector installation on a Sagem DTS 300 shearer.

5.2.6 Face Installation and Withdrawal

If the entries are driven only to a 2.7 to 3 m (9 to 10 ft.) height, it is likely that, at least in the first phase of this research development, the equipment size should be reduced accordingly. Therefore, the shield support closed height should be limited to 2 m (7 ft.) even if operating range is 3.7 to 6.1 m (12 to 20 ft.). This extra reduction may be obtained mechanically. Furthermore, installation and recovery room heights should be reduced to make these operations easier and safer.

The width of the installation room should be in accordance with the length of the closed shield. With the equipment presently available, a 7 m (22 ft.) width is the minimum expected. "Any place being driven over 20 feet in width should be supported by a Combination Roof Control Plan. The roadway should be limited to 16 feet in width on both straight and the curves to within 10 feet of the uncut face. A row of posts should be set for each 5 feet of space between the roadway posts and the ribs. Openings should not exceed 30 feet in width."[1]

The problem of height reduction is not as important for the installation as it is for the recovery. In the first case, operations are conducted in a high but relatively stable entry and equipment is installed without disturbing the environment. In the second case, operations are conducted between a caved area and a coal massif subject to the abutment pressure. The shields will have to be removed and therefore some other kind of support will be necessary. Therefore, installation should be made in the full thickness with the following safety precautions:

---

1 MSHA Underground Manual, 75.200-9, Criteria -- Combination Roof control plan.

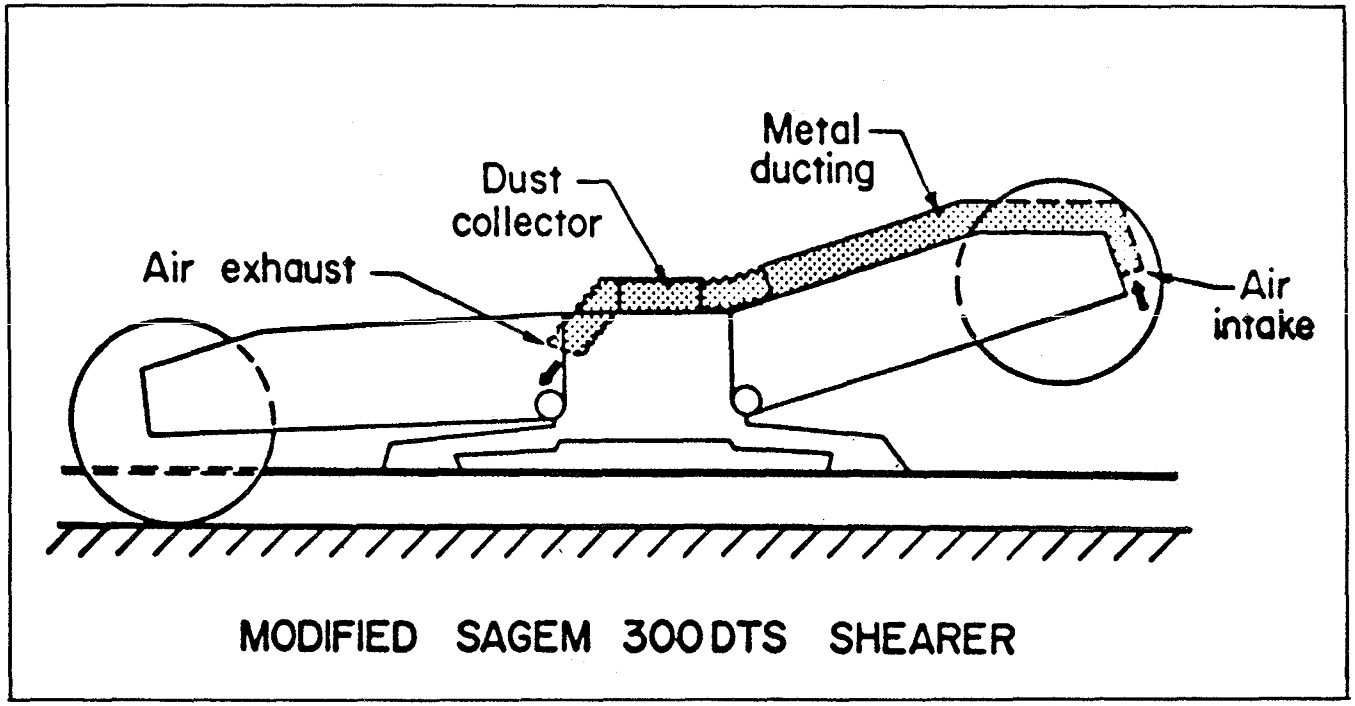


Figure 5-16. Sagem DTS 300 Shearer Concept for Secondary Exhaust Ventilation

- fiberglass bolting of the face with a pattern shown in Figure 5-17,
- installation of the face sprags before working on the face conveyor.

If it is preferred to install the face in a reduced thickness of 3.7 m (12 ft.), the mined thickness will be increased progressively at each web by also using the undercutting possibility.

For withdrawal operations, it is better to reduce the mined thickness to 3.7 to 4.0 m (12 to 13 ft.) before dismantling the face. However, if a withdrawal has to be organized with the maximum mined thickness, a minimum requirement would be bolting of the face and the eventual addition of wire mesh when the solid coal is fractured or is tending to slough.

If the solid coal is unstable even at the reduced height, mesh and bolts should be used to secure the face before starting the withdrawal operations. Fiberglass bolts or eventually wood dowels are recommended for face installation because the shearer will be able to mine through them (Figure 5-18). Wire mesh and expansion bolts are better for withdrawal operations (Figure 5-19). Figure 5-20 shows shield withdrawal operations with the use of cribs and wooden beams.

5.3 Mine Design and Equipment Specification

5.3.1 Purpose of the Specification

For the eventual first trial of this method of longwall mining up to 4.9 m (16 ft.) in a single pass, the following specifications have been established on the basis of the three mines examined during the selection process. Mining specifications are applicable to these mines, and to other mines, as long as the coal is not soft, the roof is of average quality and caves easily. The conditions of two mines, Colorado Westmoreland Inc.'s Orchard Valley Mine and SUFCo's No. 1 Mine, are therefore described as an example.

Production is aimed at a capacity of 12,000 US tons/day for 3 shifts and an available working time of 20 hours per day to insure an annual average of 4,000 to 6,000 tons/day.

The present specifications could be used for requesting proposals for the supply of complete face equipment and technical assistance during the installation and starting up of the face.

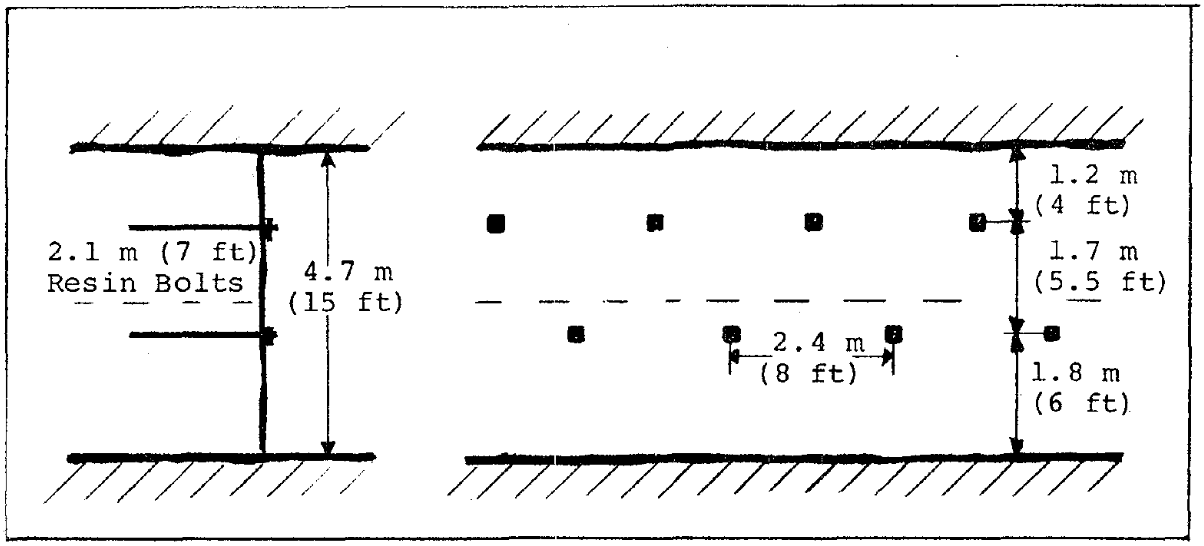


Figure 5.17. Face Bolting 15 ft. Seam with Resin Bolts

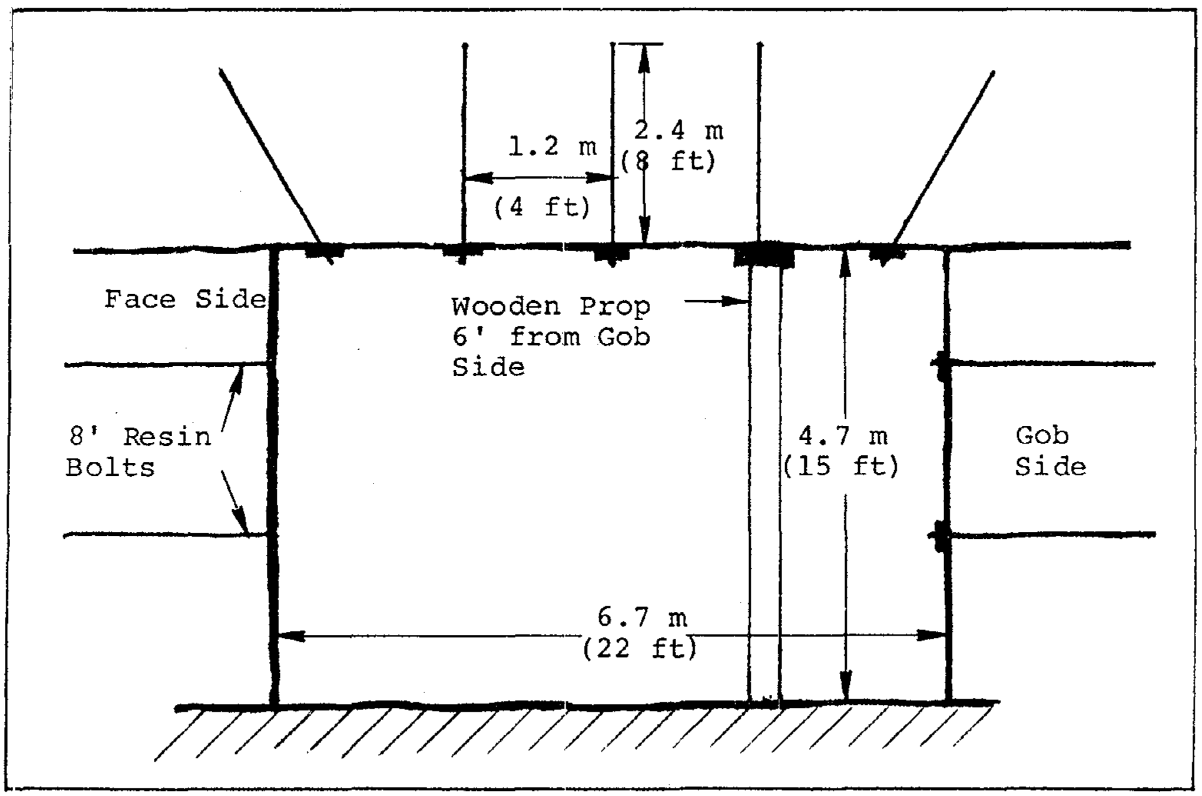


Figure 5.18. Installation Area Developed at a 22 ft. Minimum Width and a 15 ft. Height. Face bolting can vary depending on strength of coal and tendency to spall.

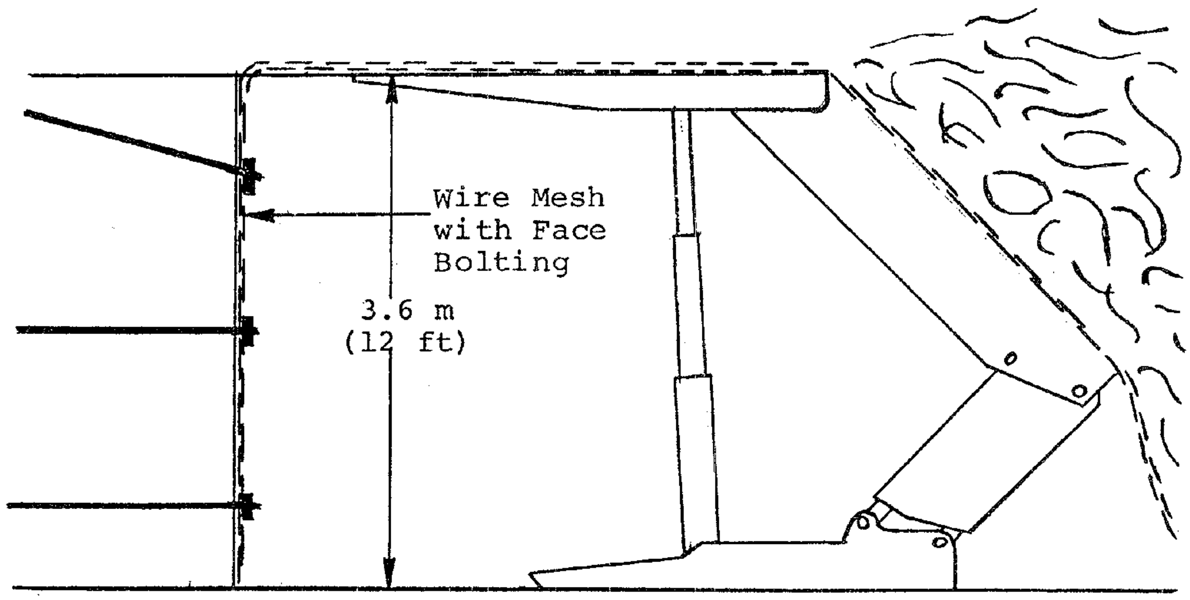


Figure 5.19. Face Withdrawal Preparation with Wire Mesh and Face Bolting. Face height reduced to 3.6 m (12 ft)

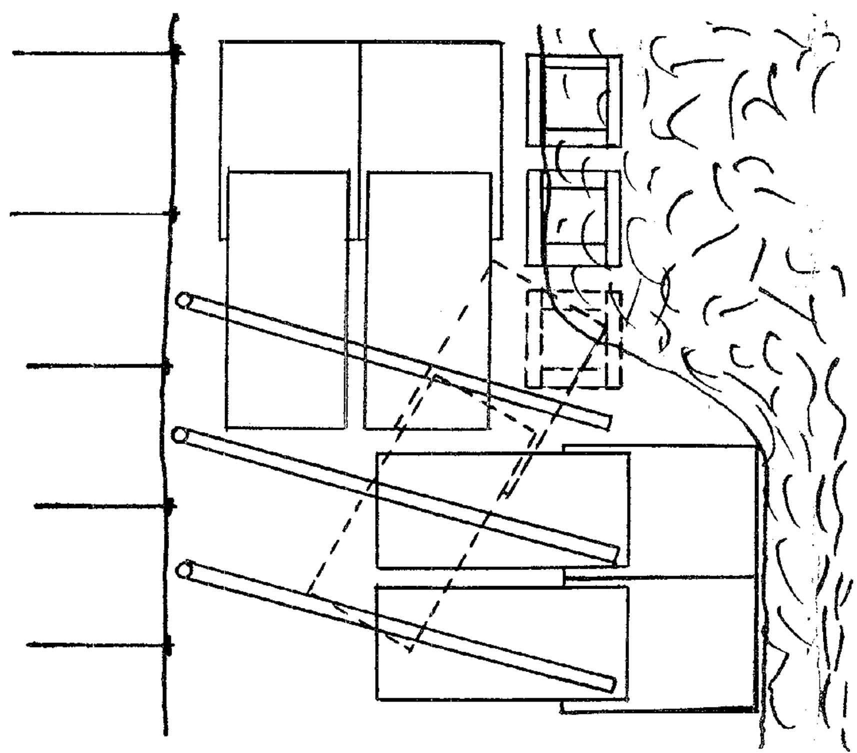


Figure 5.20. Shield Removal Sequence with Face Props and Beams. As one shield is removed a crib is set.

5.3.2 Mining Conditions at Potential Sites

5.3.2.1 Colorado Westmoreland Inc.

Coalfield Site Data:

The face equipment will be installed at Orchard Valley Mine of Colorado Westmoreland Inc., Paonia, Colorado. The coal seam to be mined is the D seam. The thickness varies from 1.5 to 7.3 m (5 to 24 ft.). The immediate roof consists of sandy shale, the upper roof of sandstone. The caving in mined areas occurred regularly when pillars were extracted. The floor consists of shale.

Compressive strength of the various strata are as follows:

- Coal: 3,200 to 3,500 psi
- Roof: 4,000 to 7,000 psi for sandy shale, 12,000 psi for sandstone
- Floor: 4,000 to 7,000 psi
- Face Length

The length of the face is 165 m (540 ft.), and is equal to the width of the worked panel plus twice 6 m (20 ft.) for the two adjacent entries.

- Seam Thickness

The worked thickness of the seam will be from 3.7 to 5.5 m (12 to 18 ft.). When the seam thickness is higher than 5.5 m (18 ft.), the bottom coal will be left.

- Gradient

The general gradient of the face will be negligible. The face will be rising at 4 degrees.

- Roadways

Entries to the face at each end consist of two 6 m (20 ft.) wide, 3.7 m (12 ft.) high roadways driven under .9 m (3 ft.) of top coal with 18.2 x 18.2 m (60 x 60 ft.) pillars. The pillars will not be mined with the face. The coal panel between entries will be 152 m (500 ft.) only.

5.3.2.2 Southern Utah Fuel Co. (SUFCo)

Coalfield Site Data

The face equipment will be installed at SUFCo Mine No. 1, Salina, Utah. The coal seam to be mined is the Upper Hiawatha.

The thickness varies from 3.4 to 5.8 m (11 to 19 ft.). The immediate roof consists of carbonaceous shale and shale-siltstone above. The top coal (0.6 to 0.9 m (2 to 3 ft.)) is left in the entries. The upper roof is of sandstone. The caving in the mined area occurred regularly when the pillars were extracted. The floor consists of hard shale.

Compressive strength of various strata are as follows:

- Coal: Is very hard with an average of 3,800 psi. Hardgrove grindability is 44.

- Roof: 7,825 to 21,000 psi in the first 5 feet

- Floor: 94,000 - 84,000 psi

- Face Length

The length of the face will be 165 m (540 ft.) and is equal to the width of the worked panel plus twice 6 m (20 ft.) for the two adjacent entries.

- Seam Thickness

The worked thickness of the seam will range from 3.7 to 5.5 m (12 to 18 ft.). When the seam thickness is higher than 5.5 m (18 ft.), the bottom coal will be left.

- Gradient

The gradient of the face will be negligible (2°).

- Roadways

Entries to the face at each end consist of two 6 m (20 ft.) wide, 3 m (10 ft.) high roadways, with 9 x 30 m (30 x 100 ft.) pillars. The pillars will not be mined with the face. Coal panel between entries will be 152 m (500 ft.) only.

A sample of the general conditions stated in a purchase contract for equipment and accessories is presented in Appendix D. Specifications for all recommended equipment are detailed in the following subsections.

5.3.3 Support System Specification

Location of the Support System

The support system covers the length of the face and also the 40 feet of bottom and top roads. It includes a certain quantity of standard supports and some supports modified to suit the ends. An additional support system shall also be supplied to reinforce the support of the entry 60 feet ahead of the face.

Operating Method

Prior to winning operation, rams are fully extended by a length which is equal to the cut and the armored conveyor is close to the face. The loading ramp edge is six inches from the face, placing the forward end of the support canopy at 12 inches maximum from the face.

When the top coal is cut, the support is immediately advanced close to the armored conveyor by a length equal to the cut. The canopy forward end is then again at 12 inches maximum from the face if good control of the shearer has been provided.

In this position,

- The man-travelway shall be designed inside the support behind the first row of props. The clearance between support and spill plates shall allow to reach hoses and electric wires supported by the spill plates, i.e., 8 inches.

- The traveling track width in the support shall not be less than 2 feet.

- The access to the shearer controls shall be possible from the travelway inside the support. Support advance by one team shall progress at a speed sufficient to follow the shearer cutting (at a speed of 20 feet per minute). If e is the distance in feet between support center lines, the time required for advancing a support shall be less than e/20 minutes.

Pump delivery and hose sizes shall be designed in accordance. After the bottom coal has been extracted, the armored conveyor will be advanced by snaking it behind the shearer loader, full advance being achieved 60 feet after the shearer.

General Support Data

The supports must comply with the following requirement:

Every unit of walking support will cover a face length of 2 to 2.25 m (6.6 to 7.4 ft.) and will present the following characteristics:

- A floor base in two parts

- One single-piece canopy

- Two hydraulically telescopic legs

- One reverse advancing ram
- Operating cylinders
- One hydraulic system including an adjacent control unit

Connecting devices of legs, advancing ram, operating cylinders and various other components shall be simple, strong, easily accessible and removable. The use of screws and bolts shall be as reduced as possible. Each shield will be equipped with a luminaire to provide the required illumination and minimize discomfort glare. The Statement of Test and Evaluation should allow the fluctuation of mined thickness to the extent allowed by the support.

Height

Supports will be used in a mined thickness from 3.7 to 5.5 m (12 to 18 ft.). They shall have the following height requirements (Figure 5.11):

Closed height: 2.1 to 2.4 m (7 to 8 ft.)

Open height: 6 m (20 ft.) (0.6 m (2 ft.) more than the shearer range)

Load Density

The load density of the support shall always be greater than 6 tons per square foot of supported roof (58.6 metric t/m^2). The surface of supported roof is equal to the product of:

- The distance between support center lines
- By the distance from coal face to the canopy rear edge

When the support is ready to be advanced after the upper cut, the support load density to be considered is the load density calculated with the nominal yielding pressure.

Base

The base shall be calculated to remain within the elastic limit under the yielding load, whatever the contact points on the floor may be. It shall have an adequate surface so that the pressure on the floor under yield load is less than 300 psi. The base will be equipped in its forward part with a very simple device enabling the support to be recentered when advanced against the armored conveyor.

Canopy

The canopy will be of the single piece type. It shall be calculated to remain within the elastic limit, whatever the contact points on the roof may be under the yielding load and it shall successfully fulfill the strength tests.

Rear Shield

To take into account the width increase and mined thickness, the stroke for the side shield will be $\pm$ 25.4 cm (10 in) (50.8 cm or 20 in. total). The side shield will be hydraulically activated by 2 jacks.

Forward Face Sprag

The forward end of the canopy will be equipped with a face sprag plate as protection against the possible fall of coal from the face. The main purpose is to hold the coal face. It should be operated by a cylinder, the stroke of which shall be sufficient to insure that the plate of the face sprag can be applied flat on the face. In the retracted position, it shall be mechanically locked by itself so that it does not interfere with the shearer, whatever the mined thickness.

Legs

Legs shall be double telescopic cylinders yielding under a constant load.

- The closed and open heights, 2.1 to 2.4 m and 6 m (6 to 7 ft. and 20 ft.), shall be reached by hydraulic or mechanical means.

- The load per square foot on setting and yielding load will be designed as requested in the paragraph on load density.

Legs will be of the double acting type, corrosion- and rust-proofed. The quality and thickness of the plating shall be stated. All the materials used shall be of first quality. Their main mechanical characteristic data shall be indicated by ASTM standards. For a pressure equal to 1.5 times the nominal yield pressure, the tubes used shall still remain within the elastic limit.

Removing legs out of the support in an operating face shall be simple and possible at any mined thickness in the operating range 14-20 feet.

Advancing Ram

The advancing ram will have the following data:

- Pull to advance support: above 3 times the support weight.
- Push on the conveyor: comprised between 15 and 20 U. S. tons.
- Stroke: 101.6 cm (40 in.).

When the armored conveyor is advanced from the shield, it will be possible to move it by:

$\pm$ 10.2 cm (4 in.) minimum along the face

$\pm$ 30.5 cm (12 in.) minimum vertically

On design, provision shall be made against packing of the advancing system by coal fines so that the above flexibility is kept.

The method for removing rams out of the support shall be simple.

Hydraulic Arrangement

The hydraulic control shall be so arranged as to enable the operator to securely achieve the operations from the adjacent shield. There shall be only one feed pressure comprised between 4,300 and 5,000 psi to feed the supports. The control valve shall be of the dead man's handle type which means that control level will automatically be brought back to neutral point. Bidirectional control may be used. However, it is expected the support will be moved successively in the direction of the ventilation only to minimize dust exposure.

Legs shall be fed through pilot-operated, non-return valves.

The data of the bleed valves used shall be indicated, as well as the approval certificates obtained from national testing laboratories. Interval of replacement shall be given.

The hoses used shall be to S.A.E. regulations and in particular:

Return Hose - two steel-ply hoses

Pressure Hose - heavy 4-steel-ply hoses

Nipples shall be of a quick-connection type.

Pressure Indicator

Each leg or group of legs will be equipped with a pressure indicator. Three 24-hour reliable pressure recorders will record the internal pressure of a leg on three shields of the face.

Face End Support

Special side shields or sprags will be designed for the two shields which are set at the face ends to hold the rib of the gate entry.

Auxiliary Gate Road Supports

100 hydraulic props will be designated for use in the entries. These props will probably be of the closed circuit type but the open circuit type may be quoted. The height range should be 2.1 to 3.7 m (7 to 12 ft.) and the yield load should not be less than 25 tons. The total weight of the individual prop must be stated in the quotation since portability is an important selection criterion.

Water Sprays

Each canopy will be fitted with at least one water spray oriented at 15 degrees on the face line in the direction of the air flow to form a water curtain dividing the air flow in two streams. This will prevent the dusty air stream along the face from spreading to the entire face area.

Each shield will be fitted with a water spray system designed to control dust from the caving.

5.3.4 Shearer Specification

General Data

The shearer shall be a double-ended ranging drum shearer equipped with chainless haulage for bi-directional cutting. It shall be able to cut a 3.7 to 5.5 m (12 to 18 ft.) thick coal seam with 2 m (6.6 ft.) diameter drums. It shall be possible to lower the drums to obtain a 25.4 cm (10 in.) undercut (Figure 5-14).

The shearer must be in compliance with Mining Enforcement Safety Administration and United States Bureau of Mine standards and regulations.

Recommended Technical Data

Power: Minimum 500 kw

Drums: Diameter 2 m (79 in.)
Width 1 m (39 in.)

	2 or 3 helical vanes: loading capacity calculations to be supplied by manufacturer. Hydraulically or mechanically operated cowls. The drum will be fitted with internal water supply and sprays.
Haulage:	Pulling force: 30 U. S. tons at a haulage speed of 5.8 m (19 ft.) per minute
Chainless Haulage:	The track system should leave the possibility of snaking the conveyor by 1 m (3.3 ft.) in 15.2 m (50 ft.) and to accept changes of gradient of ± 3 degrees at each connection.
Drum Speed:	30 rpm maximum
Steering Underframe:	The underframe shall be high enough to enable the coal produced on the tail gate side to easily pass under the shearer. The underframe shall be hydraulically inclinable by ± 6 degrees in a plane perpendicular to the face. If necessary, because of the rigidity of chainless haulage system, the haulage unit will be part of the underframe. A minimum of 30.5 cm (12 in.) clearance is necessary between the shearer top and the support canopies at the minimum mined thickness. Underframe guiding will be through guide shoes sliding on guides fastened along the gob side of the armored conveyor. Face side of the shearer will be supported by roller shoes rolling on the end of the ramp plates.

Electric Motors

The shearer will be fitted with one or two electric motors and fed by a 950 volt 60 Hz AC supply. These motors shall be flame-proof and shall comply with Mining Enforcement Safety Administration and United States Bureau of Mines standards and regulations.

Auto-Control

The shearer shall be fitted with an auto control enabling the optimization of the shearer operations.

Control Stations

The shearer shall be remote controlled with one manual control station placed in the center of the shearer. A system recording the displacement of the shearer along the face will be provided.

Electric Feeding

The shearer will be fed through one or two electric cables protected in a cable-handler together with one or two water hoses.

Dust Control

- The water supply hose and water distribution on the shearer will be calculated on the basis of a 1000 ton/hour production and an added moisture content of 3%, 30 m^3/hr or 130 gallons/minute.

- Provision will then be made to open and shut off water spraying when production starts and stops by using a pilot operated valve or a pump. A variable delivery pump or another equivalent system controlled by the shearer loader power (amps) will vary the water supply in accordance with the amount of coal produced, assumed to be related to the power consumption. The flow of water should be shut off when the shearer is moving without cutting and be maximum (3%) when both drums are in full operation.

- The sprays will be arranged to confine the dust cloud to the face or at least to avoid turbulences.

- A dust collector consisting of 5,000 to 10,000 ft^3/min fan water sprays and cyclone will be installed on the shearer.

5.3.5 Face Conveyor

The face conveyor must be capable of carrying 1200 tons per hour on a level grade. Calculations justifying the capacity should be shown. All items of the conveyor shall be of robust construction and all connections between the various pieces of equipment shall be engineered to have sufficient strength to ensure maximum performance and life. Areas subject to heavy abrasion shall be specially treated. The special treatments should be specified.

Length

The conveyor must be of sufficient length to allow the installation of the tail drive at the face end and the installation of the side discharge at the entry center.

Pan Length

The pan length will be increased to 2 m or 2.25 m (6.6 or 7.4 ft.) to match the width of the shield support.

Conveyor Carrying Frame

The conveyor carrying frame will be designed in conformity with the conveyor carrying frame developed in Saar (West Germany) and used with success at Warndt Mine (Figure 5-21). Its length will be equal to the pan length.

Chain

It will be of the double center strand chain type. The double strand of chain will be made up of matched length of chain and the link size will be 30 x 108 mm and should be supplied in 50 m (164 ft.) lengths for ease of handling in the mine. The breaking strength of the chain should be in excess of 90 tons for each strand of the chain. Scraper bars must be provided and must be securely bolted to the chains. Other fastening arrangements, such as using roll pins, are not acceptable. It is anticipated that the scraper bars will be spaced every 10 links or 1.08 m (3.5 ft.) apart.

It is preferred that the chain speed be kept as low as practical consistent with the tonnage requirement, flite bar spacing, etc., but should not exceed 1.2 m/sec. (240 ft./min.).

Special Pan Joints

The face conveyor (AFC) could be operated upon a longwall in which the entries will be 2.4 to 3 m (8 to 10 ft.) high but the coal extracted from the face will be 4.9 to 5.5 m (16 to 18 ft.) high. If mining conditions lead to the establishment of entries in the upper section of the seam, it will then be necessary to grade down from the headgate entry along the face to achieve the required mining height.

It is specifically requested that the first pan joints, including the joints between the connection rerail pan and any intermediate or special pans, be designed and constructed to allow the conveyor to flex in the vertical plane sufficiently so that the panline can accommodate the vertical curves produced when dinting from an elevation of 2.4 m (8 ft.) below the seam roof to 4.9 m (16 ft.) below the seam roof over a reasonable distance of about 15 m (50 ft.). Some allowance in the design should be made for inexperienced operator error, and special attention must be given to zones of high wear.

The purpose of the special joints is to avoid the destruction of the pans and joints at the transition points due to the

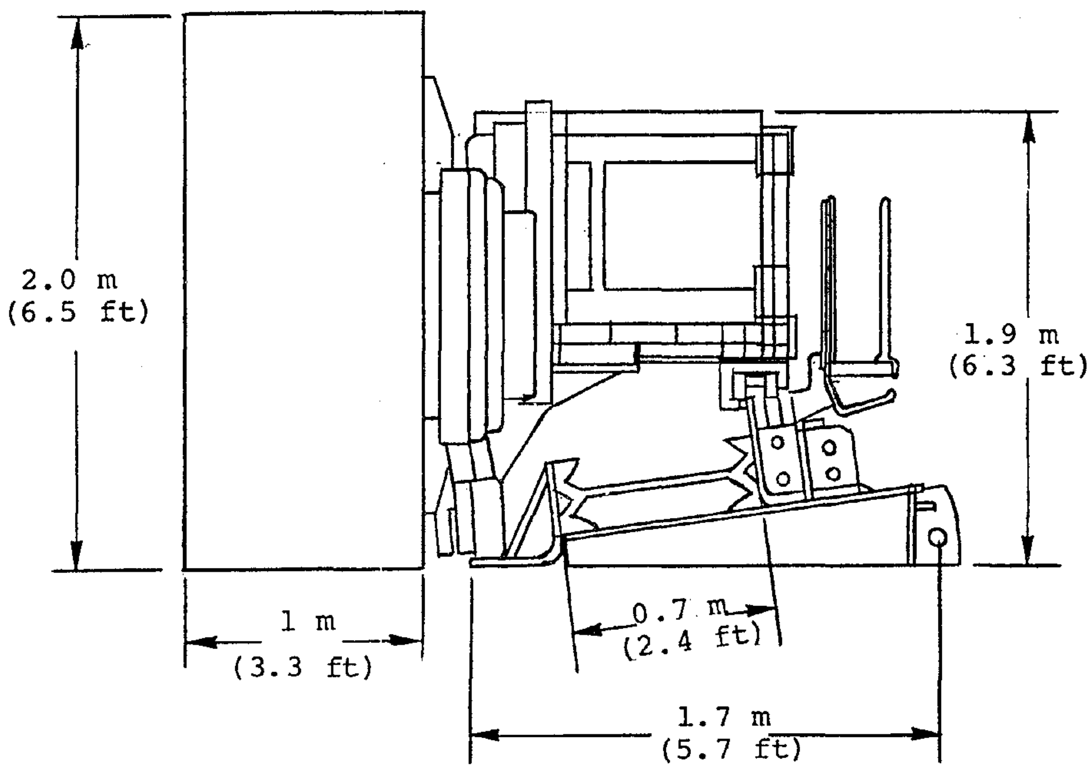


Figure 5-21. Chainless Haulage Conveyor with Carrying Frame

weight of the shearer on an unsupported length of conveyor. Until operator skill is developed, it is likely that the grading operation will be less than totally satisfactory. Please note that it is not intended to develop the face at 4.9 to 5.5 m (16 to 18 ft.) high but to gradually mine down to that height from a starting height of 3 to 3.7 m (10 to 12 ft.).

At the tail end of the face, it is intended to cut into the tailgate approximately 0.6 to 0.9 m (2 to 3 ft.) with the base of the conveyor being kept on a level plane that is 4.9 to 5.5 m (16 to 18 ft.) below the roof. The gate road will be supported upon wooden cribs and hydraulic props. Any remaining coal in the seam will be left in the floor.

Delivery onto Stage Loader

In order to deal with large lumps of coal which will be produced on this face, it is required that the delivery from AFC to stage loader be of the side discharge type and the designed clearances for the passage of coal be as large as practical bearing in mind that the gate road can be 2.4 m (8 ft.) high and 4.9 to 5.5 m (16 to 18 ft.) wide.

Connection of AFC to Stage Loader

It is intended that the stage loader will be retreated during each pushover of the headgate drive. Therefore, the stage loader must be securely connected to the AFC in a manner which will withstand the forces involved in pushing the stage loader, including a coal crusher and belt-tailpiece. The crusher and tailpiece will be addressed later.

The connection must also allow relative horizontal movement such as is required when the face needs to be slewed to allow for AFC creep.

Chain Tensioning

A chain tensioning device must be supplied. The device must be permanently installed on the headgate drive unit and should take as little space as practicable; protrusions of machine parts or hydraulic hoses should be minimized to prevent breakage.

A type of chain tensioner powered by the AFC motor is preferred. Separate hydraulic rams or other devices are not acceptable.

Spillplates

Conveyor spillplates must be provided and these must be not less than 1.2 m (4 ft.) high from the mining floor. The spillplates proposed must be of strong construction so that they will not be damaged in the event that coal rolls off the face to strike the spillplate.

The extra height of the spillplate is required to give an extra measure of safety to the personnel working on the face and to keep the travelway clean of mined coal and thus minimize wastage of coal.

The spillplate-to-spillplate connection must be designed so that there will be no gaps or spaces, which allow the wastage of coal, even when the conveyor is flexed in its extreme positions both horizontally and vertically.

The spillplates will also be designed so that they carry other services, namely:

- a 2-1/2" diameter rockdust line, from headgate to tailgate, for rock dusting tailgate
- cable to power tailgate motor
- shearer water hose to mid point of face
- shearer cable to junction base at mid point of face
- shearer trailing cable and 2" diameter water hose from mid point to both ends of the face.

Cable Handling Device

It is strongly recommended to use a Bretby cable handler of the size required to handle a 2" water hose.

Shearer Haulage System

The conveyor will also be arranged to carry the track elements of the shearer haulage system chosen.

Conveyor Motors

The conveyor will be powered by two (2) 250 kw motors, one each located at each end of the face and both on the gob side of the conveyor. They will be oriented lengthwise with the face conveyor and will transmit power to the conveyor chain through a fluid coupling and bevel spur gearbox. Both the fluid couplings and the gearboxes shall be of adequate size and capability to withstand the 250 kw transmission plus normal overloads to ensure economic durability of the equipment.

The motors must be squirrel cage induction motors, totally enclosed, fan cooled, explosion proof, and approved for use in coal mines.

Tail-End Spillplate

A special spillplate will be required which, when bolted to the tailgate drive frame, would overlap the tailgate edge of the last shield. The purpose of this plate is to prevent flushing between the last shield and the conveyor drive. The plate should be 1.8 m (6 ft.) high and 2.5 cm (1 in.) thick and suitably gussetted to maintain rigidity. The length of the plate and the location of the gussets will depend upon the general configuration of the shields in relationship with the conveyor drive unit.

Attachment of Shields to Conveyor

These attachments and attachment points must be given special attention during the design stage to ensure that the large forces involved in moving over the shields do not damage the conveyor or its furnishings. It will be considered to pull the shields from an attachment on the face side of the conveyor using the frame supporting the conveyor.

Cable Routing

It is intended that the cables and hoses will be taken onto the face from the loader but not on the longwall side of the stage loader. Therefore, no provision is needed under the main drive, although protective pipe work should be provided around the stage loader tailpiece in the form of a split pipe with the halves bolted together.

Assembly Supervision

The supplier will provide sufficient skilled personnel who have adequate knowledge of the equipment. Such skilled personnel will be provided, free of extra charge, to cover receipt of the material to ensure that all necessary equipment is received and available, during the installation period on a two shift per day basis at the rate of one man per shift and for six weeks after the startup date on a similar two shift basis. This period is deemed necessary to ensure the proper functioning of the equipment.

If assembly supervision personnel were adequately trained and skilled, it would be acceptable for individuals to supervise more than one operation, for example, the shearer assembly supervision will most likely be proficient in the assembly and operation of the face conveyor and the stage loader.

Completeness

The AFC, when installed by mine personnel under the supervision of the suppliers representative, shall be complete in every detail and be ready to operate both functionally and safely.

5.3.6 Stage Loader Specification

The face conveyor must be connected to the tailpiece of the stage loader so that when the headgate drive is pushed over the stage loader moves outby in the same operation.

The stage loader must have a carrying capacity 20 percent greater than the face conveyor. The stage loader will be powered by a 90 kw motor through a fluid coupling and bevel spur gearbox. It will be of the double center strand chain type using 30 x 108 mm chain as used on the AFC. Scraper bar spacing should be every sixth (6th) link which is at a spacing of 64.8 cm (25.5 in.) centers. The chain speed will be in the region of 1.3 m/sec. (260 ft./min.). The stage loader drive will be equipped with a chain tensioning device identical to the one on the face conveyor drive.

The greatest part of the stage loader must not be greater than 1.5 m (5 ft.) above the mine floor. The conveyor will be horizontal with the discharge end raised to deliver onto a belt conveyor tailpiece and will be approximately 21 m (70 ft.) long.

Sufficient spillplates are required over the full length of the stage loader. The spillplates should permit no coal to escape from the conveyor and special attention is required where the spillplates and the crusher come together. From the crusher outby to within 1.5 m (5 ft.) of the delivery point should be fitted with bolted top cover plates. The top cover plates should also have provision for the installation of one spray jet every 1.5 m (5 ft.) of length of cover. This provision should be in the form of a recess in the plate as would be formed by bolting a piece of pipe to the plate with the spray head in the pipe. The purpose of the sprays is for dust suppression and the recess is to prevent damage to the sprays by the coal as it is being conveyed.

The elevated portion of the conveyor from the bottom of the gooseneck to the delivery point must use closed bottom pans.

The stage loader shall be provided with a hydraulically operated advancing unit which is designed to assist the DA rams of the shields in moving the headgate hardware and also to minimize the effect of the stage loader jack-knifing due to the weight of the drive and the crusher when it is being pushed from the tail end. This unit should be in the form of two inclined hydraulic posts, one on each side of the stage loader outby the crusher, from the bottom of which two suitably sized hydraulic rams assist in pulling the crushers, etc. The hydraulic power for the advancing unit shall be from the same pumps which power the shields.

It is anticipated that the hydraulic advancing unit will have the following operating characteristics:

- o Hydraulic props inclined
- o Setting load 630 kN at 315 bar
- o Nominal load 666 kN at 330 bar
- o Stroke 1000 mm
- o Advancing cylinder
- o Pushing force 480 kN at 315 bar
- o Pulling force 237 kN at 315 bar
- o Stroke 1600 mm
- o Pushing medium - water/oil emulsion
- o Operating pressure 315 bar

Provision should be made before the conveyor starts to rise off the mine floor for a floor mounted crusher or breaker. This breaker will be approximately 3 m (10 ft.) long.

Stage Loader Crusher

A coal crusher is an integral part of this package and must be capable of sizing coal to minus 15.2 cm (6 in.) with a throughput of 1500 tons per hour. The crusher should be floor mounted and should be so arranged as to not impede the moving of the stage loader during the face retreating process. The crusher will be fitted with adequate guards to prevent injury to persons by the crusher itself and by materials being projected from the crusher due to the action of the crushing elements.

It will be powered by a 75 kw motor through a bevel spur gearbox and will be fitted with water sprays for dust suppression. The water sprays shall be located and positioned in a manner that the venturi effect of the water movement opposes the effect of the crusher's tendency to cause airflow and therefore dust dispersal.

Completeness

The stage loader, hydraulic moving unit and crusher, when installed by mine labor under the supervision of the suppliers representative, shall be complete in every detail and be ready to operate both functionally and safely.

Gate Belt Tailpiece

The belt tailpiece must be designed for use with a 54 inch wide belt and must be equipped with skirt boards and with a plow on the bottom belt. It will be coupled to the stage loader by means of a turntable and will be approximately 3.7 m (12 ft.) long. The unit will be mounted upon rubber tires and will be steerable by means of hydraulic cylinders with mechanical linkages. The hydraulic power will be provided by a hydraulic pump connected through gearing and chain drive to one of the bottom idlers in the tailpiece.

The top idlers must be of the impact absorbing type and must be troughed at 35 degrees. Top idler spacing in the area where the coal comes in contact with the belt must not exceed 30.5 cm (12 in.) centers.

Belt Storage Unit

A belt storage unit will be used to automatically take up slack belt whenever the belt tailpiece is retreated. This unit will be hydraulically powered to a predetermined tension and will move a trapped cluster of rollers along a track.

It is intended that the storage unit have a capacity for 152 m (500 ft.) of belt.

5.3.7 Electrical System Specification

The mine supply voltage will be 7200 volts, 3 phase, 60 Hertz and the longwall electrical distribution system must be designed to power the following loads,

500 kw - Shearer

500 kw - Face Conveyor in 2 motors of 250 kw each

90 kw - Stage Loader

75 kw - Crusher

120 kw - Hydraulic Pumps in 2 motors of 60 kw each

all at nominal 1000 volts, 3 phase, 60 Hertz, together with adequate spare outlets and twelve outlets at 480 volts as specified below.

The electrical supplier will provide Certification Numbers with dates of certification for all component parts of the proposed equipment. Any equipment whose certification is pending will not be considered for purchase in this package.

The electrical supplier shall provide layout drawings and detailed specifications of the electrical system proposed.

The electrical supplier shall, during the period of electrical equipment installation, conduct all necessary examinations and tests to ensure compliance with the Code of Federal Regulations, Title 30, Part 18, and will ensure that all electrical protective devices perform to the design specification and that the installation is as specified in the layout drawings and specifications.

All electrical equipment supplied under this contract, including cable, which is required to have approval plates, must carry the required designation prior to its arrival at the mine site.

All electrical equipment which is designed to be used or may be used in the last open crosscut or inby of the last open crosscut must be "permissible" or "intrinsically safe" as defined by the regulating agencies. Ease of maintenance of "permissibility" is an important factor.

In addition to the above electrical systems, ancillary equipment must also be supplied. This ancillary equipment is to include, but not be limited to, the following:

- o lighting system with all controls
- o communications and signaling
- o sequence control of device(s)
- o methane monitoring device(s)
- o master control system(s) with meters
- o pre-start warning system

The electrical system outlined above and described further below must be complete in every detail and, when installed under the direction of the supplier, must be ready to operate functionally, safely, and within the requirements of the laws and regulations governing the coal mine.

The Power Center

This unit shall be designed for underground coal mine duty. This unit shall have a structural framework to which side and top covers are attached. Unit base plate shall be a minimum 1/2 inch thick steel, side and top covers 7 gauge steel and end plates 7 gauge steel. Lifting and pulling eyes shall be supplied on all four corners of the enclosure. A drawbar box and pin, A-Frame type, shall be supplied on the center of each end of the enclosure. A 25.4 cm (10 in.) channel shall be supplied continuously welded on its edge to the base plate around the perimeter of the unit to provide the equivalent of a watertight seal to the top of the channel.

Unit approximate overall dimensions shall be 132 cm (52 in.) in height, 182.9 cm (72 in.) in width, and 7 m (23 ft.) in length.

The unit should be skid-mounted with suitable towing hooks on each end.

A 1900 KVA power center is required. The primary voltage will be 7200 volts, 3 phase, 60 Hertz.

The secondary output will have 1600 KVA capacity at 1040 volts, 3 phase, 60 Hertz and 300 KVA capacity at 480 volts, 3 phase, 60 Hertz with resistance grounded neutrals for each voltage. A 10 KVA lighting transformer will be required, the secondary voltage of which will be dependent upon the requirements of the lighting system chosen.

The power center will be provided with a suitable plug and socket with strain relief clamp suitable for the mine primary cable.

The unit will have a loadbreak fused disconnect switch to protect the transformer from single phasing.

Enough space will be provided in the unit for stand off insulators, terminals and termination of the primary mine cable.

An emergency stop button will be provided together with an adequate guard to prevent inadvertent operation. The operation of the emergency stop button will be arranged that the incoming pilot circuit is interrupted thus switching off the power to the secondary outlets, or equal protection.

All cover plates which are capable of being removed and which provide access to high voltages shall be interlocked to the ground monitoring on pilot circuits to prevent accidents to personnel.

Sufficient protection shall be provided to prevent damage to the transformer by lightning surges.

Every effort will be made to ensure that the minimum impedance is achieved. It is preferred that the impedance should not exceed 5% on either the 1040 volt or 480 volt systems.

Neutral grounding resistors with a continuous rating capacity of 15 amps shall be used on both the 1040 volt and the 480 volt windings.

<u>The 1040 Volt System</u>

Eight outgoing 1040 volt 400 amp circuits will be provided and will have nameplates: Face Conveyor #1, Face Conveyor #2, Face Conveyor #3, Stage Loader #1, Crusher #1, Crusher #2, Shearer #1, and Shearer #2. It is anticipated that the stage loader could be run from the spare crusher outlet and vice versa.

Each of the 1040 volt outlets shall be provided with the following:

- One (1) three-pole, 1000 volt circuit breaker with 800 amp thermal and 2000-4000 amp adjustable magnetic trip, and 120 volt undervoltage release.

- Ground monitor circuit with 1000 volt machine filter included. This to be acceptable to MSHA.

- Instantaneous ground trip system.

- Output gear mounted receptacle for 4/0-3/C cable, and with dust cover.

<u>The 480 Volt System</u>

Twelve outgoing 480 volt output circuits will be provided as specified below.

- Three (3) 400 amp breakers each with an adjustable setting magnetic trip range of 800-1600 amps with 120 V undervoltage release and ground monitors. Nameplates "Hydraulic Pump #1," "Hydraulic Pump #2," and "Hydraulic Pump #3" must be affixed in a noticeable location.

- One (1) 225 amp breaker with an adjustable setting magnetic trip range of 500-1000 amps. This unit shall include a 120 volt undervoltage release and a ground monitoring must be provided.

- Six (6) 100 amp breakers, each with adjustable setting magnetic trips of range 66-190 amps. Undervoltage release and ground monitoring must be provided.

Adequate protection must be provided to minimize the possibility of damage to protruding power outlets and inlets during the frequent moves of this unit within the mine. It is anticipated that the power center will be moved three to five hundred feet every two weeks over rough terrain and around two 90 degree corners.

<u>The Master Control Panel</u>

The master control panel will be a structure built on the side of the stage loader and should be designed to be not more than 60 inches high or 15 inches wide. The length of the unit is not critical provided it is not more than 30 m (100 ft.) long.

It must provide the following features:

1. Foot mounted
2. Selector switch 3 phase for shearer feed cable
3. Voltmeter shearer feed cable
4. Selector switch 3 phase for conveyor feed cable
5. Voltmeter conveyor feed cable
6. Selector switch 3 phase for stage loader feed cable
7. Voltmeter stage loader feed cable
8. Ammeter shearer
9. Ammeter stage loader
10. Ammeter tail face conveyor motor
11. Ammeter head face conveyor motor
12. Emergency stop pushbutton
13. Crusher start pushbutton
14. Crusher reverse pushbutton
15. Crusher stop pushbutton
16. Crusher run indication light
17. P.T.O. on/off selector switch
18. Shearer reset pushbutton
19. Shearer stop pushbutton
20. Shearer lockout switch
21. Shearer reset light
22. Shearer run light
23. Elapse time meter
24. Sequence by-pass pushbutton
25. Auto/Mass/Off conveyor selector switch
26. Stage loader stop pushbutton
27. Stage loader forward pushbutton
28. Stage loader reverse pushbutton
29. Stage loader run light
30. Face conveyor head/tail/both selector switch
31. Face conveyor stop pushbutton
32. Face conveyor forward pushbutton
33. Face conveyor reverse pushbutton
34. Face conveyor head run indication light
35. Face conveyor tail run indication light
36. Face conveyor elapse time meter
37. Pump #1/#2/both selector switch
38. Pump run pushbutton
39. Pump stop pushbutton
40. Pump run lights #1, #2
41. Pump reset pushbutton
42. Pump reset light
43. Provision for connection of methane monitor
44. Methane monitor control relay
45. Provision for connection of communiations system. Type to be determined later, upon selection of communications system.

The Communications System

The communications system should be provided which will perform the following functions.

Pre-Start Warning Device for Stage Loader and Face Conveyor

This device must generate a warning tone whenever the start switch is put into the "on" or "run" position. The warning tone must be sufficiently loud and distinctive that it can be heard over any other noises generated around the longwall face area. It must be able to be heard for the full length of each conveyor protected.

The starting of the conveyor must be delayed to a sufficient amount of time to enable personnel to remove themselves from dangerous positions. The delay must not be so long as to instill a lack of urgency in the attitude of the miner to move into a safe position. If the pre-start warning device is battery operated, then it must ensure that the conveyor is inoperable if the batteries are in a discharged state.

Conveyor Latch Out Device

The provision of a means of remotely preventing the operation of a machine is required. This is to enable personnel to work on sections of the conveyor distant from the main controls without the possibility of an inadvertent start of the machine. An indication of the location of the "lockout" should be available on the master control station.

Audio Communications

A paging type audio system is required and must not be capable of being connected to the general mine telephone system.

Signaling System

A pushbutton type signaling system must be provided for two-way communications from the main control station to the furthest extent of both the stage loader and the face conveyor.

System Testing

The communications system may have various features which interact with each other but the condition of the various features should be displayed at the master control station.

Methane Monitor

Not less than three (3) methane monitoring units shall be provided and these will be located, one on the second last support at the tailgate end of the face, one on the second support on the headgate end of the face and one on or close to the stage loader drive.

The methane monitor shall be designed and installed so that it will give a continuous indication at the main control that it is in working order and also indicate the percentage of methane present in the airstream at the area of the monitor head.

In the event of the monitor not being in proper working condition, it should then make the electrical controls for the system inoperable.

When that percentage of methane reaches two (2) percent, the monitor should be arranged to cut the power from the whole system irrespective of which monitor head sensed the 2% methane.

The methane monitor proposed must be robust enough to withstand the normal vibrations on machinery such as the stage loader.

Cables

The equipment shall include all control and lighting cables and the necessary conduit, fittings, glands, etc., which are necessary to suit the control, the lighting and communications systems which are proposed and their specifications and lengths should be enumerated in the proposal. All cables used must carry the approval of the appropriate regulating authority.

Sequence Control

Sequence control must be provided between the gate conveyor and the stage loader, and the stage loader and the face conveyor. The stage loader crusher should also be automatically started before the stage loader conveyor.

Lighting System

The face lighting system proposed must be approved in advance of delivery by the regulating agencies. It is very likely that none of the lighting manufacturers will have an approved STE for equipment to be used on a longwall of 4.9 to 5.5 m (16 to 18 ft.) high. It is assumed that the supplier can obtain the STE and that the lamps do not afford too much glare. The lighting system must be easy to maintain and to disconnect for face moves. The use of permissible connector is recommended.

Supervision of Installation

The installation work will be done by mine personnel, however, the installation of the electrical system must be under the direct supervision and control of a knowledgeable representative of the manufacturer. The manufacturer's representative will be made available during receipt of the equipment at the mine site and during installation and for a maximum of six (6) weeks after start up to ensure that the customer receives a working system. The date of start up shall be deemed to be the time and date approved by MSHA Approval and Certification Center, Triadelphia.

The owner's senior representative at the mine will determine the suitability or otherwise of the manufacturer's representative.

Recommendations for the Face-to-Face Move

The equipment shall include all special equipment required for the face-to-face move including lifting and hoisting devices. Convenient anchorage points will be designed for equipment handling. Directions will be provided to mine management on how the equipment should be moved.

5.3.8 Summary

For longwall mining 16 feet in a single pass under the geological conditions prevailing in U. S. coal mines, U. S. regulations lead to the selection of a retreating longwall with double entries driven on the seam floor, 9-10 feet high. Driving in all the thickness would require a single entry system. An advancing longwall will also require a single entry system.

The double entry system includes a yielding chain of pillars to ease the roof caving when the pillars are not extracted.

When selecting equipment for longwall mining a thick seam, the following recommendations apply:

- To reduce the roof area as much as possible to be supported, therefore:

  - the shearer will be using a steering underframe to control the verticality of the cut, and

  - the total length of the support canopy should not exceed 3.5 meters.

  Support will be preferably a two-leg shield. Closeness to the coal face is preferred to a high support resistance. For the same reason, the conveyor should be mounted on a frame and the support bases advanced under it.

- To prevent face sloughing, the top bench will be cut first and the support will be advanced when the bottom coal is still uncut.

In addition, each shield will be equipped with face sprags.

- o To achieve good dust control resulting in a high rate of advance.

Face-to-face move is a major problem still aggravated by the low height of the entries. Face-to-face move guidelines should be provided by the manufacturer. The equipment required for the move should always be part of any longwall face equipment order.

The given equipment specifications also include special features, such as:

- o An increase of the shield width by 50%. That will reduce the face support cost and simplify the shield design.

- o An advanced dust control system including an automatic variable water delivery system to adapt the amount of water to the quantity of coal produced.

6.0 ECONOMIC ANALYSIS

6.1 Scenario Selection

With presently available longwall mining equipment, a 5 m (16 ft.) thick seam can be worked in two ways, (1) full face cut, or (2) two bench face cuts (Section 5.1.4). The two bench method, which is beneficial to dust control and prevents face sloughing, is recommended. However, both will be considered in the economic analysis.

Some European mine layouts suitable for thick seams are very appealing, but they require single entry development. In the United States, these methods would necessitate variances from the Code of Federal Regulations which are not likely to be approved. The development method which is suggested is a two-entry development plan, or if necessary, a three-entry plan. This results in driving entries in a height limited to 9 or 10 feet for better safety. The optimum development is a two-entry system. The economic calculations will be based on this assumption.

The economic analysis of single pass longwall mining is based on the information derived from the conceptual mine design and equipment specifications which have been recommended for application to U. S. mining conditions.

Considering that the current mining operations in the selected mines are utilizing room-and-pillar methods at a height of 9 feet, it seems reasonable to take the current results of this method as a basis for comparison with the expected results of a longwall face extracting a height of 16 feet. Longwall mining only 11 feet is another possible comparison basis.

Selected scenarios are:

Plans 1 and 2 involve mining a 16-foot seam using a longwall mining system supported by two continuous miners for main, submain, panel and face development. When development work is completed to stay ahead of the longwall, the continuous miners are used for room-and-pillar mining. In Plan 1, the shearer cuts the full thickness with a 2.18 ft. wide web, when in Plan 2, two benches are successively cut with a 3.28 ft. wide web. The speed varies in accordance, 15 ft/min for Plan 1, 20 ft/min for Plan 2 with the same production of 1250 tons/hour. Reducing the shearer speed to keep a web depth of 3.28 feet would be detrimental to dust control. Keeping the same speed with a deep web is not feasible with the power and haulage force of available shearers.

Plan 3 involves mining 9 feet of the seam by using six continuous miners on a room-and-pillar basis (current mining method).

Plan 4 involves mining 11 feet of the seam by using a longwall mining system supported by two continuous miners for main, submain, panel and face development. As in Plans 1 and 2, the continuous miners are used for room-and-pillar mining when available. In Plan 4, the shearer speed is 24 ft/min. Also, shield capital and maintenance is 20% less than in Plans 1 and 2 due to the reduced seam height.

The study requires a longwall production model, a mine design, and a mining cost analysis model.

6.2 Longwall Production Model

It is difficult to assess the importance of delays which may occur, even after careful selection of face equipment and organization. Breakdowns are reduced but not eliminated through equipment selection and preventive maintenance, and a proper face organization can considerably reduce delays due to coal lumps. For the production model, delays will be summarized in a reduction of the available time.

The face conveyor capacity is sometimes considered as a limiting factor in a production model. In this case, the shearer speed is to be in accordance with the conveyor capacity. The face equipment for this project has been selected to allow a production rate of 1250 tons/hour while mining a 16 foot thick seam. It is assumed that the shearer will maintain its speed across the face with the production slowed and stopped only at the face ends to perform the operations necessary to start a new pass (cowl and drum inversion and conveyor snaking). These operations are shown in detail on Figures 6-1a and 6-1b for the bi-directional full face cut method and on Figures 6-2a and 6-2b for the two bench method. In addition, production must be stopped for the replacement of worn bits. Bits may last several webs when cutting only coal, but this is not the case when cutting in a hard floor. A hard floor may be present in some areas of SUFCo's mine.

In most of the U. S. longwall faces presently operating, the production could also be limited by the dust standard requirements if dust control is not very efficient.

Finally, longwall production is interrupted at the end of each panel when a transfer of face equipment is required.

For the studied cases, the shift production can be calculated using either equation (1) or (2).

Full Face Cut (Bi-directional)

$$\text{Production Per Shift} = P_s = A_c \times V_s \times (SH)(WS)(D) \times \frac{(FL)}{(FL) + (ST_L) \times (VS)} \qquad (1)$$

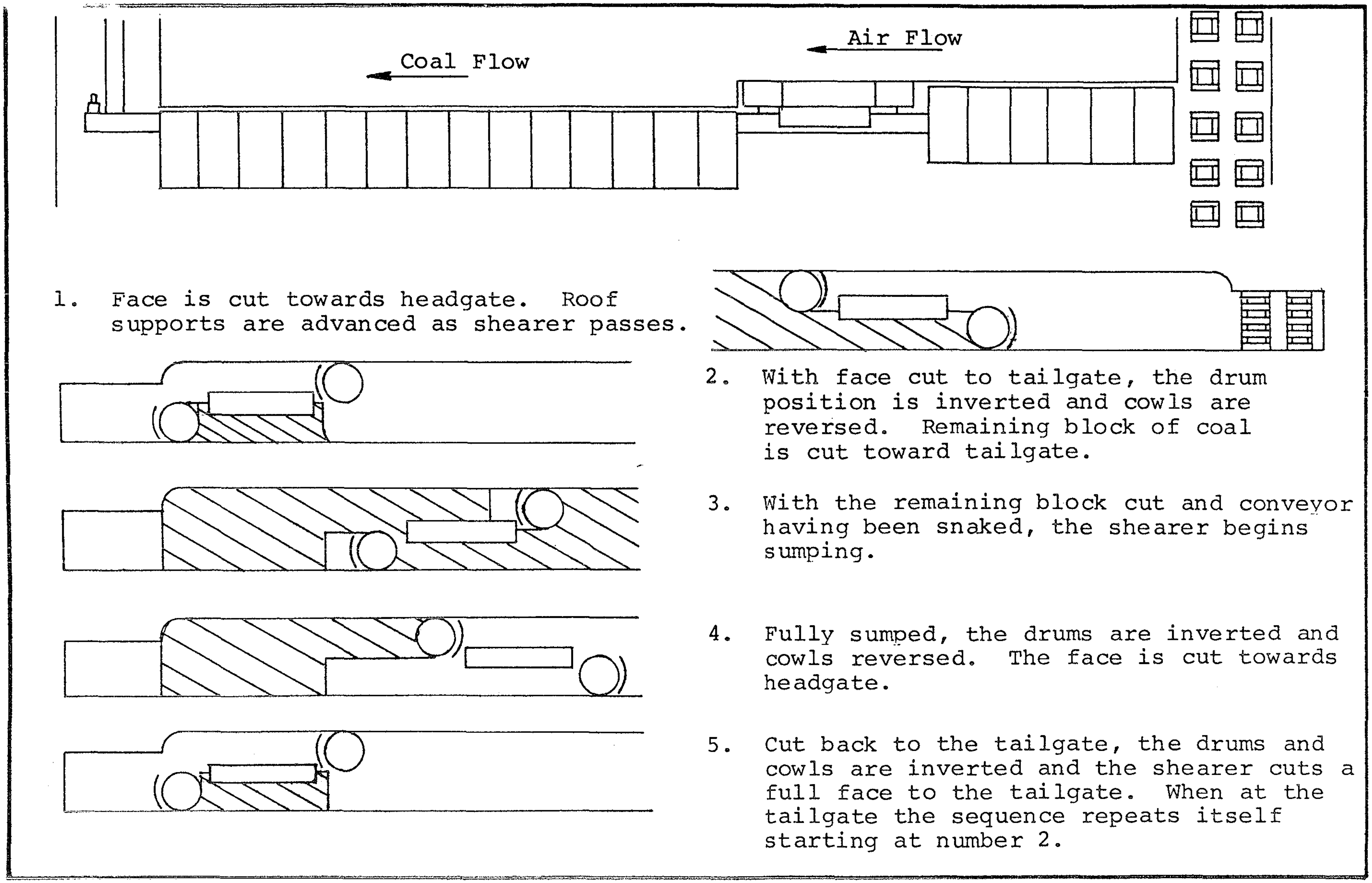


FIGURE 6-1a

Bidirectional Face Cycle with Large Diameter Drums

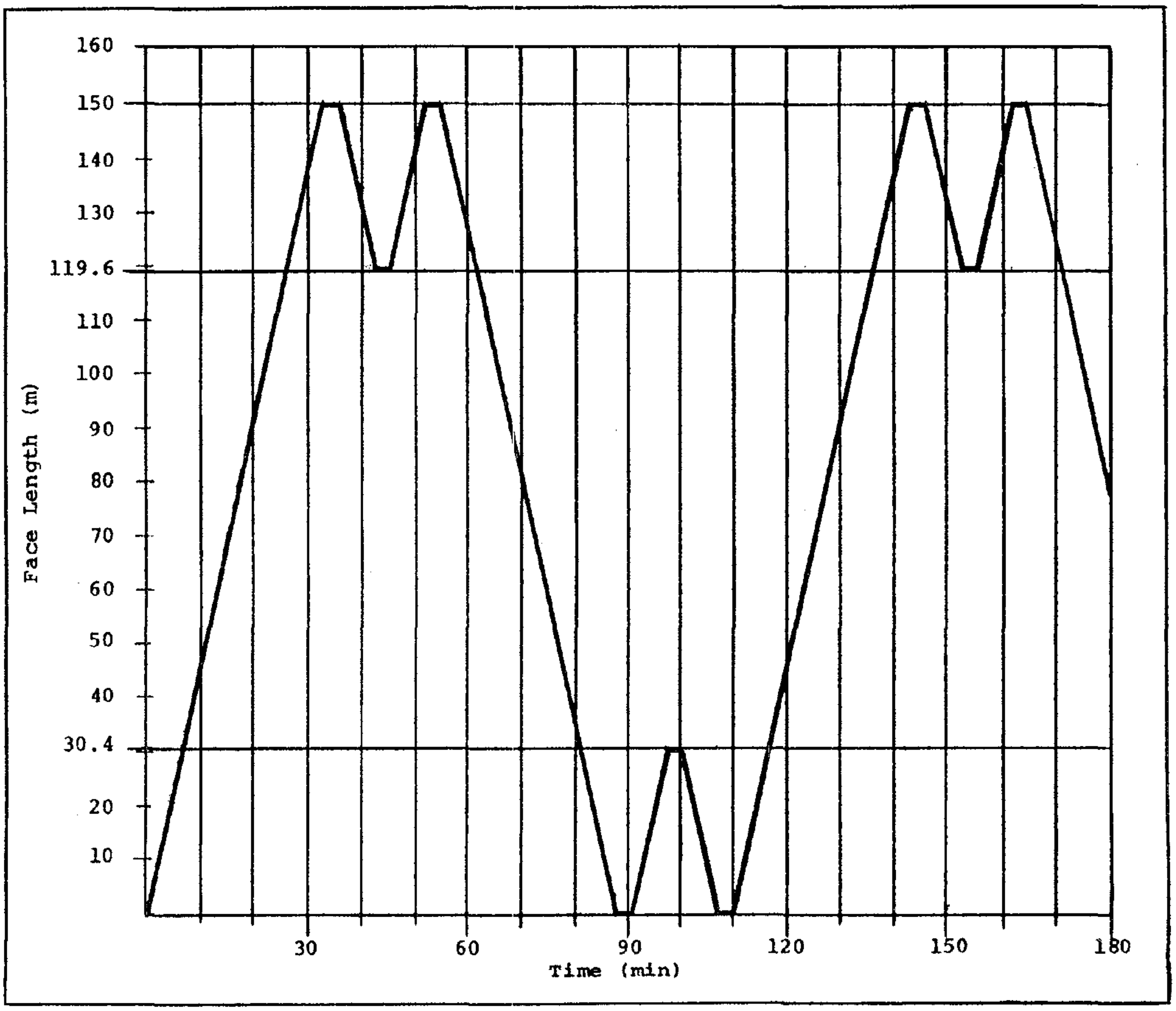


FIGURE 6-1b

Shearer Face Travel (Cutting Entire Thickness) Versus Time

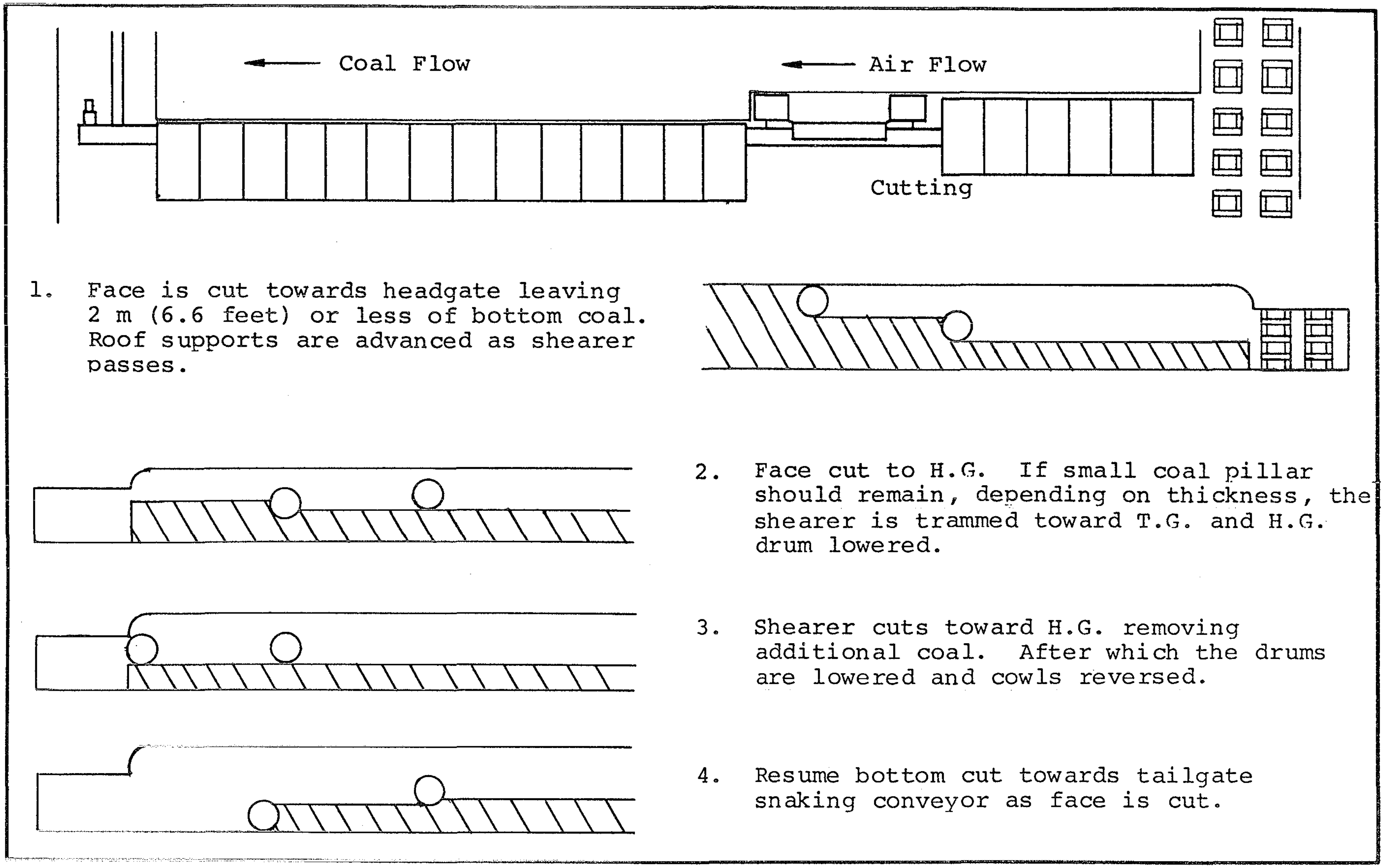


Figure 6-2a. Shearer Face Winning Cycle
(Two Bench Method)

5. After finishing bottom cut at T.G. return drums to top cut position and sump in toward H.G.

6. Once sumping is complete rear drum is raised and top 2 m (6.6 ft) is cut to T.G.

7. Drums are returned top cut position and face is cut to headgate.

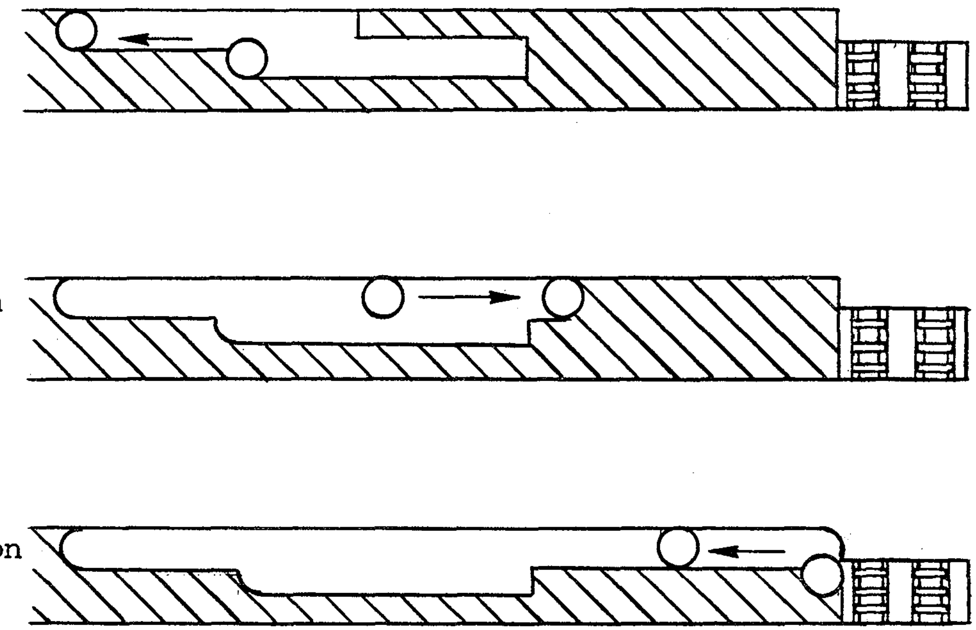


FIGURE 6-2a (Continued)

Shearer Face Winning Cycle
(Two Bench Method)

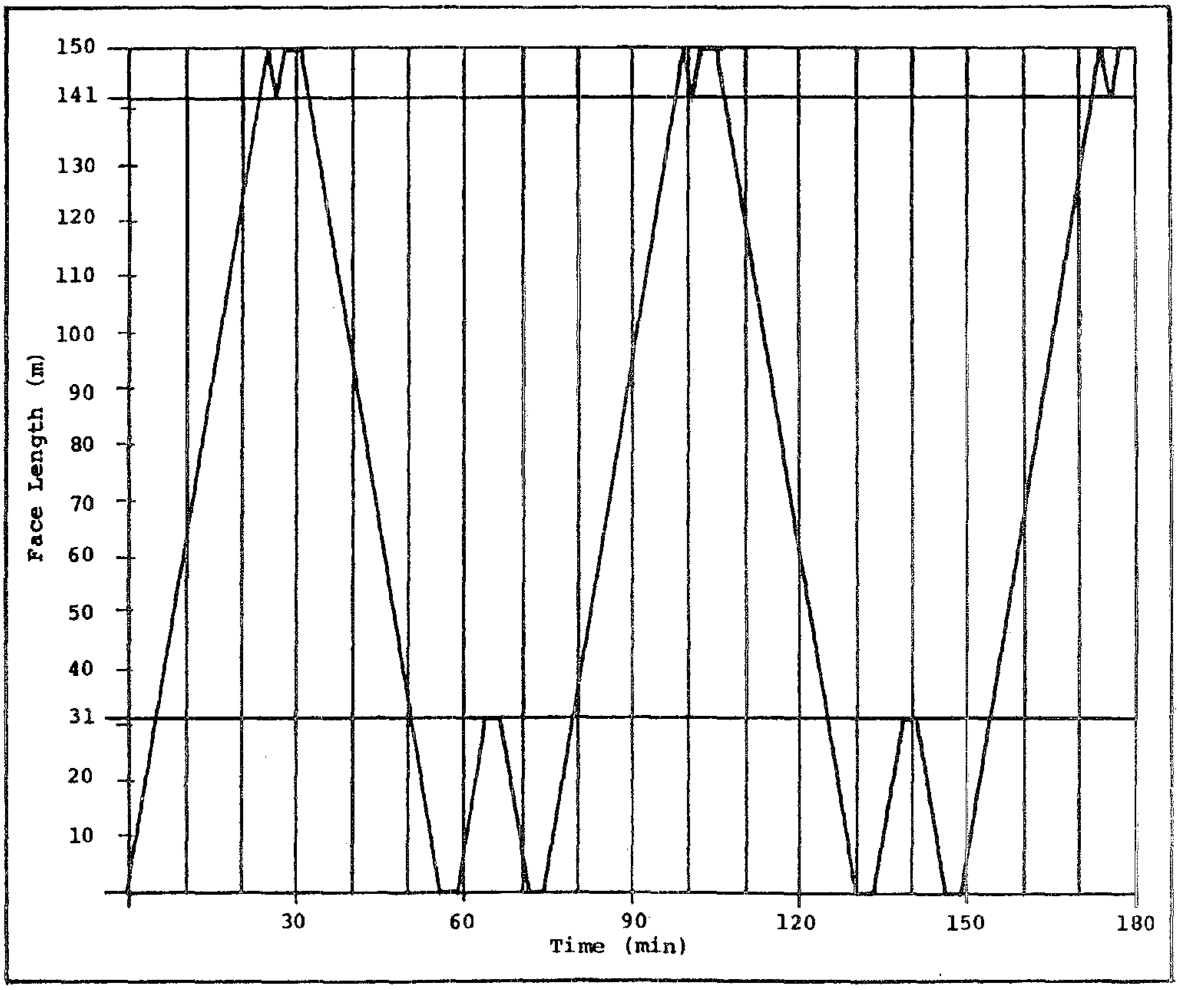


FIGURE 6-2b

Shearer Travel Versus Time
(Two Bench Method)

Two Benches (Bi-directional)

$$\text{Production Per Shift} = P_s = A_C \times V_s \times (SH)(WS)(D) \times \frac{(FL)}{2(FL) + (ST_L) \times (VS)} \quad (2)$$

A_C = Available time

V_s = Shearer speed

SL = Sump face length

FL = Face length in feet

SH = Seam height in feet

WS = Web size in feet

d = Density coal (.04 tons/cubic foot)

ST_L = Sumping Time = $\frac{2\ SL}{V_s}$ + TD

SL = Length of sump face

V_s = Shearer speed

TD = Unproductive face end operations

These equations will give a production estimate according to the various face length and assumptions of face end delay times.

Available time

The total available shift time was estimated to be 360 minutes. The available time for machine cutting was assumed to be 250 minutes, 69.4% of 360 minutes available after mantrips and lunch time deduction. That represents only 52% of the 24 hours which could be used (36% of the time available in a seven-day week).

Face End Operations

The required face end operations are different from one face organization to another. When utilizing a double ended ranging drum shearer, the sumping face distance has been estimated to be 100 feet. Raising and lowering the drums and inverting the cowls takes only 3 minutes per operation. This was called unproductive face end operations.

The productive sumping operation is twice the sumping length divided by the shearer speed.

This calculation yields the minimum time required for face end operations per web. To this time, some delay time may be added varying from 0 to 20 minutes. The main cause of delay is generally bit replacement, but may be waiting time for roof support advancement.

The time required for face end operations has an important impact on the optimum face length. Therefore, two values are considered in each case. When sumping time includes 20 minutes delay time, the machine time is reduced to about 50% of the total available shift time. This is considered as the base case of the economic analysis.

Shearer Speed

The shearer has been selected to obtain a peak production rate of 1250 tons/hour. If necessary, the drum design will have to be modified to adapt the bit lacing to the seam. The difficult point is to combine efficient dust control and shearing efficiency. The depth of cut by the bits has to reach the maximum possible to limit dust production. However, in the first trial the planned depth of cut corresponding to the maximum speed can be unrealistic and the drum will have to be modified. It has been assumed that a 20 feet/minute speed can be the normal average speed for the 500 kw shearer loader cutting two successive benches with a 1 m wide web.

Example with the following values:

Face length	= 500 feet
Seam height	= 16 feet
Web width	= 3.28 feet
Sumping face length	= 131 feet
Available machine time	= 250 minutes
Sumping time per pass	= 45 minutes
Shearer speed	= 20 feet/minute

The production per shift is:

$$P_s = 250 \times 20 \times 16 \times 3.28 \times 0.04 \frac{500}{(1000 - 131) + 51 \times 20}$$
$$= 2762 \text{ tons/shift}$$

When using 25 minutes for sumping time:

$$P_s = 3498 \text{ tons/shift}$$

The increase in time estimated for face end operations has a major effect on the shift production. Twelve minutes for unproductive face end operations is 19% of the machine cutting time. Thirty-two minutes is 50% of the machine cutting time, in this example.

Face-to-Face Moves

An important limitation to longwall mining efficiency is the cost and time necessary to move the face equipment from one panel to another. This may be a major problem if these operations are not well planned in advance. For this study, a 40 shift move time is used, which KETRON considers to be sufficient with proper planning. The required manpower will be supplied by the face crews (3 x 12 = 36) and the panel development crew (2 x 7 + 5 = 19), a total of 55 men/day or 733 man days, without adding part of the backup manpower which should be available.

Tons per year are calculated by using the following equation.

$$P_y = P_s S_n (N_w - M_y N_m)$$

P_y = tons per year of ROM coal

P_s = Tons per shift

S_n = Number of shifts per day

N_w = Number of working days per year

M_y = Number of moves per year; each move requiring N_m working days

PL = Panel length

SH = Seam height

FL = Face length

d = Coal density

Moves per year are calculated by dividing the longwall face yearly production by the total tonnage of a panel.

$$M_y = \frac{P_y}{(SH)(FL)(PL)d} = \frac{P_s S_n (N_w - N_m M_y)}{(SH)(FL)(PL)d}$$

or

$$M_y = \frac{P_s S_n N_w}{P_s S_n N_m + (SH)(FL)(PL)d}$$

According to the panel dimensions, moves taking 40 shifts reduce the average daily production by 9% to 24%. Table 6-1 shows the impact of face moves on the average annual production.

6.3 Mine Design

To compare longwall mining with room-and-pillar methods, a hypothetical mine has been designed on the basis of real mine data using two assumptions, 1) room-and-pillar method with six sections, 2) longwall mining with one longwall face and two continuous miners.

6.3.1 Room-and-Pillar Mine Design

In the first year, main entries are developed with one continuous miner section driving eight entries (18 feet by 9 feet) with a 84-foot by 100-foot center pattern. The productivity rate is 850 tons per producing shift on the basis of two producing shifts and one idle shift per day, or $850 \times 2 \div 3 = 566.7$ tons average per shift on a three-shift basis. With this productivity, the rate of advance of an eight-entry development will be approximately seven feet/shift as a result of the following calculation:

entry width x height x density (number of entries x pillar length + number of crosscuts x pillar width) =

$$18 \times 9 \times 1 \times 0.04\ (8 \times 100 + (84 - 18) \times 7) = 8178 \text{ tons}$$

for 100 feet of advance or 81.78 tons/foot advance.

To reach this productivity level, the required equipment is as follows:

One continuous miner	$ 500,000
Three teletrams @ $180,000	540,000
Other equipment	460,000
TOTAL COST	$1,500,000

The crew on both production shifts consists of seven men, including foreman. The idle shift employs a maintenance crew of five men. The total workforce per day will be 7 x 2 + 5 = 19 (including foreman). Backup labor has to be added by using a multiplying factor of two.

In the second year, a second continuous miner section is added. The number of sections are increased to four during the third year and reaches the maximum of six continuous miners the fourth year.

TABLE 6-1

LONGWALL FACE PRODUCTION
(16-Foot Seam--Two Bench Method)

1 Face Length (feet)	2 Face End Time (min.)	3 Advance Per Shift (feet)	4 Production Per Shift (tons)	5 Production Per Day (tons)	6 Yearly Average Production Per Day (including moves) (tons)					
					2000' Panel	6/ /5	3000' Panel	6/ /5	4000' Panel	6/ /5
200	25	18.4	2362	7086						
	45	12.7	1629	4887						
300	25	15.0	2892	8676	6572	76%	7237	83%	7462	86%
	45	11.0	2115	6345	5173	82%	5500	87%	5667	89%
350	25	13.8	3090	9270			7733	83%		
	45	10.3	2312	6936			6041	87%		
400	25	12.7	3258	9774			N/A			
	45	9.7	2486	7458			N/A			
450	25	11.8	3401	10203			8773	86%		
	45	9.2	2640	7920			6971	88%		
500	25	11.0	3525	10575			9188	87%		
	45*	8.7	2778	8334	7048	85%	7462	89%	7612	91%
600	25	9.7	3729	11187			9816	88%		
	45	7.8	3014	9042			8202	91%		
800	25	7.8	4020	12060			10861	90%		
	45	6.6	3373	10119			9173	91%		
1000	25	6.6	4217	12651						
	45	5.7	3633	10899						

The yearly average is 76% to 92% of the average daily production per panel. The reason is the influence of unproductive periods of face-to-face moves, which is about 10% for the base case.

* Base Case.

After sufficient development of the main eight entries, submains with six entries can be developed (9 feet high) with the same width (18 feet) and the same pattern (84-foot by 100-foot center). An advance of 100 feet with six entries produces

$$18 \text{ ft.} \times 9 \text{ ft.} \times 1 \times 0.04 \text{ t/ft.}^3 \ (6 \times 100 \text{ ft.} + (84 \text{ ft.} - 18 \text{ ft.})\ 5) = 6026.4 \text{ tons}$$

or 60.26 tons/foot. A six entry system advances at 9.404 feet/ shift in average on a three shift/day basis. Panels are then developed and pillars extracted with the same productivity.

Panel development for room and pillars uses a three entry pattern including a yielding pillar chain (Figures 6-3 and 6-4).

120 feet of advance represents:

$$0.04 \text{ t/cubic ft.} \times 18 \text{ ft.} \times 9 \text{ ft.} \times (3 \times 120 \text{ ft.} + 2 \times 42 \text{ ft.} + 30 \text{ ft.}) = 3071.52 \text{ tons}$$

This would require 3.614 shifts at 850 tons or 5.42 shifts on a three shift basis (two producing and one idle shift) or 22.14 feet/ shift.

Panel extraction by continuous miner may be represented as a 500-foot retreating face producing an average production per foot equal to 500 x 9 x 0.04 = 180 tons. This means 9.44 feet of advance is required to produce 1700 tons, or 3.148 feet of advance per shift on a three-shift basis (two producing and one idle shift). The mining sequence is shown on Figures 6-3 and 6-4.

It is possible to use an average figure for development and extraction of a room and pillar panel. The width will be 626 feet instead of 500. The panel will be extracted at 80%. One chain of 30 x 100 pillars will remain to protect the entry against the following panel.

Total tonnage per foot of panel length is:

$$9\ (626 - 30) \times 0.04 \times 0.8 = 173 \text{ tons}$$

3.3 feet are mined per shift on a three-shift basis (10 feet/ day). Panels are retreated at a slow rate of advance. That requires several panels mined simultaneously.

6.3.2 Mine Design for Longwall Mining

It is assumed that prior to longwall mining, development operations will require one year with one continuous miner section (two shifts) driving main entries (eight) at the rate of 850 tons shift which means seven feet per shift on a three-shift basis or 5000 feet per year.

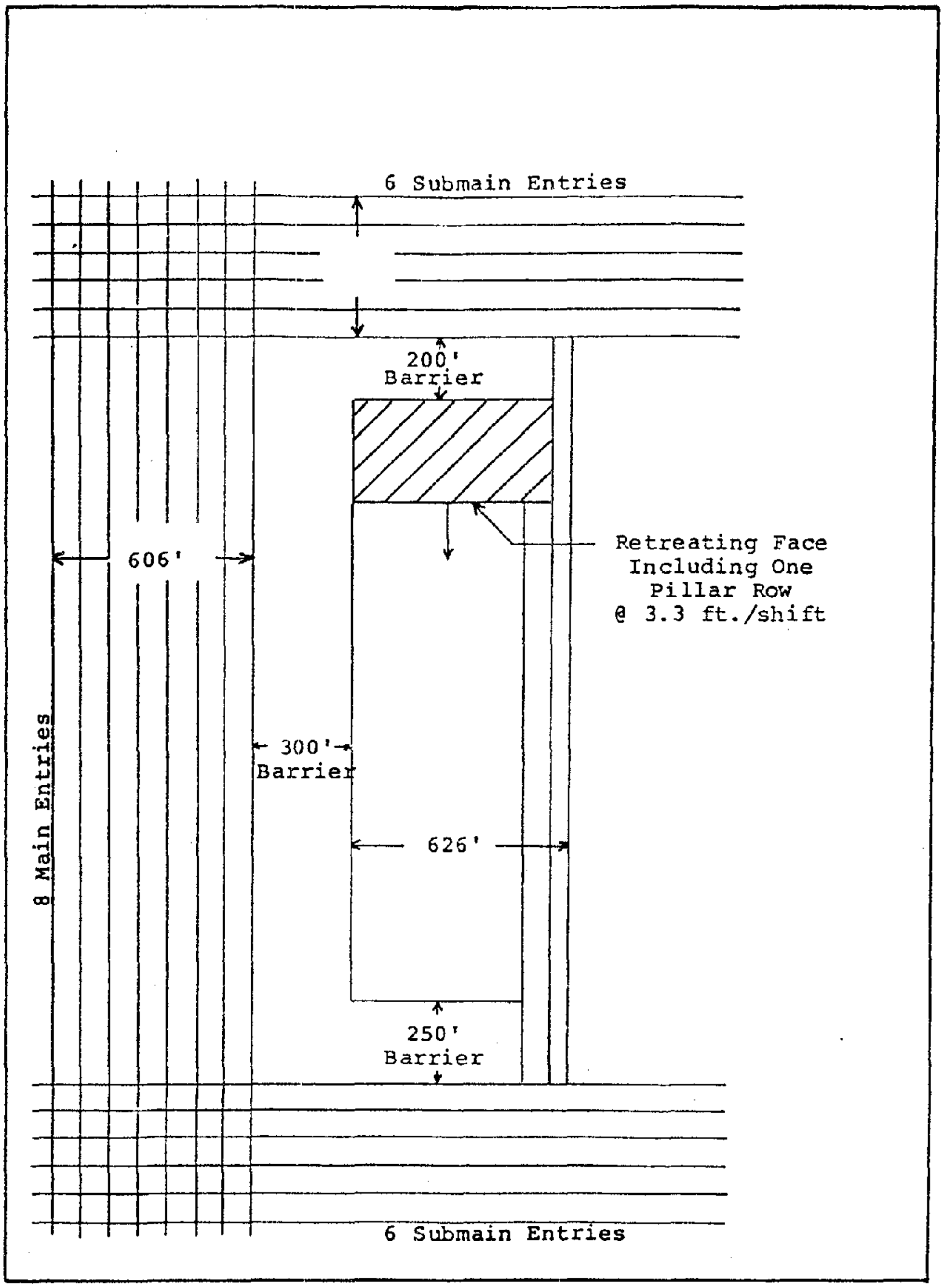


FIGURE 6-3

Layout of Room-and-Pillar Panel

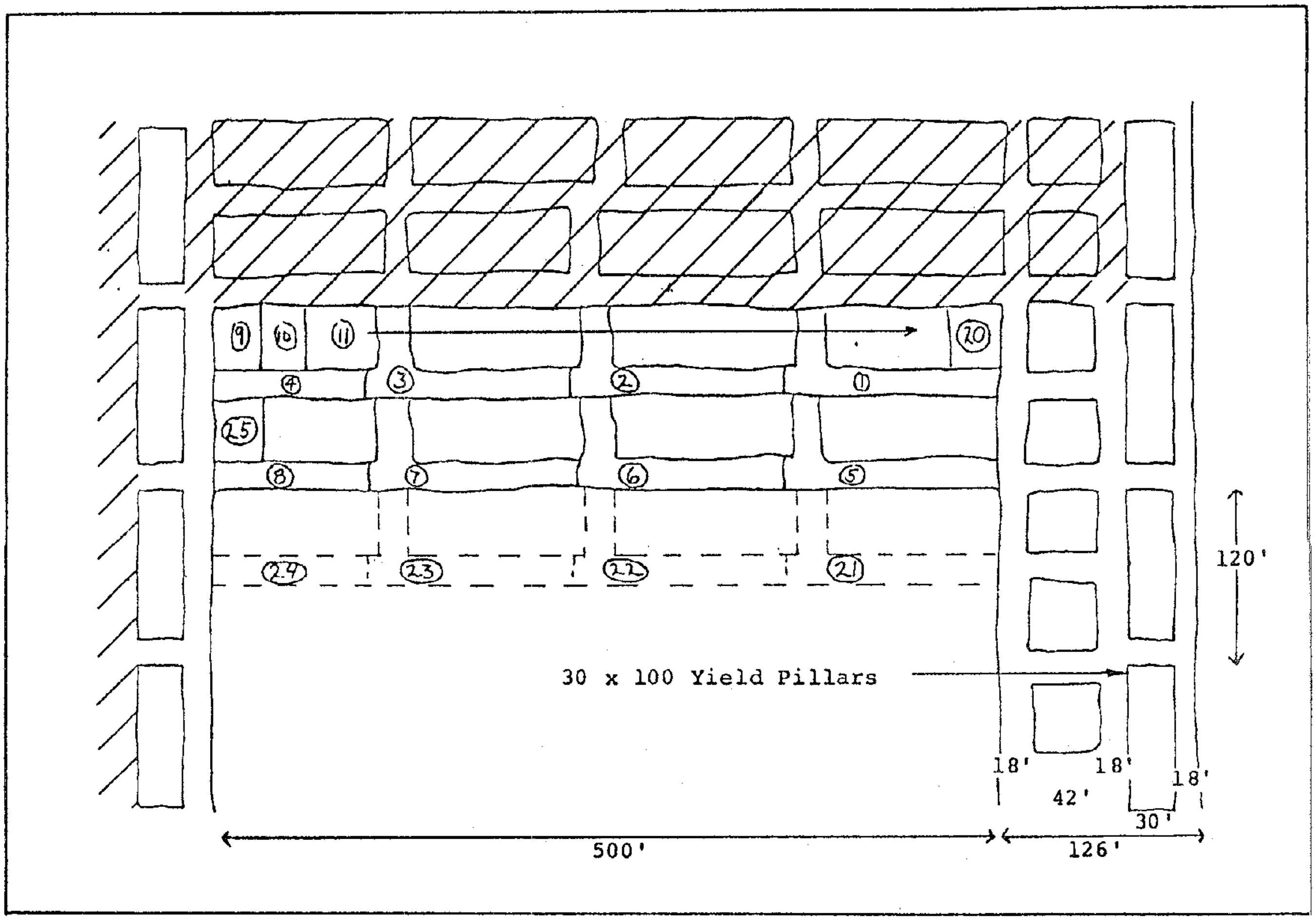


FIGURE 6-4

Mining Sequence for Room-and-Pillar Extraction

Then the second and third years utilize two continuous miner sections (two shifts producing) driving main entries (eight entries) and submains (six entries).

The six-entry development rate of advance is nine feet/shift. The total driven in three years is:

	First Year	Second Year	Third Year	Total Feet
(eight entries) mains	4,997	4,997	4,997	14,991
(six entries) submains		6,760	6,770	13,530
Total	4,997	11,757	11,767	28,521

This seems reasonable before starting a retreating longwall panel. Estimations assume 18 feet wide, 9 feet high entries with a 84-foot by 100-foot center pattern, identical with the mine design for room-and-pillar mining.

The longwall equipment will be bought at the end of year three. Year four will start with one longwall face equipment ready to install and two continuous miner sections.

Equipment

Continuous miner sections are equipped identically to the CM section for room-and-pillar methods.

Investment cost for the longwall face equipment mining a thickness of 12 feet (500 feet long):

1 AFC	2 x 30 mm chain or 1 34 mm	
	2 x 300 kW drive ends right angle delivery	600,000
	additional cost for side delivery	123,000
1 Shearer Loader	Eickhoff 300 L 27 rpm	1,400,000
	Eicotrack and ramp plates	600,000
	Shearer Breaker	75,000
104 shield 2 x 250 tons (1.8/4.00 m)		
including face end shields (unit cost 65,000)		7,000,000
1 power pack		150,000
1 stage loader		350,000
1 crusher breaker		100,000
transformer		45,000
gate boxes		100,000
cables		52,000
telephone		45,000
lighting		55,000
		10,695,000

Alternatives	shearer	AM 500	1,200,000
		lump breaker	120,000
		roll rack	600,000
		Sagem super DTS	1,000,000
		Peratrack	450,000
		Dynatrack	550,000
	four-leg	shield	6,500,000
	addition	for curved conveyor in excess	
	of stage	loader cost	150,000

The cost of the face equipment required to mine a 16-foot seam (18 feet maximum) should be increased by 2 to 2.6 million dollars, the price of shields being increased from $65,000 to $85,000 or $90,000.

An increase of the shield width from five feet to seven feet should decrease the price by 10% or $1,000,000. In this case, the 500-foot face equipment will cost about 12,000,000 dollars. The more accurate figure of $11,830,000 has been used for the base case of a 500-foot longwall in a 16-foot seam with the following breakdown:

Face conveyor	$1,458,000
Shearer	1,475,000
Shields	8,000,000
Others	897,000

- Labor: Segment Type 1 (panel development) The crew for each production shift is seven men; on the idle shift there is a maintenance crew of five men. Total per day will be 7 x 2 + 5 = 19 (including foreman). When including backup labor by using a multiplying factor of two, the manpower is 19 x 2 = 38 men/day.

- Labor: Segment Type 2 (longwall panel extraction) The longwall is assumed to be working three shifts of 12 men/crew. With the same coefficient for backup labor (coef 2), the results are 12 x 3 = 36 at the face and 36 x 2 = 72 men/day as a total workforce.

Mining Strategy

At the beginning of the fourth year after three years of development, the mining strategy is to begin with 40 shifts for longwall face equipment installation. After which, both longwall panel extraction and panel development proceeds. Panel and face development is done on a two shift per day basis while longwall extraction and longwall moves are done on a three shift per day basis for 240 days per year. When the development of the next panel, which is ahead of the longwall panel being extracted, is completed, other development proceeds (mains or submains), except during the face equipment move which is assumed to take 40 shifts and requires additional manpower. A panel cycle is considered to include a 40 shift longwall move and the longwall panel extraction.

Length of development required to develop longwall panels varies with the size of panels. We use the basic pattern shown above. (See Figures 6-5 and 6-6.)

That means, a main entry with 2 x 300' barriers is 606 feet wide + 600 = 1206 feet. A submain entry with 450 feet (200 + 250) of barrier is 438 + 450 = 988 feet wide.

If a main is used for 10 longwall panels (five each side) the development length required for one panel is:

$$\frac{L + 988}{10} \text{ of main}$$

$$w + \frac{300}{5} \text{ of submain,}$$

if L is the panel length and w its width including the two entries.

Example: panel length: 3000 face length: 500

$$\frac{3000 + 988}{10} = 398.8 \text{ feet of main}$$

$$(500 + 84 + 18) + \frac{300}{5} = 662 \text{ feet of submain}$$

In a model as well as in a real mine it is difficult to match development and panel extraction. If a continuous miner section could remain as a spare section for a while, the men normally working with this equipment have to be employed. It is therefore assumed that some panels will still be mined by rooms and pillars even in a mine using longwall mining. These panels will preferably be of small size and/or of irregular shape, for instance, in the vicinity of faults.

Mine Tree

The previously described organization is represented by the two following mine trees with six continuous miner sections (Figure 6-7) and with two continuous miner sections and one longwall (Figure 6-8).

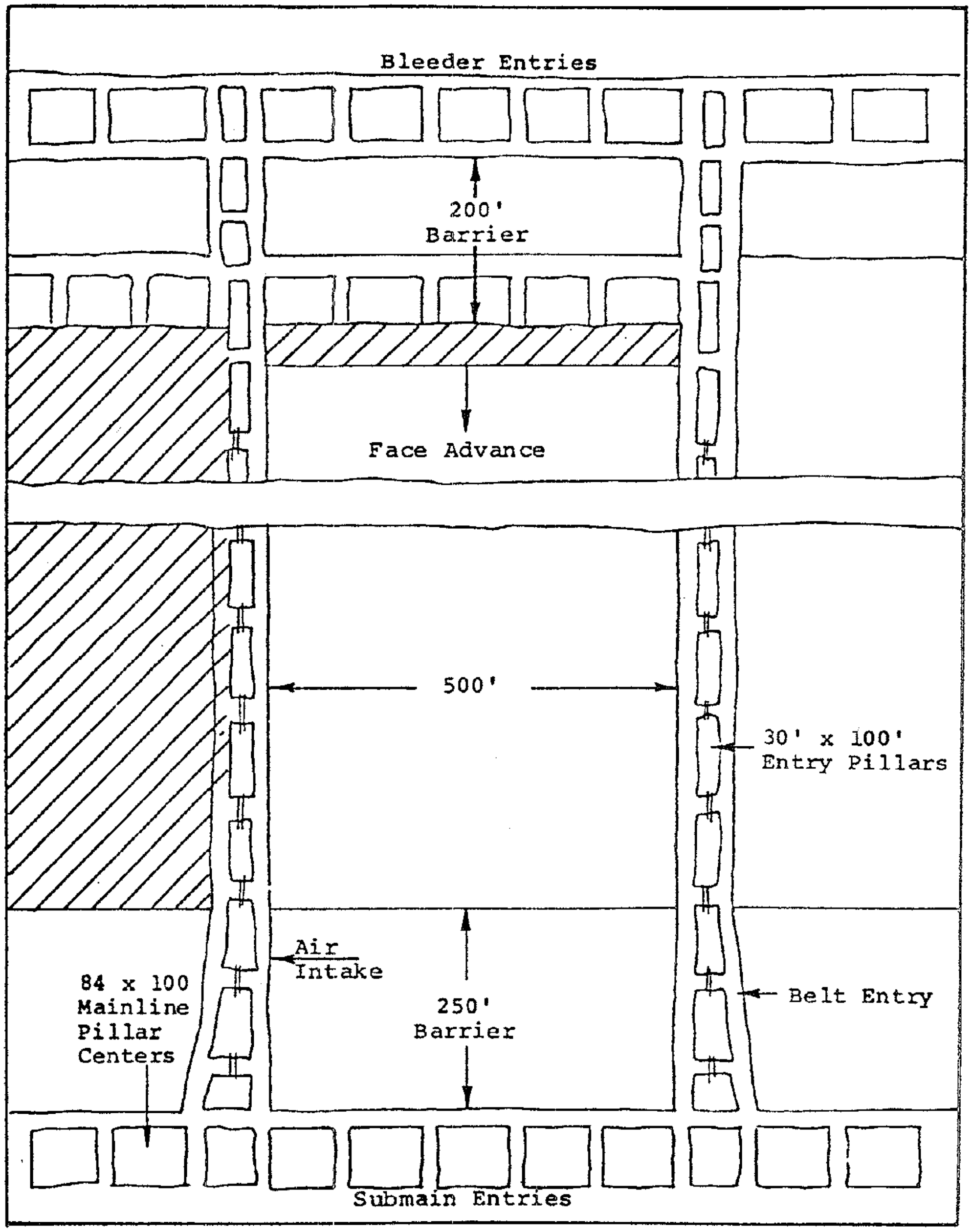


FIGURE 6-5

Longwall Two-Entry Development

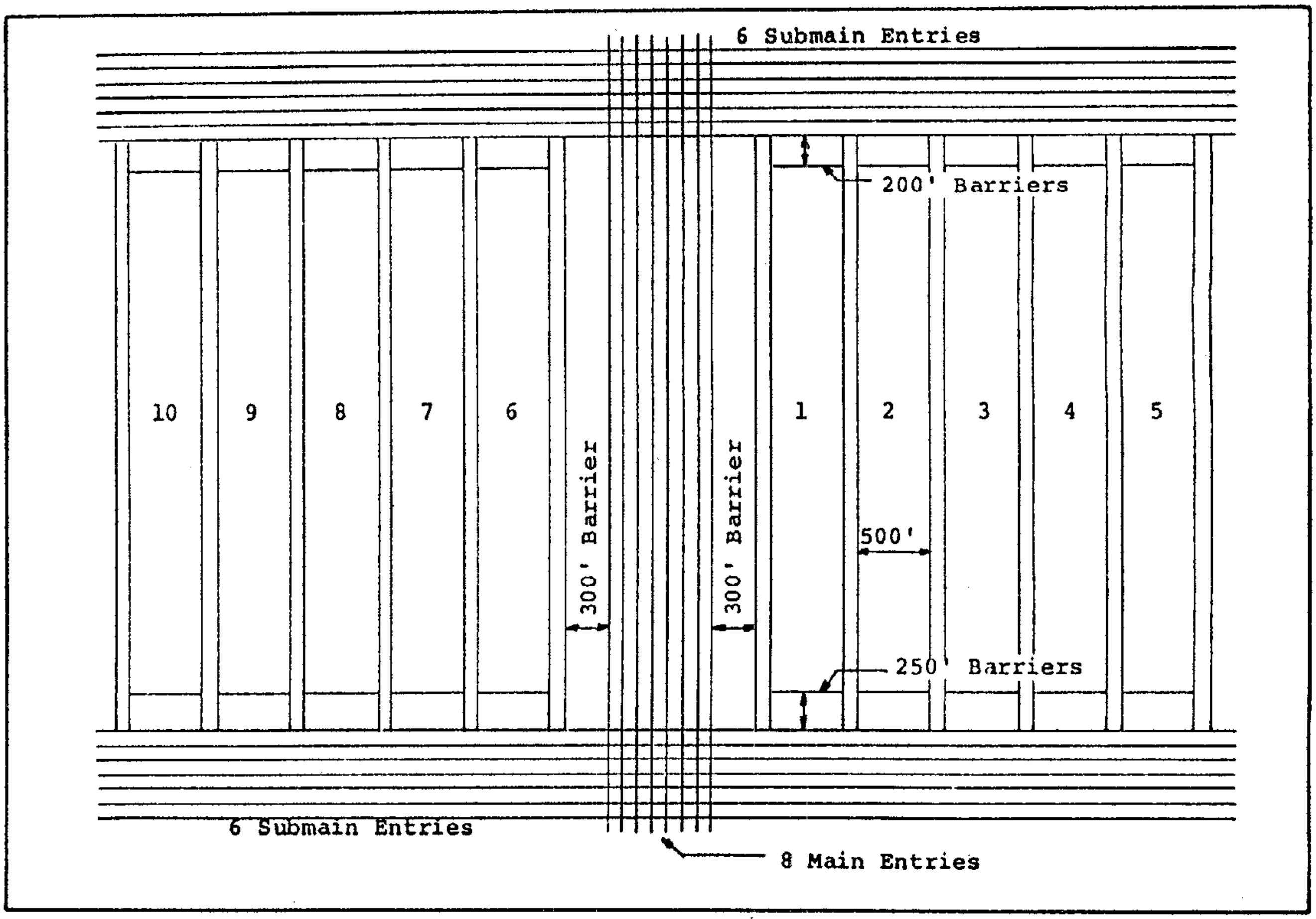


FIGURE 6-6

10-Panel Two-Entry Longwall Layout

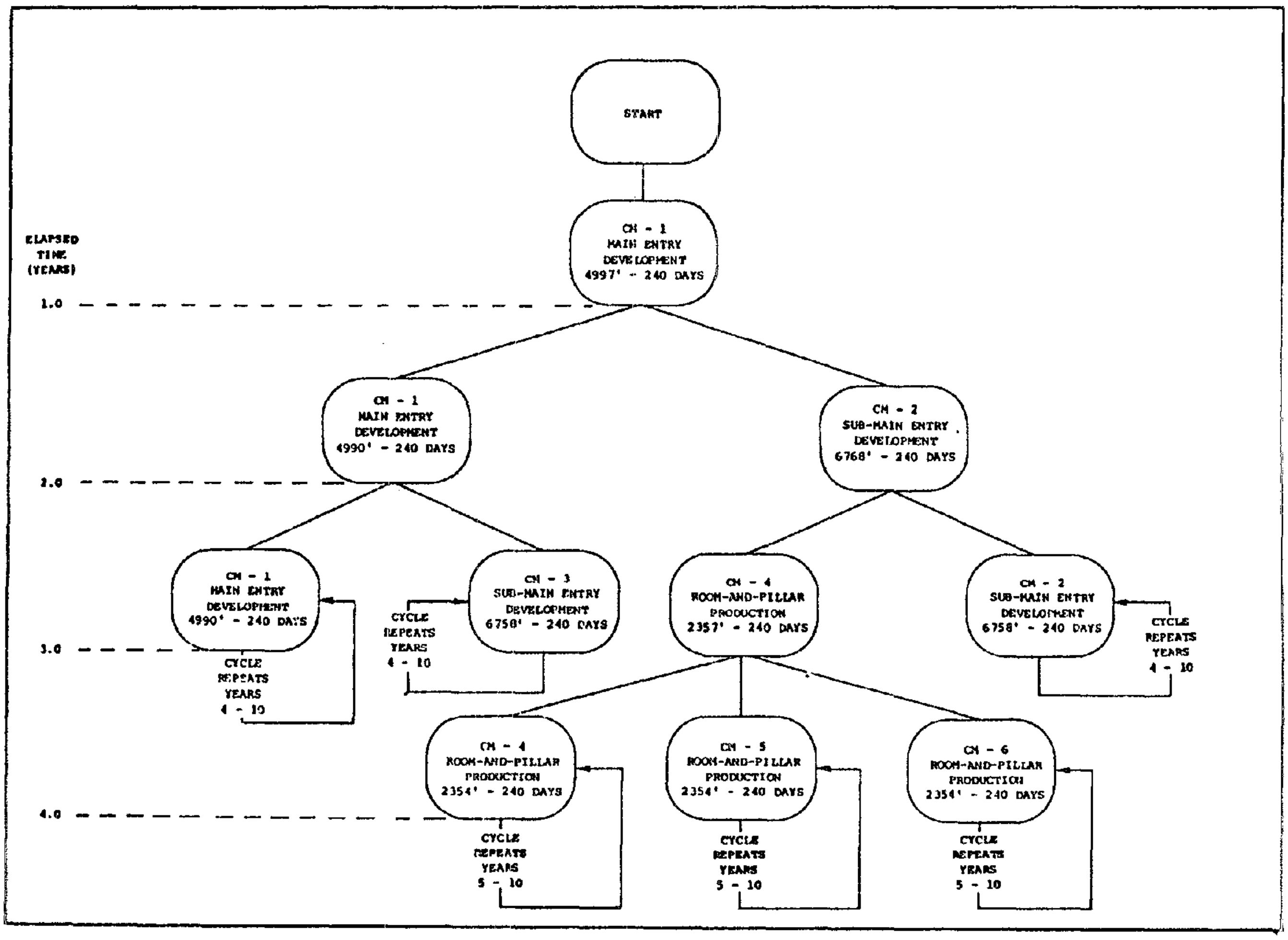


FIGURE 6-7

Mine Tree for Six Continuous Miner Sections (Plan 1)

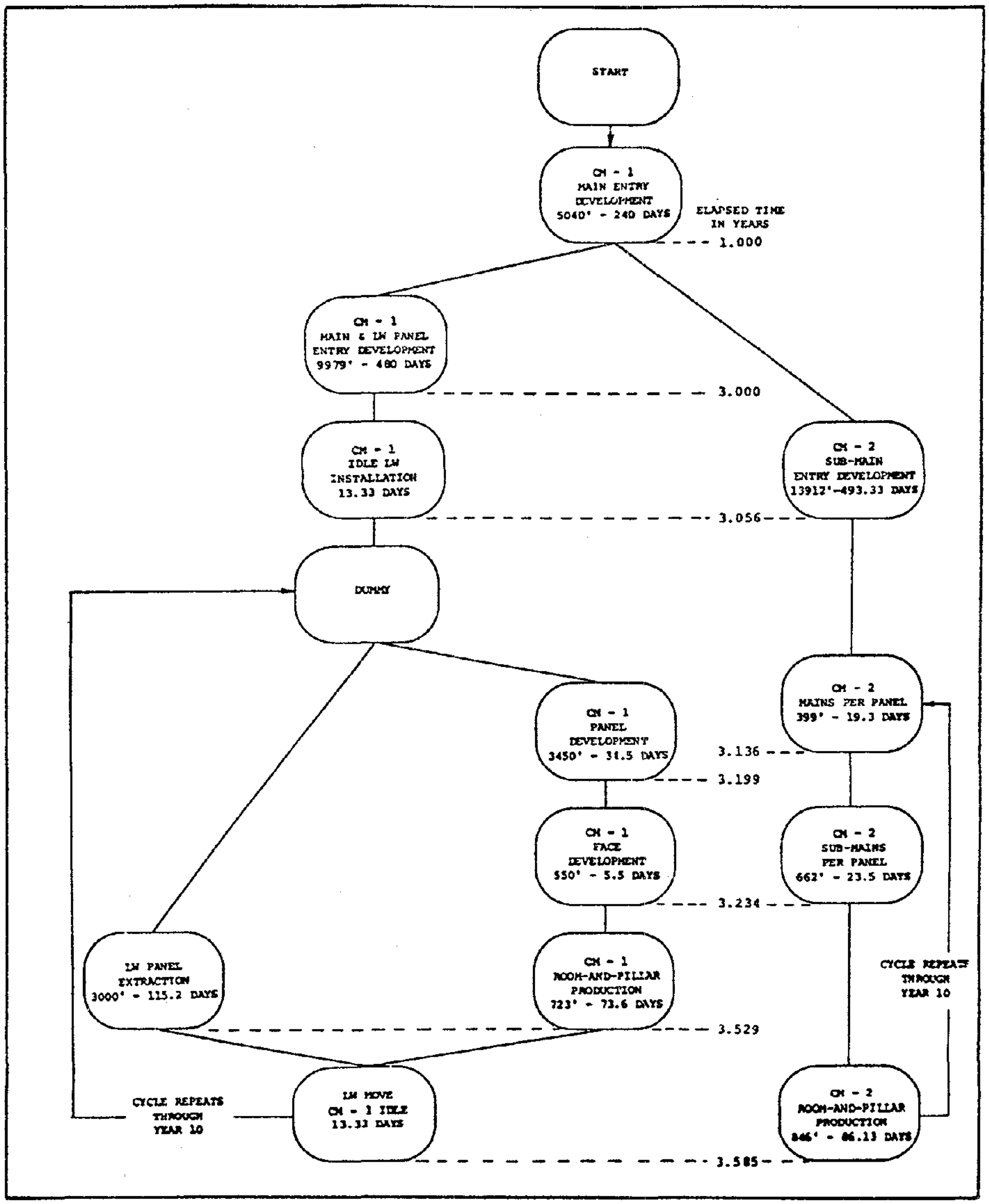


FIGURE 6-8

Mine Tree for Two Continuous
Miner and One Longwall Section
(Plan 2 -- Base Case)

6.3.3 Data and Assumptions for Mine Cost Analysis

6.3.3.1 Wages

For the contract year 1980-1981, the hourly and daily wages are as follows:

Contract Year: 1980 - 1981				
UMWA Labor Grade	Hourly Rate ($)	Per Hour Shift Differential (¢)		Daily Wage ($)
5	9.97	20	(2)	(1) 84.52
		30	(3)	(2) 86.12
				(3) 86.92
4	9.56	-		(1) 81.28
				(2) 82.88
				(3) 83.68
3	9.19	-		(1) 78.34
				(2) 79.94
				(3) 80.74
2	8.97	-		(1) 76.56
				(2) 78.16
				(3) 78.96
1	8.90	-		(1) 75.98
				(2) 77.58
				(3) 78.38

(1) 1st Shift (2) 2nd Shift (3) 3rd Shift

To take into account some additional costs such as overtime paid, an average daily wage of 87 dollars is used, with a coefficient of 1.9 for overhead and benefits. The cost of a man/day is 165.3 dollars when overhead and benefits are included.

A continuous miner section with 19 men/day has a manpower cost of 3140.70 dollars and with backup labor at a factor of 2, the total manpower cost is 6281.40 dollars/day. A longwall face section with 36 men/day has a manpower cost of $5950.80/day and with backup labor at a factor of 2, the total manpower cost is $11901.60/day, including supervision.

6.3.3.2 Equipment Cost Assumptions

A continuous miner costs $500,000, is assumed to have a five-year life and a maintenance cost of 25%/year, or $125,000 per year ($173.61 per shift). A teletram costs $180,000 with a five-year life and a maintenance cost of 22%/year ($166.65 per shift for three teletrams). Other CM section equipment costs $460,000 with a five-year life and a maintenance of 12%. Supplies are $2.44/ton.

Longwall roof support costs $8,000,000 (for 500 feet), has an eight-year life, with a maintenance cost of 6%/year (666.667 per shift). Shearer loader is $1,475,000, has a five-year life, with a maintenance cost of 20%/year (409.72 per shift). Conveyor costs $1,458,000, has a two-year life, with a maintenance cost of 10%/year (202.5 per shift). Other longwall equipment costs $897,000, with a maintenance of 10%, or 124.58 per shift. Supplies are 1.29/ton.

6.3.3.3 Other Costs and Assumptions

Selling price: $18/ton
Royalty : 0.1667% of selling price ($3/ton)
Depletion : 10%
Income tax : 2%
Ten million dollars pre-productive development

The value of $10 million of pre-productive development for a ten year study seems reasonable. It must be pointed out that if this investment has a notable importance on financial results, it does not seriously alter the cost comparison between plans (Table 6-2).

TABLE 6-2

EFFECT OF PRE-PRODUCTION DEVELOPMENT COSTS

PREPRODUCTION DEVELOPMENT COSTS	PLAN 2 LONGWALL REQUIRED SELLING PRICE	PLAN 3 CONTINUOUS MINER REQUIRED SELLING PRICE	DIFFERENCE
0	$ 8.75/ton	$10.05/ton	1.3
BASE 10	11.56/ton	12.45/ton	0.9
20	14.54/ton	15.25/ton	0.7

6.4 Mine Cost Analysis Model

To measure economic efficiency of alternate mining methods, the effect of the investment must be evaluated on the entire mine and for a long period, if not the entire life of the mine.

The KETRON, INC. Mining Economic Model was used to simulate production and determine the financial results for each mine plan and various design parameters.

The features of this model are:

1. Simulate annual production based on mine design and production strategy.

2. Compute standard financial measures.

3. Provide flexibility in analyzing mining scenarios.

The sequence of mining operations must be defined to simulate production. Inputs are required that specify the advance and production rates per shift for each mining operation. Figures 6-7 and 6-8 show the mining sequence and timing for the operations involved for Plans 2 and 3.

Inputs to the financial portion of this model include operating costs, capital costs, discount rate and the selling price anticipated for production. Production is translated into revenue flows and the costs of the operations are determined, including taxes, depreciation, etc.

This model provides numerous outputs, including the following financial reports:

1. Capital expenditures
2. Production and revenues
3. Cash operating costs
4. Non-cash operating expenses
5. Net profit
6. Cash flow
7. Summary statistics, including summaries of the above, and return on investment (for the estimated selling price), net present value, and the required selling price to provide a return on investment of 25% (in this study).

Detailed information concerning this model is documented in Oil Shale Cost Analysis, Volume III, Oil Shale Mining Economic Model, Draft Users Guide, November 7, 1980, Contract No. U. S. DOE ET-77-C-01-8915 (13/14).

Rate of Return on Investment

The Rate of Return on Investment (ROI) technique evaluates the rate of return that results from a capital investment, based upon predicted annual (net) cash flows. This may be interpreted as the interest rate that the investor would have to charge on a capital loan of equivalent risk, in order to be indifferent between the investment and the loan.

This calculated "interest rate" can be used by a decision-maker for two purposes:

1. to determine if the investment provides a sufficient rate of return to compensate him for his cost of capital and for the degree of "risk" involved in the investment, and

2. to permit meaningful comparisons of alternate investments. From a purely economic point of view, the investment with the highest ROI would be selected.

The Rate of Return on Investment is evaluated by solving the following equation for r:

$$P = \frac{S_1}{(1 + r)} + \frac{S_2}{(1 + r)^2} + \ldots. \frac{S_n}{(1 + r)^n}$$

where:

P = Initial Capital Investment

S_1, S_2, ... S_n = Expected Cash Flow in Year 1, 2,...n

r = Rate of Return on Investment

<u>Present Value</u>

The present value is obtained from the cash flow equation in the following manner:

$$PV = \frac{S_1}{(1 + i)} + \frac{S_2}{(1 + i)^2} + \ldots. \frac{S_n}{(1 + i)^n}$$

where:

PV = Present Value

S_1, S_2, ... S_n = Expected Cash Flow per Year

i = Pre-selected "discounting" Rate

As stated, the factor i is the minimum acceptable rate of return that a decision-maker is willing to accept for his initial capital investment, given the degree of risk involved. For example, by choosing i = 0.25, the decision-maker demands a rate of return on investment of at least 25%. If the present value calculated with this value of i is greater than the initial capital investment, then the criteria is met.

Alternate investments which are otherwise equivalent can be evaluated by comparing their excess Present Value over their required capital investment; the investment with the highest excess Present Value would be selected.

Both Present Value and Rate of Return techniques take into account the time value of money and are equally valid for evaluating investments.

Rate of Escalation

All calculations have been made in 1981 dollars with price escalation on the basis of January 1981, prices and wages. Costs and coal selling prices were not escalated. Thus dollars represent 1981 values.

6.5 Results

When studying the replacement of continuous miner operations by longwall mining 16 feet in a single pass, two questions arise: What would be the result of extracting only 11 feet (Plan 4); and what is the effect of differing face organizations (full face cut, two bench method) and panel dimensions?

Table 6-3 shows a comparison of Plan 1 (16 feet full face cut), Plan 2 (16 feet two bench method), and Plan 4 (extracting 11 feet only). Plan 2 is the most favorable. Therefore, the remaining part of the economic study has been made by varying panel dimensions (Table 6-4), using only Plan 2 and comparing to Plan 3. The sensitivity analysis has been made around the base case of Plan 2 (500 feet x 3000 feet panel).

1. Plans 1 and 2 (longwall with two continuous miners) have economic advantages over Plan 3, (six continuous miners using room-and-pillar mining).

2. Plan 4, longwall mining extracting only 11 feet from a 16 foot seam, shows moderate reduction in economic advantages from Plan 1 which was mining 16 feet. This is due to the reduced level of production per shift.

3. Plans 2 and 3 have essentially identical 10-year production; however, Plan 3 has some production (thus revenue) sooner than Plan 2 (Figure 6-9). Plan 3 (six

TABLE 6-3

ECONOMIC MEASURES (1) 500' x 3000' PANEL

ECONOMIC MEASURE	PLAN 1	PLAN 2	PLAN 3	PLAN 4
Total 10 year production	18.0 M Tons	19.9 M Tons	20.0 M Tons	17.1 M Tons
Required selling price	$12.38/ton	$11.56/ton	$12.45/ton	$12.58/ton
Present values at 25% investment discount rate when the required selling prices are realized (2)				
- PV Gross Revenues	$61.0 M	$62.3 M	$70.2 M	$59.5 M
- PV Net Revenues	$51.0 M	$51.9 M	$58.5 M	$49.5 M
- PV Cash Expenses	$27.1 M	$27.7 M	$38.8 M	$26.6 M
- PV Capital Expenditures	$21.0 M	$21.0 M	$18.2 M	$20.2 M
- PV Taxable Income	$ 7.5 M	$ 8.0 M	$ 4.1 M	$ 7.1 M
- PV Operating Cash Flow	$21.0 M	$21.0 M	$18.2 M	$20.2 M
- PV Net Profit	$ 4.6 M	$ 4.9 M	$ 2.6 M	$ 4.3 M

(1) Plan 1 - Longwall plus two continuous miners (16 feet--full face cut)
Plan 2 - Longwall plus two continuous miners (16 feet--two bench method)
Plan 3 - Six continuous miners
Plan 4 - Longwall plus two continuous miners (extracting 11 feet only)

(2) Plans 1 and 2 realize 25% return on investment when the required selling prices are realized.

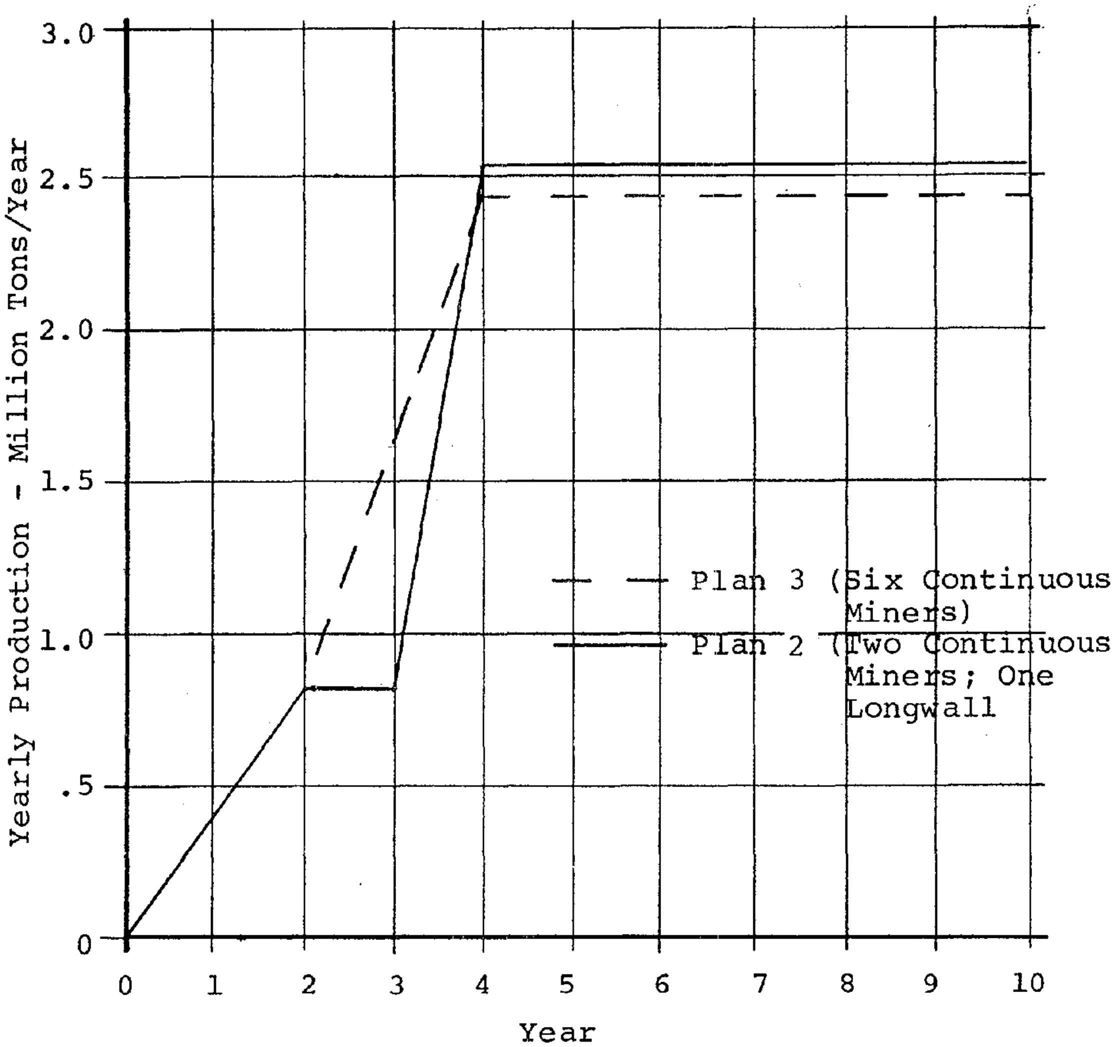
TABLE 6-4

THICK SEAM ECONOMIC SUMMARY

MINE PLAN	PANEL LENGTH feet	FACE LENGTH feet	SUMP TIME min.	% BASE PRODUCTION	ROI %	REQUIRED SELLING PRICE* $/TON
1	2000'	300'	42.3	100%	N/A	$13.52
	2000	800	42.3	100	N/A	12.18
	2000	300	22.3	100	N/A	12.14
	2000	800	22.3	100	N/A	11.11
	3000	300	42.3	100	N/A	13.26
	3000	500	42.3	100	N/A	12.38
	3000	800	42.3	100	N/A	11.99
	3000	300	22.3	100	N/A	11.77
	3000	500	22.3	100	N/A	11.08
	3000	800	22.3	100	N/A	10.86
	4000	300	42.3	100	N/A	13.12
	4000	800	42.3	100	N/A	11.85
	4000	300	22.3	100	N/A	11.58
	4000	500	22.3	100	N/A	10.91
	4000	800	22.3	100	N/A	10.76
2	2000	300	45	100	41.43	12.72
	2000	500	45	100	45.07	11.85
	2000	300	25	100	46.28	11.59
	3000	300	45	100	43.49	12.27
	3000	350	45	100	44.09	12.10
	3000	450	45	100	45.99	11.68
Base	3000	500	45	100	46.45	11.56
	3000	600	45	100	47.13	11.40
	3000	800	45	100	47.53	11.31
	3000	300	25	100	49.09	11.05
	3000	350	25	100	49.42	10.95
	3000	450	25	100	50.77	10.67
	3000	500	25	100	51.14	10.50
	3000	600	25	100	51.58	10.48
	3000	800	25	100	51.67	10.47
	4000	300	45	100	43.78	12.19
	4000	500	45	100	47.08	11.43
	4000	300	25	100	49.40	10.96
3	--	--	--	--	46.42	12.45
4	3000	500	46	100	41.96	12.58
	3000	400	26	100	47.12	11.44

1 - Longwall plus 2 continuous miners (16 feet-full face cut)
2 - Longwall plus 2 continuous miners (16 feet-two bench method)
3 - 6 continuous miners
4 - Longwall plus 2 continuous miners (extracting 11 feet only)

*Required selling price for 25% ROI; the ROI shown above is realized when coal sells for $18 per ton.



The longwall face equipment is installed only after three years development, when in a room-and-pillar mining, more sections can be added after two years. The result is a delay in production.

FIGURE 6-9

Plan 2 and Plan 3 Yearly Production Levels continuous miners) has an appreciably larger annual operating cost than Plan 2 (longwall) but Plan 3 (Figure 6-10) has significantly lower capital costs than Plan 2 (Figure 6-11). The return on investment is essentially the same (46.4%) for both plans, yet the required selling price for coal (to return 25% on investment) is higher for Plan 3 ($12.45 per ton) than Plan 2 ($11.56 per ton) (Table 6-3). For a 25% ROI, Plan 2 (longwall) is more economical than Plan 3 (six continuous miners).

4. Comparison of the ROI and the required selling price per ton (to earn 25% ROI) between cases, where the longwall parameters in Plan 1 were varied, is shown on Table 6-5.

The results of longwall parametric variations to Plan 2 (Figure 6-12 and Table 6-4) are summarized as follows:

Panel length. The longer the panel, the the more economical. This reduces the nonproductive longwall moves. Costs that would occur as the panel length becomes excessive are not included.

Face length. The longer the face length, the more economical. This reduces the nonproductive longwall sumping times. The increased production per shift, by reducing the face turn-around sumping times, offsets the increased capital and maintenance costs for shields and face conveyors as the face is lengthened, but costs that would occur as the face length becomes excessive are not included.

Sumping time. Plan 2, base case, uses a longwall sumping time of 45 minutes per sump. This includes 20 minutes of delay time during sumping, in addition to the 12 minutes required for nonproductive sumping operations. When this nonproductive sumping time is reduced to 12 minutes for a total sumping time of 25 minutes, production increases per shift and, of course, economy increases by about $1.

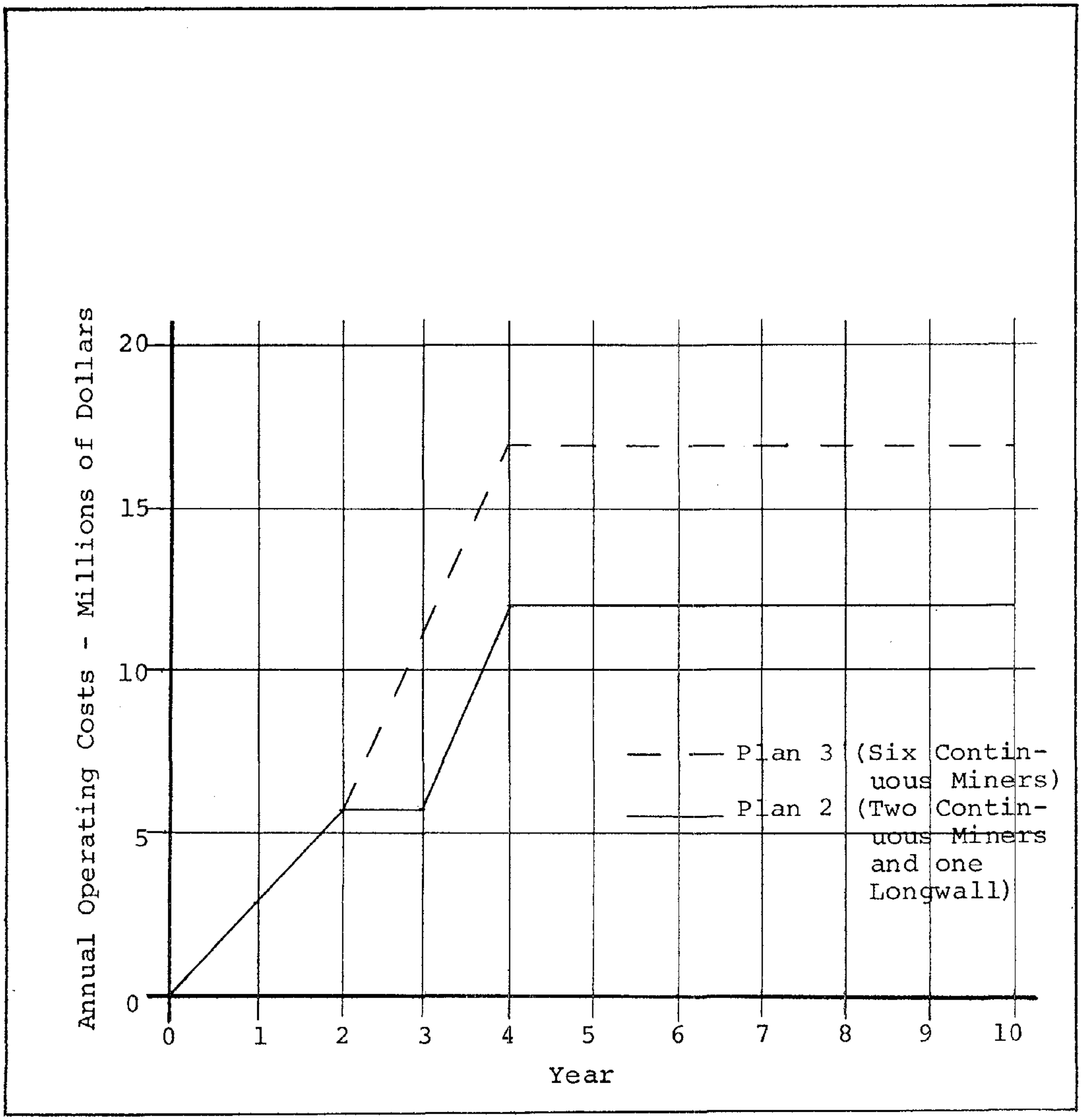


FIGURE 6-10
Plan 2 and Plan 3 Annual Operating Cost Levels
(in millions of dollars per year)

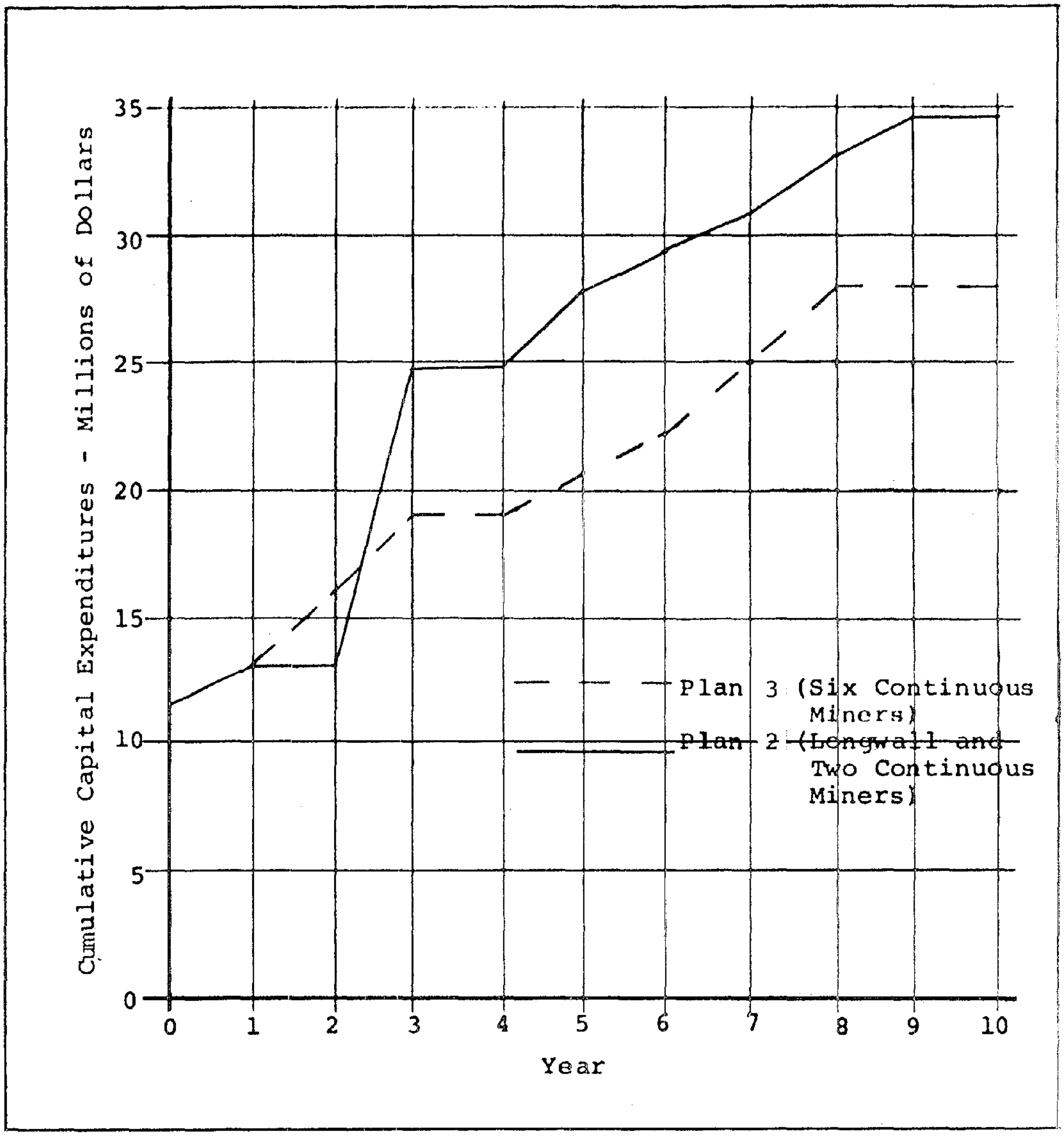


FIGURE 6-11

Plan 2 and Plan 3 Cumulative Capital Expenditures

TABLE 6-5

SENSITIVITY ANALYSIS - PLAN 2

REQUIRED SELLING PRICE FOR PARAMETER VARIATIONS

BASE = 500 FOOT X 3000 FOOT PANEL

Longwall Parameter	Value	% of Base	Required Selling Price $/Ton
Longwall Production Rate (tons per shift)	1389	50%	$14.52
	2362	85	12.28
	2778	Base	11.56
	3195	115	10.99
Sump Time (minutes)	25	61	10.50
	45	Base	11.56
Face Length (feet)	300	60	12.27
	350	70	12.10
	450	90	11.68
	500	Base	11.56
	600	120	11.40
	800	160	11.31
Panel Length (feet)	2000	67	11.85
	3000	Base	11.56
	4000	133	11.43

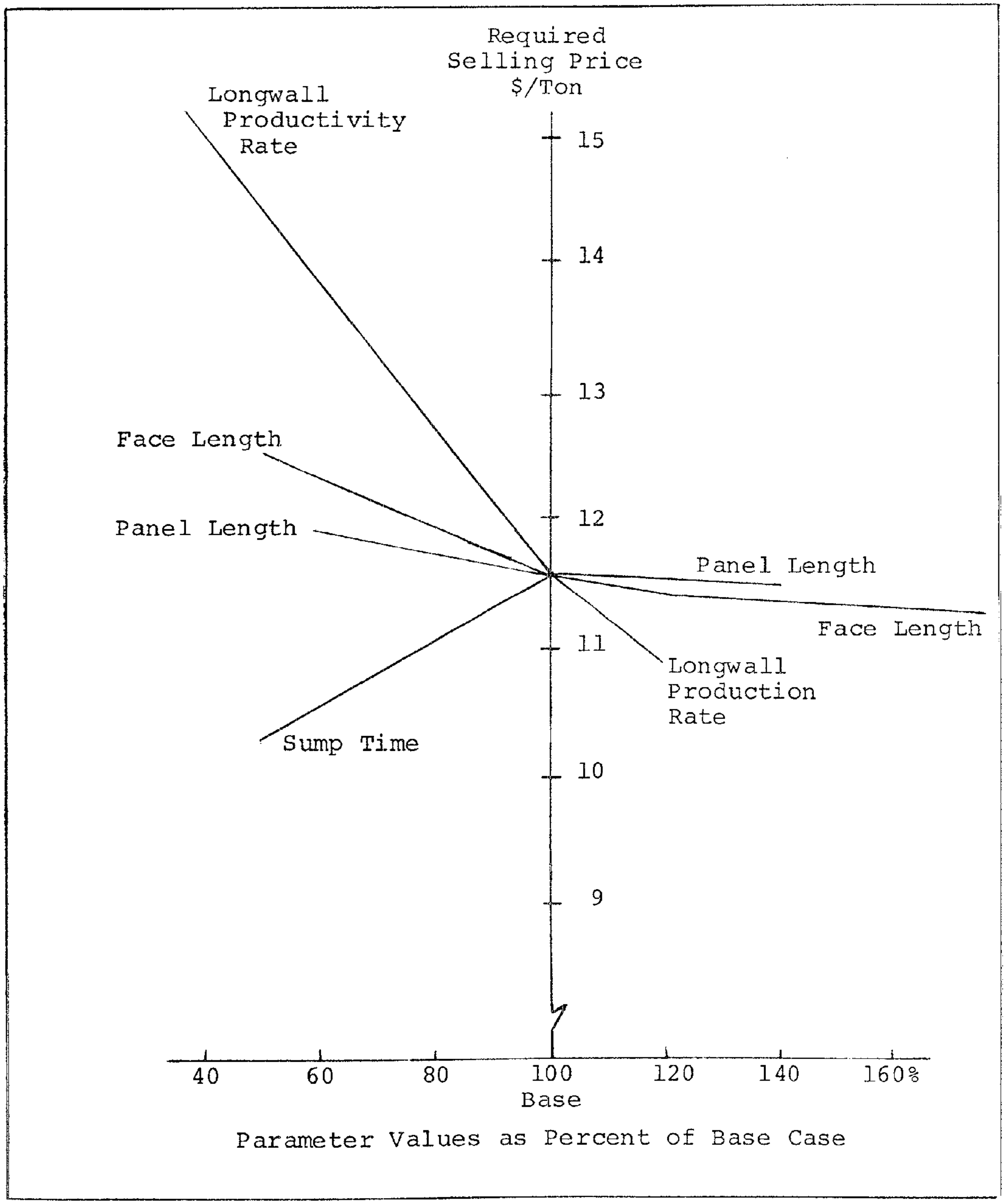


FIGURE 6-12

Required Selling Price Sensitivity With Long-wall Parameters Varied One at a Time - Plan 2

Production. Ten million dollars of pre-production development and other up-front capital costs are included, making production per shift a sensitive positive factor towards economy. The production from the longwall Plan 2 base case was varied from 50% to 115% production to observe the economic sensitivity to longwall production. Results vary from \$14.52 (50%) to \$10.99 (115%) (Table 6-5). Plan 2 versus Plan 3 crossover point occurs at about 83%.

In the selected case, 6200 tons per day is the average annual production a 500 foot longwall should yield to compete with a room-and-pillar method producing an average of 850 tons/shift. This is feasible (with effective dust control) if sloughing of the face does not cause important delays. The application of a "one web back" shield support and a two bench method, will help to achieve a satisfactory control of the coal face.

The results of this study indicate that the production and economics of such longwall mining operations can be better than room-and-pillar or multi-slice longwall mining methods in the extraction of such thick seams.

Resource recovery of thick coal reserves will increase through single pass longwall mining method by 50%.

7.0 CONCLUSIONS

The states of Alaska, Colorado, Montana, New Mexico, Utah and Wyoming contain an estimated 125 billion tons of in place coal reserves which can possibly be mined in a single pass (10 to 20 feet in thickness). This figure represents approximately 25% of the total identified reserves in these states.

The present practice of extracting coal by the room-and-pillar method limits the mined thickness to a height of 9 feet. When longwall mining is considered, total extraction of the seam is not planned, because of uncertainties stemming from the application of an untested system. The maximum thickness appears to be 4.3 m (14 ft.).

Several other countries are currently longwall mining thick seams. Among these are the USSR, Poland, Germany, Australia and the United Kingdom. These countries have developed methods to mine the thick seams by longwall mining in sucessive slices or in one pass, with or without sublevel caving. Their experience shows a steady increase of the upper limit of extraction height from an economic point of view. Shearers have been developed up to 5.4 m and shield roof supports have been manufactured for 6 m heights.

Geologic parameters such as coal strength, presence of cleats and fissures, composition and strength of the strata, and type of coal are important factors in mining thick seams.

In the United States, where the coal mines are not as deep as in Europe, the conditions are technically favorable for successful implementation of a single pass longwall method up to 16 feet. On the other hand, market conditions are such that a single pass longwall system should be very productive in the United States.

The economic study shows that 6,200 tons per day is the average annual production that a 500 foot longwall should yield to compete with a room-and-pillar method producing an average of 850 tons/shift. This is feasible (with effective dust control) if sloughing of the face does not cause important delays. To avoid face sloughing, it is recommended that instead of cutting the entire thickness with two large diameter drums, the top bench be mined first with a double ended drum shearer equipped with smaller diameter drums. The application of "one web back" shield supports advancing before mining the bottom bench will help to achieve a satisfactory control of the coal face.

U. S. regulations have governed the selection of a retreating longwall with a multiple entry driven on the seam floor, 9 to 10 feet high. A single entry system or an advancing longwall system would be required for driving entries in the total thickness.

The double entry system has been preferred to a three entry system because it includes a yielding chain of pillars to ease roof caving when the pillars are not extracted.

In the selection of equipment for longwall mining a thick seam, the following are recommended:

- o Reduction of the unsupported roof area by using a steering shearer underframe to control the angle of the cut.
- o Limitation of the total length of the support canopy at 3.5 meters. Selection of two-leg shields.
- o Use face conveyor underframe with the support bases advancing under the conveyor for closeness to the coal face.
- o Increase the shield width by 50%, thereby reducing the face support cost and simplifying the shield design.
- o Dust control by using an automatic variable water delivery system which adapts the amount of water to the quantity of coal produced.

The results of this study indicate that the production and economics of such longwall mining operations would be better than room-and-pillar or multi-slice longwall mining methods in the extraction of such thick seams.

Resource recovery of thick coal reserves will increase through the use of the single pass longwall mining method. This fact is not included in the economic analysis.

It is anticipated that some of the mine operators in the western United States will commence mining 16-17 feet thick seams and the technical and economic information contained in this study will be utilized in such operations.

8.0 REFERENCES

Adam, R., R. Coeuillet, and M. Jacob, "How to Select Powered Roof Support," Proceedings of the Third International Conference on Strata Control, New York, 1964.

Adam, R., "Mechanization of Longwall Faces with Sublevel Caving," Gluckauf, October 25, 1973.

Adam, R., "French Thick Seams Mining Practices," Symposium on Thick Seam Mining by Underground Methods, Australian Institute of Mining and Metallurgy (AIMM), Rockhampton, Australia, September, 1976.

Adam, R., and J. F. Raffoux, "Roof Support and Strata Control in Thick or Steep Seams," Proceedings of Luxembourg Symposium on High Output Coal Winning Districts, September 29, 1976.

Adam, R., and J. F. Raffoux, "Experience with Roof Bolting in French Coal Mines," Gluckauf, Vol. 108, No. 23, 1972.

Ahcan, R., "Progress Made in the Mechanical Extraction of Thick Coal Seams in Yugoslavia," Gluckauf, Vol. 112, No. 3, 1976, pp. 127-132.

Ahcan, R., and S. Janezic, "Recent Experience with Sublevel Caving Techniques in Yugoslavia," presented at The First International Symposium on Thick and Steep Seam Coal Mining, London, England, May 18-21, 1980.

Arabagian, O., "The Mining of Thick Seams in Slices, in the Coal District Petrosani," paper presented at the Hemscheidt-Symposium Wuppertal, West Germany, October 30 and 31, 1979, not published.

Auriol, P., "Roadway Planning and Maintenance and Layout of the Workings," Fifth International Strata Control Conference, 1972.

Bartholmai, H., Mining Methods for Thick Strata in Deep Mining in Eastern Europe and the Soviet Union, Journal Bergbau-Wissenschaften 18, No. 12, 1971, pp. 399-402.

Beckmann, K., "The Extraction by Longwall Mining," Gluckauf, Vol. 114, No. 18, 1978, pp. 796-801.

Benech, M., "Recent Developments in 'Soutirage' Sublevel Mining," paper presented at The First International Symposium on Thick and Steep Seam Coal Mining, London, England, May 18-21, 1980.

Boldt, H., R. Sander, and F. Eichbaum, "The Cutting Disc Loader as Forerunner in Integrated Longwall Face Mechanization," Gluckauf, Vol. 114, No. 20, 1978, pp. 889-896.

"Calcul de la Puissance Motrice des Convoyeurs Blindes aux H.B.L.," Publications Techniques des Charbonnages de France, Envoi No. 1, 1979, pp. 53-57.

Claes and Weinheimer, "Face Slewing in German Coalmines," Gluckauf, June 5, 1980.

Clanzy, T. K., "Safety in Mines, Past, Present and Future, or Can We Profit from Experience?" Mining Technology, August, 1979, pp. 269-381.

Courtney, W. G., "Preventing Frictional Ignitions," Coal Mining and Processing, January, 1981.

Curth, E. A., Safety Aspects of Longwall Mining in the Illinois Coal Basin, U. S. Bureau of Mines, IC 8776, 1978.

Dietrich, W., "The Mining of Thick and Inclined Coal Deposits in Warndt Mine," ("Abbau Machtiger Floze in Geneigter Lagerung auf dem Bergwerk Warndt") Gluckauf, Vol. 113, 1977, pp. 111-114.

Dietrich, W., and E. Weyand, "Steep Seam Longwall Mining in Thick Seams with Full Caving in the Warndt Mine," presented at The First International Symposium on Thick and Steep Seam Coal Mining, London, England, May 18-21, 1980.

Euler, W. J., "Improving Longwall Entry Development and Operation," Mining Congress Journal, May, 1979, pp. 21-27.

Fissgus, S., "The Mining of Thick Seams in the Bituminous Coal Mining Industry of Rumania," Gluckauf, Vol. 114, No. 19, 1978, pp. 852-854.

Gapanovic, L. N., et al, "Methods for Improving the Mining of Thick Coal Seams," Ugol 42, No. 7, 1967, pp. 53-57.

Gapanovic, L. N., "Prospects for the Future Development of Full Mechanization in Connection with the Extraction of Thick Flat Bedded Coal Seams without Division into Slices," Journal Naucn. Soobsc. JGD Skocinskogo, No. 145, 1977, pp. 45-52.

Goettig, P., "Shield Supports and Cutting-Type Winning as Prerequisities for the Extraction of Very Thick Seams," Gluckauf, Vol. 113, No. 5, 1977, pp. 259-266.

Grotowsky, U., "New Techniques for Control in Haulageways," paper presented at the Sixth International Strata Control Conference, Banff, Canada, September 23-28, 1977.

Haarmann, K., and V. Mertens, "Maschineller Flozstreckenvortrieb," Gluckauf, Vol. 115, No. 9, May, 1979, pp. 397-403.

Hahn, L., "Recent Development of Face Supports," paper presented at the Sixth International Strata Control Conference, Banff, Canada, September 23-28, 1977.

Hess, H., "Report on the Face Finding Mission of a Ruhrkohle Delegation to the USSR from June 12 to June 30, 1978," not published.

Hughes, E., "Experience at Daw Mill Colliery," paper presented at The First International Symposium on Thick and Steep Seam Coal Mining, London, England, May 18-21, 1980.

Irresberger, H., "Strata Control in Face and Road at Deep Levels," Gluckauf, Vol. 116, March, 1980, pp. 88-91.

Jacobi, O., G. Everling, N. Buschmann, et al, "Practice of Ground Control," Gluckauf, Edition 1976, Essen, West Germany, C4, p. 190, and D4, pp. 248-249.

Jankowski, J., and S. Gajewski, "One Million Tons of Coal from a Longwall Face Worked by the Caving Method in a 4.5 m Thick Seam at the Zabrze Colliery, (Poland)," Gluckauf, June 5, 1980.

Jaspal, J. S., "Two Lift Longwall Mining Method for Mid Continent Resources, Inc.," paper presented at the AIME Annual Meeting, Chicago, February, 1981.

Josien, J. P., and C. Gouilloux, "Present and Future Roof Control and Support in Longwall Faces in French Coal Mines," Colliery Guardian International, October, 1978, pp. 49-58.

Kammer, W., "Calculation of Convergence and Support Systems Planning in Gateroads by Means of an EDP Program," Gluckauf, Vol. 116, March, 1980, pp. 91-94.

Konetzny, H., "Extraction of a Thick Seam Using Shield Support and Double-Ended Ranging Drum Shearer Loader," Gluckauf, Vol. 110, No. 12, 1974, pp. 470-476.

Kraft, B., "Training to Cover all the Bases," Coal Mining and Processing, May, 1978, pp. 50-53.

Kundel, H., "High Capacity Longwall Faces in West German Bituminous Coal Mines," Gluckauf, Vol. 115, No. 12, 1979, pp. 593-594.

Kundel, H., "Status of the Longwall Face Technology in the West German Bituminous Coal Mining Industry in 1977," Gluckauf, Vol. 114, No. 18, 1978, pp. 787-795.

Kundel, H., "Status of the Longwall Face Technology in the West German Bituminous Coal Mining Industry in 1978," Gluckauf, Vol. 115, No. 15, 1979, pp. 748-758.

Kurrat, D., "Longwall Caving Operation Working to the Dip in a 3 to 4 m Thick Coal Seam," Gluckauf, Vol. 112, No. 13, 1978, pp. 746-752.

Kuschel, K. H., "Experience Gained in the Mining of Thick Seams in the Ruhr Coal Mining Industry," Berg- und Huttenmannische Monatshefte, 115, No. 8., 1970, pp. 202-211.

Lehmann, G., "Longwall Mining in Big and Pitching Seams in Germany," Mining Congress Journal, September, 1980.

Levitt, E., "Subsidence Consideration Over Steep and Thick Underground Coal Workings in the U.K.," paper presented at The First International Symposium on Thick and Steep Seam Coal Mining, London, England, May 18-21, 1980.

Matson, T. K., and D. H. White, Jr., The Reserve Base of Coal for Underground Mining in the Western United States, USBM IC8678, 1975.

Mills, L. J., "The Influence of Mining Machinery Development on Production Safety and Health," Mining Technology, August, 1979, pp. 383-394.

Mundell, R. L., R. A. Jankowski, et al, "Respirable Dust Control on Longwall Mining Operations in the United States," presented at the Second International Mine Ventilation Congress, Reno, Nevada, November, 1980.

"National Reports," Sixth International Strata Control Conference Banff, Canada, September 23-28, 1977.

Oil Shale Mining Cost Analysis, Vol. I, Surface Restoring Process, DOE-ET-12532-1, January, 1981.

Parkes, D. M., "History of Hydraulic Mining at Kaiser," presented at The First International Symposium on Thick and Steep Seam Coal Mining, London, England, May 18-21, 1980.

Pluta, L., Hard Coal Mining in Poland, Ministry of Mining, The Polish People's Republic, Central Mining Institute, Katowice, October, 1979.

Sandru, L., V. Popescu, and R. Ghiata, The Mining of Thick Lignite Seams in the Socialistic Republic of Romania, Report ST/ECE/Coal/29, Vol. II, 1968, pp. 53-61.

Schmidt-Koehl, W., "Betriebskonzentration im Saarbergbau," ("Concentration in Saar Coalfield") Gluckauf, Vol. 112, 1976, pp. 555-564.

Steinbach, H., "Utilization of Shield Support in Critical Face Support Areas," Gluckauf, Vol. 112, No. 10, 1976, pp. 565-569.

Sudoplatov, A. P., et al, Proposals for Improved Mining Methods Applying to Thick Coal Seams in the Soviet Union, Report ST/ECE/Coal/29, Vol. II, 1968, pp. 81-100.

Szczurowski, A., and J. Logas, "Growth Trends in Mining Thick Seams in the Polish Coal Mining Industry," presented at The First International Symposium on Thick and Steep Seam Coal Mining, London, England, May 18-21, 1980.

Szczurowski, A., and J. Logas, "Longwall Mining of Coal Seams in Poland," Colliery Guardian Coal International, January, 1979, pp. 44-49.

Thrush, P. W., A Dictionary of Mining, Mineral, and Related Terms, U. S. Department of the Interior, 1968.

Wilson, A. H., "Support Load Requirements on Longwall Faces," The Mining Engineer, June, 1975, pp. 479-491.

Wilson, A. H., "Various Aspects of Longwall Roof Support," Colliery Guardian, Vol. 226, April, 1978, pp. 50-56.

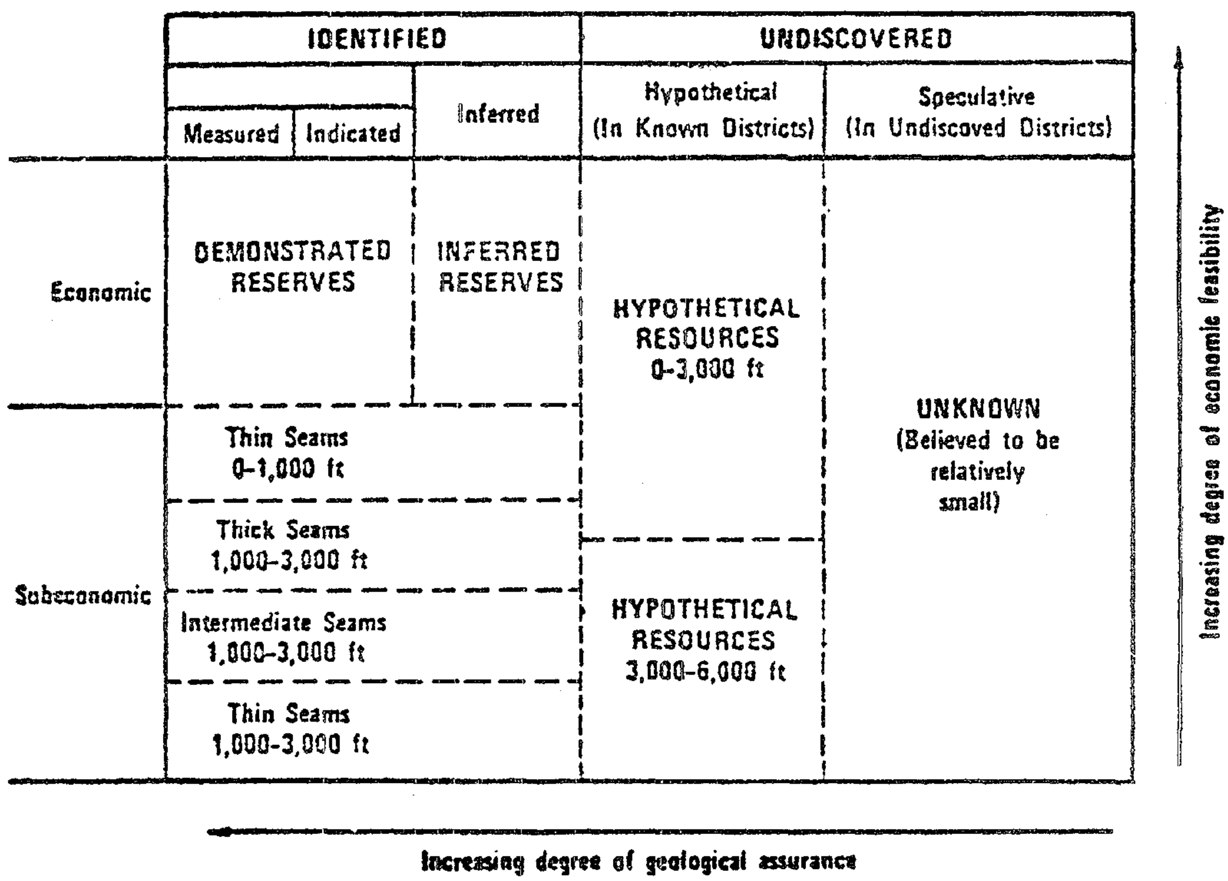
APPENDIX A

RESOURCE CLASSIFICATIONS

AND DEFINITIONS

Resource Classifications

The U. S. Bureau of Mines (USBM) and the U. S. Geological Survey (USGS) have defined a method of classification for coal resources. This system classifies coal beds in terms of their degree of geological identification and economic or technological feasibility of recovery. The system is depicted below.



Continental U. S. Coal Resources

For the purpose of this study, only identified resources are considered. Estimates of undiscovered resources are too unreliable to justify inclusion in this study. Inferred resources are also included.

The terms used in the classification system are defined as follows:

Resource -- A concentration of coal in or on the earth's crust in such form that economic extraction is currently or potentially feasible.

Identified resources -- Specific bodies of coal where location, quality and quantity are known from geologic evidence supported by engineering measurements with respect to the demonstrated category.

Undiscovered resources -- Unspecified bodies of coal surmised to exist on the basis of broad geologic knowledge and theory.

Reserve -- That portion of the identified coal resource that can be economically and legally mined at the time of determination -- also referred to as "recoverable reserve."

Reserve base -- The component of the identified coal resource from which the reserve is derived by recoverability calculations.

The following definitions for measured, indicated, and inferred are applicable to both the Reserve and Identified-Subeconomic resource components.

Measured -- Coal for which estimates of the quality and quantity have been computed, within a margin of error of less than 20 percent; from sample analyses and measurements from closely spaced and geologically well-known sample sites.

Indicated -- Coal for which estimates of the quality and quantity have been computed partly from sample analyses and measurements and partly from reasonable geologic projections.

Inferred -- Coal in unexplored extensions of "demonstrated" resources for which estimates of the quality and size are based on geologic evidence and projection.

Identified-subeconomic resources -- Coal beds that are not "reserves," but may become so as a result of changes in economic and legal conditions.

Paramarginal -- The portion of "subeconomic resources" that (1) borders on being economically producible, or (2) is not commercially available solely because of legal or political circumstances.

Submarginal -- The portion of "subeconomic resources" that would require a substantially higher price (more than 1.5 times the price at the time of determination) or a major cost reducing advance in technology.

Hypothetical resources -- Undiscovered coal that may reasonably be expected to exist in a known mining district under known geologic conditions. Exploration that confirms their existence and reveals quantity and quality will permit their reclassification as a "reserve" or "identified-subeconomic resource."

Speculative resources -- Undiscovered coal that may occur either in known types of deposits in a favorable geologic setting where no discoveries have been made, or in as yet unknown types of deposits that remain to be recognized. Exploration that confirms their existence and reveals quantity and quality will permit their reclassification as "reserves" or "identified-subeconomic resources."

APPENDIX B

THICK SEAM DATA BASE

PART I -- BITUMINOUS RESOURCES

For bituminous coal, USGS tonnage estimates are given for resources greater than 42 inches in thickness.

STATE COLORADO

COUNTY ARCHULETA, MONTEZUMA and LA PLATA

COAL FIELD/REGION SAN JUAN REGION -- DURANGO FIELD/FRUITLAND FORMATION

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
----	500-1000	Bit	10-20	N.A.	N.A.	1,001.2	48.8	488.59		
----	1-2000	Bit	10-20	N.A.	N.A.	1,888.8	48.8	921.73		
----	2-3000	Bit	10-20	N.A.	N.A.	14,285.3	48.8	6,971.23		
							COUNTY TOTAL	8,381.55		

SOURCES:

Resource and Thickness data from: Fassett and Hinds, 1971, pp. 55-70

STATE COLORADO

COUNTY DELTA

COAL FIELD/REGION SOMERSET/GRAND MESA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
B	0-1000	Bit	6-25	40.90	51.43	92.33	52.6	48.57		
B	1-2000	Bit	6-25	17.78	85.27	103.05	52.6	95.43		
B	2-3000	Bit	6-25	----	.29	.29	52.6	.15		
C	0-1000	Bit	6-25	11.45	17.81	29.26	52.6	15.39		
C	1-2000	Bit	6-25	12.20	47.84	60.04	52.6	31.58		
C	2-3000	Bit	6-25	----	10.34	10.34	52.6	5.44		
							COUNTY TOTAL	196.56		

SOURCES:

Thickness: Keystone, 1979, p. 479
Keystone, 1978, p. 468

STATE COLORADO

COUNTY GARFIELD

COAL FIELD/REGION BOOK CLIFFS/GRAND HOGBACK

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Allen	0-1000	Bit	14-22	27.04	15.77	42.81	75	32.11	To 80	
Allen	1-2000	Bit	14-22	29.75	20.28	50.03	75	37.52	To 80	
Allen	2-3000	Bit	14-22	28.28	25.02	53.30	75	39.98	To 80	
D	0-1000	Bit	max 12	15.19	5.08	20.27	23.5	4.76	To 80	
D	1-2000	Bit	max 12	18.65	8.89	27.54	23.5	6.47	To 80	
D	2-3000	Bit	max 12	19.41	10.90	30.31	23.5	7.12	To 80	
E	0-1000	Bit	max 18	14.23	12.34	26.57	55.2	14.67	To 80	
E	1-2000	Bit	max 18	16.66	16.63	33.29	55.2	18.38	To 80	
E	2-3000	Bit	max 18	13.15	16.82	29.97	55.2	16.54	To 80	
							COUNTY TOTAL	CONT.		

SOURCES:

Thickness: Erdmann, 1934
Gale, 1910, pp. 109-136

Dip: Hileman, 1970, pp. 38-39

STATE COLORADO

COUNTY GARFIELD

COAL FIELD/REGION BOOK CLIFFS/GRAND HOGBACK

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Wheeler	0-1000	Bit	18-45	48.59	38.79	87.38	7	6.12	To 80°	
Wheeler	1-2000	Bit	18-45	50.56	56.02	106.58	7	7.46	To 80°	
Wheeler	2-3000	Bit	18-45	48.47	71.99	120.46	7	8.43	To 80°	
Cameo	0-1000	Bit	max 21	115.31	123.60	238.91	57.1	136.42	5-20	
Cameo	1-2000	Bit	max 21	27.52	103.58	131.10	57.1	74.86	5-20	
Cameo	2-3000	Bit	max 21	----	27.95	27.95	57.1	15.96	5-20	
							COUNTY TOTAL	426.80		

SOURCES:

Thickness: Erdmann, 1934
Gale, 1910, pp. 109-136

Dip: Hileman, 1970, pp. 38-39

STATE COLORADO

COUNTY GUNNISON

COAL FIELD/REGION SOMERSET

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
B	0-1000	Bit	6-25	192.24	98.01	290.25	52.6	152.67		
B	1-2000	Bit	6-25	147.80	316.48	464.28	52.6	244.21		
B	2-3000	Bit	6-25	34.79	146.32	181.11	52.6	95.26		
C	0-1000	Bit	6-25	90.91	97.89	188.80	52.6	99.31		
C	1-2000	Bit	6-25	49.36	113.31	162.67	52.6	85.56		
C	2-3000	Bit	6-25	6.39	65.18	71.57	52.6	37.65		
							COUNTY TOTAL	714.66		

SOURCES:

Thickness: Keystone, 1979, p. 479.
Keystone, 1978, p. 468.

STATE COLORADO

COUNTY MESA

COAL FIELD/REGION BOOK CLIFFS

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Cameo	0-1000	Bit	max 21	96.52	112.47	208.99	57.1	119.33	5-20	
Cameo	1-2000	Bit	max 21	12.63	129.46	142.09	57.1	.81.13	5-20	
Cameo	2-3000	Bit	max 21	----	48.91	48.91	57.1	27.93	5-20	
							COUNTY TOTAL	228.39		

SOURCES:

Thickness: Erdmann, 1934

STATE COLORADO

COUNTY MOFFAT

COAL FIELD/REGION DANFORTH HILLS

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Black Diamond Group	0-1000	Bit	5-18	3.93	----	3.93	61.5	2.42		
Black Diamond Group	1-2000	Bit	5-18	6.83	----	6.83	61.5	4.20		
Fairfield Group	0-1000	Bit	6.5-22	338.92	----	338.92	64.5	218.60		
Fairfield Group	1-2000	Bit	6.5-22	127.85	----	127.85	64.5	82.46		
Goff Group	0-1000	Bit	max 18	56.83	92.72	149.55	55.2	82.55		
Classed By Zone	0-3000	Bit	0-22	----	----	3,525.88	38.5	1,357.46		
							COUNTY TOTAL	1,747.69		

SOURCES:

Thickness: Keystone, 1977, p. 585
Hancock, 1930

STATE COLORADO

COUNTY MOFFAT

COAL FIELD/REGION YAMPA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Black Diamond Group	0-1000	Bit	max 12.5	61.24	----	61.24	27.8	17.02		
Black Diamond Group	1-2000	Bit	max 12.5	4.18	----	4.18	27.8	1.16		
Lower Group	0-1000	Bit	max 12.5	34.37	----	34.37	27.8	9.55		
Fairfield Group	0-1000	Bit	max 16	835.37	----	835.37	48	400.98		
Fairfield Group	1-2000	Bit	max 16	28.16	----	28.16	48	13.52		
Middle Group	0-1000	Bit	max 16	28.31	----	28.31	48	13.59		
F Zone	0-1000	Bit	max 16	62.83	----	62.83	48	30.16		
F Zone	1-2000	Bit	max 16	16.58	----	16.58	48	7.96		
H Zone	0-1000	Bit	6-11	69.84	----	69.84	20	13.97		
H Zone	1-2000	Bit	6-11	90.88	----	90.88	20	18.18		
							COUNTY TOTAL	CONT.		

SOURCES:

Thickness: Keystone, 1977, p. 585

STATE COLORADO

COUNTY MOFFAT

COAL FIELD/REGION YAMPA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Wadge	1-2000	Bit	max 12	----	77.71	77.71	23.5	18.26		
Classed By Zone	0-3000	Bit	ave 5 max 16	----	----	7,896.21	27	2,131.98		
							COUNTY TOTAL	2,677.33		

SOURCES:

Thickness: Keystone, 1977, p. 585

STATE COLORADO

COUNTY RIO BLANCO

COAL FIELD/REGION DANFORTH HILLS

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Black Diamond Group	0-1000	Bit	5-18	42.20	12.98	55.18	61.5	33.94		
Black Diamond Group	1-2000	Bit	5-18	45.81	39.04	84.85	61.5	52.18		
Black Diamond Group	2-3000	Bit	5-18	18.06	25.86	43.92	61.5	27.01		
Fairfield Group	0-1000	Bit	6.5-22	265.10	227.76	492.86	64.5	317.89		
Fairfield Group	1-2000	Bit	6.5-22	120.46	409.46	529.92	64.5	341.80		
Fairfield Group	2-3000	Bit	6.5-22	30.12	122.30	152.42	64.5	98.31		
Goff Group	0-1000	Bit	max 18	111.15	155.20	266.35	55.2	147.03		
Goff Group	1-2000	Bit	max 18	37.85	106.81	144.66	55.2	79.85		
Goff Group	2-3000	Bit	max 18	5.17	30.98	36.15	55.2	19.95		
							COUNTY TOTAL	CONT.		

SOURCES:

Thickness: Keystone, 1977, p. 585
Hancock, 1930

STATE COLORADO

COUNTY RIO BLANCO

COAL FIELD/REGION DANFORTH HILLS

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Lion Canyon Group	0-1000	Bit	4-34	68.99	22.73	91.72	33.3	30.54		
Lion Canyon Group	1-2000	Bit	4-34	39.04	17.25	56.29	33.3	18.74		
Lion Canyon Group	2-3000	Bit	4-34	27.38	15.67	43.05	33.3	14.34		
							COUNTY TOTAL	1,181.58		

SOURCES:

Thickness: Keystone, 1977, p. 585
Hancock, 1930

STATE COLORADO

COUNTY ROUTT

COAL FIELD/REGION YAMPA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
H Zone	0-1000	Bit	6-12.5	139.22	----	139.22	38.5	53.60		
H Zone	1-2000	Bit	6-12.5	100.30	----	100.30	38.5	38.62		
J	0-1000	Bit	5-12	66.51	----	66.51	28.6	19.02		
Wadge	0-1000	Bit	3.5-12	547.50	156.01	703.51	23.5	165.32		
Wadge	1-2000	Bit	3.5-12	236.80	216.49	453.29	23.5	106.52		
Wadge	2-3000	Bit	3.5-12	16.84	32.65	49.49	23.5	11.63		
Bear River	0-1000	Bit	6-12	71.50	----	71.50	33.3	23.81		
Bear River	1-2000	Bit	6-12	37.48	----	37.48	33.3	12.48		
G Zone	0-1000	Bit	6-11	46.59	12.96	59.55	33.3	19.83		
Bed P		Bit	5-11					---		
							COUNTY TOTAL	CONT.		

SOURCES:

Thickness: Bass, N.W. et al, 1955, plates

STATE COLORADO

COUNTY ROUTT

COAL FIELD/REGION YAMPA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
H Zone	0-1000	Bit	6-12	20.52	----	20.52	33.3	6.83		
H Zone	1-2000	Bit	6-12	----	58	.58	33.3	.19		
#1 Bed	0-1000	Bit	3.5-13	130.16	----	130.16	31.6	41.13		
#1 Bed	1-2000	Bit	3.5-13	84.82	----	84.82	31.6	26.80		
#2 Bed	0-1000	Bit	3.5-15	156.01	----	156.01	43.5	67.86		
#2 Bed	1-2000	Bit	3.5-15	68.12	----	68.12	43.5	29.63		
Wolf Creek	0-1000	Bit	3.5-17	497.11	309.85	806.96	51.9	418.81		
Wolf Creek	1-2000	Bit	3.5-17	335.91	503.84	839.75	51.9	435.83		
Wolf Creek	2-3000	Bit	3.5-17	----	80.43	80.43	51.9	41.74		
Bed P		Bit	5-11					----		
							COUNTY TOTAL	CONT.		

SOURCES:

Thickness: Bass, N.W. et al, 1955, plates

STATE COLORADO

COUNTY ROUTT

COAL FIELD/REGION YAMPA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Lorella		Bit	3.5-11					---		
Seymour		Bit	3.5-18					---		
Uncorre-lated	0-1000	Bit	Ave 6.4 3.5-18	529.98	---	529.98	34.5	182.84		
Uncorre-lated	1-2000	Bit	Ave 6.4 3.5-18	348.20	---	348.20	34.5	120.06		
Classed By Zone	0-3000	Bit	Ave 6.4 3.5-18	N.A.	N.A.	177.52	34.5	61.25		
							COUNTY TOTAL	1,883.80		

SOURCES:

Thickness: Bass, N.W. et al, 1955, plates

STATE New Mexico

COUNTY Colfax

COAL FIELD/REGION Raton

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Raton		Bit	14.5 max					East 1-5° West 0 to overturned		
Vermejo		Bit	2-10					East 1-5° West 0 to overturned		
Sugarite		Bit	3.5-5.9					East 1-5° West 0 to overturned		
York Canyon		Bit	3-13					East 1-5° West 0 to overturned		
Tin Pan		Bit	8 max					East 1-5° West 0 to overturned		
Yankee		Bit	4 (ave)					East 1-5° West 0 to overturned		
Upper Left Fork		Bit	5-8					East 1-5° West 0 to overturned		
Lower Left Fork		Bit	7 max					East 1-5° West 0 to overturned		
Cotton-wood Canyon		Bit	7 max					East 1-5° West 0 to overturned		
Ancho Canyon		Bit	9 max					East 1-5° West 0 to overturned		
							COUNTY TOTAL	Cont.		

SOURCES:

Thickness: Lee, 1924, pp. 152-165
Pillmore, 1969, pp. 131-141

Dip: Pilmore, 1969, p. 131

STATE New Mexico

COUNTY Colfax

COAL FIELD/REGION Raton

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Chimney Divide		Bit	4 max						East 1-5° West 0 to overturned	
Uncorre-lated	0-1000	Bit	to 14.5	642.69	235.12	877.81	26.5	232.62	East 1-5° West 0 to overturned	
Uncorre-lated	1-2000	Bit	to 14.5	69.40	715.30	784.70	26.5	207.95	East 1-5° West 0 to overturned	
Uncorre-lated	2-3000	Bit	to 14.5	---	376.90	376.90	26.5	99.88	East 1-5° West 0 to overturned	
							COUNTY TOTAL	540.45		

SOURCES:

Thickness: Lee, 1924, pp. 152-165
Pillmore, 1969, pp. 131-141

Dip: Pilmore, 1969, p. 131

STATE Utah

COUNTY Carbon County

COAL FIELD/REGION Book Cliffs

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Lower Sunnyside	0-3000	Bit	3-18	N.A.	N.A.	589.94	55	324.47	Average4-8 Max 15	
Upper Sunnyside	0-3000	Bit	Max 6.5						Average4-8 Max 15	
Kenilworth	0-3000	Bit	to 20						Average4-8 Max 15	
Rock Canyon	0-3000	Bit	4-10						Average4-8 Max 15	
Gilson	0-3000	Bit	4-13	N.A.	N.A.	52.08	33	17.19	Average4-8 Max 15	
Castlegate C	0-3000	Bit	6-13						Average4-8 Max 15	
Royal Blue	0-3000	Bit	5-9						Average4-8 Max 15	
Castlegate B	0-3000	Bit	to 13						Average4-8 Max 15	
Castlegate A	0-3000	Bit	to 20						Average4-8 Max 15	
Fish Creek	0-3000	Bit	to 5						Average4-8 Max 15	
							COUNTY TOTAL	CONT.		

SOURCES:

Thickness Keystone, 1979, p. 565

Dip Keystone, 1979, p. 565

STATE Utah

COUNTY Carbon County

COAL FIELD/REGION Book Cliffs

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Beck-with Zone	0-3000	Bit	to 6.5						Average4-8 Max 15	
Uncorre-lated	0-1000	Bit	to 20	735.34	----	735.34	37.5	275.75	Average4-8 Max 15	
Uncorre-lated	1-2000	Bit	to 20	625.80	349.20	975.00	37.5	365.63	Average4-8 Max 15	
Uncorre-lated	2-3000	Bit	to 20	154.00	622.50	776.50	37.5	291.19	Average4-8 Max 15	
							COUNTY TOTAL	1,274.23		

SOURCES:

Thickness Keystone, 1979, p. 565

Dip Keystone, 1979, p. 565

STATE Utah

COUNTY Carbon County

COAL FIELD/REGION Wasatch

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Bob Wright	0-2000	Bit	4-15	N.A.	N.A.	157.89	45	71.05	flat, locally to 20°	
Castlegate A	0-2000	Bit	5-19	N.A.	N.A.	388.85	64	248.86	flat, locally to 20°	
Hiawatha	0-2000	Bit	6-25	N.A.	N.A.	261.10	53	138.38	flat, locally to 20°	
Tank	0-2000	Bit	6	----	----	----	----	----	flat, locally to 20°	
Wattis	0-2000	Bit	to 30	N.A.	N.A.	95.28	38	36.21	flat, locally to 20°	
Royal Blue	----	Bit	to 6	----	----	----	----	----	flat, locally to 20°	
Uncorrelated	0-3000	Bit	3.5-30	N.A.	N.A.	1083.19	33.3	360.70	flat, locally to 20°	
							COUNTY TOTAL	855.20		

S O U R C E S:

Thickness Keystone, 1979, p. 565; Spieker, 1931, p. 113

Dip Spieker, 1931, p. 53

STATE Utah

COUNTY Duchesne

COAL FIELD/REGION Tabby Mountain

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Frontier BED #3	0-3000	Bit	15 Max	13.37	----	13.37	43.5	5.82	20-58°	
Frontier BED #4	0-3000	Bit	10 Max	15.88	6.42	22.30	0	----	20-58°	
Lower Mesaverde	0-3000	Bit	4 Ave	9.52	4.68	14.20	0	----	20-58°	
Middle Mesaverde	0-3000	Bit	8 Ave	17.47	9.36	26.83	0	----	20-58°	
Upper Mesaverde	0-3000	Bit	13 Ave	21.17	22.39	43.56	50	21.78	20-58°	
Winchester	0-3000	Bit	5 Max	N.A.	N.A.	10.70	0	----	20-58°	
							COUNTY TOTAL	27.60		

SOURCES:

Thickness — Keystone, 1979, p. 568
Contact with Utah Geological Survey

Dip — USGS, 1969, p. 43.

STATE Utah

COUNTY Emery

COAL FIELD/REGION Book Cliffs

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Sunnyside	0-3000	Bit	4-18	N.A.	N.A.	267.78	57	152.63	$3^{\circ} - 12^{\circ}$	
Upper Sunnyside	0-3000	Bit	4-6.5	N.A.	N.A.	33.38	0	0	$3^{\circ} - 12^{\circ}$	
Lower Sunnyside	0-3000	Bit	8-18	N.A.	N.A.	58.98	80	47.18	$3^{\circ} - 12^{\circ}$	
							COUNTY TOTAL	199.81		

SOURCES:

Thickness Keystone, 1979, p. 565

Dip USGS, 1969, p. 45

STATE Utah

COUNTY Emery

COAL FIELD/REGION Wasatch

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Blind Canyon	0-3000	Bit	4-15	N.A.	N.A.	398.07	45.5	180.94	flat, locally 10-20°	
Bob Wright	0-3000	Bit	4-15	N.A.	N.A.	146.59	45.5	66.70	flat, locally 10-20°	
Candland	0-3000	Bit	4-16	N.A.	N.A.	89.93	50	44.97	flat, locally 10-20°	
Castlegate A	0-3000	Bit	5-19	N.A.	N.A.	468.32	64.3	301.13	flat, locally 10-20°	
Castlegate A	0-2000	Bit	5-19	N.A.	N.A.	22.37	64.3	14.38	flat, locally 10-20°	
Castlegate A	2-3000	Bit	5-19	N.A.	N.A.	62.64	64.3	40.28	flat, locally 10-20°	
Hiawatha	0-3000	Bit	6-25	N.A.	N.A.	1242.88	52.6	653.75	flat, locally 10-20°	
Hiawatha	0-2000	Bit	6-25	N.A.	N.A.	42.28	52.6	22.24	flat, locally 10-20°	
Wattis	0-2000	Bit	to 30	N.A.	N.A.	35.76	37.7	13.48	flat, locally 10-20°	
Upper Hiawatha	0-3000	Bit	5-17	N.A.	N.A.	11.87	58.3	6.92	flat, locally 10-20°	
							COUNTY TOTAL	CONT		

SOURCES:

Thickness Keystone, 1979, p. 565; Spieker, 1931, pp. 154, 156.

Dip Spieker, 1931, p. 53.

STATE Utah

COUNTY Emery

COAL FIELD/REGION Wasatch

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Bear Canyon	0-3000	Bit	5-16	N.A.	N.A.	236.27	54.5	128.77	flat, locally 10-20°	
Upper Bear	0-3000	Bit	Max 3.5	N.A.	N.A.	46.99	----	----	flat, locally 10-20°	
Cottonwood	0-3000	Bit	to 5	N.A.	N.A.	101.66	----	----	flat, locally 10-20°	
Uncorrelated	0-3000	Bit	3.5-30 Ave 10	N.A.	N.A.	201.46	25	50.37	flat, locally 10-20°	
							COUNTY TOTAL	1,523.93		

SOURCES:

Thickness Keystone, 1979, p. 565; Spieker, 1931, pp. 154, 156.

Dip Spieker, 1931, p. 53.

STATE Utah

COUNTY Emery

COAL FIELD/REGION Emery

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
A		Bit	4-13						<5° but up to 11° locally	
C		Bit	5-12						<5° but up to 11° locally	
F		Bit	5-7.5						<5° but up to 11° locally	
Lower Coal Zone (A-F)	0-1000	Bit	to 13	68.80	----	68.80	25	17.20	<5° but up to 11° locally	
Lower Coal Zone (A-F)	Unclas-sified	Bit	to 13	208.52	----	208.52	25	52.13	<5° but up to 11° locally	
H		Bit	to 5						<5° but up to 11° locally	
I		Bit	5-13						<5° but up to 11° locally	
J		Bit	4-12						<5° but up to 11° locally	
M		Bit	4-10						<5° but up to 11° locally	
Uncorre-lated	0-1000	Bit	To 13	14.90	----	14.90	25	3.73	<5° but up to 11° locally	
							COUNTY TOTAL	CONT		

SOURCES:

Thickness Keystone, 1979, p. 566

Dip USGS, 1969, p. 46

STATE Utah

COUNTY Emery

COAL FIELD/REGION Emery

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Uncorre-lated	Unclas-sifed	Bit	To 13	131.49	372.88	504.37	25	126.09	<5° but up to 11° locally	
							COUNTY TOTAL	199.15		

S O U R C E S:

Thickness Keystone, 1979, p. 566

Dip USGS, 1969, p. 46

STATE Utah

COUNTY Garfield

COAL FIELD/REGION Kaiparowits Plateau

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Alvey Coal Zone	0-3000	Bit	5-16.5	173.84	303.88	477.72	56.5	269.91	Average 7° Max 25°	
Alvey Coal Zone	0-2000	Bit	5-16.5	85.62	28.95	114.57	56.5	64.73	Average 7° Max 25°	
Alvey Coal Zone	0-1000	Bit	5-16.5	148.23	27.83	176.06	56.5	99.47	Average 7° Max 25°	
Christensen Hendersen Zones	0-1000	Bit	4-30	630.15	232.45	862.60	38.5	332.10	Average 7° Max 25°	
Christensen Hendersen Zones	0-3000	Bit	4-30	393.36	921.59	1314.95	38.5	506.26	Average 7° Max 25°	
Rees Coal Zone	0-3000	Bit	2-12	123.37	41.08	164.45	23.5	38.65	Average 7° Max 25°	
Rees Coal Zone	0-2000	Bit	2-12	22.40	----	22.40	23.5	5.26	Average 7° Max 25°	
Rees Coal Zone	0-1000	Bit	2-12	4.03	----	4.03	23.5	.95	Average 7° Max 25°	
Lower Zone		Bit	5 Max	----	----	----	0	----	Average 7° Max 25°	
Uncorre-lated	0-1000	Bit	3.5-30 Average 10	28.46	----	28.46	25	7.12	Average 7° Max 25°	
							COUNTY TOTAL	1,324.45		

SOURCES:

Thickness — Keystone, 1979, p. 567
Contact with Utah Geological Survey

Dip — Keystone, 1979, p. 567

STATE Utah

COUNTY Garfield

COAL FIELD/REGION Henry Mt.

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Emery Coal Zone	0-2000	Bit	4-13	4.51	18.27	22.78	33	7.52		
Emery Coal Zone	0-1000	Bit	4-13	115.01	9.09	124.10	33	40.95		
Emery Coal Zone	0-3000	Bit	4-13	N.A.	N.A.	46.80	33	15.44		
Ferron Coal Zone	0-2000	Bit	4-9	21.12	----	21.12	0	----		
Ferron Coal Zone	0-1000	Bit	4-9	4.01	----	4.01	0	----		
							COUNTY TOTAL	63.91		

SOURCES:

Thickness Keystone, 1979, P. 567

STATE Utah

COUNTY Iron

COAL FIELD/REGION Kolob

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Lower Zone Ord Area	0-2000	Bit	2-7	----	16.02	16.02	0	---		
Upper Culver Zone	0-3000	Bit	5-12	----	27.24	27.24	29	7.90		
Upper Culver Zone	0-2000	Bit	5-12	245.53	139.54	385.07	29	111.67		
Upper Culver Zone	0-1000	Bit	5-12	5.18	254.45	259.63	29	75.29		
							COUNTY TOTAL	194.86		

SOURCES:

Thickness Keystone, 1979, p. 569

STATE Utah

COUNTY Kane

COAL FIELD/REGION Alton

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Bald Knoll Coal Zone	0-1000	Bit	4-9	264.10	42.95	307.05	0	----		
Bald Knoll Coal Zone	1-2000	Bit	4-9	28.57	189.52	218.19	0	----		
Bald Knoll Coal Zone	2-3000	Bit	4-9	----	105.09	105.09	0	----		
Smirl Coal Zone	0-1000	Bit	18 Max	313.36	288.72	602.08	55.2	332.35		
Smirl Coal Zone	1-2000	Bit	18 Max	12.96	142.86	155.82	55.2	86.01		
Smirl Coal Zone	2-3000	Bit	18 Max	-----	82.29	82.29	55.2	45.42		
							COUNTY TOTAL	463.78		

SOURCES:

Thickness — Keystone, 1979, p 567
Contact with Utah Geological Society

STATE Utah

COUNTY Kane

COAL FIELD/REGION Kaiporowits

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Alvey Coal Zone	0-3000	Bit	2-7	----	----	----	0	----	$<10^{o}$ except locally	
Christensen Henderson Zones	0-3000	Bit	5-25	31.02	190.13	221.15	50	110.58	$<10^{o}$ except locally	
Christensen Henderson Zones	0-1000	Bit	5-25	545.73	577.90	1123.63	50	561.82	$<10^{o}$ except locally	
Christensen Henderson Zones	1-2000	Bit	5-25	----	434.48	434.48	50	217.24	$<10^{o}$ except locally	
John Henry Member	0-3000	Bit	Max 12	1144.45	482.00	1626.45	23.5	382.22	$<10^{o}$ except locally	John Henry Member includes
John Henry Member	0-1000	Bit	Max 12	9.86	4.81	14.67	23.5	3.45	$<10^{o}$ except locally	the Rees and the Alvey Coal Zones
Lower Coal Zone	0-1000	Bit	Max 5	----	----	----	0	----	$<10^{o}$ except locally	
Rees Coal Zone	0-3000	Bit	Max 12	3.05	----	3.05	23.5	.72	$<10^{o}$ except locally	
Rees Coal Zone	0-1000	Bit	Max 12	88.60	150.07	238.67	23.5	56.09	$<10^{o}$ except locally	
Rees Coal Zone	1-2000	Bit	Max 12	6.32	----	6.32	23.5	1.49	$<10^{o}$ except locally	
							COUNTY TOTAL	Con't		

SOURCES:

Thickness Keystone, 1979, p. 567
Contact with Utah Geological Survey

Dip USGS, 1969, p. 48

STATE Utah

COUNTY Kane

COAL FIELD/REGION Kaiporowits

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Uncorre-lated	0-3000	Bit	3.5-25 Ave 8	17.17	----	17.17	29.4	5.05	$<10^{o}$ except locally	
							COUNTY TOTAL	1,338.66		

SOURCES:

Thickness Keystone, 1979, p. 567
Contact with Utah Geological Survey

Dip USGS, 1969, p. 48

STATE Utah

COUNTY Kane

COAL FIELD/REGION Kolob

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Lower Zone Ord Area	0-2000	Bit	to 7	2.93	----	2.93	0	----		
Lower Zone Ord Area	0-1000	Bit	to 7	37.36	----	37.36	0	----		
Upper Culver Zone	0-2000	Bit	5-12	----	107.99	107.99	29	31.32		
Upper Culver Zone	0-1000	Bit	5-12	217.23	41.97	259.20	29	75.17		
Upper Culver Zone	1-2000	Bit	5-12	178.81	469.57	648.38	29	188.03		
Upper Culver Zone	2-3000	Bit	5-12	----	201.53	201.53	29	58.44		
							COUNTY TOTAL	352.96		

SOURCES:

Thickness Keystone, 1979, p. 569

STATE Utah

COUNTY Sanpete

COAL FIELD/REGION Wasatch

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Hiawatha	0-3000	Bit	5-9	N.A.	N.A.	2.68	0	----	horizontal, locally to 20°	
Candland	2-3000	Bit	4-16	----	41.76	41.76	50	20.88	horizontal, locally to 20°	
Castlegate A	2-3000	Bit	5-19	----	62.64	62.64	64.3	40.28	horizontal, locally to 20°	
Muddy #1	0-3000	Bit	3-9	N.A.	N.A.	10.55	0	----	horizontal, locally to 20°	
Muddy #2	0-3000	Bit	4-6	N.A.	N.A.	9.59	0	----	horizontal, locally to 20°	
Upper Hiawatha		Bit	5-17	----	----	----	----	----	horizontal, locally to 20°	
Blind Canyon		Bit	4-15	----	----	----	----	----	horizontal, locally to 20°	
Bear Canyon		Bit	5-16	----	----	----	----	----	horizontal, locally to 20°	
Uncorre-lated	0-3000	Bit	3.5-19	N.A.	N.A.	187.98	45	84.59	horizontal, locally to 20°	
							COUNTY TOTAL	145.75		

S O U R C E S:

Thickness Keystone, 1979, p. 565

Dip Spieker, 1931, p. 53

STATE Utah

COUNTY Sevier

COAL FIELD/REGION Salina Canyon

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Upper Ivie	0-3000	Bit	6-15	N.A.	N.A.	82.84	50	41.42	10-15° locally greater	
							COUNTY TOTAL	41.42		

SOURCES:

Thickness USGS, 1969, p. 46

Dip USGS, 1969, p. 46

STATE Utah

COUNTY Sevier

COAL FIELD/REGION Emery

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
A		Bit	4-13						locally to 11°	
C		Bit	5-12						locally to 11°	
F		Bit	5-7.5						locally to 11°	
Lower Coal Zone (A-F)	0-1000	Bit		136.54	47.91	184.45	25	46.11	locally to 11°	
Lower Coal Zone (A-F)	Unclassified	Bit		5.18	----	5.18	25	4.97	locally to 11°	
H		Bit	3.5-5						locally to 11°	
I		Bit	5-13						locally to 11°	
J		Bit	4-12						locally to 11°	
M		Bit	4-10						locally to 11°	
Uncorrelated	0-3000	Bit		N.A.	N.A.	120.46	25	30.12	locally to 11°	
							COUNTY TOTAL	Cont		

SOURCES:

Thickness Keystone, 1979, p. 566

Dip USGS, 1969, p. 46

STATE Utah

COUNTY Sevier

COAL FIELD/REGION Emery

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Uncorre-lated	0-2000	Bit		----	16.50	16.50	25	4.13	locally to 11°	
Uncorre-lated	0-1000	Bit		3.59	----	3.59	25	.90	locally to 11°	
Uncorre-lated	Unclassified	Bit		2.88	208.03	210.91	25	52.73	locally to 11°	
							COUNTY TOTAL	138.96		

SOURCES:

Thickness Keystone, 1979, p. 566

Dip USGS, 1969, p. 46

STATE Utah

COUNTY Sevier

COAL FIELD/REGION Wasatch

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Hiawatha	0-3000	Bit	5-9	N.A.	N.A.	54.86	0	----		
Upper Hiawatha	0-3000	Bit	5-17	N.A.	N.A.	103.55	58	60.06		
Upper Ivie	0-3000	Bit	6-14	N.A.	N.A.	145.76	50	72.88		
Ivie		Bit	5-10					----		
Muddy #2		Bit	4-6					----		
Muddy #1		Bit	3-9					----		
Uncorre-lated	0-3000	Bit	To 17	N.A.	N.A.	292.07	37.5	109.53		
Uncorre-lated	0-3000	Bit	To 17	---	310.75	310.75	37.5	116.53		
							COUNTY TOTAL	359.00		

SOURCES:

Thickness Keystone, 1979, p. 565

STATE Utah

COUNTY Wasatch

COAL FIELD/REGION Tabby Mt.

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Frontier Bed #3	0-3000	Bit	15 max	13.37	----	13.37	43.5	5.82	20-58°	
Frontier Bed #4	0-3000	Bit	10 max	15.88	6.42	22.30	0	----	20-58°	
Lower Mesaverde	0-3000	Bit	4.5 ave	9.52	----	9.52	0	----	20-58°	
Middle Mesaverde	0-3000	Bit	8 ave	17.47	----	17.47	0	----	20-58°	
Upper Mesaverde	0-3000	Bit	13 ave	21.17	10.69	31.86	50	15.93	20-58°	
Uncorre-lated	0-3000	Bit	3.5-15	N.A.	N.A.	73.52	28.7	21.10	20-58°	
							COUNTY TOTAL	42.85		

SOURCES:

Thickness — Keystone, 1979, p. 568
Contact with Utah Geological Survey

Dip — USGS, 1969, p. 43

STATE Utah

COUNTY Washington

COAL FIELD/REGION Kolob

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Lower Zone Ord Area	0-2000	Bit	2-7	----	16.02	16.02	0	----		
Upper Culver Zone	0-2000	Bit	5-12	----	90.79	90.79	28.5	25.88		
							COUNTY TOTAL	25.88		

SOURCES:

Keystone, 1979, p. 569

STATE Wyoming

COUNTY Lincoln

COAL FIELD/REGION Hams Fork Region (Kemmerer)

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
A Seam		Bit	6.5 max							
Lower Spring Valley		Bit	5							
Lower Willow Creek		Bit	2-4							
Kemmerer		Bit	8-21							
Upper Spring Valley		Bit	6							
Vail		Bit	10 max							
Willow Creek #5		Bit	2-11							
Uncorre-lated	0-1000	Bit	to 21	506.90	12.39	519.29	35.4	183.83		
Uncorre-lated	1-2000	Bit	to 21	498.69	4.65	503.34	35.4	178.18		
Uncorre-lated	2-3000	Bit	to 21	437.69	37.98	475.67	35.4	168.39		
							COUNTY TOTAL	530.40		

SOURCES:

Thickness: Glass, 1978

STATE WYOMING

COUNTY SWEETWATER

COAL FIELD/REGION GREEN RIVER

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Rock Springs#1		Bit	To 13.8						5-20°	Thickness of Rock Springs
Rock Springs#3		Bit	To 13.8						5-20°	Formation Coals Range
Rock Springs#7		Bit	4.5 Ave						5-20°	From 2-13.8 ft. & the average
Rock Springs#11		Bit	4.5 Max						5-20°	is 6 ft. (Berryhill, 1950,
Other Rock Sprg		Bit	2-13.8						5-20°	p. 25)
Uncorre-lated	0-1000	Bit	To 13.8	2,664.31	565.13	3,229.44	25	807.36	5-20°	
Uncorre-lated	1-2000	Bit	To 13.8	840.60	962.95	1,803.55	25	450.89	5-20°	
Uncorre-lated	2-3000	Bit	To 13.8	160.70	1,497.49	1,658.19	25	414.55	5-20°	
							COUNTY TOTAL	1,672.80		

SOURCES:

Dip: Berryhill, 1950, p. 25.
Thickness: Glass, 1978; Berryhill, 1950.

STATE WYOMING

COUNTY UINTA

COAL FIELD/REGION HAMS FORK (Kemmerer)

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
A Seam		Bit	6.5 Max							
Lower Spring Valley		Bit	5							
Lower Willow Creek		Bit	2-4							
Kemmerer		Bit	8-21							
Upper Spring Valley		Bit	6							
Vail		Bit	10 Max							
Willow Creek #5		Bit	2-11							
Uncorre-lated	0-1000	Bit	To 21	478.75	---	478.75	35.5	169.96		
Uncorre-lated	1-2000	Bit	To 21	478.58	---	478.58	35.5	169.90		
Uncorre-lated	2-3000	Bit	To 21	480.25	---	480.25	35.5	170.49		
							COUNTY TOTAL	356.39		

S O U R C E S:

Dip: Berryhill, et al, 1950, p. 28.
Thichness: Glass, 1978

APPENDIX B

THICK SEAM DATA BASE

PART II -- SUBBITUMINOUS RESOURCES

For subbituminous coal, USGS tonnage estimates are given for resources greater than 10 feet in thickness.

STATE Alaska

COUNTY --

COAL FIELD/REGION Barrow - North Slope

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Uncorre- lated	0-3000	Subbit	to 30'	470.32	43,567.62	44,037.94	50	22,018.97		
Uncorre- lated	0-1000	Subbit	to 30'	74.30	1,433.72	1,508.02	50	754.01		
Uncorre- lated	1-2000	Subbit	to 30'	---	1,920.00	1,920.00	50	960.00		
							COUNTY TOTAL	23,732.98		

SOURCES:

Thickness: Barnes, 1967, pp. B16-B18

STATE Alaska

COUNTY ---

COAL FIELD/REGION Kenai - Cook Inlet (Beluga, Copp, Christna)

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Uncorre-lated	0-1000	Subbit	to 50	1,847.48	---	1,847.48	25	461.87		Lignite & sub-bituminous beds to 50'
							COUNTY TOTAL	461.87		

SOURCES:

Thickness: Barnes, F.F., 1967, p. B24

STATE Alaska

COUNTY --

COAL FIELD/REGION Matanuska - Susitna (Yentna River)

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Uncorre-lated	0-1000	Subbit	> 10'	125.12	---	125.12	100	125.12		
							COUNTY TOTAL	125.12		

SOURCES:

Thickness: Barnes, 1967, p. B25

STATE Alaska

COUNTY --

COAL FIELD/REGION Nenana

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Uncorre-lated	0-1000	Subbit	max 60	2,022.55	1,378.22	3,400.77	20	680.15	mostly low to moderate	
Uncorre-lated	1-2000	Subbit	max 60	706.80	470.67	1,177.47	20	235.49	mostly low to moderate	
Uncorre-lated	2-3000	Subbit	max 60	216.19	366.02	582.21	20	116.44	mostly low to moderate	
							COUNTY TOTAL	1,032.08		

SOURCES:

Thickness: Barnes, 1967, p. B23

STATE COLORADO

COUNTY BOULDER

COAL FIELD/REGION DENVER BASIN

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Upper & Lower	0-1000	Subbit	to 20	27.75	----	27.75	100	27.75		
3 and 4	0-1000	Subbit	to 20	.91	----	.91	100	.91		
							COUNTY TOTAL	28.66		

SOURCES:

Thickness: Boreck & Murray, 1979, Figure 6
Keystone, 1979, p. 470-471

STATE COLORADO

COUNTY DELTA

COAL FIELD/REGION GRAND MESA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
A	0-1000	Subbit	to 12	64.65	----	64.65	100	64.65		
A	1-2000	Subbit	to 12	.69	----	.69	100	.69		
B	0-1000	Subbit	to 23	11.38	37.45	48.83	76.9	37.55		
B	1-2000	Subbit	to 23	20.64	21.51	42.15	76.9	32.41		
C	0-1000	Subbit	to 13	7.63	71.49	79.12	100	79.12		
C	1-2000	Subbit	to 13	----	7.04	7.04	100	7.04		
D	0-1000	Subbit	to 26	18.26	51.95	70.21	62.5	43.88		
D	1-2000	Subbit	to 26	1.45	57.24	58.69	62.5	36.68		
D	2-3000	Subbit	to 26	----	6.33	6.33	62.5	3.96		
							COUNTY TOTAL	305.98		

SOURCES:

Thickness: Murray, Fender and Jones, 1977
Hileman, 1970

STATE COLORADO

COUNTY GUNNISON

COAL FIELD/REGION TONGUE MESA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Uncor-related	0-1000	Subbit	to 40	212.08	83.69	295.77	33.3	98.49		
							COUNTY TOTAL	98.49		

SOURCES:

Thickness: Keystone, 1979, p.477
Keystone, 1978, p.468

STATE COLORADO

COUNTY JACKSON

COAL FIELD/REGION NORTH PARK

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Below Riach	0-1000	Subbit	Max 12	8.34	5.84	14.18	100	14.18	to 85°	
Below Riach	1-2000	Subbit	Max 12	.34	11.51	11.85	100	11.85	to 85°	
Below Riach	2-3000	Subbit	Max 12	1.72	18.97	20.69	100	20.69	to 85°	
Below Riach Double	0-1000	Subbit	Max 18	17.63	13.64	31.27	100	31.27	to 85°	
Below Riach Double	1-2000	Subbit	Max 18	14.14	19.95	34.09	100	34.09	to 85°	
Below Riach Double	2-3000	Subbit	Max 18	----	33.25	33.25	100	33.25	to 85°	
Capron	0-1000	Subbit	Max 12	19.98	24.40	44.38	100	44.38	to 85°	
Capron	1-2000	Subbit	Max 12	----	15.53	15.53	100	15.53	to 85°	
							COUNTY TOTAL	205.24		

SOURCES:

Thickness: Beekly, 1915, pp. 93-111

STATE COLORADO

COUNTY JEFFERSON

COAL FIELD/REGION DENVER BASIN

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Uncor-related	0-1000	Subbit	< 14'	17.60	----	17.60	100	17.60		
Uncor-related	1-2000	Subbit	< 14'	14.85	----	14.85	100	14.85		
							COUNTY TOTAL	32.45		

SOURCES:

Thickness: Boreck & Murray, 1979, Figure 6

STATE COLORADO

COUNTY MOFFAT

COAL FIELD/REGION YAMPA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Blevins Group	0-1000	Subbit	to 20	29.11	71.29	100.40	100	100.40		
K	0-1000	Subbit	to 20	25.00	----	25.00	100	25.00		
S	0-1000	Subbit	to 20	79.43	----	79.43	100	79.43		
Twenty Mile Group	0-1000	Subbit	to 20	15.88	----	15.88	100	15.88		
Classed By Zone	0-3000	Subbit	to 40	N.A.	N.A.	2924.64	25	731.16		
							COUNTY TOTAL	951.87		

SOURCES:

Thickness: Keystone, 1979, pp.472-473
Keystone, 1978, pp.471-472

STATE COLORADO

COUNTY MONTROSE

COAL FIELD/REGION TONGUE MESA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Uncor-related	0-1000	Subbit	to 40	96.60	895.40	992.00	33.3	330.34		
							COUNTY TOTAL	330.34		

SOURCES:

Thickness: Keystone, 1979, p.477
Keystone, 1978, p. 468

STATE COLORADO

COUNTY OURAY

COAL FIELD/REGION TONGUE MESA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Uncor-related	0-1000	Subbit	to 40	734.98	246.94	981.92	33.3	326.98		
							COUNTY TOTAL	326.98		

S O U R C E S:

Thickness: Keystone, 1979, p.477
Keystone, 1978, p.468

STATE COLORADO

COUNTY ROUTT

COAL FIELD/REGION YAMPA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						Comments
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Dry Creek	0-1000	Subbit	< 20	19.19	95.80	114.99	100	114.99		No Evidence
Dry Creek	1-2000	Subbit	< 20	----	11.46	11.46	100	11.46		of Any Beds
K	0-1000	Subbit	< 20	102.27	----	102.27	100	102.27		> 20' Thick
K	1-2000	Subbit	< 20	----	12.00	12.00	100	12.00		
P	0-1000	Subbit	< 20	18.38	----	18.38	100	18.38		
Uncor-related	0-1000	Subbit	< 20	31.99	12.86	44.85	100	44.85		
							COUNTY TOTAL	303.95		

SOURCES:

Thickness: Bass, N. W. et al, 1955, plates

STATE COLORADO

COUNTY WELD

COAL FIELD/REGION DENVER BASIN

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
NO 3	0-1000	Subbit	14 max	17.63	----	17.63	100	17.63		
NO 4	0-1000	Subbit	< 14	.46	----	.46	100	.46		
NO 5	0-1000	Subbit	< 14	2.44	----	2.44	100	2.44		
							COUNTY TOTAL	20.53		

SOURCES:

Thickness: Boreck & Murray, 1979, Figure 6

STATE Montana

COUNTY Big Horn

COAL FIELD/REGION Tullock Creek, Crow Indian Reservation, Forsyth, Sheridan

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Roland		Subbit	13 max							
Smith		Subbit	7-20							
Anderson		Subbit	3-27							
Deitz #1		Subbit	47 max							
Deitz #2		Subbit	15 max							
Canyon		Subbit	6-24							
Monarch		Subbit	25 max							
Davis		Subbit	3-32							
Sawyer		Subbit	20 max							
Knoblock		Subbit	20 max							
							COUNTY TOTAL	cont.		

SOURCES:

Thickness: Thom, 1935, p. 87
Dobbin, 1929, plates
Rogers, 1923, plates
Baker, 1929, pp. 32-66

STATE Montana

COUNTY Big Horn

COAL FIELD/REGION Tullock Creek, Crow, Forsyth

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Shoedel		Subbit	14 max							
(M) Robinson		Subbit	28 max							
(Q) Rosebud		Subbit	23 max							
P		Subbit	20 max							
Wall		Subbit	13.5 max							
Powers		Subbit	12.5 max							
Uncorre-lated	0-1000	Subbit	10-47 ave. 16	17,053.59	1,806.56	18,860.15	62.9	11,863.03		
							COUNTY TOTAL	11,863.03		

S O U R C E S:

Thickness: Thom, 1935, p. 87
Dobbin, 1929, plates
Rogers, 1923, plates
Baker, 1929, pp. 32-66

STATE Montana

COUNTY Custer

COAL FIELD/REGION Ashland, Rosbud

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Knoblock		Subbit	6-24							Does not appear to be any other coal beds > 10ft. thick.
Uncorre-lated	0-1000	Subbit	10-24	566.19	---	556.19	71.4	404.26		
							COUNTY TOTAL	404.26		

SOURCES:

Thickness: Bass, 1932, plates
Pierce, 1936

STATE Montana

COUNTY Musselshell

COAL FIELD/REGION Bull Mountain

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Mammoth & Rehder		Subbit	7-13.5						to 36°, but gener-ally < 5°	Mammoth where combined with Rehder is only bed in-dicated which exceeds 10ft. in thickness.
Uncorre-lated	0-1000	Subbit	> 10	1166.25	---	1166.25	100	1166.25		
							COUNTY TOTAL	1166.25		

S O U R C E S:

Thickness: Woolsey, 1917, pp. 43-44
Dip: Woolsey, 1917, p. 32

STATE Montana

COUNTY Powder River

COAL FIELD/REGION Ashland

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Knoblock		Subbit	11-22						nearly horizontal	
Sawyer		Subbit	11-22						nearly horizontal	
"D"		Subbit	max 18						nearly horizontal	
"E"		Subbit	max 19.5						nearly horizontal	
Flowers Goodale		Subbit	max 12						nearly horizontal	
"A"		Subbit	max 15						nearly horizontal	
Uncorre-lated	0-1000	Subbit	10-22 ave. 14.2	12,462.42	5,062.09	17,524.51	87.5	15,333.95	nearly horizontal	
							COUNTY TOTAL	15,333.95		

SOURCES:

Thickness: Bass, 1931, plates 1-37
Dip: Bass, 1931

STATE Montana

COUNTY Rosebud

COAL FIELD/REGION Rosebud, Ashland, Forsyth, Sheridan

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)					Dip (degrees)	COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.		
Rosebud		Subbit	max 28						$< 3^{o}$	
Terret		Subbit	max 12						$< 3^{o}$	
Sawyer		Subbit	max 20						$< 3^{o}$	
Knoblock		Subbit	max 25						$< 3^{o}$	
Canyon		Subbit	max 13						$< 3^{o}$	
Wall		Subbit	max 32						$< 3^{o}$	
Brewster Arnold		Subbit	max 17						$< 3^{o}$	
Anderson		Subbit	max 27						$< 3^{o}$	
Deitz		Subbit	max 13						$< 3^{o}$	
Smith		Subbit	max 15						$< 3^{o}$	
							COUNTY TOTAL	Cont.		

SOURCES:

Thickness: Baker, 1929, pp. 32-66
Pierce, 1936, pp. 74-80
Bass, 1931-32, plates
Dobbin, 1929, plates

Dip: Pierce, 1936, p. 67

STATE Montana

COUNTY Rosebud

COAL FIELD/REGION Rosebud, Ashland, Forsyth, Sheridan

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Uncorre-lated	0-1000	Subbit	ave. 15 10-32	8,889.85	3,877.68	12,767.53	64.7	8,260.59		
							COUNTY TOTAL	8,260.59		

SOURCES:

Thickness: Baker, 1929, pp. 32-66
Pierce, 1936, pp. 74-80
Bass, 1931-32, plates
Dobbin, 1929, plates

Dip: Pierce, 1936, p. 67

STATE Montana

COUNTY Treasure

COAL FIELD/REGION Forsyth, Tullock

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Rosebud		Subbit	20 max							
M		Subbit	20 max							
P		Subbit	20 max							
Uncorre-lated	0-1000	Subbit	10-20	438.49	---	438.49	100	438.49		
							COUNTY TOTAL	438.49		

SOURCES:

Thickness: Dobbin, 1929
Rogers, 1923, plates

STATE Montana

COUNTY Yellowstone

COAL FIELD/REGION Bull Mountain

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Mammoth & Rehder		Subbit	7-13.5						< 5°	Mammouth where combined with Rehder is only bed indicated which exceeds 10ft. in thickness.
Uncorrelated	0-1000	Subbit	> 10	198.51	---	198.51	100	198.51	< 5°	
							COUNTY TOTAL	198.51		

SOURCES:

Thickness: Woolsey, et. al., 1917, pp. 43-44
Dip: Woolsey, et. al., 1917, p. 32

STATE New Mexico

COUNTY McKinley, Rio Arriba, San Juan, Sandoval

COAL FIELD/REGION San Juan Basin/Fruitland Formation

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Uncorre-lated	500-1000	Subbit	10-20	N/A	N/A	4,503.8	60.2	2,711.29		
Uncorre-lated	1-2000	Subbit	10-20	N/A	N/A	7,771.2	60.2	4,678.26		
Uncorre-lated	2-3000	Subbit	10-20	N/A	N/A	18,026.7	60.2	10,852.07		
							COUNTY TOTAL	18,241.62		

SOURCES:

Resource and Thickness Data: Fasset, 1971, pp. 68-70

STATE WYOMING

COUNTY CAMPBELL

COAL FIELD/REGION POWDER RIVER

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Anderson	0-1000	Subbit	10-50	500.89	45.91	546.80	25	136.7	$2-3^{\circ}$	
Canyon	0-1000	Subbit	11-65	211.89	.05	211.94	18.2	38.6	$2-3^{\circ}$	
Felix	0-1000	Subbit	5-50	686.68	---	686.68	25	171.67	$2-3^{\circ}$	
Smith	0-1000	Subbit	5-13	68.00	---	68.00	100	68.00	$2-3^{\circ}$	
VLM #2	0-1000	Subbit	7-30	15.64	---	15.64	50	7.82	$2-3^{\circ}$	
Wall	0-1000	Subbit	5-25	3.34	---	3.34	66.7	2.23	$2-3^{\circ}$	
Uncorre-lated	0-1000	Subbit	To 65	5,060.52	16,603.90	21,664.42	34.5	7,474.22	$2-3^{\circ}$	
Uncorre-lated	1-2000	Subbit	To 65	---	412.16	412.16	34.5	142.20	$2-3^{\circ}$	
							COUNTY TOTAL	8,041.44		

SOURCES:

Dip Glass, 1978, p. 20.
Thickness Glass, 1978.

STATE WYOMING

COUNTY CARBON

COAL FIELD/REGION HANNA

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Bed 24		Subbit	Avg 18-20						10-25°	Accord. to Berryhill
25		Subbit	Avg 22						10-25°	Coal in the Ferris Form.
50		Subbit	15-19						10-25°	Does not exceed 23.4' in
62		Subbit	>9	Ferris Formation					10-25°	Observed Thickness
64		Subbit	>6						10-25°	
80		Subbit	15.5-24						10-25°	
82		Subbit	Avg 9						10-25°	
Brooks		Subbit	7.5-15						12-20°	
Hanna 1		Subbit	15-30	Hanna Formation					12-20°	
Hanna 2		Subbit	30-36						12-20°	
							COUNTY TOTAL	cont.		

SOURCES:

Dip: Berryhill, 1950, p. 22
Thichness: Glass, 1978

STATE WYOMING

COUNTY CARBON

COAL FIELD/REGION GREEN RIVER

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Almond Coal Beds		Subbit	To 12						$20-50^{\circ}$	
Black Butte & Maxwell		Subbit	16-22						$20-50^{\circ}$	
Creston #2 & #3		Subbit	Avg 18						$20-50^{\circ}$	
Deadman		Subbit	30						$20-50^{\circ}$	
Little Valley		Subbit	To 15						$20-50^{\circ}$	
Uncorre-lated	0-1000	Subbit	10-36	441.96	25.25	467.21	54.8	256.03	$20-50^{\circ}$	
Uncorre-lated	1-2000	Subbit	10-36	164.24	9.99	174.23	54.8	95.48	$20-50^{\circ}$	
Uncorre-lated	2-3000	Subbit	10-36	67.57	22.46	90.03	54.8	49.34	$20-50^{\circ}$	
							COUNTY TOTAL	400.85		

SOURCES:

Dip: Glass, 1978, p. 34
Thickness: Glass, 1978

STATE WYOMING

COUNTY CONVERSE

COAL FIELD/REGION POWDER RIVER

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Anderson		Subbit	10-50						$2-3^{\circ}$	
Badger		Subbit	17-20						$2-3^{\circ}$	
Canyon		Subbit	11-65						$2-3^{\circ}$	
Deitz #2		Subbit	Avg 12						$2-3^{\circ}$	
Deitz #3		Subbit	10-25						$2-3^{\circ}$	
Felix		Subbit	5-50						$2-3^{\circ}$	
Monarch		Subbit	5-25						$2-3^{\circ}$	
School		Subbit	22-38						$2-3^{\circ}$	
Smith		Subbit	5-13						$2-3^{\circ}$	
Sussex		Subbit	Avg 11.8 max 50						$2-3^{\circ}$	
							COUNTY TOTAL	cont.		

SOURCES:

Dip: Glass, 1978, p.20
Thickness: Glass, 1978

STATE WYOMING

COUNTY CONVERSE

COAL FIELD/REGION POWDER RIVER

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Uncorre-lated	0-1000	Subbit	10-65	447.06	42.71	489.77	39.5	193.46	2-3°	
							COUNTY TOTAL	193.46		

S O U R C E S:

Dip: Glass, 1978, p. 20
Thickness: Glass, 1978

STATE WYOMING

COUNTY FREEMONT

COAL FIELD/REGION WIND RIVER BASIN

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Downey		Subbit	28						15-40 So. 40-85 No.	
Signor		Subbit	16.5 max						15-40 So. 40-85 No.	
Welton		Subbit	16 max						15-40 So. 40-85 No.	
Uncorre-lated	0-1000	Subbit	Avg 15 10-28	8.85	---	8.85	69.2	6.12	15-40 So. 40-85 No.	
Uncorre-lated	1-2000	Subbit	Avg 15 10-28	50.05	---	50.05	69.2	34.63	15-40 So. 40-85 No.	
Uncorre-lated	2-3000	Subbit	Avg 15 10-28	69.44	190.28	259.72	69.2	179.73	15-40 So. 40-85 No.	
							COUNTY TOTAL	220.48		

SOURCES:

Dip: Glass, 1978, p. 72
Thickness: Glass, 1978

STATE WYOMING

COUNTY HOT SPRINGS

COAL FIELD/REGION BIG HORN BASIN

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Mayfield		Subbit	8-38						15-50	Only observed seam to exceed
Uncorrelated	0-1000	Subbit	10-38	9.20	---	9.20	35.7	3.28	15-50	10 feet (see Glass)
							COUNTY TOTAL	3.28		

SOURCES:

Dip: Glass, 1978, p. 68
Thickness: Glass, 1978, p. 70

STATE WYOMING

COUNTY JOHNSON

COAL FIELD/REGION POWDER RIVER

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Healy	0-1000	Subbit	5-200	---	---	---	Do Not Include	Do Not Include	2°-5° Up to 20°	Too thick to include
Lower Cameron	0-1000	Subbit	3-19	31.69	---	31.69	100	31.69	2°-5° Up to 20°	
Ucross	0-1000	Subbit	22 max	41.16	549.94	591.10	83	492.58	2°-5° Up to 20°	
Walters	0-1000	Subbit	20-35	---	---	---	0	---	2°-5° Up to 20°	
Uncorre-lated	0-1000	Subbit	To 35	232.75	191.80	424.55	53	255.01	2°-5° Up to 20°	
Uncorre-lated	1-2000	Subbit	To 35	---	2.71	2.71	53	1.44	2°-5° Up to 20°	
							COUNTY TOTAL	750.72		

SOURCES:

Thickness: Glass, 1978

STATE WYOMING

COUNTY LINCOLN

COAL FIELD/REGION HAMS FORK*

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Adaville #1		Subbit	88						17-45°	
Adaville #2		Subbit	16						17-45°	
Adaville #3		Subbit	34-42						17-45°	
Adaville#3 Middle Rider		Subbit	11						17-45°	
Adaville#4 Lower Bench		Subbit	12						17-45°	
Adaville#4 Rider		Subbit	11						17-45°	
Adaville #6		Subbit	18						17-45°	
Adaville #11		Subbit	13						17-45°	
Uncorre-lated	0-1000	Subbit	10-88 Avg 25	355.69	---	355.69	33.3	118.44	17-45°	
Uncorre-lated	1-2000	Subbit	10-88 Avg 25	96.49	68.55	165.04	33.3	54.96	17-45°	
							COUNTY TOTAL	173.40		

SOURCES:

Dip: Glass, 1978, p. 58
Thickness: Glass, 1978, p. 57

*Part of Labarge Ridge Field is also in Lincoln County. According to Berryhill (1950, P. 26) "the average thickness of 12 sections measured in various parts of the field is 4.7 feet and the maximum is 8.3"

STATE WYOMING

COUNTY SHERIDAN

COAL FIELD/REGION POWDER RIVER BASIN

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Anderson	0-1000	Subbit	10-50	196.99	---	196.99	25	49.25	1-4°	
Arvada	0-1000	Subbit	12 max	.20	---	.20	100	.20	1-4°	
Canyon	0-1000	Subbit	11-65	153.91	.05	153.96	18	27.71	1-4°	
Smith	0-1000	Subbit	5-13	15.79	---	15.79	100	15.79	1-4°	
Healy	0-1000	Subbit	5-25	215.29	293.26	508.55	66	508.55	1-4°	
Ucross	0-1000	Subbit	4-18	12.59	204.48	217.07	100	217.07	1-4°	
Walters	0-1000	Subbit	4-18	.22	16.79	17.01	100	17.01	1-4°	
Uncorre-lated	0-1000	Subbit	10-65 ave. 19	1,164.41	1,796.72	2,961.13	51.5	1,524.98	1-4°	
Uncorre-lated	1-2000	Subbit	10-65 ave. 19	---	3.12	3.12	51.5	1.61	1-4°	
							COUNTY TOTAL	1,526.59		

SOURCES:

Dip: Berryhill, 1950, p. 12

Thickness: Glass, 1978; Berryhill, 1950, pp. 12, 13

STATE WYOMING

COUNTY SWEETWATER

COAL FIELD/REGION GREEN RIVER

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons) Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	COMMENTS
Creston #2 & #3		Subbit	18 Avg						To 20°	
Deadman		Subbit	30						To 20°	
Little Valley		Subbit	To 15						To 20°	
Fort Union Coals		Subbit	10-26						To 20°	
Lebar		Subbit	8-12						To 20°	
Black Butte and Maxwell		Subbit	16-22						To 20°	
Upper Cherokee		Subbit	10-18						To 20°	
Lower Cherokee		Subbit	20-32						To 20°	
Uncorre-lated	0-1000	Subbit	To 32	114.63	18.18	132.81	59.2	78.62	To 20°	
Uncorre-lated	2-3000	Subbit	To 32	---	---	3.76	59.2	2.23	To 20°	
							COUNTY TOTAL	80.85		

S O U R C E S:

Dip: Glass, 1978, p. 34
Thickness: Glass, 1978

STATE WYOMING

COUNTY UINTAH

COAL FIELD/REGION HAMS FORK/GREEN RIVER

Bed Name	Overburden (feet)	Rank	Thickness (feet)	RESOURCES (Millions of Short Tons)						COMMENTS
				Measured & Indicated	Inferred	TOTAL	Percent 10 - 20 ft.	Estimated Resources Between 10 - 20 ft.	Dip (degrees)	
Almy		Subbit	2.5 to 28						10-20	
Uncorre-lated	0-1000	Subbit	To 28	34.67	---	34.67	55.5	19.24	10-20	
Uncorre-lated	1-2000	Subbit	To 28	54.60	---	54.60	55.5	30.30	10-20	
Uncorre-lated	2-3000	Subbit	To 28	34.82	---	34.82	55.5	19.33	10-20	
							COUNTY TOTAL	68.87		

SOURCES:

Dip: Berryhill, 1950, p. 28
Thickness: Glass, 1978, p. 61; Berryhill, 1950, p. 28

APPENDIX B

PART III -- BIBLIOGRAPHY

Barnes, F. F., Coal Resources of Alaska, USGS Bulletin 1242-B, 1967.

Bass, N. W., J. B. Eby, and M. R. Campbell, Geology and Mineral Fuels of Parts of Routt and Moffat Counties, Colorado, USGS Bulletin 1027-D, pp. 143-250.

Bass, N. W., The Ashland Coal Field: Rosebud, Powder River and Custer Counties, Montana, USGS Bulletin 831(b), 1931.

Beekly, A. L., Geology and Coal Resources of North Park, Colorado, USGS Bulletin 596, 1915.

Berryhill, H. L., Jr., et al, Coal Resources of Wyoming, USGS Circular 81, 1950.

Boreck, D. L. and D. K. Murray, Colorado Coal Reserves Depletion Data and Coal Mine Summaries, Colorado Geological Survey, Open File Report 79-1, 1979.

Coal Facts, National Coal Association, Washington, DC, 1979.

Collins, B. A., "Coal Deposits of the Carbondale, Grand Hogback and Southern Danforth Hills Coal Fields, Eastern Piceance Basin, Colorado," Colorado School of Mines Quarterly, Vol. 71, No. 1, 1976.

Dobbin, C. E., The Forsyth Coal Field, Rosebud, Treasure and Big Horn Counties, Montana, USGS Bulletin 812(a), 1929.

Erdmann, C. E., The Book Cliffs Coal Field in Garfield and Mesa Counties, Colorado, USGS Bulletin 851, 1934.

Fassett, J. E. and J. S. Hinds, Geology and Fuel Resources, Fruitland Formation and Kirtland Shale, San Juan Basin, USGS Prof. Paper 676, 1971.

Gale, H. S., Coal Fields of Northwestern Colorado and Northeastern Utah, USGS Bulletin 416, 1919.

Glass, G. B., Mining in the Hanna Coal Field, Geological Survey of Wyoming, 1972.

Glass, G. B., Wyoming Coal Fields, 1978, Geological Survey of Wyoming, Public Information Circular 9, 1978.

Hancock, E. T. and J. B. Eby, Geology and Coal Resources of the Meeker Quadrangle, Moffat and Rio Blanco Counties, Colorado, USGS Bulletin 812-C, pp. 191-242, 1930.

Hileman, D. H., B. A. Collins, and S. R. Wilson, Coal Production from the Uinta Region, Colorado and Utah, USBM IC 8497, 1970.

Keystone Coal Industry Manual, McGraw-Hill, New York, 1977, 1978, 1979.

Lee, W., Coal Resources of the Raton Field, Colfax County, New Mexico, USGS Bulletin 752, 1924.

Matson, T. K. and D. H. White, Jr., The Reserve Base of Coal for Underground Mining in the Western United States, USBM IC 8678, 1975.

Murray, D. K., H. B. Fender, and D. C. Jones, "Coal and Methane Gas in the Southeastern Part of the Piceance Creek Basin, Colorado," Exploration Frontiers of the Central and Southern Rockies, Rocky Mountain Association of Geologists Field Conference Guidebook, pp. 379-405, 1977.

Pierce, W. G., The Rosebud Coal Field, Rosebud and Custer Counties, Montana, USGS Bulletin 847(b), 1936.

Pillmore, C. L., "Geology and Coal Deposits of the Raton Coal Field, Colfax, New Mexico," Mountain Geologist, 6:3, pp. 125-142, 1969.

Read, C. B., et al, Coal Resources of New Mexico, USGS Circular 89, 1950.

Rogers, G. S. and W. Lee, Geology of the Tullock Creek Coal Field, Rosebud and Big Horn Counties, Montana, USGS Bulletin 749, 1923.

Spieker, E. M., The Wasatch Plateau Coal Field, Utah, USGS Bulletin 819, 1931.

Thom, W. T. Jr., et al, Geology of Big Horn County and the Crow Indian Reservation, Montana, USGS Bulletin 856, 1935.

Mineral and Water Resources of Colorado, USGS, 1968.

Mineral and Water Resources of Utah, USGS, 1969.

Woolsey, L. H., R. W. Richards, and C. T. Lupton, The Bull Mountain Coal Field, Musselshell and Yellowstone Counties, Montana, USGS Bulletin 647, 1917.

Numerous references not cited here were utilized in eliminating those counties and coal beds with thick seam resources.

APPENDIX C

INPUT DATA SUMMARY

AND

LONGWALL VARIABLES

INPUT DATA SUMMARY

Length of sump face: 131.2 ft.

Velocity of shearer: 20 ft./min.

Sumping time, excluding delay time: 13.12 min.

Available cutting time: 250 min.

Seam height: 16 ft.

Web size: 3.28 ft.

Coal density: .04 tons/cubic ft.

Face conveyor capital cost: $1458 K per 500 ft. of face length

Shields capital cost: $8000 K per 500 ft. of face length

Annual MTCE % applied to shields capital cost: 6%

Annual MTCE % applied to conveyor capital cost: 10%

Number of shifts per work day: 3

Number of work days per year: 240

Main entry (8) adv. rate: 6.93 ft./shift
(on a 3 shift basis)

Sub-main entry (6) adv. rate: 9.4 ft./shift
(on a 3 shift basis)

LONGWALL VARIABLES
100% OF LONGWALL OPTIMUM PRODUCTION

FACE LENGTH	SUMPING TIME	ADV/ SHIFT	TONS/ SHIFT
200	25	18.4518	2361.84
200	45	12.725	1628.8
600	25	9.71104	3729.04
600	45	7.8514	3014.94
800	25	7.8514	4019.92
800	45	6.58952	3373.83
1000	25	6.58952	4217.29
1000	45	5.6771	3633.34

COSTS RELATED TO FACE LENGTH

VARIABLE EQUIPMENT COSTS $K PER UNIT			VARIABLE OPR. COSTS $ PER SHIFT	
FACE LENGTH	FACE CONVEYOR	SHIELDS	SHIELD MAINT.	CONVEYOR MAINT.
200	583.2	3200	266.667	81
600	1749.6	9600	800	243
800	2332.8	12800	1066.67	324
1000	2916	16000	1333.33	405

LONGWALL VARIABLES
100% OF LONGWALL OPTIMUM PRODUCTION

FACE LENGTH	SUMPING TIME	ADV/ SHIFT	TONS/ SHIFT
300	25	15.0625	2891.99
300	45	11.0156	2114.99
350	25	13.7954	3090.17
350	45	10.3223	2312.19
400	25	12.725	3257.6
400	45	9.71104	2486.03
450	25	11.8088	3400.92
450	45	9.16816	2640.43
500	25	11.0156	3524.99
500	45	8.68276	2778.48

COSTS RELATED TO FACE LENGTH

	VARIABLE EQUIPMENT COSTS $K PER UNIT		VARIABLE OPR. COSTS $ PER SHIFT	
FACE LENGTH	FACE CONVEYOR	SHIELDS	SHIELD MAINT.	CONVEYOR MAINT.
300	874.8	4800	400	121.5
350	1020.6	5600	466.667	141.75
400	1166.4	6400	533.333	162
450	1312.2	7200	600	182.25
500	1458	8000	666.667	202.5

LONGWALL VARIABLES
85% OF LONGWALL OPTIMUM PRODUCTION

FACE LENGTH	SUMPING TIME	ADV/ SHIFT	TONS/ SHIFT
300	25	12.8031	2458.19
300	45	9.36325	1797.74
350	25	11.7261	2626.65
350	45	8.77392	1965.36
400	25	10.8163	2768.96
400	45	8.25438	2113.12
450	25	10.0374	2890.78
450	45	7.79293	2244.37
500	25	9.36325	2996.24
500	45	7.38035	2361.71

COSTS RELATED TO FACE LENGTH

VARIABLE EQUIPMENT COSTS $K PER UNIT			VARIABLE OPR. COSTS $ PER SHIFT	
FACE LENGTH	FACE CONVEYOR	SHIELDS	SHIELD MAINT.	CONVEYOR MAINT.
300	874.8	4800	400	121.5
350	1020.6	5600	466.667	141.75
400	1166.4	6400	533.333	162
450	1312.2	7200	600	182.25
500	1458	8000	666.667	202.5

LONGWALL VARIABLES
50% OF LONGWALL OPTIMUM PRODUCTION

FACE LENGTH	SUMPING TIME	ADV/ SHIFT	TONS/ SHIFT
300	25	7.53123	1446
300	45	5.50779	1057.54
350	25	6.89771	1545.09
350	45	5.16113	1156.09
400	25	6.36251	1628.8
400	45	4.85552	1243.01
450	25	5.90438	1700.46
450	45	4.58408	1320.21
500	25	5.50779	1762.49
500	45	4.34138	1389.24

COSTS RELATED TO FACE LENGTH

	VARIABLE EQUIPMENT COSTS $K PER UNIT		VARIABLE OPR. COSTS $ PER SHIFT	
FACE LENGTH	FACE CONVEYOR	SHIELDS	SHIELD MAINT.	CONVEYOR MAINT.
300	874.8	4800	400	121.5
350	1020.6	5600	466.667	141.75
400	1166.4	6400	533.333	162
450	1312.2	7200	600	182.25
500	1458	8000	666.667	202.5

LONGWALL VARIABLES
75% OF LONGWALL OPTIMUM PRODUCTION

FACE LENGTH	SUMPING TIME	ADV/ SHIFT	TONS/ SHIFT
300	25	11.2968	2168.99
300	45	8.26169	1586.24
350	25	10.3466	2317.63
350	45	7.74169	1734.14
400	25	9.54376	2443.2
400	45	7.28328	1864.52
450	25	8.85657	2550.69
450	45	6.87612	1980.32
500	25	8.26169	2643.74
500	45	6.51207	2083.86

COSTS RELATED TO FACE LENGTH

	VARIABLE EQUIPMENT COSTS $K PER UNIT		VARIABLE OPR. COSTS $ PER SHIFT	
FACE LENGTH	FACE CONVEYOR	SHIELDS	SHIELD MAINT.	CONVEYOR MAINT.
300	874.8	4800	400	121.5
350	1020.6	5600	466.667	141.75
400	1166.4	6400	533.333	162
450	1312.2	7200	600	182.25
500	1458	8000	666.667	202.5

APPENDIX D

PURCHASE CONTRACT GENERAL CONDITIONS

APPENDIX D

PURCHASE CONTRACT GENERAL CONDITIONS

The supplier shall offer all equipment which will ensure continuous and safe operation and will comply with the data and guarantees stated even if parts of the equipment are omitted in this specification.

All equipment and accessories must comply with the regulations and standards of the United States Bureau of Mines (USBM) Schedule 2G and with the Mining Enforcement and Safety Administration (MESA) Code of Federal Regulations, Title 30, Mineral Resources, Chapter I together with the individual issues of the Federal Register. USBM approval plates shall be attached to the equipment. All material shall conform to ASTM standards.

Assembly Drawings

The purchase contract will require the provision of four (4) complete sets of assembly drawings to be delivered to the owner one month prior to receipt of any of the equipment. The drawings will show in as simple a manner as practical the method of assembly of the equipment. Exploded drawings may be used and part numbers must be used whether or not part names are used. All parts must be numbered both on the drawings and on the part itself. Distinctive numbering must be applied by welding or with durable paint or other appropriate marker. Loose labels or tags are not acceptable since they are liable to become displaced and lost.

The drawings must be complete enough to allow the equipment to be assembled by mining personnel who may not be familiar with the equipment.

Fabrication or manufacturing drawings are usually not suitable for this purpose and are therefore specifically not wanted.

Spare Parts

A spare parts list is required for each type of equipment proposed. Such a list will comprise the parts necessary to ensure that downtime is minimized. The spare parts agreed upon will be purchased as part of the whole package. Six (6) complete parts books will be provided for each major piece of equipment proposed.

Technical Assistance

During the period of receipt of and assembling the equipment, the start up of the face, and for six working weeks thereafter the supplier shall supply, free of charge, technical assistance composed at the minimum of:

- One electrical service engineer
- Two service engineers for support
- Two service engineers for shearer
- Two service engineers for conveyors

Compatibility

It will be the responsibility of the prime contractor to ensure the compatibility of the different equipment. Any failure to ensure compatibility which results in increased costs to the owner will be reimbursed to the owner by the contractor.

Equipment Warranties

All equipment proposed to be supplied will be guaranteed to be free from defect in material, design and construction for a specified period depending upon the type of equipment, but at least twelve (12) months. Such warranty must not be contingent upon the buyer returning the defective part to the seller's factory.

Prices

All quotations shall be in U. S. dollars and must be F.O.B. minesite with packing cases and materials included. The terms of payment must be specified with the quotation.

Training

It is the explicit intent of any contract resulting from this request for proposal that the assembly supervision or service representatives will show by demonstrating and explaining how the equipment or machinery is best operated. These demonstrations will be conducted for the coal miners whose job it will be to operate the equipment as well as to the management training personnel if mine management so require.

Maintenance Training

In order that the miner personnel may become competent to maintain and repair the equipment, specially oriented sessions should be held prior to installation which would then be supplemented by hands-on experience during installation and debugging.

Special Tools

Any special tools necessary for the installation and/or maintenance of the proposed equipment shall be included in this package.

Delivery

The owner will have a need for the subject longwall equipment by ______(Date)______.

In order to minimize potential delays in delivery, your proposals shall identify the termination dates of any labor contracts which would adversely affect the performance of any contract resulting from these proposals.

Your proposals upon action to be taken to further minimize the risk of delays are solicited.

Pert-CPM Diagram

Contractor shall provide a preliminary Pert-CPM diagram to the Company as a part of the bid documents. This Pert-CPM diagram shall cover the scheduled status of all functions of the project including, but not limited to, the following:

- o Engineering
- o Procurement
- o Fabrication
- o Compatibility Testing
- o Transportation
- o Installation
- o Testing

Contractor shall, within 25 days after receipt of notification of contract award, provide Company with a detailed Pert-CPM diagram and, after work has started, deliver monthly to the Company an updated Pert-CPM diagram showing changes in the project scope, planning, or scheduling which reflects the project status as of the end of the previous month. This diagram shall be delivered on or before the 10th of each month.

☆U.S. GOVERNMENT PRINTING OFFICE: 1982-646-069 286