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College of Earth and Mineral Sciences

AN ANALYSIS OF UNDERGROUND EXTRACTION TECHNIQUES FOR THICK COAL SEAMS

by

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ABSTRACT

Less than two percent of this nation's annual coal production comes from underground mines in the West. A contributing factor to the low production appears to be the unfavorable economics of deep mining western coal seams. Additionally, a large fraction of the western coal reserves that may be deep mineable are thick seams. Thick coal seams do not lend themselves readily to extraction with methods common to the coal industry in the U.S.A. However, coal production from western deep mines has to increase to alleviate the current energy crisis. Although the development of new technology and equipment may be in order for the long run, attention must be directed, for short-term gains, to the adoption of proven technology.

The problems, technical and non-technical, which are associated with the mining of thick coal seams are analyzed in this thesis. Mining methods currently practiced abroad are reviewed. A discussion of North American thick-seam mining, based on information gathered from mine-site visits and published literature, is also included. On the basis of an inventory of geologic and mining conditions in the West, four methods are recommended for potential application. An economic evaluation of the proposed methods, on a panel basis, has been conducted. A comparative analysis of the proposed methods with typical conventional and continuous sections is provided.

The thesis has arrived at a number of conclusions. Although there are several thick-seam methods practiced abroad, these methods cannot be adapted, for the most part, without variations for U.S. conditions. This is due, mainly to the different economic and political climate of the United States. However, thick-seam methods can be

designed to be economically competitive with methods employed in seams of average thickness. Further, the recovery rates are also acceptable. New equipment, such as shields, should find increased acceptance whereas standard U.S. equipment, such as shuttle cars, will have limited applications.

I. INTRODUCTION

General

The U.S. energy consumption during the last 25 years has grown approximately at an average annual rate of four percent. The domestic energy production experienced little improvement during the last five years, and for two decades prior to 1970, the average yearly increase has been only three percent. In recent years, imported fuel, primarily oil, has made up the difference. Figure 1 presents the fuel-mix pattern of consumption, and illustrates the trend away from coal over the past half century. This growing disparity between the national energy consumption and production, coupled with the uncertainty of energy supply at the stated price from extra-national sources, has precipitated the need for an accelerated development of domestic energy resources.

Coal is the only conventional energy resource which still exists in great abundance, as shown in Table 1 (Ford Foundation, 1974). Barring the discovery of new and novel sources of energy, the depletion of petroleum and natural gas should turn the United States back to the use of coal as a primary energy source. It is theorized that coal will play an important role in the energy policies of the nation. In fact, future demands for coal are conservatively predicted to rise to 700 million tons (635 million metric tons)[1,2] by 1980, and to anywhere between one and three billion tons (0.9 and 2.7 billion metric tons) by the year 2000. Therefore, this thesis concerns itself with one aspect

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[1] In the body of this text, equivalent metric units are shown in parentheses following the English measurements.

[2] All tonnage figures are given in short tons (1 short ton = 2000 pounds).

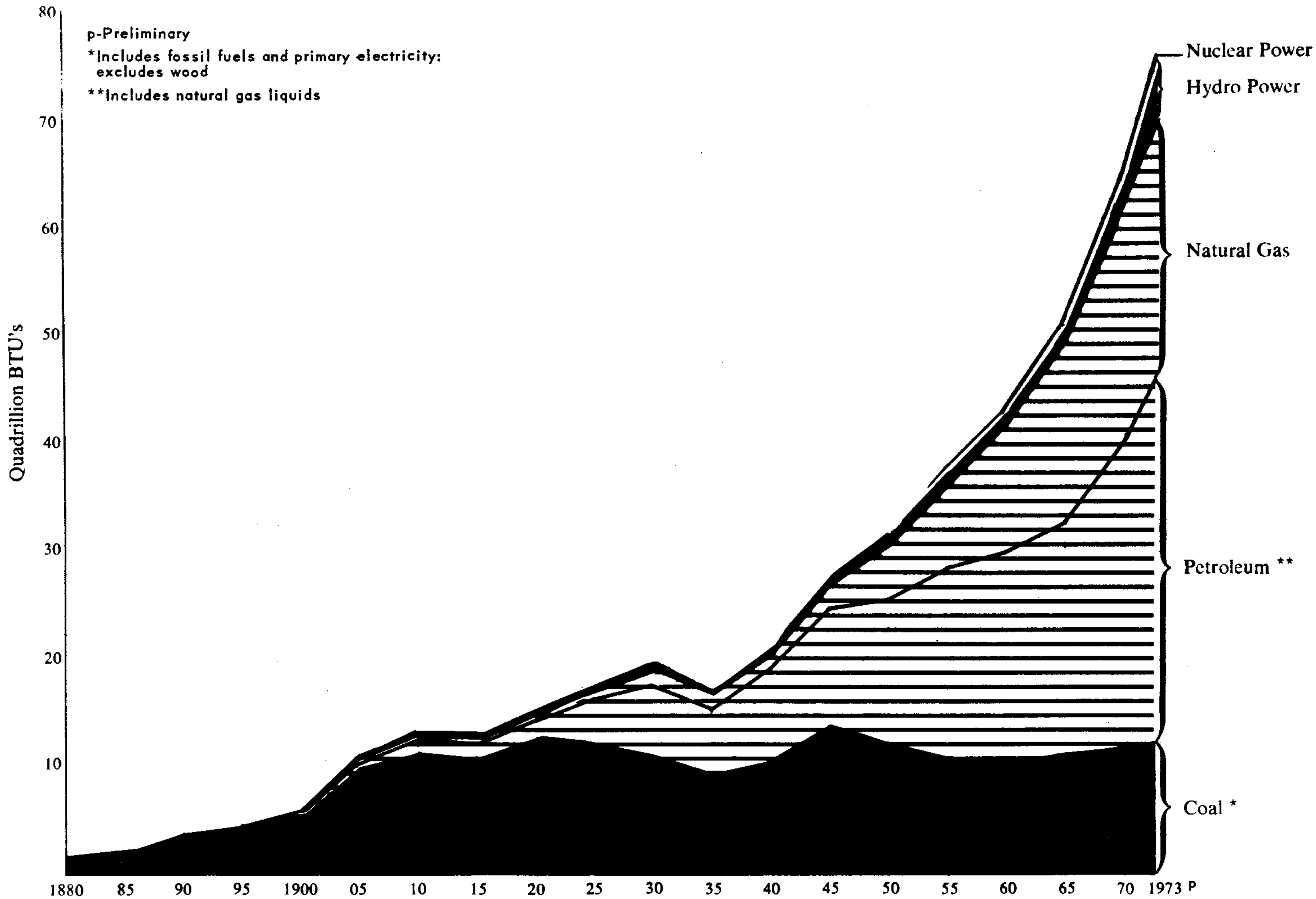


FIG. 1 ENERGY CONSUMPTION IN THE U.S.A. BY SOURCES (1880 - 1973)
(Ford Foundation, 1974)

Table 1. Major Energy Resources of the U.S.A. (after Ford Foundation, 1974)

	1973 Consumption (Quadrillion Btu)	Cumulative Production (Q Btu)	Reserves (Q Btu)	Recoverable Resources (Q Btu)	Remaining Resource Base (Q Btu)
Petroleum	34.7	605	302	2,910	16,790
Shale Oil	----	---	(465)	N/A	975,000
Tar Sands	----	---	---	N/A	168
Natural Gas	23.6	405	300	2,470	6,800
Coal	13.5	810	4,110	14,600	64,000
Strippable coal	N/A	N/A	925	2,600	2,600
Low-sulfur coal	N/A	N/A	2,390	N/A	38,200
Uranium					
Used in light-water reactors	.85	2	228	600	3,200
Used in breeders	----	---	17,700	47,000	200,000,000
Thorium used in breeders	----	---	4,200	17,500	570,000
Hydropower	2.9			5.8*	

N/A not available

* ultimate capability

Note: The terms "Reserves," "Recoverable Resources," and "Remaining Resource Base" are geological estimates. "Reserve" estimates are based on detailed geologic evidence, usually obtained through drilling, while the other estimates reflect less detailed knowledge and more geologic inference. All of these estimates are based on assumptions about technology and economics. They may increase over time as technology improves or prices increase.

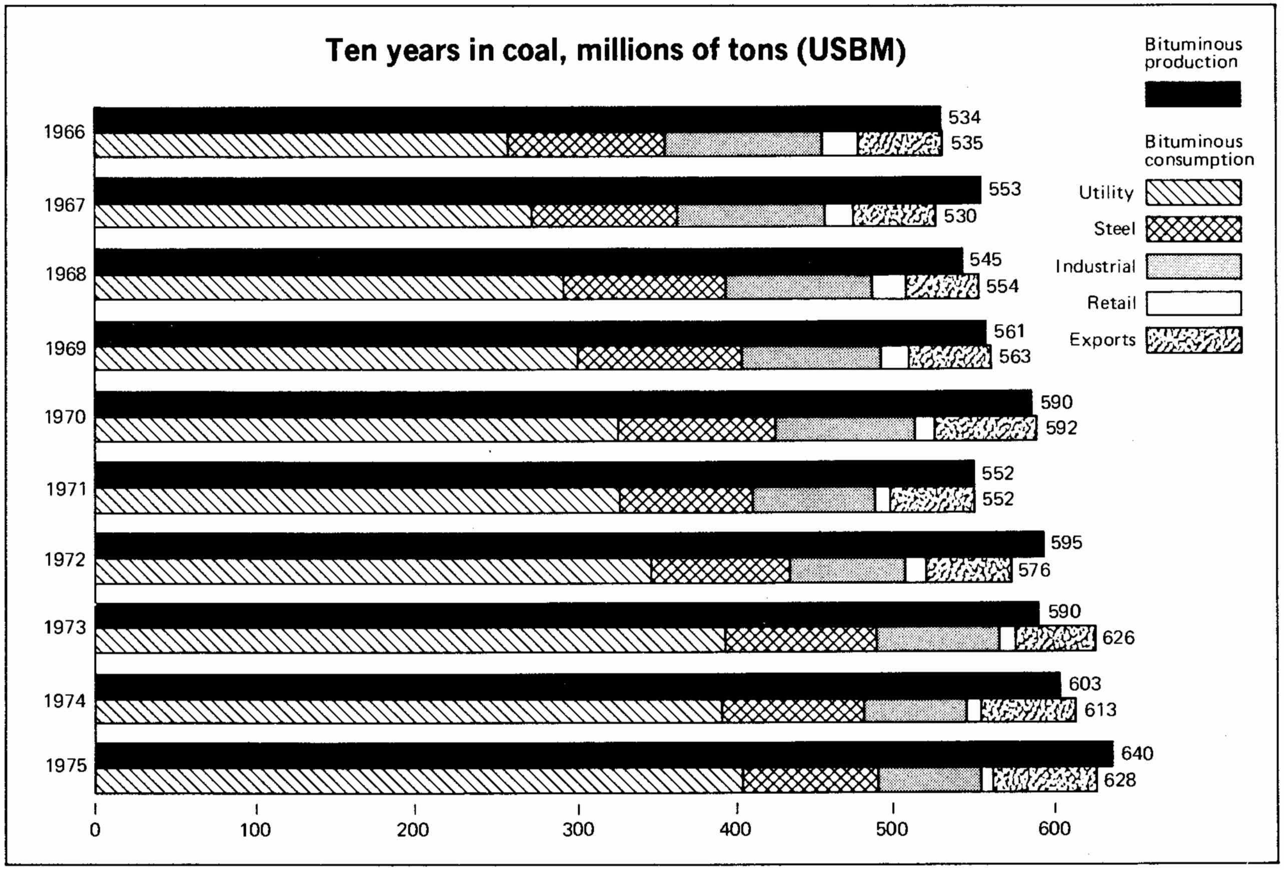
of supplying the predicted demand, i.e., extracting coal from thick seams by underground methods.

Coal Industry Background

In light of the above predictions for coal, it is necessary to review several coal-industry developments. During the Fifties, the loss of the railroad and home-heating markets led to an accelerated decrease in coal production. The coal output plummeted from a record 630 million tons (571 million metric tons) in 1947 to 391 million tons (355 million metric tons) in 1954. Though slow in recovery,the industry not only grew during the Sixties, but appeared poised to reach a new zenith. However, during the last ten years, there has not been a significant increase in production (Figure 2). The following is a selected and not an all-inclusive list of important developments during this period which may have worked to curb coal's resurgence:

1. The National Environment Policy Act, 1969.
2. The Coal Mine Health and Safety Act, 1969.
3. The Clean Air Act of 1970.
4. Surface mine legislations.
5. Foreign tax credit for oil companies.
6. The extreme lead times for mine development and equipment procurement.
7. Uncertain fuel use patterns and unpredictable price structure to permit long term planning.
8. Shortage of trained manpower.
9. Increased labor, capital, and operating costs.
10. Increased social concern for environmental protection.

Salient statistics on production, employment, and productivity in the coal industry for the years 1965 through 1975 are presented in

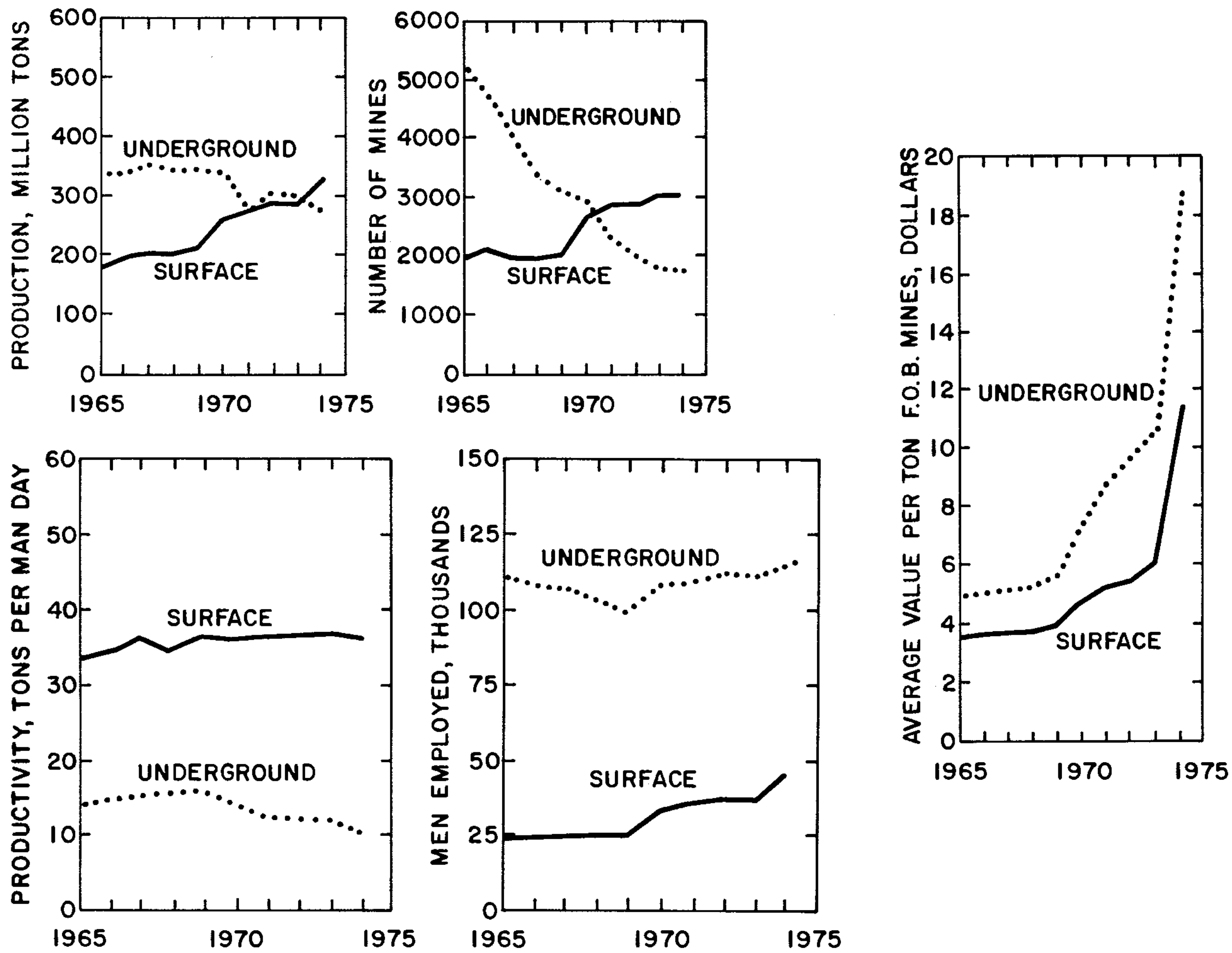


1 Short Ton = 0.907 Metric Tons

FIG. 2 PRODUCTION AND CONSUMPTION OF U.S. BITUMINOUS COAL (1966 - 1975) (Merritt, 1976)

Figure 3. With regard to underground coal mining, a dramatic downward trend in productivity since 1969 is evident, a situation caused by decreased production and increased manpower. From 1965 onwards, surface mines have made impressive gains both in numbers and in production. Surface-mined coal is projected to play an even more important role in the future (Federal Energy Administration, 1974). However, to meet the energy demands in the later years of this century, it has been projected that production from underground mines must also significantly increase (Table 2). Additionally, these mines will probably be of capacities ranging from one to three million tons (0.9 to 2.7 million metric tons) per year. Continuous and conventional mining methods are still the most common, and account for over 95% of the underground production (Stefanko, 1976). Longwall mining, though becoming increasingly popular, accounts for less than 2% of the total coal production. The shortwall method has evoked considerable interest as it attempts to combine the advantages of the longwall with the capabilities of a continuous miner. Since there are only six faces in operation, it is difficult to make any predictive statement. However, during the last five years, several projects have been funded to develop better equipment and methods. Even then, to transfer research and development results to operating equipment and procedures requires both lead time and resources. Extensive deployment may follow later, but only through demonstrated success.

One area that has received little or no research attention in the United States is the mining of thick coal seams by underground methods. The United States Bureau of Mines has recently established



1 Short Ton = 0.907 Metric Tons

FIG. 3 TRENDS IN THE U.S. COAL MINING INDUSTRY (1965 - 1975)
(Falkie, 1976)

Table 2. New Mine Requirements (1975 - 1990)*
(after Federal Energy Administration, 1974)

	Business as Usual***	Accelerated Development***
Underground Mines:		
1 million tons	153	445
3 million tons	74	190
Surface Mines:		
1 million tons	110	195
3 million tons	25	90
5 million tons**	98	219
Total	460	1129

* Including new mines to replace depleted productive capacity and new mines to increase existing productive capacity.

** Although there are now 10-million ton surface mines in the West, and others are on the drawing board, for the purpose of this report nothing larger than a 5-million ton mine was considered. Checks with western surface mine operators indicate that the economy of scale is such that the cost of producing coal at a 10-million ton mine was considered the equivalent of two 5-million ton mines for the purpose of determining minimum selling prices, manpower requirements, equipment and supply requirements, etc.

*** Production targets for two cases

1 Short ton = 0.907 metric tons

a demonstrated coal reserve base[1] of 434 billion tons (394 billion metric tons) (Murphy, *et al.*, 1974). The majority of the coal in this country (54%) is located west of the Mississippi River. Though the emphasis today in the West is on surface mining, about 56% of the western reserves can only be extracted by underground mining methods (Figure 4).

The potential of the western reserves for meeting the energy needs has been recognized for quite some time. Many constraints have limited both surface and underground operations in the West. Primary among these was the great distances to the large volume markets of the East and Midwest, which placed the western coal at a competitive disadvantage when compared to that from the Appalachian fields. There are, however, several technical reasons for the low volumes of deep-mined coal. Western coalfields have seams which are greater in thickness than those presently mined in the United States. In fact, 45 billion tons (41 billion metric tons) of the deep-mineable western reserves are located in deposits greater than 10 ft (3m) in thickness

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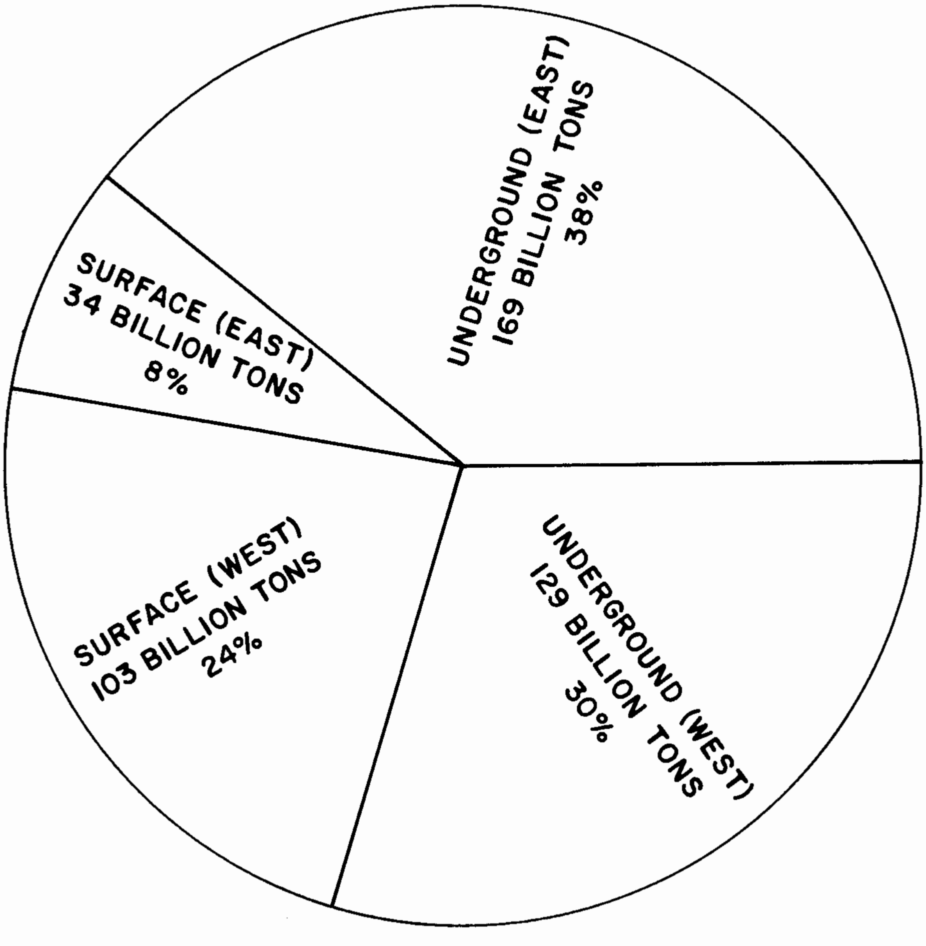
[1] "... The reserve base includes beds of bituminous coal and anthracite 28 inches (711.2 millimeters) or more thick and beds of subbituminous coal 60 inches (1524 millimeters) or more thick that occur at depths to 1,000 feet (304.8 meters), as well as beds of lignite at depths no greater than 120 feet (36.58 meters). Also included are thinner and/or deeper beds that presently are being mined or for which there is evidence that they could be mined commercially at this time... Demonstrated reserves...[are]...the sum of the measured and indicated reserves...measured reserves...[are]...computed from dimensions revealed in outcrops, trenches, mine workings and drill holes. The points of observation and measurement are so closely spaced and the thickness and extent of coalbeds are so well defined that the calculated tonnage is judged to be accurate within 20% of true tonnage... Indicated reserves ...[are]... computed partly from specified measurements and partly from projection of visible data for a reasonable distance on the basis of geologic evidence..." (Matson and White, p.6).



1 Short Ton = 0.907 Metric Tons

FIG. 4 DEMONSTRATED COAL RESERVE BASE OF THE U.S.A.
(Murphy, et al., 1974)

(Smith and White, 1975). Also, the tectonic activities associated with mountain building have altered the attitude of seams such that they often pitch to a greater extent than their eastern counterparts. Finally, the coal mining equipment manufactured today is, primarily, designed for eastern deep mining [12 ft (3.6m), maximum; 5 ft(1.5m) average] where seams are basically tabular. Therefore, it is not surprising that western deep mines account for less than two percent of the nation's annual output (Yancik, 1975). The need to increase coal production over the next few years, combined with the changing energy market, makes it imperative that a sound evaluation of the deep-mineable western reserves be carried out. Of equal importance is an effort to evaluate mining methods for thick coal seams, in general, and their applicability to western deposits, in particular. Although new technology may be available in the long run, short term gains can result only from adapting present technology, with or without modifications, to thick seams.

Purpose and Scope of Work

The projected role of coal in achieving energy self-sufficiency and the existence of abundant coal reserves which are unrecoverable with present mining methods are two important factors that led to the present study. To the best knowledge of this author, there has not been any organized or widely known study, to date, on the underground mining of thick coal seams which occur in the western United States. Since early 1975, several studies have been initiated, and a greater volume of literature on the subject may become available. However, most of these studies are site specific and are, therefore, likely to be limited in scope. The objective of this study is the development of some broad concepts for the extraction of thick seams with varying degrees of thickness and pitch. As a prerequisite to the development and potential application of the methods, several ancillary areas needed to be researched. Therefore, in this study, the specific work done is listed below:

1. A general description of the western coalfields, with particular attention to reserves and geologic conditions, is developed.
2. A review of the thick-seam mining methods practiced in other countries is provided.
3. A discussion of the deep-mining practices in the United States and Canada, based on published literature and on data collected during field trips, is provided.
4. An analysis of the various equipment that are in use today, for possible application in thick seam mining is included.
5. Safety considerations, that are peculiar to thick-seam extraction, are briefly reviewed.
6. Four mining methods are proposed for possible application in thick seams with varying pitches and thicknesses.
7. A comparative analysis among the four proposed mining methods, as well as a conventional mining and continuous mining application in a seam of average thickness, is also provided. This analysis considers safety and economic factors.

A thick coal seam is defined to be over 12 ft (3.6m) in thickness, since this height is greater than that mineable with the normal range of most presently available equipment. The following additional definitions are provided to clarify the discussion in subsequent chapters of this thesis (Thrush, 1968):

1. Dip: The angle at which a coal seam is inclined from the horizontal. For the purposes of this thesis, it is further subdivided into the following classifications (Cochrane, 1972):

   Tabular (0° to 3°)
   Gently Pitching (3° to 15°)
   Moderately Pitching (15° to 25°)
   Inclined (25° to 45°)
   Steeply Pitching (45° to 90°)

2. Entry (Drift) (Sublevel): Names given to a development heading driven in a coal seam. Entry is more commonly used in tabular seams while drift is used in reference to headings driven in pitching seams. Sublevel is used where the heading is driven such that, on retreat, the coal is recovered by caving.

3. Slice (Lift): The division of a thick coal seam, for exploitation purposes, into two or more layers, either parallel to the bedding plane of the seam or along the horizontal, is common. Each such layer is known as a slice or a lift.

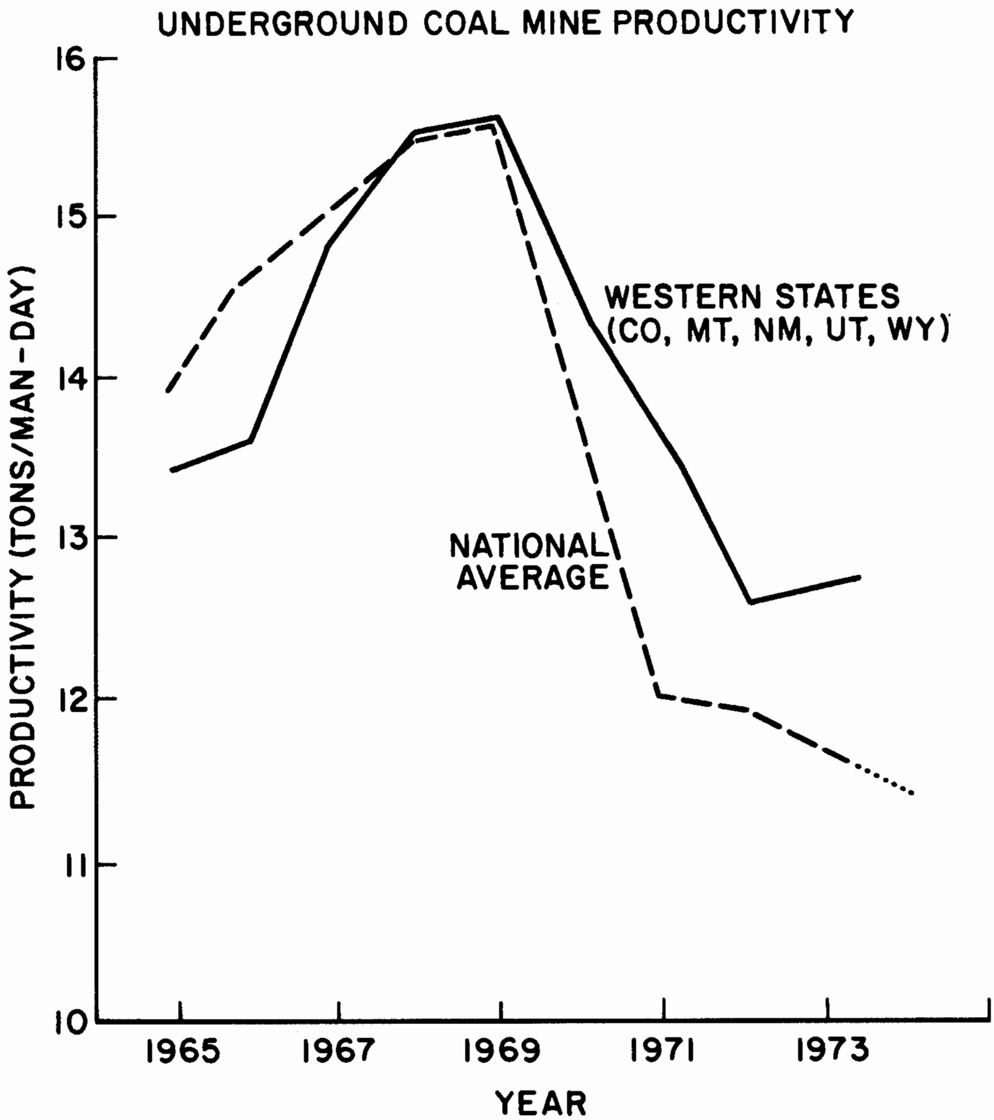
4. Stowing (Packing): A ground control practice often used with thick-seam mining where the void created by the extraction is backfilled with a non-clayey material, such as sand.

II. WESTERN COAL RESERVES

The Project Independence Report has called for the doubling of the nation's coal production by 1985. The projection for the western coal's contribution to meet this increased supply is rather significant. The estimate suggests an increase of nearly 400% in the annual production from the West, from 52 million tons (47 million metric tons) at the present time to 240 million tons (218 million metric tons) by 1985 (Yancik, 1975).

There may be several reasons for the projected four-fold increase in production. Nearly 90% of the nation's low-sulfur coal is located in the West (Yancik, 1975). The underground production per manshift for western coal has generally followed the national trend, although the rate of decrease in productivity has been smaller than the national rate (Figure 5). Additionally, more than 50% of the western deep-mined coal is shipped to the metallurgical market whereas, at the national level, only 17% of the total underground production is consumed for metallurgical purposes (Yancik, 1975). It is apparent that the need to examine the potential of underground mining of the western coal is important.

Although the evaluation of western coal entails many factors, the geological factors, either directly or indirectly, greatly affect their economic potential. These factors have a tremendous impact, particularly on deep-mineable reserves, because the whole mining system is geologically enclosed. Therefore, the geological aspects of the western thick seams are examined, though not in any great detail. The geological conditions in the coalfields of Alberta and British Columbia are similar to those in the western United States. Therefore, the reserves and mining conditions in these Canadian provinces are also reviewed.



1 Short Ton = 0.907 Metric Tons

FIG. 5 UNDERGROUND COAL MINE PRODUCTIVITY (1965 - 1973)
(Yancik, 1975)

Western Coalfields

The number of underground coal mines in the West in operation today is around 50, a far cry from the 500 in operation during the early 1930's (Yancik, 1975). Despite this decreasing number of underground mines, annual production from the West increased until 1948 (Figure 6). Through the 1950's, western production followed the downward trend of the coal industry already referred to in the previous chapter. However in recent years, due to the low-sulfur characteristics of western coal, contribution of the western reserves to the total coal production has started to increase. Surface mining, rather than deep mining, has accounted for the bulk of the tonnage. The relative ease with which the shallow-burden thick coal seams of the West can be extracted through surface mining methods can hardly be overemphasized. Yet, there is more coal in the West that must be deep mined than there is surface-mineable coal (Murphy, *et al.*, 1974). The reserves in the Rocky Mountain coalfields are broken down into underground and surface-mineable reserves, by states, in Figure 7. While reserves that must be deep mined are greater than those that must be surface mined in all the states, the underground mineable reserves of Utah and Colorado far outweigh the surface-mineable reserves.

The location of the coalfields in the Rocky Mountain states is shown in Figure 8. A brief description of the major fields, with particular reference to the thick seams in each of the states, follows.

Utah

Nearly all of Utah's reserves, which have been classified into 18 fields, must be recovered by deep-mining methods. The major coal

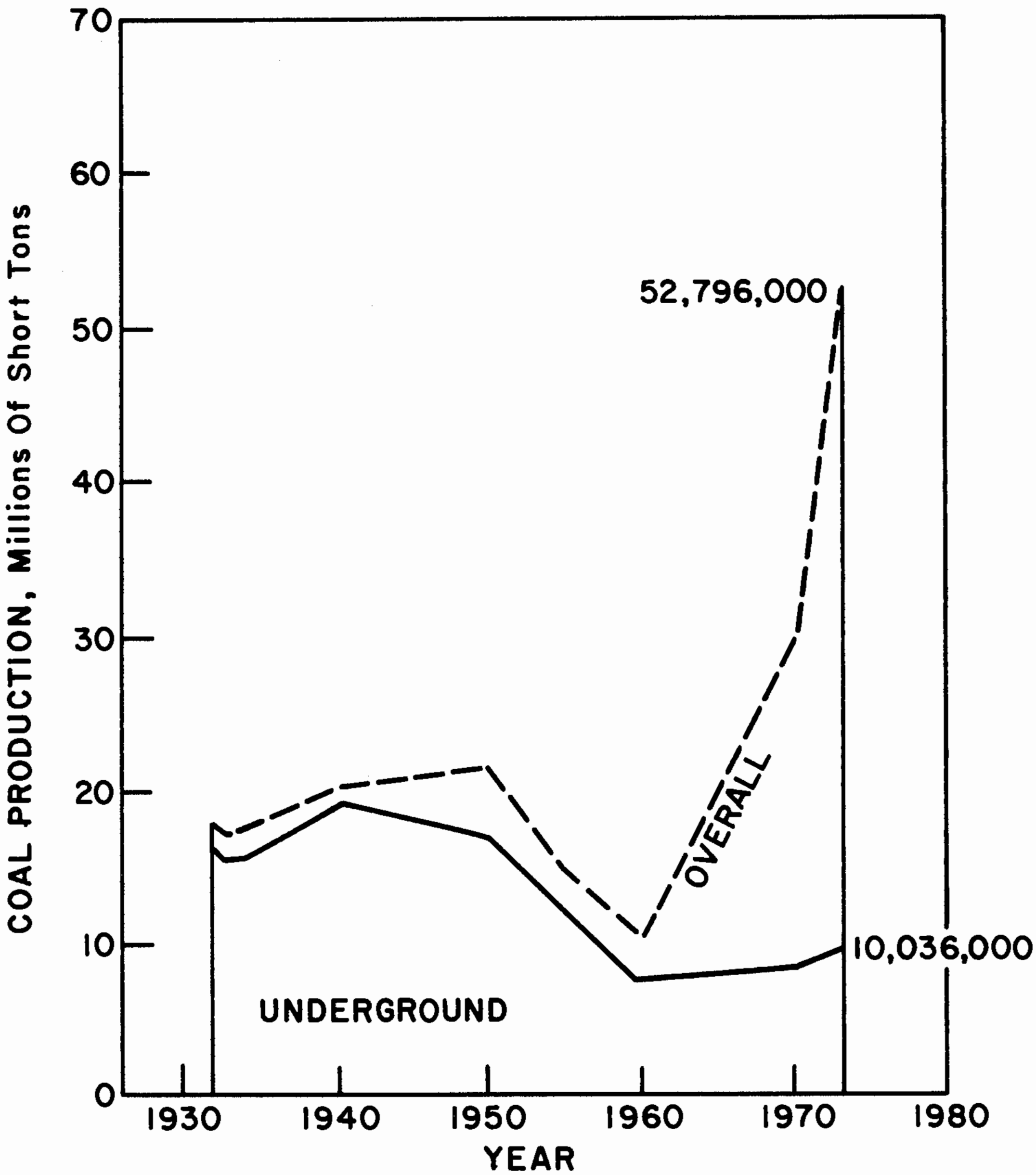


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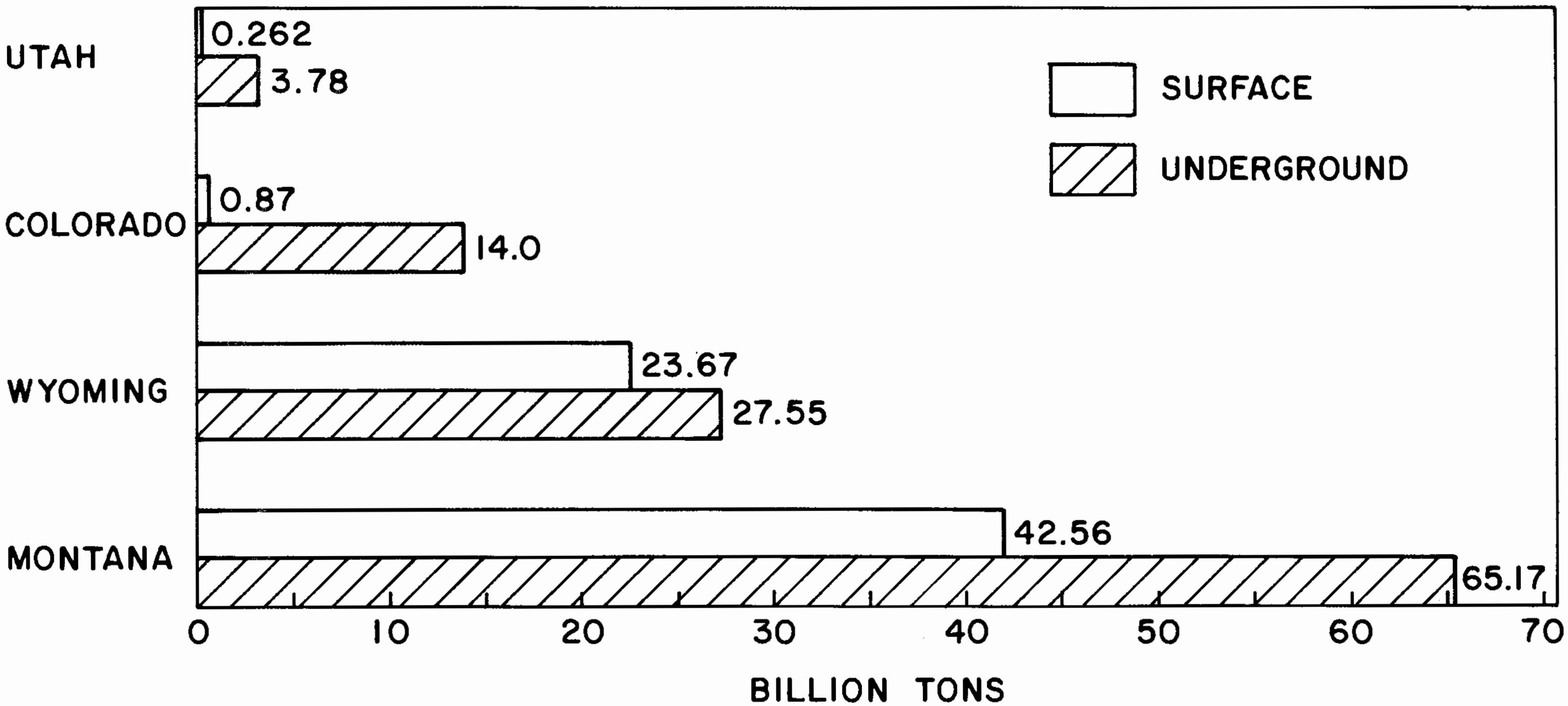
FIG. 6 ANNUAL PRODUCTION FROM WESTERN MINES 1930 - 1973)
(Yancik, 1975)



1 Short Ton = 0.907 Metric Tons

FIG. 7 SURFACE AND UNDERGROUND RESERVES, U.S. ROCKY MOUNTAIN STATES (Murphy, et al., 1974)

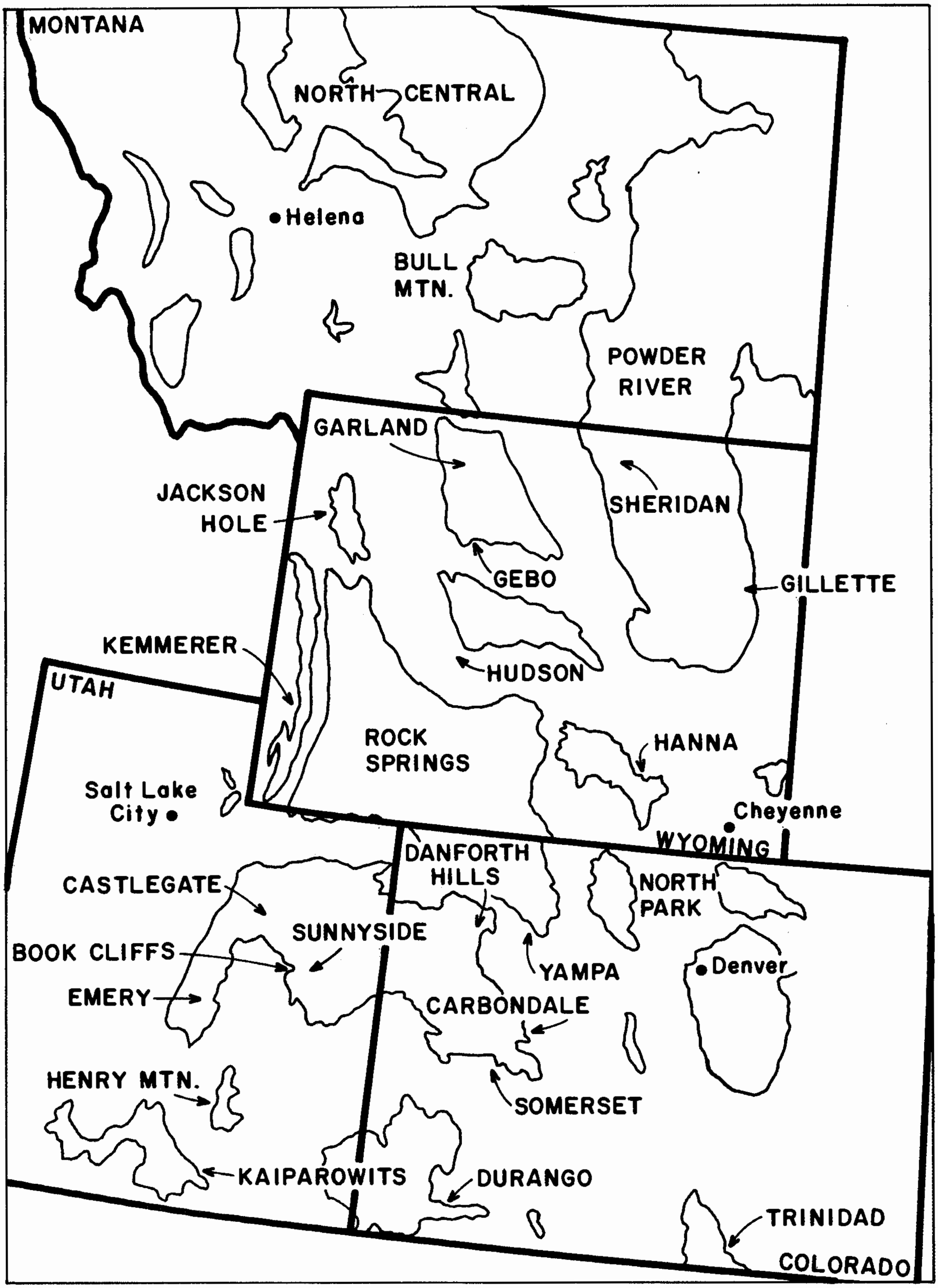


FIG. 8 ROCKY MOUNTAIN COALFIELDS OF THE U.S.A.

fields are Book Cliffs, Emery, and Kaiparowits (Keystone Coal Industry Manual, 1974).

Over 60% of the state's tonnage comes from the Book Cliffs field. It supplies the western metallurgical markets with high-quality coal. The major operations in the field are captive to steel companies. One seam of particular interest is the Lower Sunnyside. It ranges in thickness from 7 to 18 ft(2.1 to 5.5m), dips from 7° to 15°, and is characterized by numerous faults. Ground control problems are aggravated by the massive Castle Gate sandstone, which lies approximately 100 ft (30m) above the Lower Sunnyside, and by the depth of the workings, which range up to 2500 ft (758m) (Huntsman, 1974).

The Emery field is 80 miles (128km) long and 10 miles (16km) wide, and is located along the Castle Valley in central Utah. It contains over 750 million tons (680 million metric tons) of demonstrated reserves (Keystone Coal Industry Manual, 1974). Of particular interest is the I-J seam (two converging seams) which varies in thickness from 13 to 21 ft (4 to 6.4m). The immediate roof consists of the 41-ft (12.4m) Ferron sandstone. This seam dips gently (3°), and its depth does not exceed 800 ft (242m).

The Kaiparowits field, which has multiple seams ranging up to 30 ft (9m) in thickness, received a great deal of attention when Kaiser Industries announced its intent to create a 10-million-tpy (9-million-metric tpy) mining complex. At the present time, there are no active mines in the area and environmentalists have won the initial battle to prevent any new development.

Colorado

Ninety-five percent of Colorado's reserves must be deep mined

(*Keystone Coal Industry Manual*, 1974). Coals of western Colorado exhibit good coking qualities. The fields of interest to thick-seam mining are Yampa, Somerset, Carbondale, Danforth Hills, and North Park (Hornbaker and Holt, 1973).

The Yampa field, located in the northwestern part of the state, has seams in excess of 15 ft (4.5m) and, in one area, has a seam in excess of 40 ft (12.1m) (Hornbaker and Holt, 1973).

Seams in the Somerset field, whose estimated mineable reserves is 3.3 billion tons (3 billion metric tons) range up to 25 ft (7.6m) in thickness. Of primary interest is the B seam, a 22-ft (6.7m) seam of good coking quality. The geology in this area is similar to that of the Book Cliffs area of Utah (Watson, 1974).

Fifty percent of the coal in the Carbondale field, where seams ranging up to 16 ft (4.8m) in thickness are reported, is of coking quality. Demonstrated reserves in this field total 1.1 billion tons (1 billion metric tons) (Hornbaker and Holt, 1973).

The Danforth Hills field has seams of up to 34 ft (10m) in thickness. Mining has been conducted in the 23-ft (7m) Collom seam. The Collom seam has a dip of less than 4°, and occurs at depths ranging from 200 ft to 750 ft (60 to 227m). An 18-ft (5.5m) seam is located only 60 ft (18.2m) below the Collom seam. Therefore, the method and sequence of extraction in the Collom seam become important to conserve and safely recover the lower coal seam (Hornbaker and Holt, 1973).

Although there are no active mines in the North Park field, there are several important seams. In the northeastern part of the basin, the Sudduth seam varies in thickness from 10 ft (3m) to 58 ft (17.5m) and dips from 20° to 85°. In the southwest part of the basin, the seams dip up to 20° and one seam, the Riach, varies in thickness from 22 to 77 ft (6.7 to 23m). Mining in the past has been conducted primarily in the Riach seam (Hornbaker and Holt, 1973).

Wyoming

Although emphasis in Wyoming has been on surface mining, deep-mineable reserves are located in the Kemmerer, Hanna, and Rock Springs fields. Further, 85% of the state's deep-mineable reserves are in thick seams (NUS Corporation, 1976).

The coals in the Kemmerer field range in thickness from 6 to 118 ft (1.8 to 36m). Even though these reserves are surfaced mined today, coal was deep mined in this field many years ago. Since the formation dips at 18°, these coal seams must eventually be recovered by deep-mining methods (Figure 9) ("Subbituminous for Power...,"1963).

The Hanna and Rock Springs coalfields have seams which range in thickness from 7 to 35 ft (2.1 to 10.6m). Interest has been shown by operating companies in one of the seams which dips at 20°, and has a thickness of 30 ft (9m) (Keystone Coal Industry Manual, 1974).

Montana

As shown in Figure 7, the deep-mineable reserves in Montana are greater than the surface-mineable reserves. However, only one underground mine was reported in the state in 1973 (Yancik, 1975). Attention at the present time has been drawn to the tremendous reserves of thick seams which are recoverable by surface mining methods. Additionally, only 1.4 billion tons (1.3 billion metric tons) of the state's underground reserves, amounting to approximately two percent, is of bituminous quality (Matson and White, 1975). Emphasis in this state, therefore,

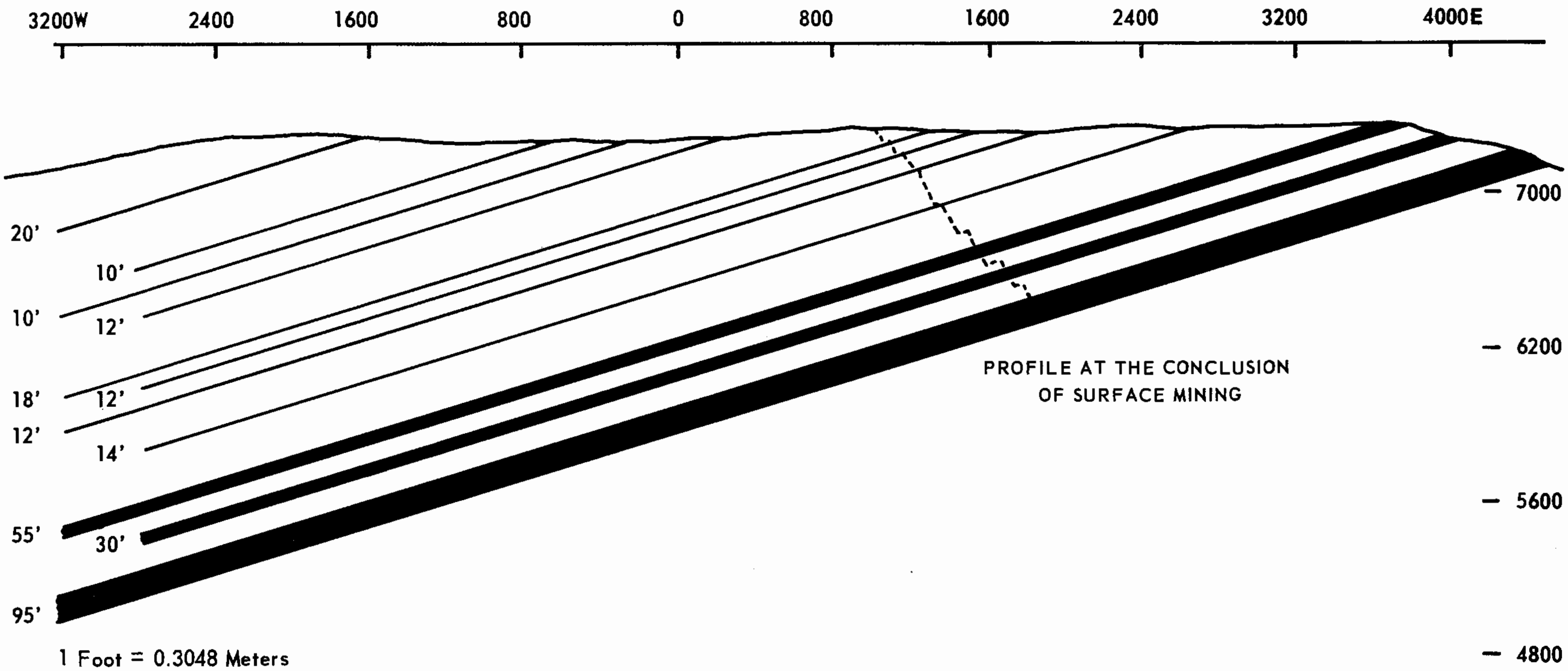


FIG. 9 TYPICAL CROSS SECTION OF A WYOMING COAL PROPERTY IN THE KEMMERER COALFIELD (Ferko, 1974)

has been toward delineating factors, such as geological conditions and seam characteristics, for the reserves recoverable by surface methods. Exploration geared toward deep mining has not been as complete. In time though, the deep-mineable coal reserves of Montana should become an important factor in western coal development.

Alberta and British Columbia

The successful mining of the coalfields at the northern extreme of the Rocky Mountains, specifically in the Canadian provinces of Alberta and British Columbia, will be very dependent upon the development of underground thick-seam mining methods. Therefore, it is of interest to review the current state of coal mining in these provinces (Figure 10).

A recent study (Heron, 1974) has placed the thick-seam coal reserves recoverable by underground methods in Alberta at 2.8 billion tons (2.5 billion metric tons). In this province, coal is found in three fields-the Plains, Foothills, and Mountains. The thickness of the seams in the Mountain and Foothill regions ranges up to 45 ft (14m). Frequently, the seams have been tectonically altered to even greater thicknesses. Coal seams in the Plains are at the lower end of the scale with an average thickness of 20 ft (6m). These coals are usually tabular. In the Mountains and Foothills, the coal seams dip from 0° to 90°, though the average dip is between 25° to 40°. These coals are generally of low and medium-volatile bituminous rank (Horachek, *et al.*, 1974).

The Mountain field is the only producing area in British Columbia. Five to nine different seams have been identified, varying in thickness from 10 to 50 ft (3 to 15m). The Balmer seam, a 45-ft (13.6m) deposit which pitches from 30° to 60°, is presently being mined by Kaiser Resources.

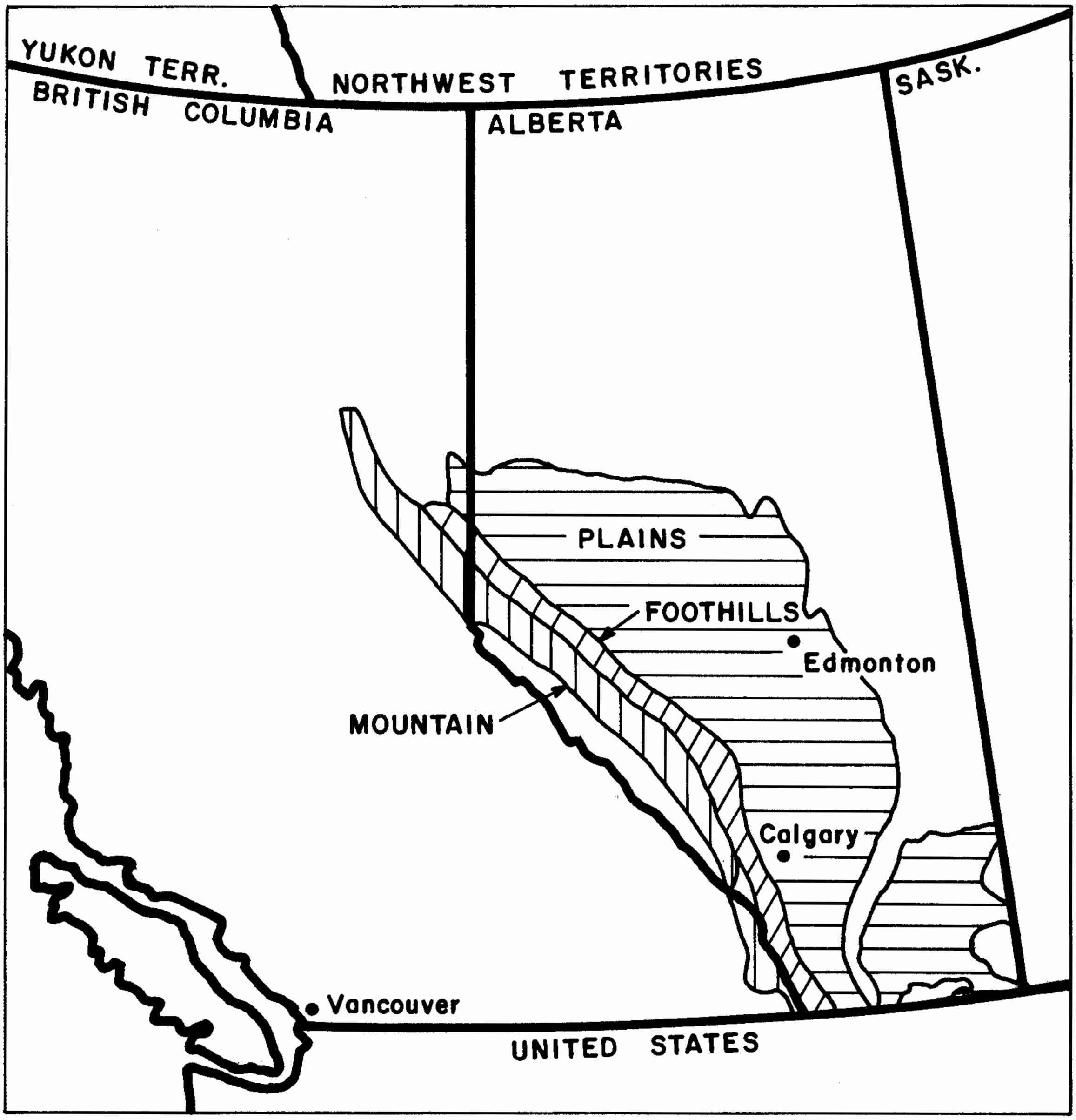


FIG. 10 ROCKY MOUNTAIN COALFIELDS OF CANADA

The coals in British Columbia are medium to low-volatile bituminous, with low sulfur content. Six billion tons (5.4 billion metric tons) of the province's measured reserves are classified in the thick-seam category (James, 1974).

Summary

In this chapter, detailed site-specific descriptions of geological and mining conditions were not done. In some instances, like Montana, there is little published information. This review, however, has been provided only to illustrate the magnitude of the western reserves, and generally identify the thicknesses and pitches that may be encountered. In the next chapter, the methods that are practiced worldwide for underground extraction of coal under similar conditions are reviewed.

III. THICK SEAM MINING METHODS

During the past decade, two major symposia have been held on the underground mining of thick coal seams. "The International Symposium on the Methods of Working Thick Coal Seams," held at the Indian School of Mines, Dhanbad, India, was a collection of papers reflecting the state of the art, in 1964, in coalfields worldwide. There were 33 papers describing the mining methods in seven countries. However,only one paper dealt with a U.S. operation. In September, 1966, "The Symposium on the Methods of Working Thick Coal Seams" was held in Bucharest under the sponsorship of the United Nations. The Symposium proceedings not only discussed methods practiced in theparticipating countries, but showed the impact of thick seam extraction on the annual output of each nation. No papers were presented from the United States. The countries that were most active in the deep mining of thick coal seams were France, Poland, Romania, Czechoslavakia, Yugoslavia, and the U.S.S.R.[1]

It appears that France, Romania, Japan, and the U.S.S.R. are the only countries that have exploited pitching seams to a great degree. Gaponovich, *et al.* (Cochrane, 1972) summarized the production and conditions under which thick coal was extracted a decade ago (Tables 3 and 4). Cochrane (1972) has stated that Yugoslavaia, Czechoslavakia, and Romania produce most of their thick coal from low-rank lignite deposits.

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[1] The "Second Conference on Mine Productivity", jointly sponsored by The University of Arizona, The Pennsylvania State University, and The University of Missouri-Rolla at Tucson, Arizona in May, 1975, dealt with the extraction of western thick fossil fuel deposits. Although much emphasis was directed toward surface mining, Smith (1975), Welzel (1975), and Yancik (1975) presented papers which were directed toward the impact and possible extraction of the thick, deep-mineable coal reserves in the West.

Table 3. Foreign Production from Underground Mining of Thick Seams - 1965 (after Cochrane, 1972)

Country	Minimum Seam Thickness (Ft)	Total Thick Coal Production (million tons)	Total Underground Production (million tons)	% of Country's Production - Thick Coal
U.S.S.R.	11.5	64	477	13
Poland	9.9	49	131	37
Japan	7.4	28	55	50
Czechoslavakia	9.9	24	51	47
Yugoslavia	9.9	21	24	87
France	13.2	9	60	13
Romania	9.9	7	10	69

1 ft = 0.3048m

1 short ton = 0.907 metric tons

Table 4. Foreign Production According to Dip of Thick Seams - 1965 (after Cochrane, 1972)

Country	Flat, Moderately Pitching 0 - 25° (million tons)	(%)	Inclined 26 - 45° (million tons)	(%)	Steep 46 - 90° (million tons)	(%)
U.S.S.R.	37	58	12	20	14	22
Poland	44	91	5	9	--	--
Japan	19	68	7	24	2	8
Czechoslavakia	24	100	--	--	--	--
Yugoslavia	18	84	1	5	2	11
France	5	50	3	38	1	12
Romania	5	67	--	--	2	33

1 short ton = 0.907 metric tons

Therefore, it can be concluded that only the U.S.S.R., Poland, France, and Japan mine, to any significant extent, thick-seam bituminous coal.

All of the thick-seam underground mining methods can be classified into three major categories: full-face, slicing, and caving (Cochrane, 1972). There are numerous variations within each classification since individual applications are dictated by such factors as the thickness and dip of the coal, the condition of the roof and floor, the level of mechanization, the availability of labor, the proximity of other coal seams, the availability of packing materials and the desirability of packing (Table 5).

Full-Face Systems

Full-face mining is practiced primarily in the United States, Canada, and Europe. The coal face is driven the full seam thickness and, often, in one machine pass. At the present time, full-face mining in the United States is limited to room-and-pillar applications in seams that are basically tabular. In some cases, only a part of the total thickness is recovered. The equipment used in these applications incorporate some modifications in existing equipment to extend their range. For example, for use in an 18-ft(5.5m) tabular seam in Virginia, conventional equipment has been modified to increase its range ("A Mine of Tomorrow",1957). Continuous miners have also been used in U.S. thick seams in a method referred to as benching. In this method, a tabular thick seam, less than 20 ft (6m) in thickness, is extracted by a two-pass room and pillar procedure. The upper part of the seam is extracted on development while the lower part, as well as any mineable coal pillar, is taken on retreat by ramping down into the lower part (Huntsman, 1974).

Table 5. Classification of Thick-Seam Mining Methods

Methods	Variations	Operational Dip Constraint	Thickness Constraint	Ground Control	Haulage Constr.	Practicing Countries
Full Face	Longwall	0° to 15°	<16'	Caving Stowing	Conveyors	Germany Poland
	Conventional	Depends on Haulage	<20'	Caving Stowing	Shuttle Cars<12° Conveyors <15°	U.S.A. India
Slicing	Longwall	0° to 15°	10'/pass (max. 40')	Caving Stowing	Chain Conveyors	France Poland
	Continuous	Depends on Haulage	10'/pass (max.20')	Caving	Shuttle Cars<12° Conveyors <15°	U.S.A.
	Conventional	Depends on Haulage	10'/pass (max.20')	Caving Stowing	Chain Conveyors	India
Caving	Longwall	0° to 15°	<40'	Caving	Chain Conveyors	France
	Conventional	Dip is used	<20'	Caving	None	U.S.S.R.
	Hydraulic	>2.5°	Penetration up to 60'	Caving	Fluming >2.5°	Canada U.S.S.R.

1 ft = 0.3048 m

Application of the traditional continuous and conventional mining equipment is limited by the thickness and pitch of the seams. More than two or three benches, or operation in a pitch greater than 20°, may not be economically feasible. As such, this type of thick-seam mining can only be projected for rather specialized conditions.

In Europe, the trend in full-face mining of thick coal seams is toward the use of longwall equipment (Cochrane, 1972). In Germany, 13-ft (3.9m) seams have been mined by the full-face longwall method (Welzel, 1975). Single-pass extraction of thicknesses greater than this are within equipment capabilities. For example, the recent development of a double-drum shearer that extracts 16 ft (4.8m) in one pass is a major advance (Barnard, 1976). In Poland, a height of 13 ft (3.9m) has been taken in an area where the roof is non-caving by combining mechanized cutting and loading with simultaneous hydraulic backfilling (Figure 11) (Cochrane, 1972).

As with room-and-pillar methods, the difficulties of full-face longwalling become apparent as the dip of the seam increases over 20°. Though there are operations in pitches greater than 20°, longwall mechanization is in the experimental stages, particularly with regard to roof control (Cochrane, 1972).

Slicing Systems

A common method of mining thick seams is by slicing. The extraction of two or more slices, in an ascending or descending order, is accomplished primarily by longwalling each slice (Figure 12). When the seam is tabular, the slices are taken parallel to the floor. When the seam is pitching, the slices are taken horizontally along the strike.

In the ascending method of slicing, the floor of the first lift

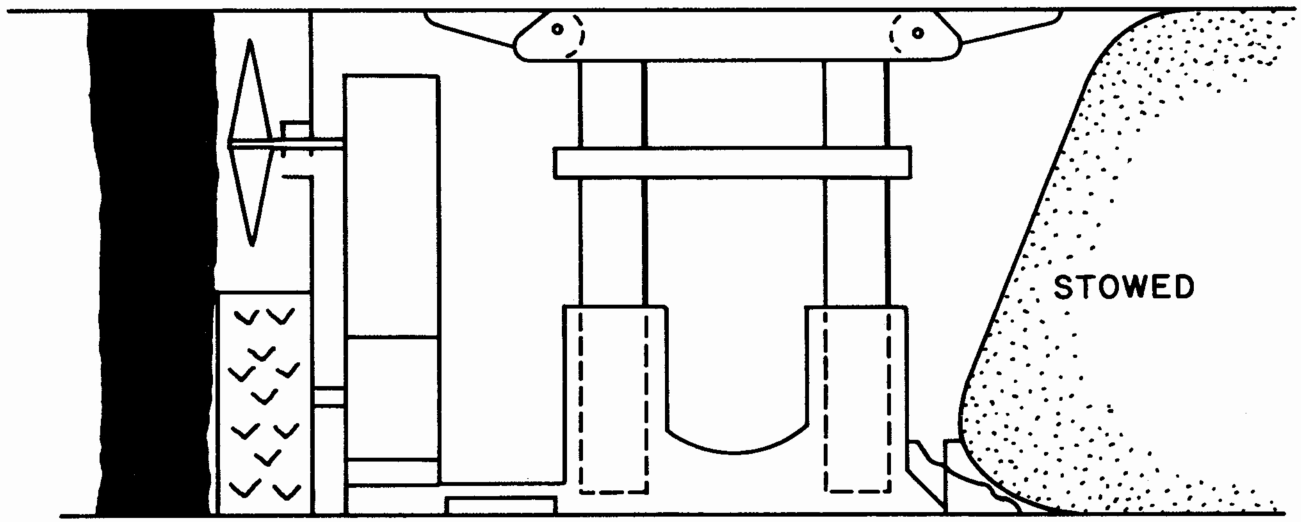
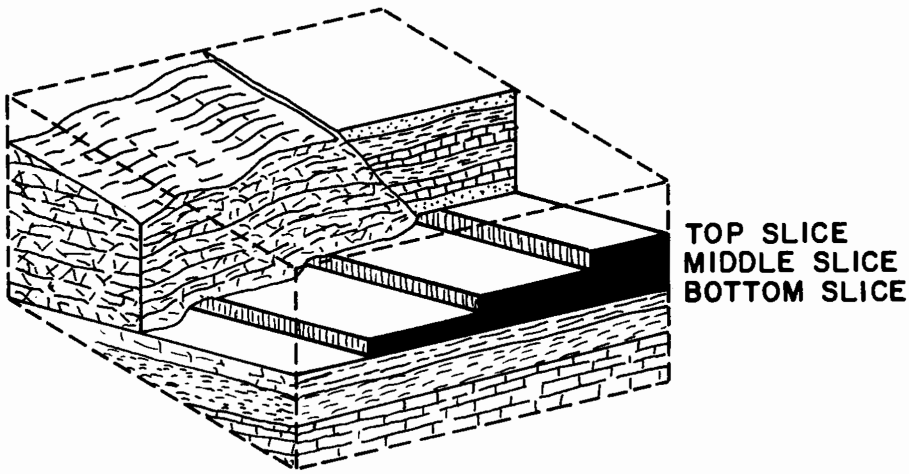
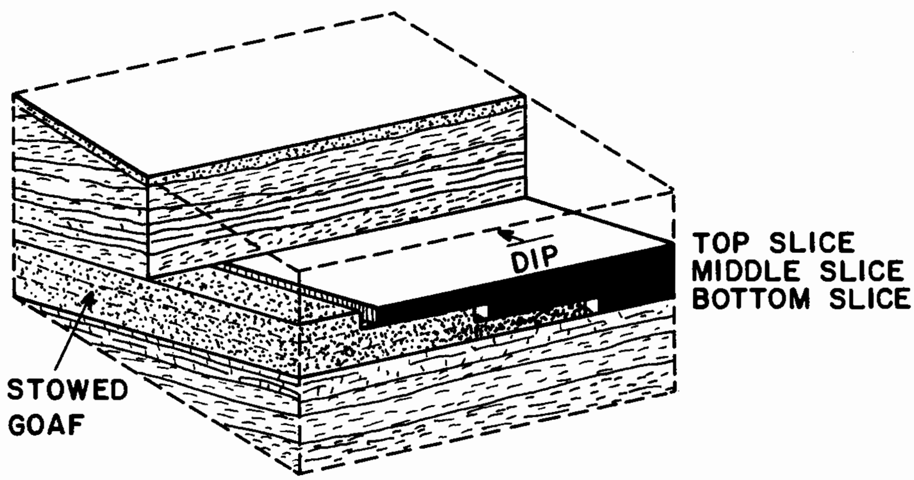


FIG. 11 FULL FACE MINING WITH SIMULTANEOUS BACKFULLING
(Cochrane, 1972)



a) Descending with Caving



b) Ascending with Stowing

FIG. 12 SLICING SYSTEMS
(Vorobjev and Deshmukh, 1966)

is the seam floor and the roof is the remaining coal. The difficulty in working with this method is that, for subsequent slices, hydraulic fill becomes the floor. The stowing operation can create a floor that is uneven and, therefore, difficult to work upon. If there are several slices to be taken, the coal in the top slices may actually be detached from the roof, giving rise to hazardous conditions.

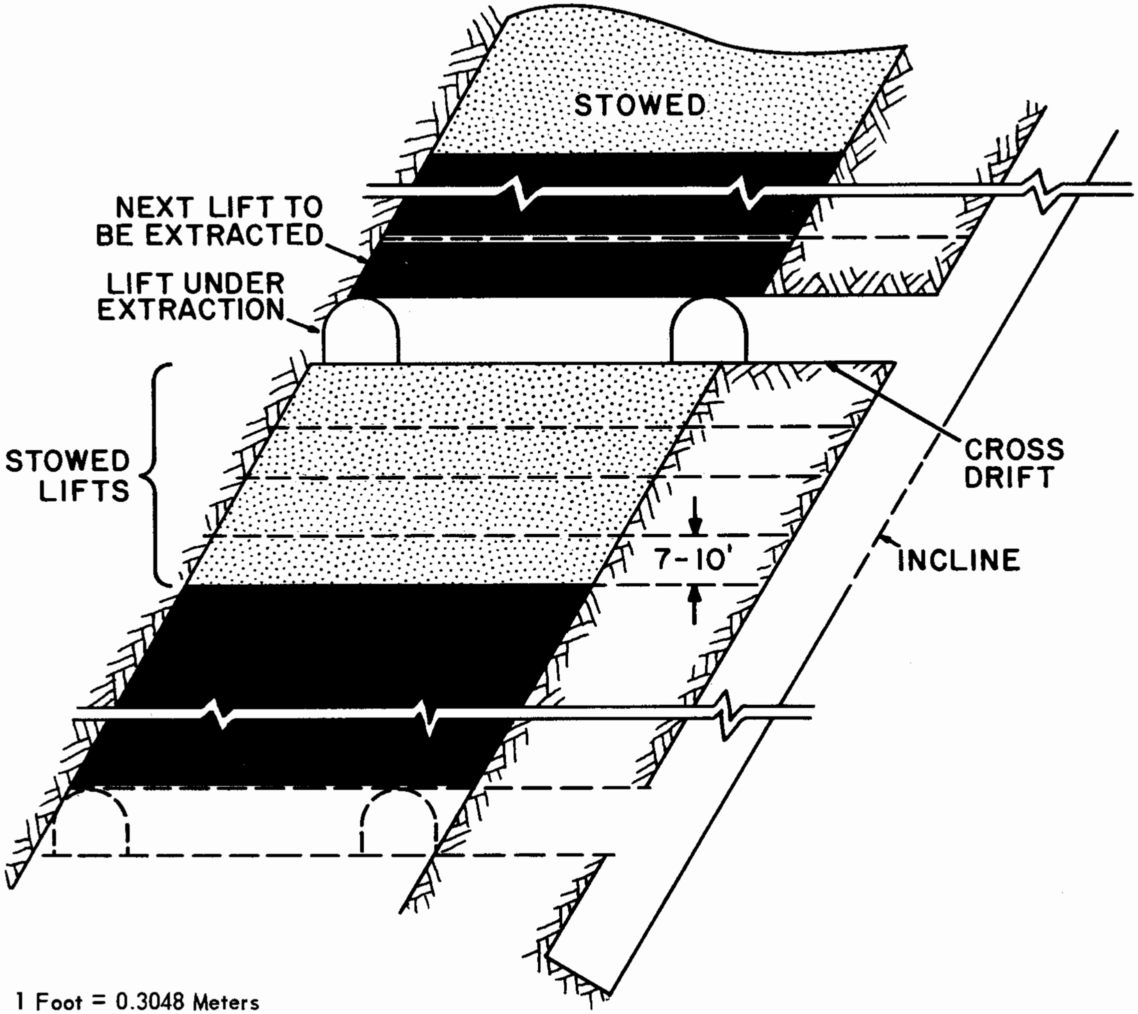
Horizontal slicing by ascending lifts in a pitching seam is similar to the hardrock method of cut-and-fill mining (Figure 13). Basically, the development drifts are driven in the footwall and cut into the coal seam at prescribed intervals (Vorobjev and Deshmukh, 1966). As each horizontal lift is completed, the mined-out area is stowed.

Due to the high development and stowing costs, the ascending order of slicing is used primarily where massive non-caving roofs are found, such as in the Upper Silesian Basin of Poland, or where surface damage must be minimized.

In the descending method of slicing, solid coal forms the floor for the first slice, and for each subsequent slice except for the last. As the first slice is taken, a flexible mat is placed on the floor. This mat, with the broken top over it, becomes the artificial roof for the lower slices (Figure 14). In variations of this method, two or more slices are taken simultaneously (Callier, 1972). The method is used where the roof caves easily. Unlike the ascending order method, this method is usually limited to tabular seams.

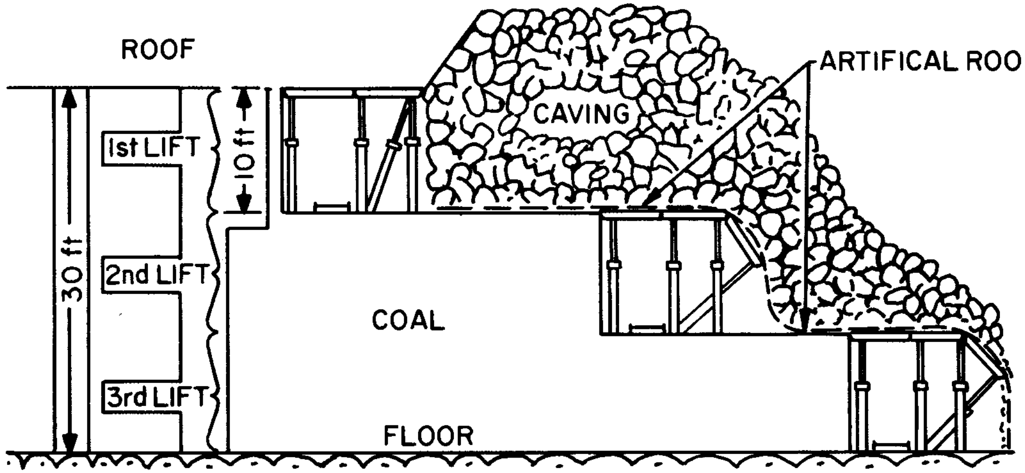
Caving Systems

Caving systems are applicable where the seams are very thick and irregular. Unlike the full-face and slicing methods, caving methods are



1 Foot = 0.3048 Meters

FIG. 13 HORIZONTAL SLICING IN A PITCHING SEAM
(Vorobjev and Deshmukh, 1966)



1 Foot = 0.3048 Meters

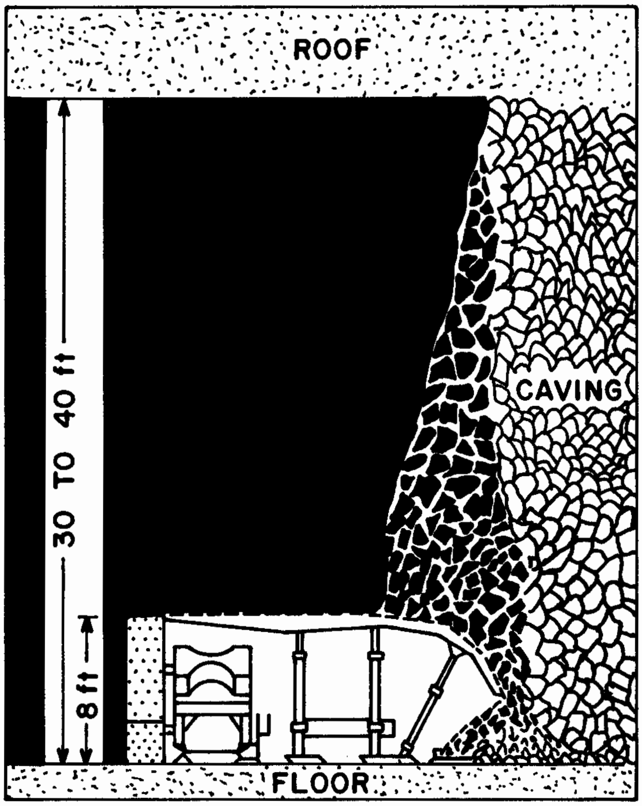
FIG. 14 DESCENDING METHOD OF SLICING WITH CAVING
(Callier, 1972)

practiced in thick seams at any pitch. Where seams are tabular to gently pitching, the integrated caving method is applicable (Çallier, 1972). In this system, one slice is taken at the bottom of the seam and the rest of the coal is caved into the excavated cut (Figure 15). In moderately thick seams that are steeply pitching, a sublevel caving method, similar to that used in hardrock mining, has been successful. Where seams are very thick and pitching, they are extracted in horizontal slices in descending order with caving.

The face equipment for the integrated caving method consists of a shearer, two chain conveyors, powered roof supports, and non-recoverable wire mesh. When the shearer cuts the bottom slice, a 2.0 in. x 2.0 in. (50.8m x 50.8m) wire net is placed between the coal roof and the chocks. As the face advances, the coal roof caves onto the mat. The front conveyor transports the sheared coal while the caved coal is directed to the gobside conveyor by conveniently cutting the mesh for drawing the coal. To maximize recovery and to limit dilution, proper drawing of the caved coal is most important. At the same time, it is also the most difficult operation (Callier, 1972).

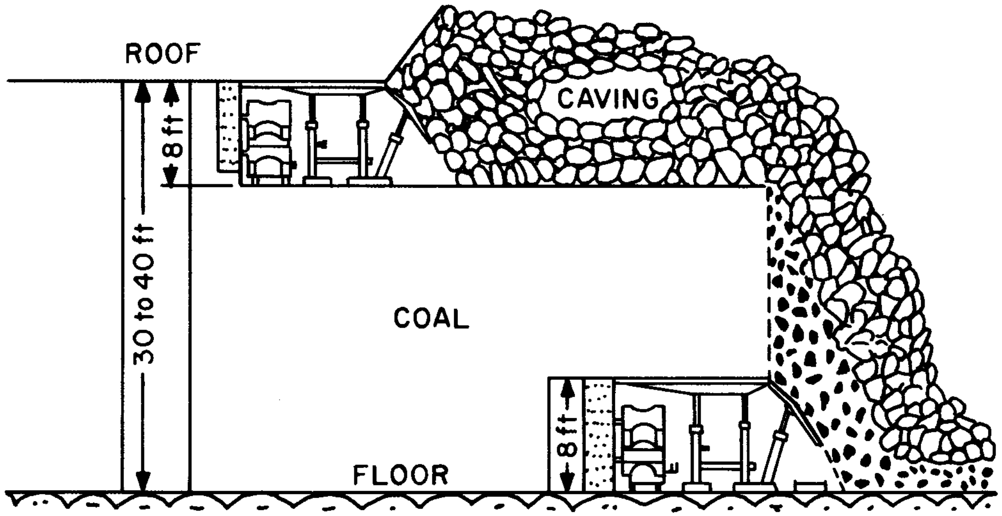
Where the coal is hard, sometimes two slices are taken in the integrated caving method, a slice at the top and one at the bottom (Figure 16). The coal in between the two slices is destressed and caved (Callier, 1972).

Sublevel caving is practiced in steep seams with strong country rock. A method commonly practiced in the Kuzbass region of the Soviet Union, where the seams dip nearly vertically, is known as the moving steel support method (Figure 17). In this method, each sublevel is supported at the face area by a steel frame, which overhangs into the gob.



1 Foot = 0.3048 Meters

FIG. 15 INTEGRATED CAVING METHOD
(Callier, 1972)



1 Foot = 0.3048 Meters

FIG. 16 DESTRESSED CAVING METHOD
(Callier, 1972)

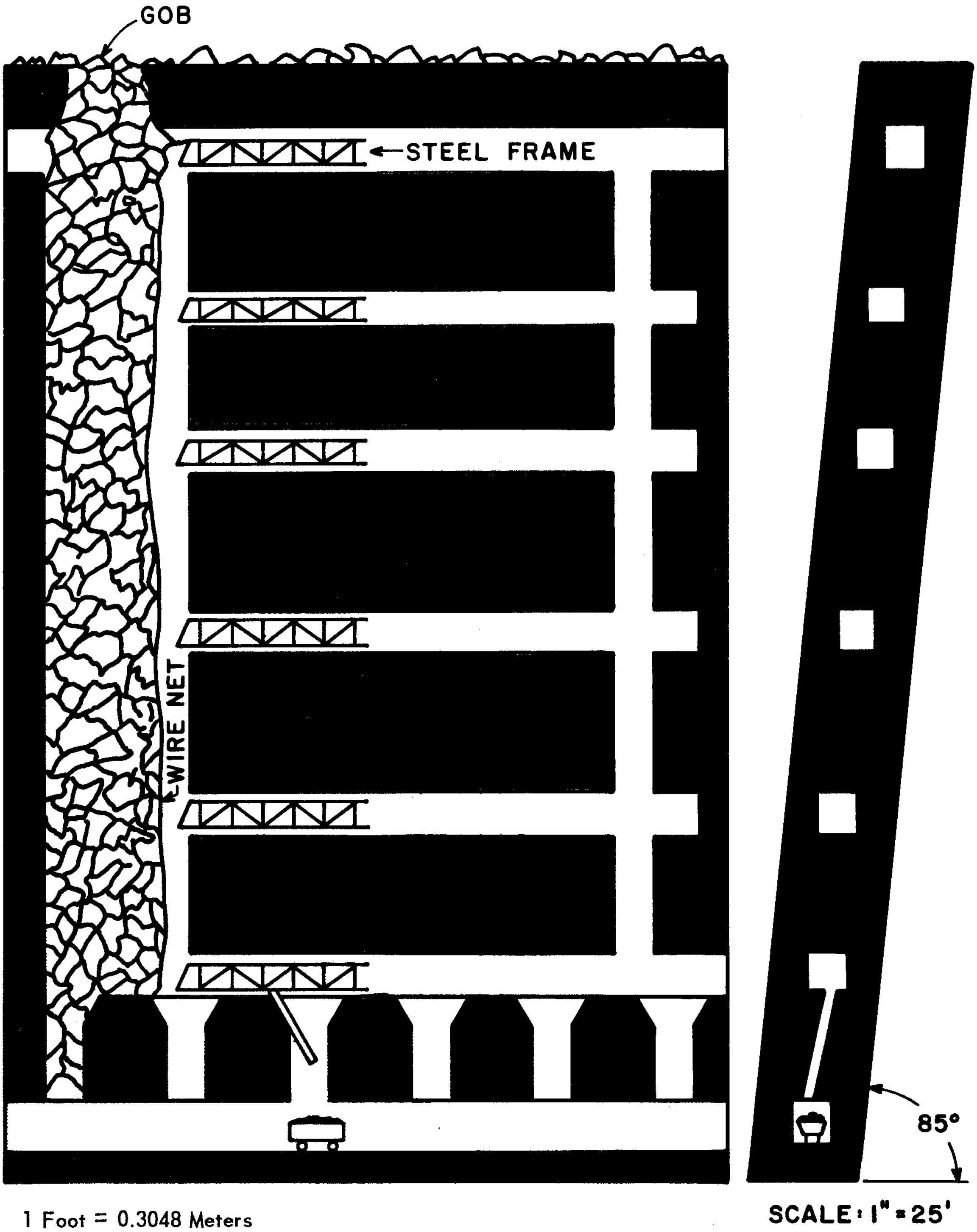


FIG. 17 SUBLEVEL CAVING
(Vorobjev and Deshmukh, 1966)

Wire netting is attached to the top frame and is draped between the production areas and the gob to limit dilution. The coal, above and below the sublevels, is either blasted or hydromechanically cut. The broken coal gravitates to the haulage level and, after the cycle is completed in all the sublevels, the steel frames are pulled back (Vorobjev and Deshmukh, 1966).

A horizontal sublevel caving method is used in France for pitching seams where the horizontal width is greater than 40 ft (12m). The panel consists of two inclined entries, one of which is located along the hangingwall for fresh air and coal transportation, while the other is located along the footwall and is used for return air. At each level, a breakthrough is driven between the two entries. From this point, two airways are driven in each direction on strike, one adjacent to the footwall and the other, adjacent to the hangingwall, for return and fresh air (Figure 18). At the end of the property or panel, these airways are connected by crosscuts usually 8 ft (2.4m) high, under 25 ft (7.5m) of roof.

Thus, two caving longwall faces are established in each horizon. These faces are retreated back from the boundary to the inclined drift (Coates, *et al.*, 1972).

Summary

Full-face operations, either room and pillar or longwall, are most favorable for extraction of tabular seams in thicknesses less than 20 ft (6m). An increase in both thickness and dip, however will severely affect coal recovery and equipment efficiency. In fact, in seams which pitch at an angle greater than 15°, continuous miners should be confined to operations in the strike direction.

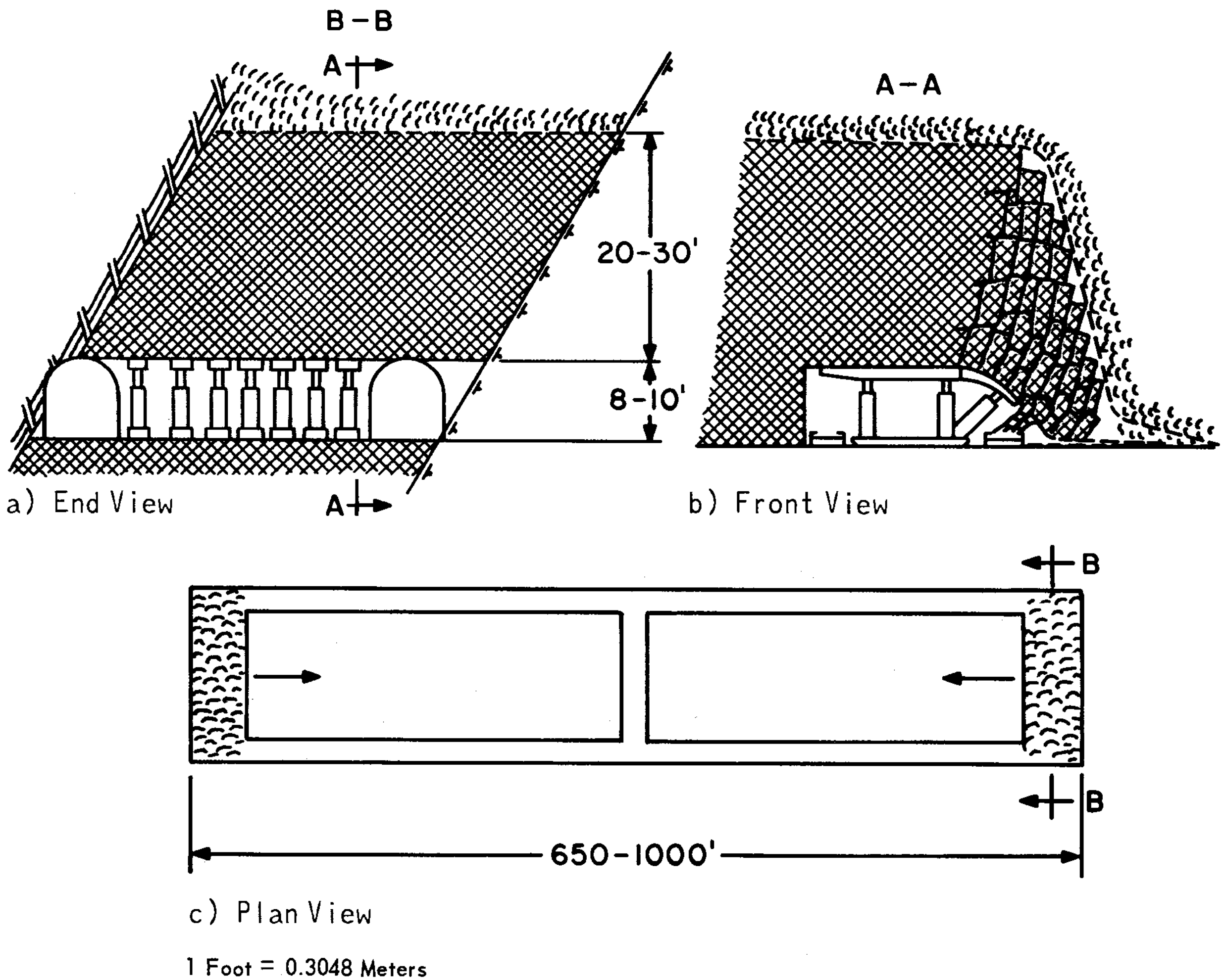


FIG. 18 HORIZONTAL SUBLEVEL CAVING
(Coates, et al., 1972)

Slicing methods have greater coal recovery though, here again, the dip of the seam will affect equipment application much in the same manner as in the full-face method. Slicing with caving is common abroad. In the United States, the economics of steel matting and the associated manpower may make the method uneconomical. For example, the cost for the steel mesh, alone, is approximately $\$0.15/ft^2$ ($\$1.62/m^2$) for a U.S. application (Heers, 1975). Although extraction with stowing has great benefits, the availability of stowing material and trained personnel and the cost of stowing are factors that require careful evaluation.

Caving methods, though widely practiced abroad, are low in productivity by U.S. standards. For example, the Rozelay Mine in France, which is often referred to in literature on the subject, achieves less than nine gross tons (8.2 metric tons) per manshift (Barron, 1974).

In the United States, the objective of economic performance is vital for the selection of any system. Deep-mined thick coal must be competitive in the energy market because of the free enterprise system. As such, many of the methods practiced abroad cannot be readily transferred to U.S. conditions because of factors such as economic climate, national policies, and the availability of manpower and material.

This chapter has generally reviewed the applicability and limitations of thick-seam mining. In the next chapter, the methods that are practiced in selected North American mines are described.

IV. OPERATIONS IN NORTH AMERICA

Published information on thick-seam underground mining is rather scant. Oftentimes, it is difficult to visualize the conditions encountered from the descriptions. Therefore, one aspect of this study was to visit thick-seam operations in the United States and Canada and gain an understanding of the applications of equipment and methods. In all, nine mines were visited. Although a few of these mines could not be considered thick-seam operations, due to either a thickness less than 12 ft (3.6m) or the absence of a full-seam recovery method, some of their operational features were considered worthy of evaluation for possible adaptation to thick-seam mining.

In the following, a description of the geological conditions, mining methods, and equipment is provided to gain an insight into the state of the art of underground thick-seam mining in North America. This description is also necessary to understand the basis for projecting the types of problems that may be encountered in the future, and for circumventing these problems in the methods that are recommended.

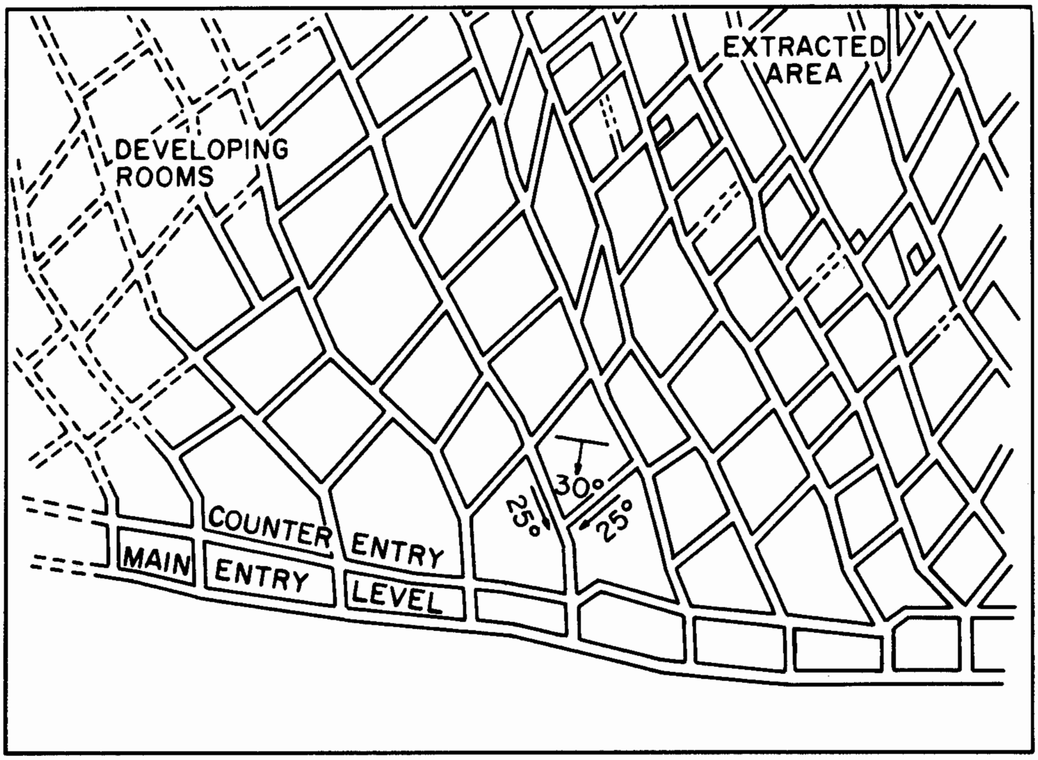
Vicary Creek Mine

The Vicary Creek Mine, operated by Coleman Collieries, Limited, is located in the southwestern corner of Alberta near the town of Coleman. Established in the 1950's, Coleman embarked on a mechanization program for its mines in 1966 (Chamberlin, 1972). The changeover from hand mining to mechanization was completed in 1971.

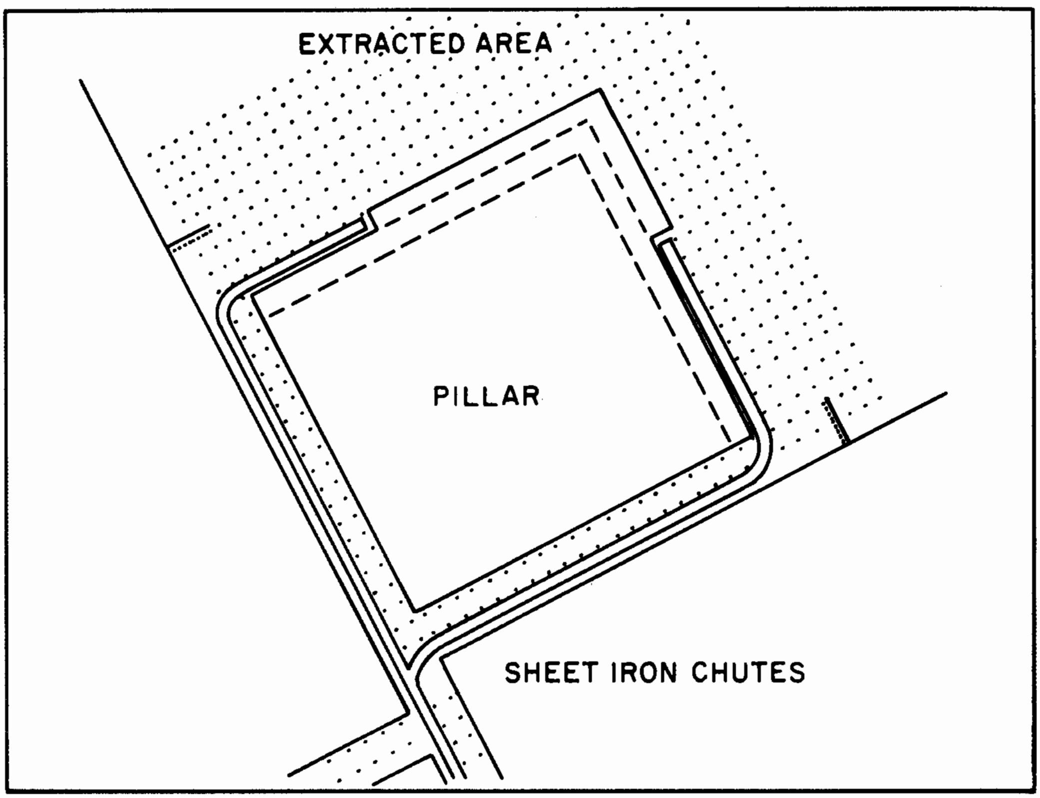
The Vicary Creek seam dips up to 40°, the most common pitch being in the range of 15° to 30°. The thickness of the seam in the area that is presently being mined is 20 ft (6m). There are areas where the thickness encountered ranges up to 40 ft (12m). The seam is overlain by laminated sandstone with the overburden thickness ranging up to approximately 400 ft (121m). The floor is a hard shale.

Prior to 1966, two incline drifts were driven in each panel at a very small angle to the strike direction. These drifts were connected by crosscuts driven at full pitch (Figure 19a). Rooms, driven as continuations of the crosscuts, were advanced 25° to the left of full rise and were connected by breakthroughs, at a predetermined distance, driven 25° to the right of full rise. The rooms and crosscuts were advanced by pairs of miners using hand-held air-operated picks. As the miners worked updip, they supported the roof along the highside ribs with timbers or, where necessary, with three-piece timber sets. The men would then lay a sheet iron chute in the rooms and crosscuts to facilitate gravity flow of the mined coal to the haulage road for loading into mine cars. After development, the pillars in the rooms were extracted in a similar manner (Figure 19b). Due to the difficulty of pitch haulage, a large labor force was needed to haul supplies to the sections. Therefore, production per manshift was low and averaged only between six and eight tons (5.4 and 7.3 metric tons)(Chamberlain, 1972).

To increase both production and recovery, Coleman Collieries began experimenting with hydraulic jet cutting of coal at Vicary Creek in 1962 ("Hydraulic Pitch...," 1964). The method consisted of drilling a 4.75 in.-diameter (120mm diameter) hole, parallel to the seam dip at a equal distance between the hangingwall and the footwall, from one drift to a lower drift (Figure 20) ("Revolutionary Coal...," 1964). The drifts were driven parallel to the strike, and were up to 600 ft (182m) apart.

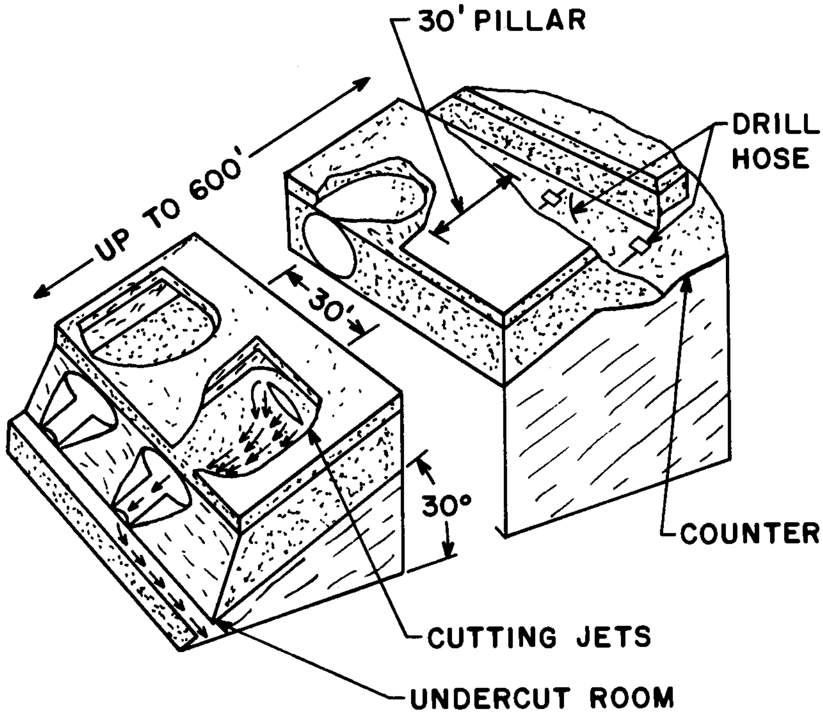


a) Panel Layout



b) Pillaring Plan

FIG. 19 HAND MINING, VICARY CREEK MINE
(Chamberlin, 1972)



1 Foot = 0.3048 Meters

FIG. 20 HYDRAULIC JET MINING, VICARY CREEK MINE
(""Hydraulic Pitch ...,'' 1964)

After the hole intersected the lower drift, the drill bit was replaced with a special nozzle which, when rotated, was designed to emit a high-pressure water jet normal to the hole. When the nozzle was retracted, the rotating jet would enlarge the hole to 20 ft (6m) in diameter. The cut coal dropped to the lower drift where it was flumed to a screening room for sizing. The drill holes were spaced on approximately 30-ft (9m) centers.

The longhole hydraulic jet cutting concept, the first application of its type in a North American coal mine, was eventually abandoned because of the many problems encountered. First, the soft coal tended to collapse around the mined-out holes. Also, the coal did not always flow properly and, in time, clogged up the holes. Finally, there were too many faults and undulations in the coal seam which affected the centering of the longhole. Although this experiment was not successful at Vicary Creek, under suitable conditions this technique has excellent potential for the safe recovery of thick and steep seams.

The present method of extraction at Vicary Creek Mine is the mechanized mining method, introduced in 1966 (Chamberlain, 1972). Essentially, a panel is developed by two drifts, a supply-intake and a belt-return, which are driven along the strike (Figure 21). The drifts are separated by 60-ft-wide (18m-wide) pillars and are connected by crosscuts driven every 180 ft (55m). Since the crosscuts have to be driven against the dip, a 12° gradient is maintained for favorable equipment operation (Chamberlain, 1972).

On advance, a Joy[1] 6CM ripper continuous miner begins the cut

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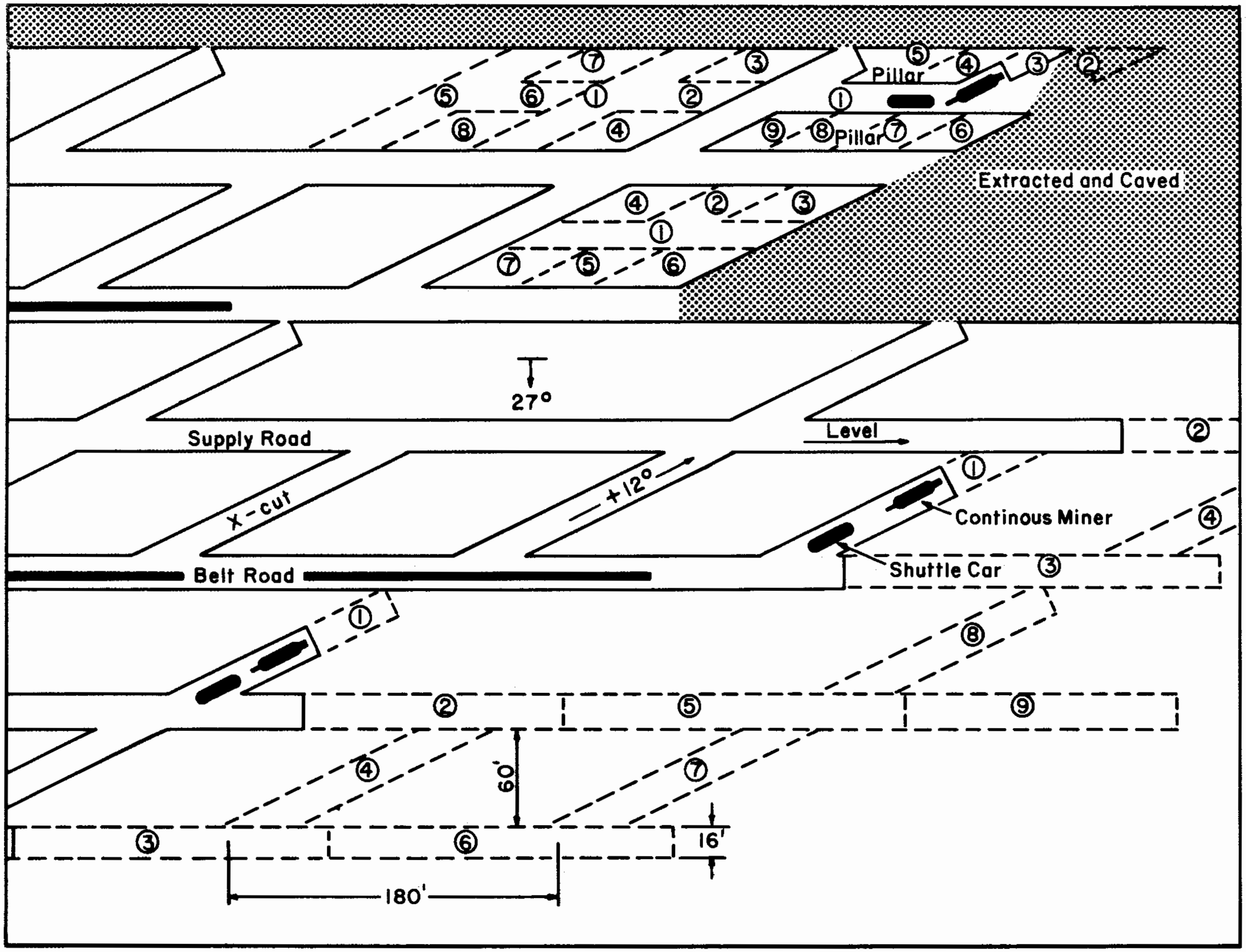
[1] Reference to the manufacturer as used in the text are for identification purposes only and do not imply endorsement by the author.

sequence by completing the inby crosscut from the belt road to the supply road (Figure 21). The supply road is then advanced 60 ft (18m) beyond the crosscut interval. The miner then backs into the belt road to drive the entry up and connect it with the supply road (Chamberlain, 1972). The drifts are driven 16 ft (4.8m) wide and the height, due to the pitch, can vary from 18 ft (5.5m) on the highside rib to 4 ft (1.2m) on the lowside rib (Figure 22).

Since only one Joy 10SC shuttle car is used for face haulage, all of the face operations are performed between the loading operations. After the entries are advanced five ft (1.5m) [four 8-ton (7.3 metric ton) shuttle car loads], a row of four 6 ft (1.8m) roof bolts are installed by four crew members with Gardner-Denver RB83 stopers. Additionally, between shuttle car loads, supports (timber and lagging) are installed along the highside rib, and the air and water lines are extended. Since auxiliary fans are used for face ventilation, tubing is also extended (Chamberlain, 1972).

On retreat, the miners extract the pillars to the rise (Figure 21). The row of pillars between the drifts is initially split in half by either a lift taken parallel to the drifts or to the crosscuts. Then, each pillar split is reduced according to one of three plans most suited to local conditions, as shown in Figure 21. Similarly, the barrier pillar between the active panel and the up pitch panel, which has already been mined, is also extracted (Chamberlain, 1972).

The typical face crew consists of seven men. For both advance and retreat mining, one section supervisor, two equipment operators (to operate the miner and stopers), two facemen (to assist with timbering, etc.), one shuttle car operator and one mechanic are employed. At



1 Foot = 0.3048 Meters

FIG. 21 MECHANIZED MINING PANEL LAYOUT, VICARY CREEK MINE
(Chamberlin, 1972)

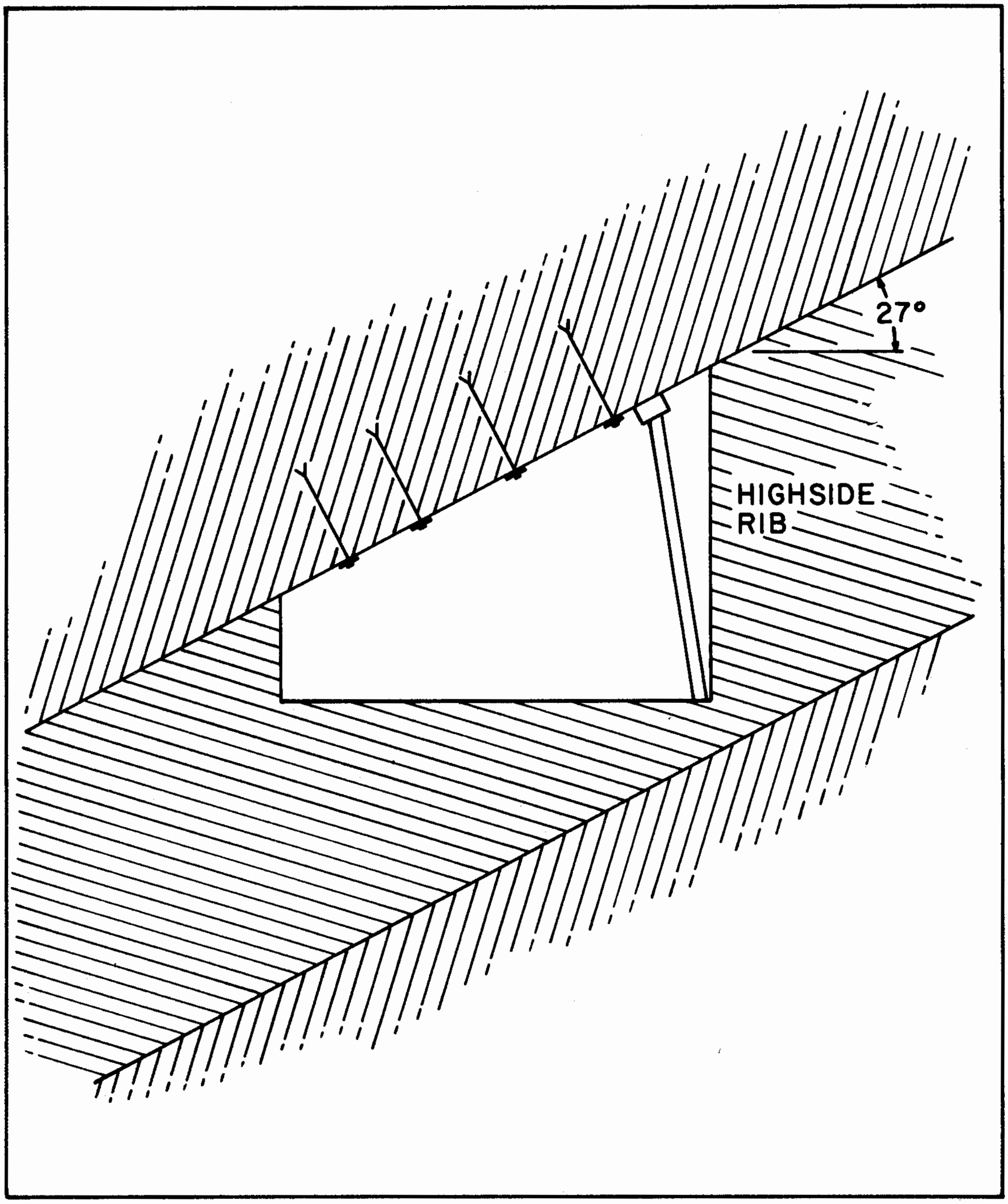


FIG. 22 STRIKE ENTRY SHOWING THE VARIABILITY OF RIB HEIGHT, VICARY CREEK MINE
(Chamberlin, 1972)

the time of the mine visit, the mine production was 250 tons (227 metric tons) per shift.

Although the mine is designed for mechanized mining, most of the tonnage, at the time of the mine visit, came from the previously abandoned method utilizing hand-held air picks. This situation resulted because all three ripper miners in use were rendered inoperative due to problems caused by a roof fall, and by the exceedingly high pitch in one area.

In general, strata control problems in the mine are severe. Entries driven along the strike have to contend with a highside rib of 18 ft (5.5m), due to the seam pitch. Since the coal is soft, extensive support of this rib is required. It is costly not only in terms of material but because of the ensuing production delays. Floor heave is also quite common. For example, floor heave in one area of the mine reduced a 16-ft-high (4.8m high) entry to approximately 5 ft (1.5m). In all, the mining conditions at the Vicary Creek Mine are quite difficult.

Michel Colliery

Kaiser Resources' Michel Colliery, located near Sparwood, British Columbia, consists of two operations: the Balmer North Mine and the Hydraulic Mine. Both of the mines produce metallurgical grade coal from the 45-ft(13.6m) Balmer seam. The seam is synclinal with Balmer North located on the northern side and the Hydraulic Mine located on the southern side of the fold. The dip of the seam, in both areas, is 35°. The overburden thickness averages up to about 400 ft (121m). The coal is overlain by a competent laminated shale roof, and is underlain by a fireclay. Development in both mines is either within or at the top of the seam to avoid degradation of the bottom by water. There are a few friable partings in the seam which, because of their relative softness, aid in the hydraulic mining process. The coal, itself, is somewhat soft.

The Balmer North Mine has been in operation since 1900 (Parkes and Grimley, 1975). Although the present operation is fully mechanized with continuous miners, shuttle cars, conveyor belts, etc., originally a sublevel caving system was practiced. The former method consisted of driving a sublevel at the footwall, then blasting the top coal on retreat with longholes. The broken coal was then loaded out with duckbill loaders; at a later time, continuous miners replaced the loaders.

Balmer North presently produces between 1800 and 2500 tons (1633 and 2268 metric tons) per day from four machine shifts. Two miners, a Lee-Norse narrow head milling-type miner and a Joy 6CM ripper-type miner, account for the production. Behind the miners in each section is a Joy shuttle car for face haulage, while the outby haulage is provided by 36-in.(914mm) conveyor belts. Roof, in all instances, is secured by 7-ft (2.1m) bolts on 4-ft (1.2m) centers. The total mine work force is 70 men, with 7 men per production crew.

The mine workings are entered by a horizontal drift from the surface. The drift was driven through rock until the pitching seam was intersected. In the panels, the crosscuts are driven up the dip at 60° so that the pitch never exceeds 12°.

In each panel, three 16-ft (4.8m) entries are driven on 75-ft (23m) centers in the top 10 ft (3m) of the seam (Figure 23). The belt conveyor is located in the center entry, from which the 60° crosscuts

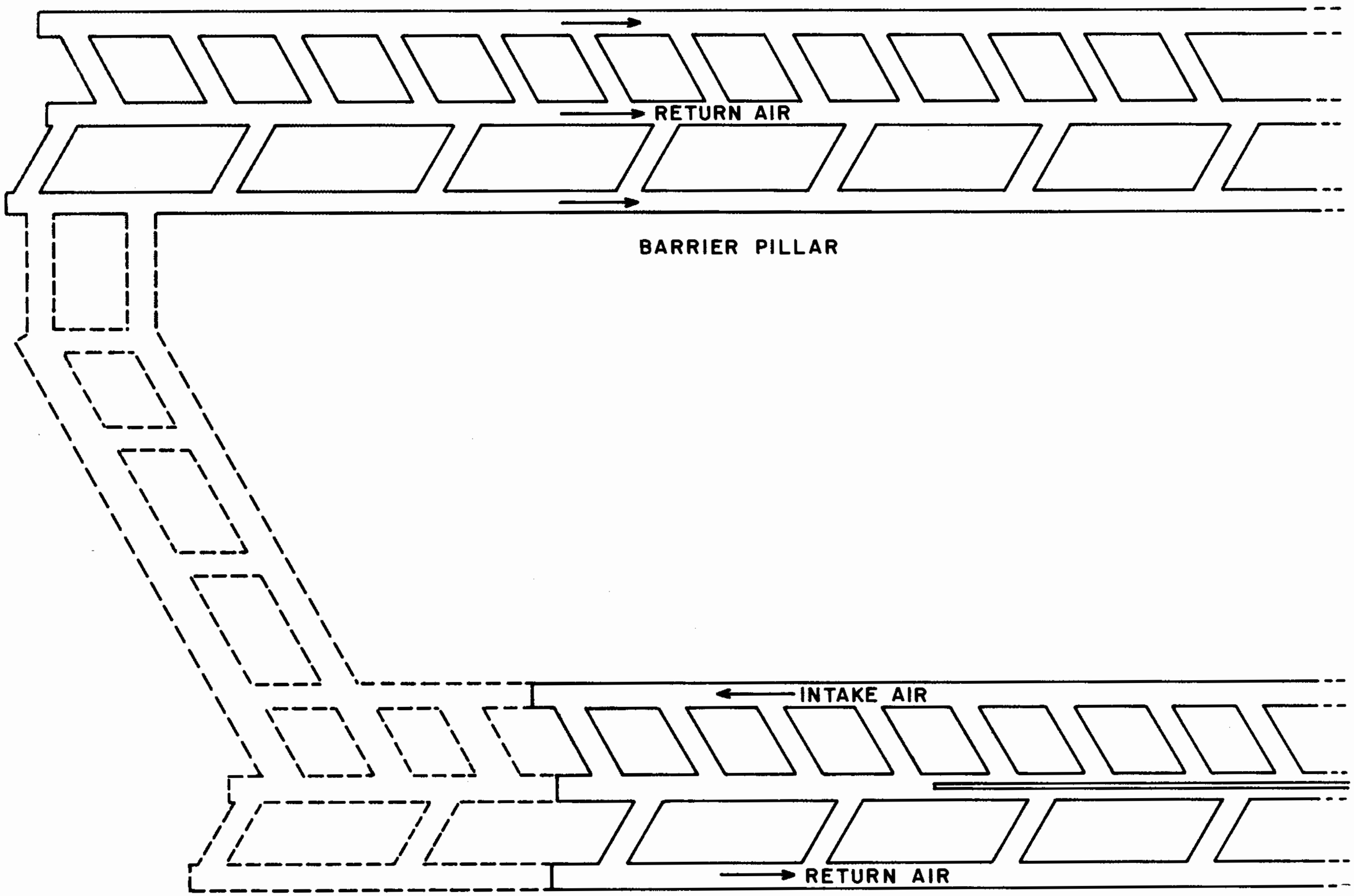


FIG. 23 PANEL LAYOUT DURING DEVELOPMENT, BALMER NORTH MINE

are driven. At the top end of the panel, a set of bleeder entries are driven up the pitch for a distance of approximately 450 ft (136m) until the previously mined panel is intersected. Then, the panel return is changed to an intake and the retreat procedure is started.

Retreat mining at Balmer North merely involves the developing and deepening of rooms on the up pitch side and deepening of the panel entries and crosscuts (Figure 24). Pillars are not extracted. To begin the mining in the rooms, the panel crosscuts on only the right side are extended 400 ft (121m) with 16-ft-wide (4.8m wide) roadways driven 10 ft (3m) high at the top of the seam (Figure 25, Step 1). Four breakthroughs are usually driven on 100-ft (30m) centers from the room toward the previously mined (inby) room. When this is completed, the miner drops back down the room for 100 ft (30m) and, after center posts are set, widens the last 100 ft (30m) of the room to 30 ft (9m) (Figure 25, Step 2). After widening that portion of the room, the miner then drops back another 50 ft (15m) from the face and ramps down into the floor coal on the left side of the room (Figure 25, Step 3). Each successive pass is taken to a width which is less than the previous pass for rib control. The miner then swings 45° to the right for the remaining coal on the second bench (Figure 25, Step 4) and ramps down for the third lift (Figure 26, Step 5). The remaining coal on the third lift is extracted by swinging the miner 45° to the right once more (Figure 26, Step 6). Before dropping back out of the room, a small amount of the coal can be recovered by repeating the ramping process (Figure 27, Steps 7,8,9). Figure 27 shows a front view of the total bottom coal recovered between two breakthroughs in a room (Figure 27, Section 3-3). As is readily apparent, only one-third of the total

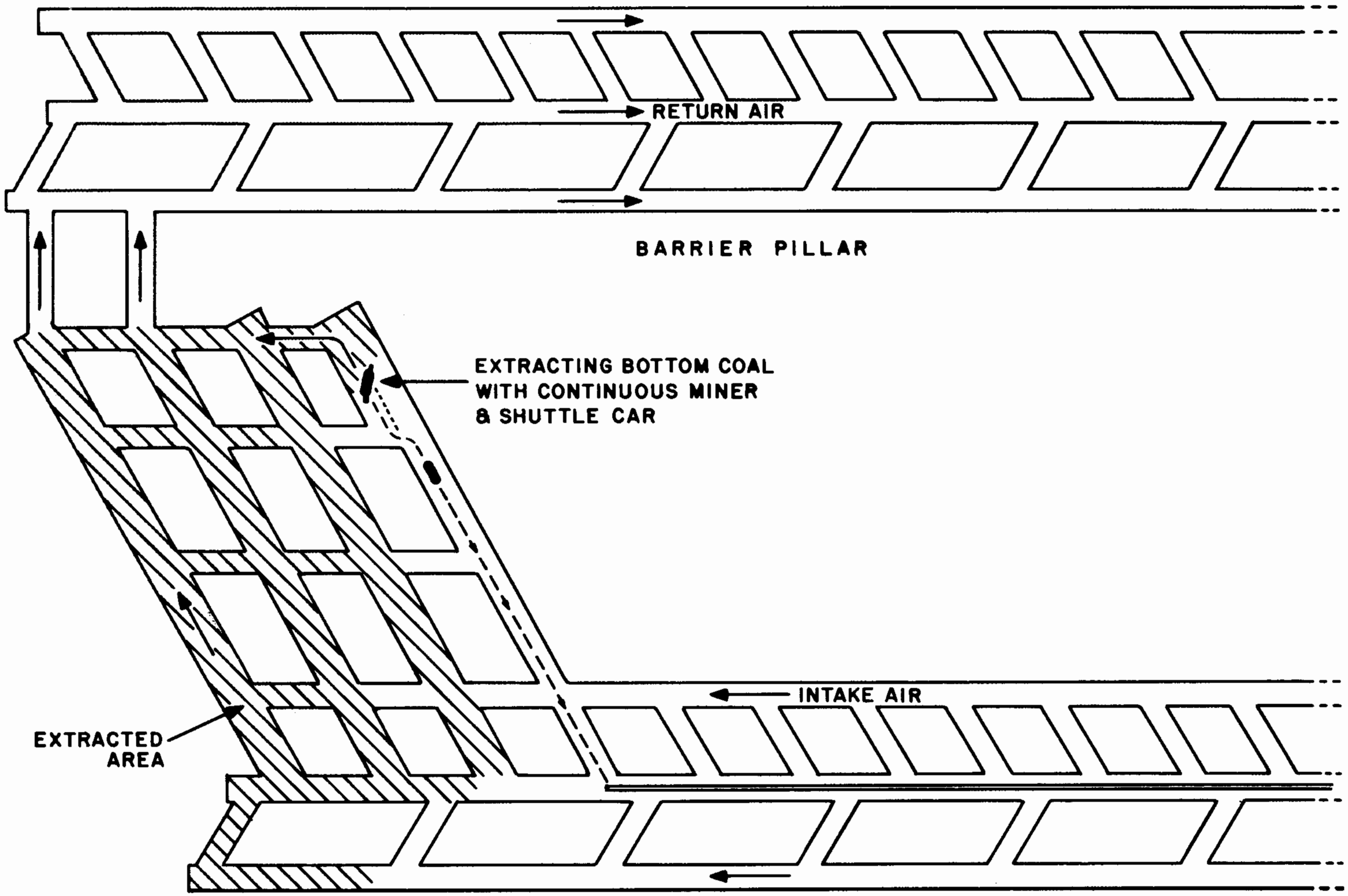


FIG. 24 PANEL LAYOUT DURING RETREAT, BALMER NORTH MINE

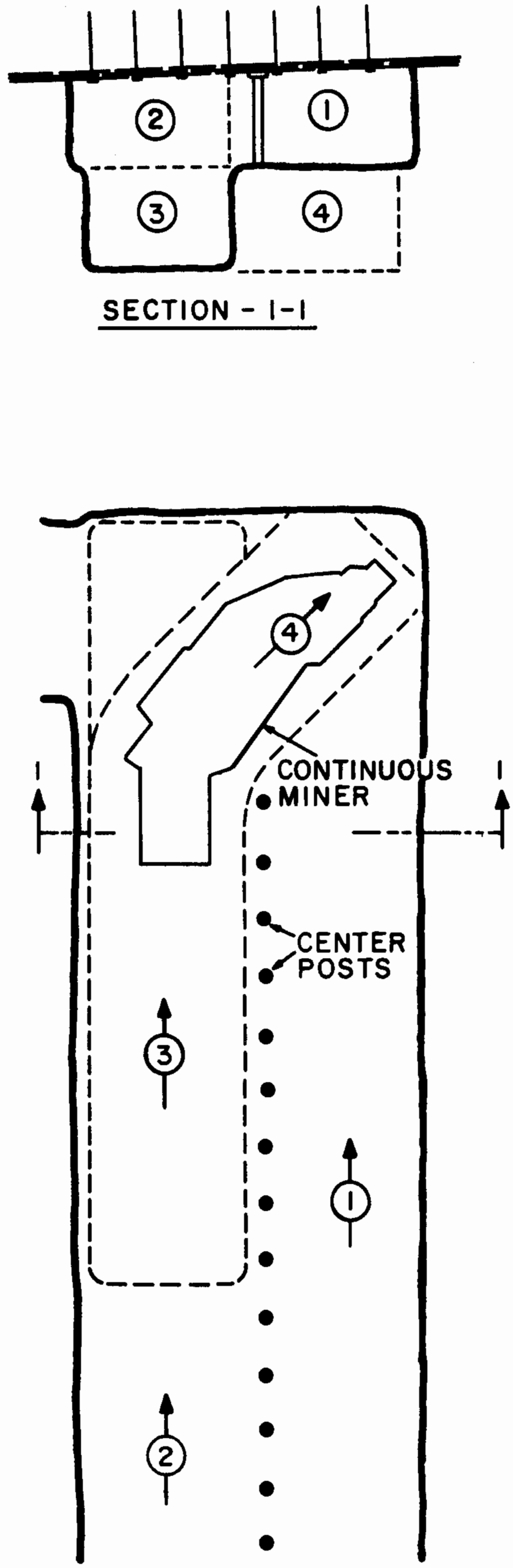
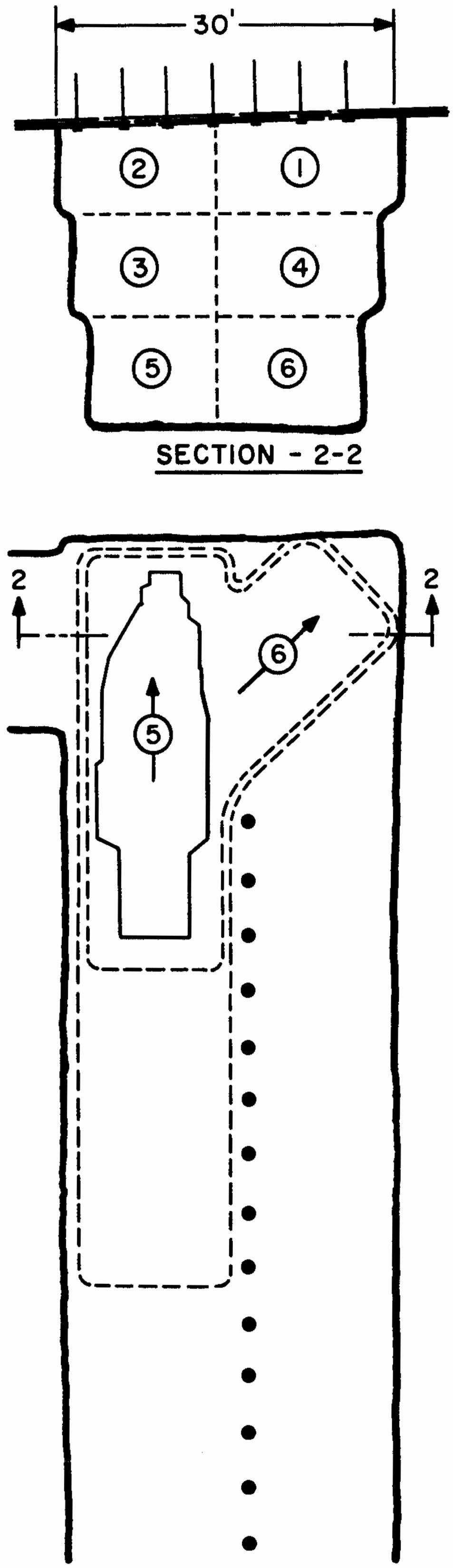
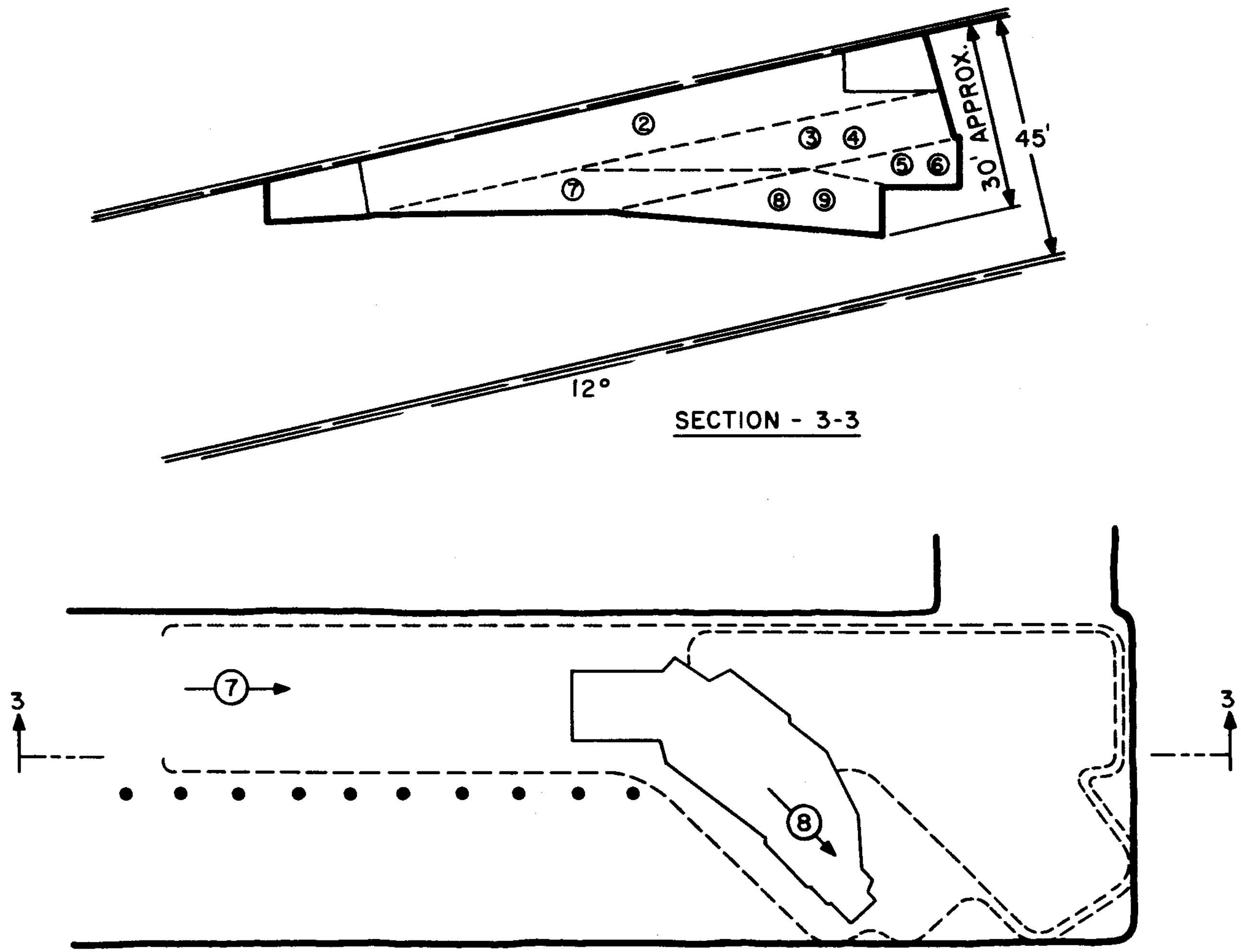


FIG. 25 FIRST PHASE OF FLOOR COAL EXTRACTION, BALMER NORTH MINE



1 Foot = 0.3048 Meters

FIG. 26 SECOND PHASE OF FLOOR COAL EXTRACTION, BALMER NORTH MINE



1 Foot = 0.3048 Meters

FIG. 27 FINAL PHASE OF FLOOR COAL EXTRACTION, BALMER NORTH MINE seam thickness in the room is eventually recovered. As deeper cuts are taken in the dip crosscut, only a limited number of passes can be taken in the strike entry, due to the high face resulting from advancing in the updip direction. The crosscuts in the entries follow the same procedures used in the rooms.

Although the Balmer North Mine has an excellent safety record and is, compared to other mines under similar conditions, quite productive, the inadequacy of continuous miners and shuttle cars from extracting very thick and pitching seams is clearly revealed. The coal recovery from a panel is less than 20%. The pitch limits the face haulage to one shuttle car behind the miner and the roadways have to be maintained in excellent condition. Furthermore, the 12° gradient in the roadways slows down the shuttle car, leading to a large cycle time and, consequently, to a large miner wait time. Additionally, breakdowns under these difficult circumstances drastically affect production and, therefore, call for a very high level of maintenance delays. The mine visit provided an insight into the severe problems encountered with continuous miners and shuttle cars in thick, pitching seams. In fact their application, from a production and a resource recovery standpoint, is not desirable.

The Hydraulic Mine has been in operation for over five years. During the latter part of the 1960's, Kaiser Resources and Japan's Mitsui Coal Mining Co. agreed to test hydraulic mining at the Balmer Seam. In this system, a high pressure water jet is used to cut, break, load, and transport the coal (Parkes and Grimley, 1975). Water, at 2000 psi (137.9×10^5 Nm^{-2})[1] pressure, is provided by a 2500 hp

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[1] Nm^{-2} = Newtons per square meter

seven-stage centrifugal pump, installed near the mine portal at a rate of 1800 gpm (6813 lpm)[1]. The water is delivered from the pump through high-pressure steel pipes to the monitor at the face. The pressurized water is directed by the nozzle of the monitor to the solid coal. In addition to fracturing and loosening the coal, the water flushes the broken coal to a feederbreaker for sizing. The sized coal is then transported out of the mine as a slurry in an open steel flume. Low-pressure water is also provided to maintain the consistency of the slurry. To close the circuit, the slurry is dewatered and the water is fed back to the pump. The method is appropriate for mining under conditions which are not suitable for extraction with continuous or conventional methods (Grimley, 1974).

Entrance to the mine is by a drift from which two 16-ft (4.8m) entries, an intake-flume road and a return, are driven near the footwall of the seam at seven degrees up the pitch to facilitate gravity flow of the coal slurry in the flume. Off these main headings, 800-ft-long (243m-long) panel headings, also at a prescribed gradient but in the opposite direction, are driven on 80-ft (24m) centers (Figures 28 and 29). As more panels were developed, the gradient of the sublevels was reduced, such that the newer sublevels are at a 4° gradient. These sublevels, like the mains, are driven with Joy ripper miners (1CM or 6CM) and are supported by steel arches set on 5-ft (1.5m) centers with 2-in.(50.8mm) lagging.

On development, the mined coal is trammed by a shuttle car to the

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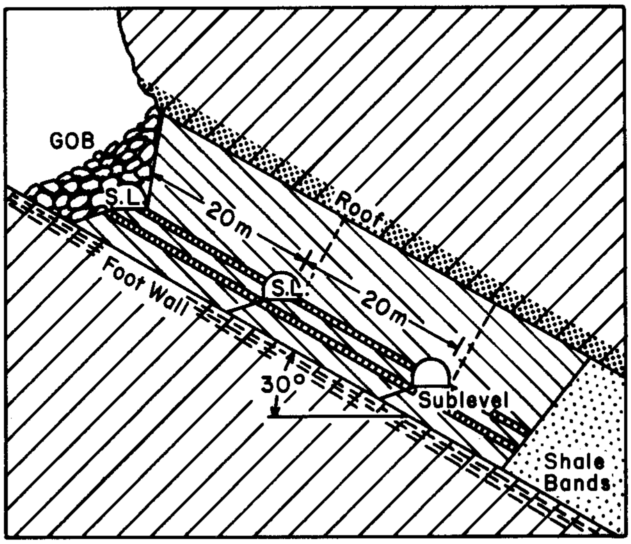
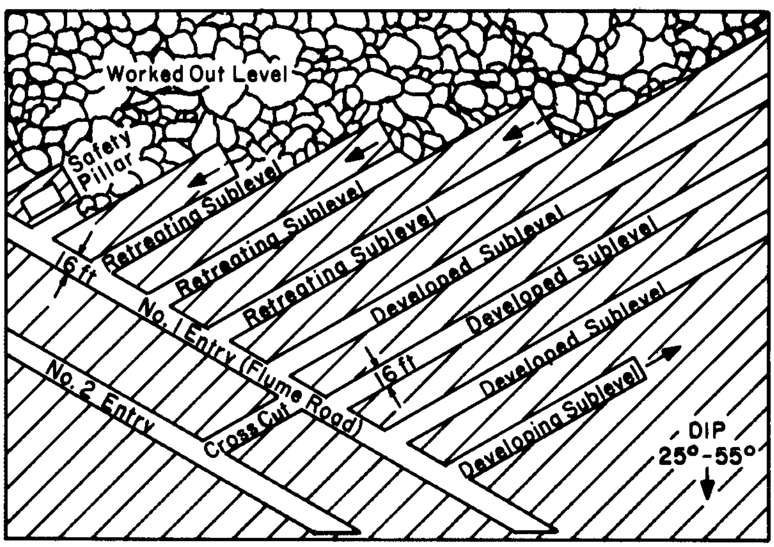
[1] lpm = liters per minute



1 Foot = 0.3048 Meters

FIG. 28 END VIEW OF PANELS, HYDRAULIC MINE
(Grimley, 1974)



1 Foot = 0.3048 Meters

FIG. 29 PLAN VIEW OF PANELS, HYDRAULIC MINE
(Grimley, 1974)

feederbreaker where it is sized, mixed with water which is delivered under low pressure, and flumed out of the mine. The flume is an effective replacement for conveyor belts and can be set in an intake entry, thereby affording easy access (Parkes and Grimley, 1975).

Seven men work on the development crew and average between 20 and 40 ft(6 and 12m) of advance per shift. Face ventilation is provided by an auxillary fan with flexible tubing.

On retreat, two men operate the hydraulic monitor and the feederbreaker as the 60-ft (18m) pillar between sublevels is extracted (Figures 30 and 31). To begin pillaring, eight arches are withdrawn to expose the pillar coal. The hydraulic jet then follows a cut sequence, mining inby the last arch, and is directed by the operator to exploit the weak partings in the seam to induce breakage (Grimley, 1974). The caved coal is flushed to the feederbreaker, sized, then flumed out of the mine. When all of the coal is extracted, or when the roof collapses, the units are pulled back and another eight arches are removed. While most of the rise side of the levels is extracted at any one time, an attempt is made to recover some coal from the dip side of the sublevel as long as a proper gradient can be maintained for flow of the broken coal (Figure 32). Though only one monitor is operated per shift, other monitors are available as standbys.

Two important pieces of face equipment, the feederbreaker and the monitor, are specially designed to operate under these conditions. The monitor is built in Japan to Kaiser's specifications. The skid-mounted feederbreaker, is made in the United States to Kaiser's specifications and is designed to work almost submerged in coal and water.

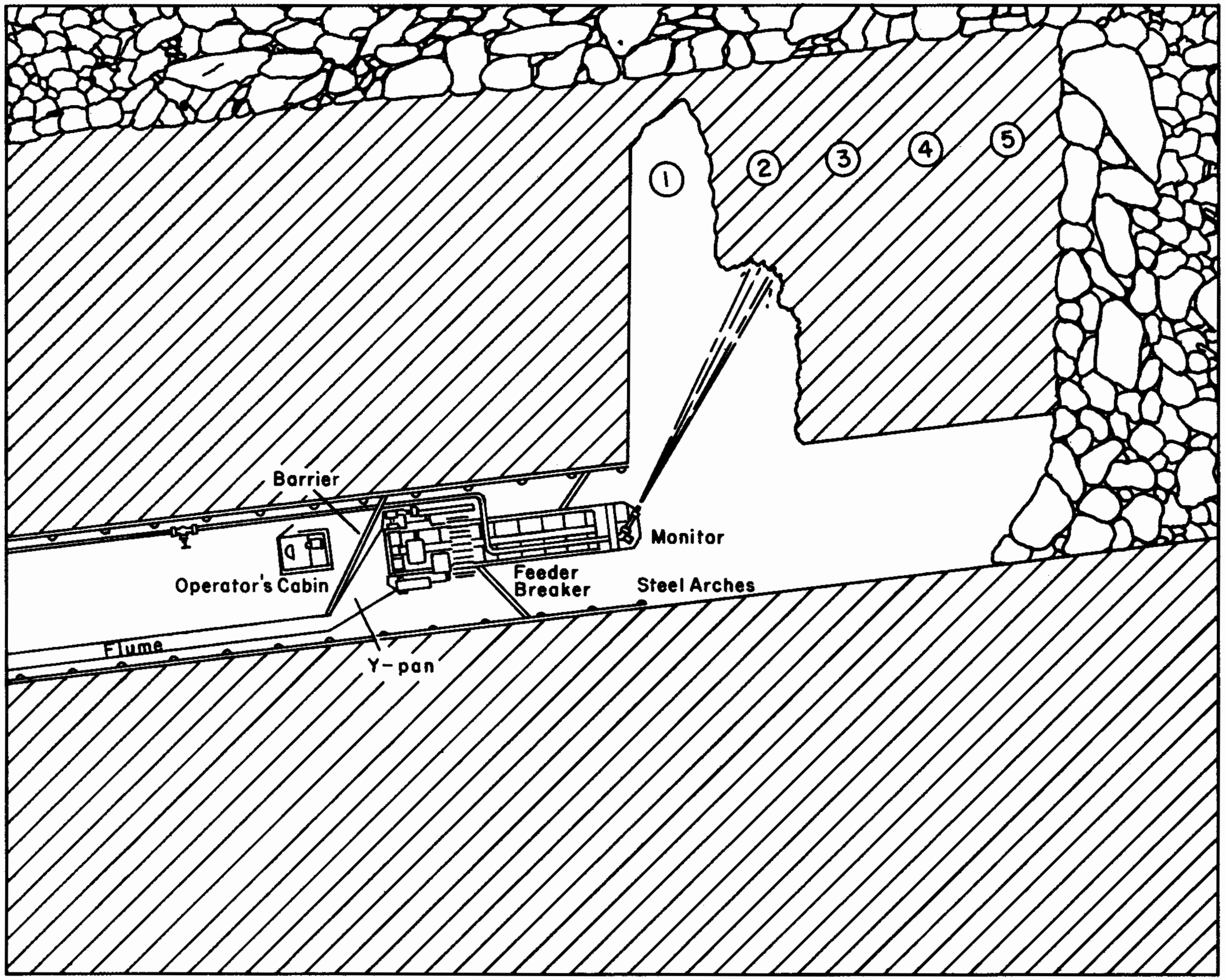


FIG. 30 PLAN VIEW OF RETREATING FACE AREA, HYDRAULIC MINE
(Grimley, 1974)

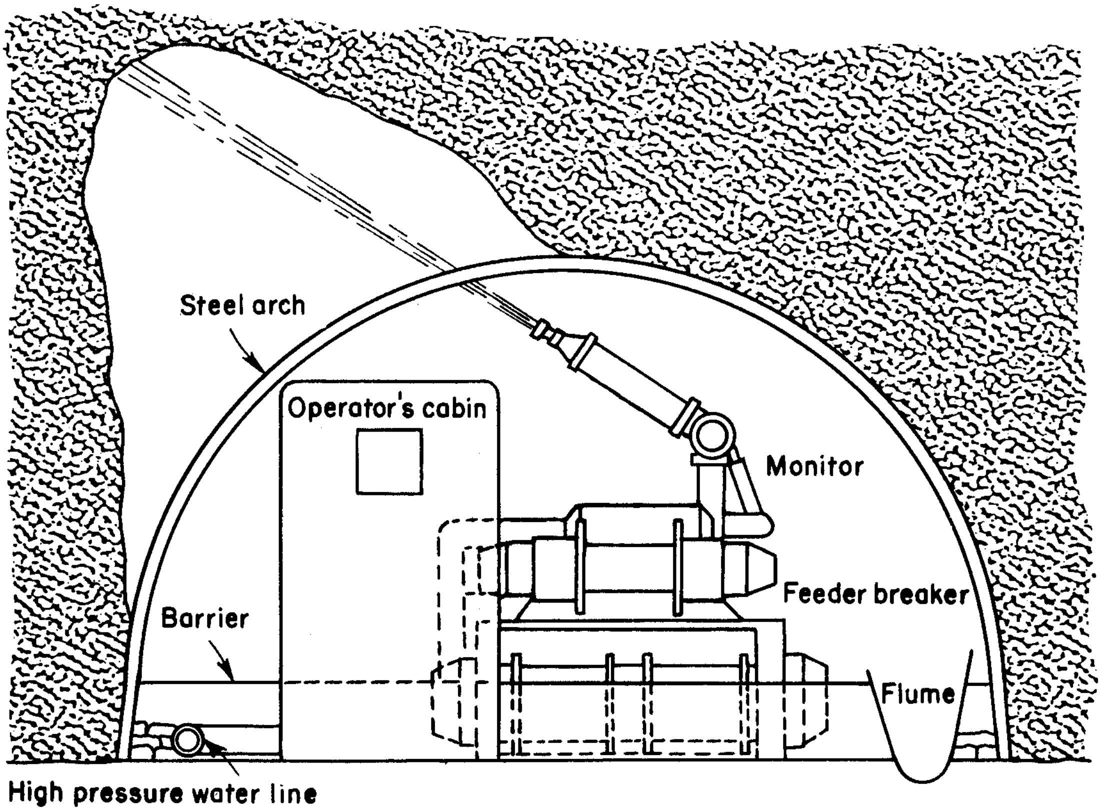


FIG. 31 END VIEW OF RETREATING FACE AREA, HYDRAULIC MINE
(Parkes and Grimley, 1975)

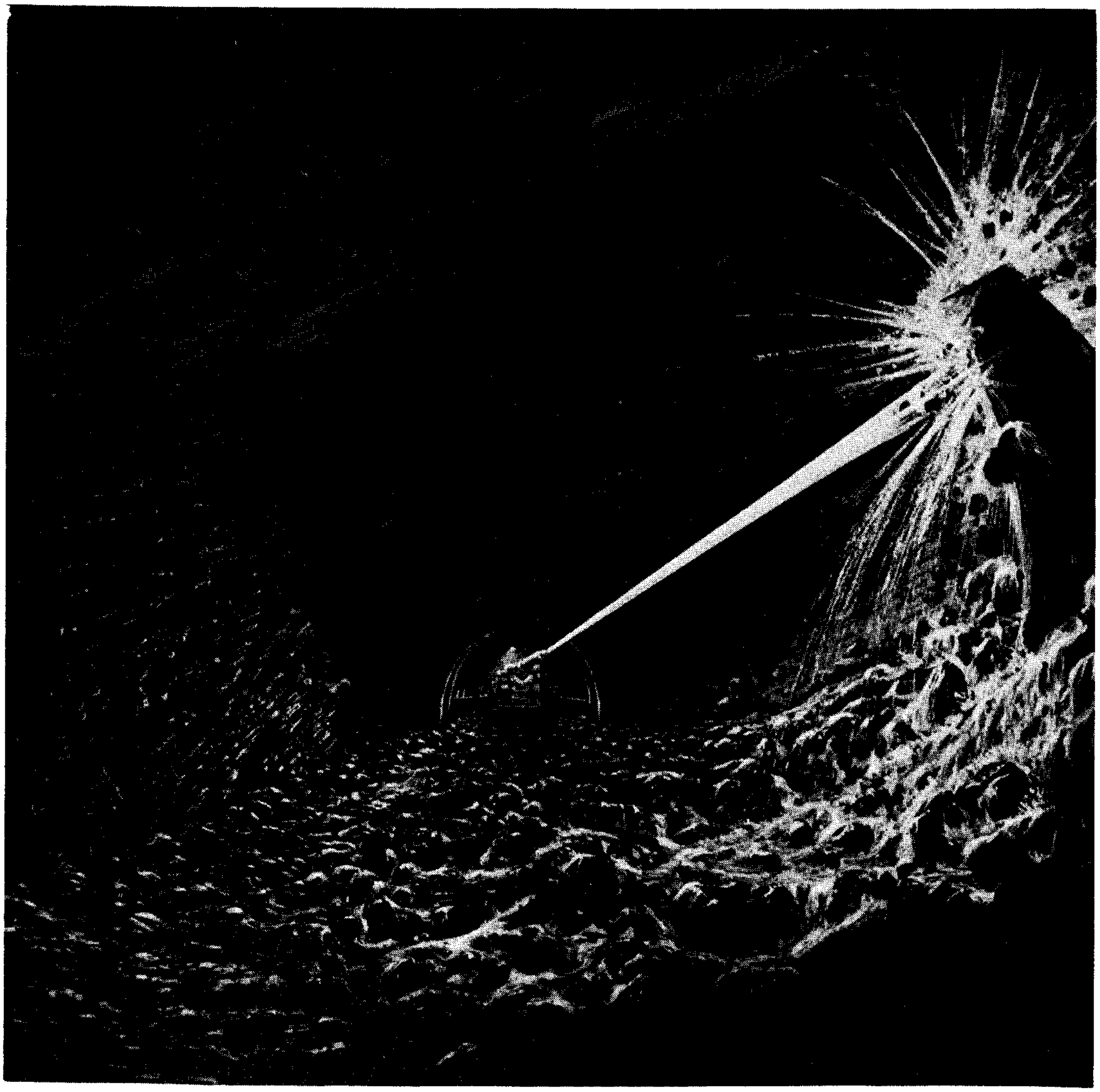


FIG. 32 GOBSIDE VIEW OF RETREAT MINING, HYDRAULIC MINE
(Artist's Conception)

It can handle 15 tons (13.6 metric tons) of coal per minute (Parkes and Grimley, 1975).

While the record-setting performances for a single shift and a three-shift period are 3500 tons (3175 metric tons) and 7900 tons (7165 metric tons), respectively, a representative figure for shift production will be between 2200 and 2500 tons (1995 and 2268 metric tons). In 1974, just over 900,000 tons (816,300 metric tons) were produced, of which hydraulic mining accounted for slightly over 800,000 tons (725,600 metric tons). Since the total mine personnel is approximately 150, the average productivity is above 25 raw tons (22.7 metric tons) per manshift. The hydraulic mining system also claims high resource recovery. The Hydraulic Mine reports a recovery of 70% in the panels and an overall recovery of 55% (Parkes and Grimley, 1975).

Michel Colliery won the 1971 and 1973 Ryan Trophy for having the fewest lost time accidents per manshift of any underground coal mine in Canada (Parkes and Grimley, 1975). This is a testimony to the safety of a system employed under very difficult conditions. As such, the most hazardous part of the operation is not due to falls of roof and rib but to the handling of heavy and awkward-sized materials, such as pipe and flumes. Some of the reasons for the mine's safety record according to Grimley (1974) are as follows:

1. All roadways are supported by steel arches.

2. The monitor operator is 35 ft (10.6m) behind the monitor in a control room under the arches. Also, the feederbreaker operator is 100 ft (30m) from the face. Because of this relative remoteness to the actual jetting point, they are not endangered by caving at the face.

3. The coal is broken, loaded, and transported by water. Thus, dust is minimized and sparking is eliminated.

4. The operators are always in the intake air. The intake passes over the operators, up the sublevel, and out through the gob.

5. Less men are required for the desired production,which results in less man-hours of exposure.

In analyzing the safety of the system, attention is drawn to one aspect of the present layout. If a fire starts near the neck of a sublevel, no alternate escapeway is provided for face personnel. Thus, for applications in the United States, a variation of the system incorporating an escapeway is needed, or a variance would have to be obtained.

Some important requirements for the application of hydraulic mining according to Parkes and Grimley (1975) are:

1. The coal should be soft or have friable bands to aid in the cutting sequence. Where the coal is hard, blasting can aid the water jet.

2. The roof should be competent and part easily enough from the coal to allow for maximum extraction before the roof collapses.

3. The floor must be able to resist degradation by water.

4. The dip of the roadways should be sufficient to allow the flow of coal in the flumes. The present dip in use is four degrees.

5. Overall economics improves with greater seam thicknesses.

Under proper conditions, this system of mining is very efficient, from safety and productivity points of view.

<u>Canmore Mines, Limited</u>

Located 62.5 miles (100km) west of Calgary, Alberta, Canmore Mines, Ltd. operates two mines in a pitching, thick seam. The Canmore No. 2 Mine was opened in 1961, while the Riverside Mine is a relatively new operation. In both of the mines, the coal is semi-anthracite. The Wilson seam is very gassy and is characterized by extreme variability in thickness and pitch (Stephenson, *et al.*, 1972). The seam thickness

ranges from 7 to 30 ft (2.1 to 9.1m) with an average of 13 ft (4m). The gradient varies from 5° to 30° with an average of 15°. A rock band, which varies from a few inches to 3 ft (1m) in thickness, occurs 8 ft (2.4m) above the floor of the seam. For example, in a 13 ft (4m) cross section, the band is about 1 ft (.3m) thick. The immediate roof, which consists of grey and black shales, is fairly competent (Figure 33).

The No. 2 Mine is designed to extract a block of coal 8000 ft (2425m) long and 1500 ft (455m) wide. Two 1400 ft (425m) slopes, driven at an apparent pitch of 17°, are used for men, materials, and coal transportation as well as ventilation. At the bottom of the haulage slope along the strike, there is a locomotive haulage gangway.

In the past, the mine operated with contract miners. Two Lee-Norse continuous miners were also used for development purposes. The entries were driven 16 to 18 ft (4.8 to 5.5m) wide. The gassiness of the seam, and consequently the ventilation requirements for the mine, dictated the other dimensions of the operations. Methane emmissions, which required 20,000 to 30,000 cfm (9.4×10^6 to $14.2 \times 10^6 cm^3$ per sec) at the face, directed by an auxillary fan and tubing, limited the length of drivage. The property was laid out in 100 by 100 ft (30 by 30m) pillars with an initial in-place recovery of 30%. These pillars eventually proved to be too small for efficient recovery with the continuous miners (Stephenson, *et al.*, 1972).

Before the pillars could be extracted, an abandoned and inaccessible mine,160 ft (48m) above the Wilson seam, had to be dewatered. Over a one year period (12/69 - 11/70), an estimated 360 million gallons (1363 million liters) of water was pumped out of the mine (Stephenson, *et al.*, 1972).

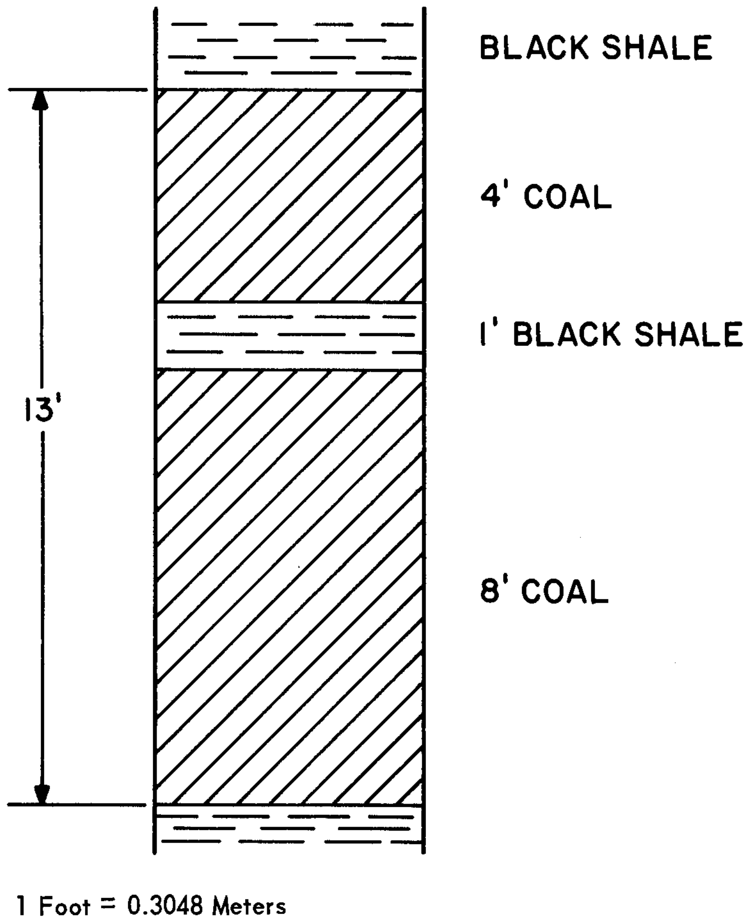


FIG. 33 TYPICAL CROSS-SECTION OF THE WILSON SEAM, CANMORE NO. 2 MINE (Stephenson, et al., 1972)

Initially, the pillars were extracted with continuous miners (Figure 34). Face haulage was provided by Joy 10SC-26 shuttle cars, with one unit operating behind each miner. The coal was discharged by each shuttle car onto the floor near the receiving end of the panel chain conveyor. A Joy loading machine, either an 8BU or a 14BU, was used to transfer the payload from the pile to the 20-in.(508mm) chain conveyor. The loader was required to properly meter the coal for the chain conveyor. The face haulage system - one shuttle car, loader, and the chain conveyor - may seem inefficient but it did not affect the overall operation greatly because of the other necessary, but time consuming, functions such as roof support and methane control at the workings (Stephenson, *et al.*, 1972).

The pillars were extracted downdip in slices, a procedure dictated by the 15° gradient. Strike pillaring or dip to rise pillaring resulted in equipment operational problems due to poor traction. If the floor became wet, even rise to dip traction became difficult. In fact, 18° for the shuttle car and 20° for the miners were found to be the upper limit for operation. However, even under good conditions, shuttle car haulage was seriously affected at pitches of 13° to 15°. Roof bolts and planks were used to support the roof, but many caves originated far above bolt anchorage. Eventually, the roof control problems increased and tonnage dropped from an initial figure of 300 tons (272 metric tons) per machine shift to 160 tons (145 metric tons) per machine shift. Pillar recovery also suffered, dropping from 55% to 40% within a span of three months. Not long after, a heavy cave occurred and buried one of the continuous miners. This led Canmore to seek a new method of pillar recovery (Stephenson, *et al.*, 1972).

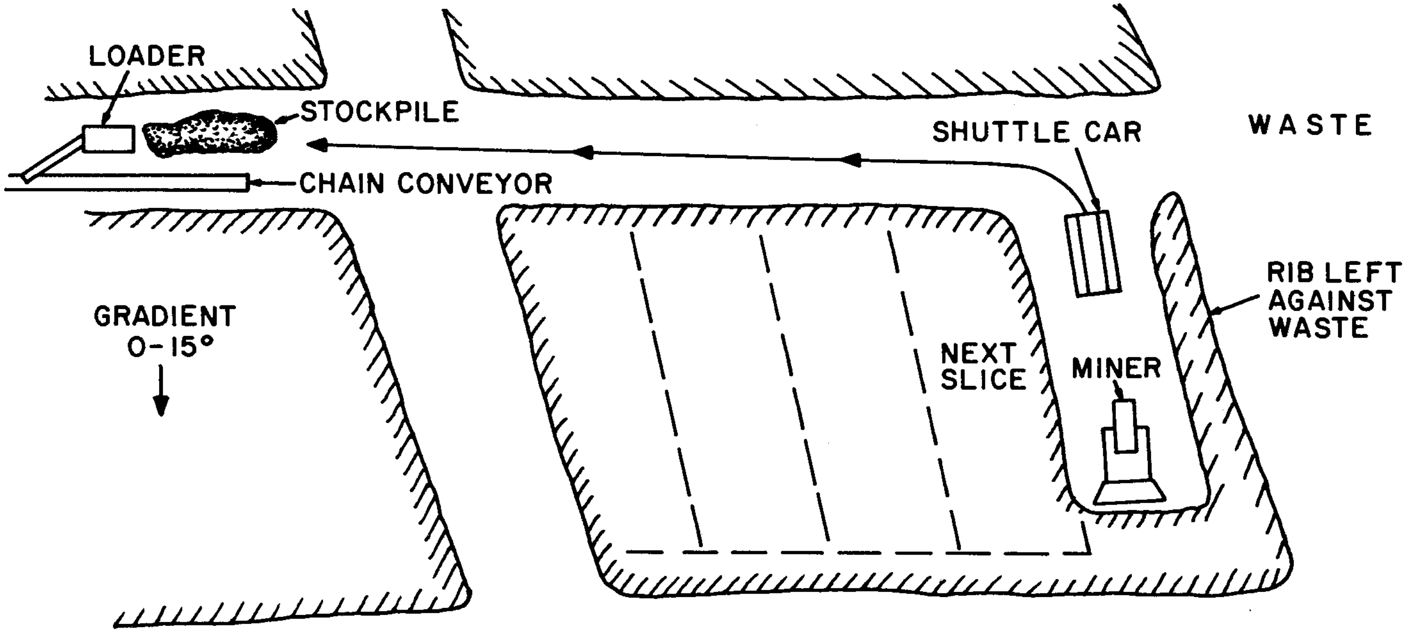


FIG. 34 PILLAR EXTRACTION WITH A CONTINUOUS MINER, CANMORE NO. 2 MINE
(Stephenson, et. al., 1972)

The two continuous miners were replaced with four electric and five compressed-air slushers (Figure 35) (Stephenson, *et al.*, 1972). To begin the mining cycle, an eye-bolt anchor for the slusher is secured at the far end of the pillar. The pillar is then drilled and shot to permit the recovery of a 5-ft (1.5m) slice. During the extraction operation, gravity causes the slusher bucket to be dragged along the face to be loaded out. Two men in each of the seven man crew are classified as production miners and are paid both a set wage and an incentive bonus based upon tonnage produced. The other men help with the equipment and roof support. As the face is worked, timber support is advanced to within 8 ft (2.4m) of the working face. The remaining area is bolted. After the slice is extracted, the eye-bolt anchor for the bucket is reset and the cycle is started over again. This system has proven to be safer than the method employing continuous miners because the workmen's exposure at the face is limited only to the roof support function. The slusher operator works 50 to 150 ft (15 to 45.5m) away from the face in a well-secured area, eliminating the dangers previously confronting the production crew.

Face productivity with the new system is averaging nearly 50 tons (45 metric tons) per manshift, with pillar recovery increasing to 55%. The operational problems are so minimal that, in a development section, a slusher is used to load out the blasted coal. In summary, Canmore's method of pillar recovery in a pitching thick seam is not only productive but does not need extensive capital equipment or roof support.

The Riverside Mine, at the time of the visit, was only developed for six crosscuts. As such, the eventual conditions that the mine will

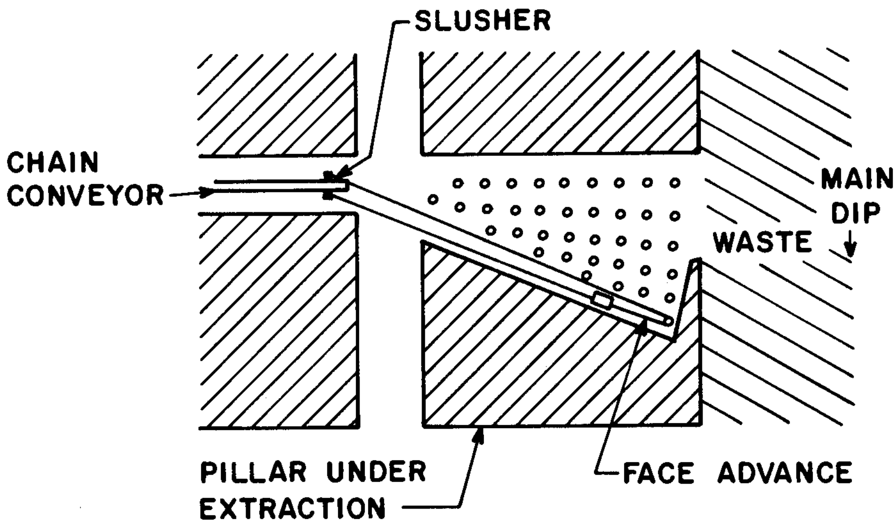


FIG. 35 PILLAR EXTRACTION WITH A SLUSHER, CANMORE NO. 2 MINE
(Stephenson, et al., 1972)

encounter are difficult to determine. However, the seam height at the immediate area of mining differs from that encountered at the No. 2 Mine. The height appears to range between 8 and 12 ft (2.4 and 3.6m) in this portion of the Wilson Seam. There are three operating sections at the mine and they work every shift. By incorporating the slusher method for development drivage, the daily production averages 350 tons (317 metric tons) for the mine.

Entrance to the mine is by a 7.5° slope. This slope contains a conveyor belt for the coal and a monorail system for the transportation of men and materials. Since the three faces visited were on advance, it was felt that a more regular pillar layout could be achieved with slushers than the layout that was produced at the No. 2 Mine with continuous miners. Riverside also has to contend with considerable methane liberation. In one section visited, a newly shot face had to be ventilated for one hour before the loading cycle could begin. Entries are driven 20 ft (6m) wide on 150-ft (45.5m) centers. Bolting with stopers and timbering are the primary means of roof support.

Sunnyside No. 1 Mine

The Sunnyside No. 1 Mine, the largest of a three mine complex at Sunnyside, Utah, is operated by the Kaiser Steel Corporation. The property was purchased by Kaiser in 1950 to supply most of the coking coal requirements for its steel plant at Fontana, California. Production at the mine is slightly over one million tons (907,000 metric tons) per year, making it the largest producer in the state (Keystone Coal Industry Manual, 1974). The mine began as a room-and-pillar operation, but eventually became well known for its longwall faces (Huntsman, 1974).

Although the present operations are in the Lower Sunnyside seam, in the past the Upper Sunnyside seam was extracted in conjunction with the Lower Sunnyside.

Both seams outcrop along the Book Cliffs, a prominent escarpment in the area. They also pitch from 3° to 10°downward into the mountain range. The Upper Sunnyside seam ranges from 3 to 6 ft (1.0 to 1.8m) in thickness, while the Lower Sunnyside varies from 5 to 14 ft (1.5 to 4.2m). Separation between the two seams varies from a few inches to 45 ft (13.6m) and, as the parting increases, it changes from a weak shale to a laminated sandstone. Where the seams were closed together and the preparation plant could economically handle the rock parting, the Upper Sunnyside was taken along with the Lower Sunnyside (Huntsman, 1974).

The massive Castle Gate sandstone, ranging up to 200 ft (60.6m) in thickness, is located 150 ft (45.5m) above the Lower Sunnyside seam. Most of the cover is sandstone and varies from several hundred feet to 2500 ft (758m). The hilly terrain causes the overburden thickness to vary considerably over relatively short distances, (Peperakis, 1958). For example, the 2000 ft (606m) cover line for the No. 1 Mine is only 2700 ft (818m) from the outcrop. Because of the massive sandstone beds in the overburden, unless the roof caves regularly with the advance, bump conditions are created. The immediate roof is either shale or sandy shale, which is less competent than the main roof. This tends to cause a large number of roof falls when bumps occur (Peperakis, 1958).

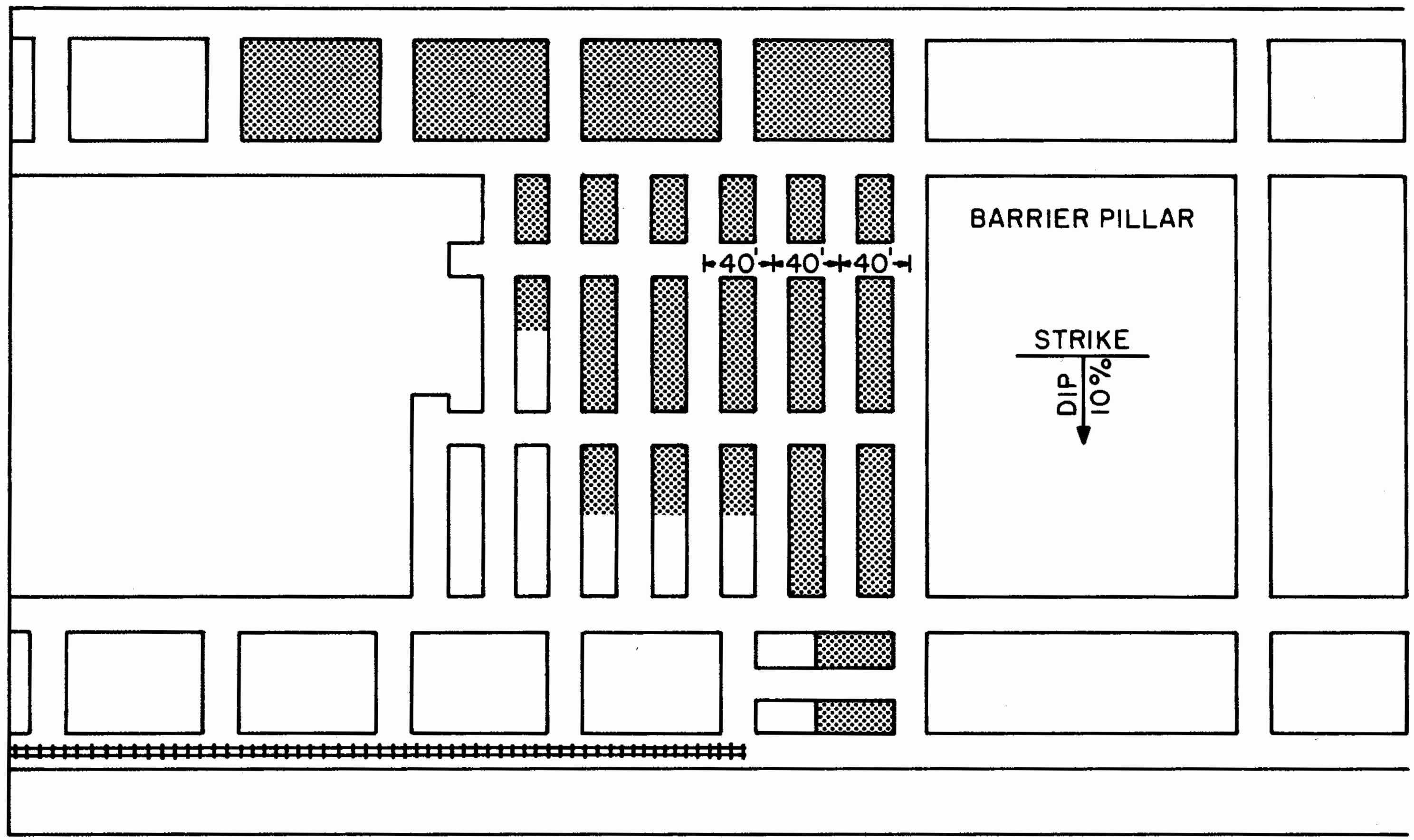
The immediate floor of the Lower Sunnyside seam is a thin shale, under which lies a massive sandstone bed whose thickness varies from 20 ft (6m) to 50 ft (15m). This bed is also thought to aggravate the bump conditions. Since the seam pitches, there is an additional problem of lateral movement accompanying the crushing (Peperakis, 1958).

In the past, as previously noted, the room-and-pillar method was employed at Sunnyside. This included both continuous and conventional mining.

In a continuous section, two development entries were driven along the strike. Upon completion of the entries, a 200-ft (61m) barrier was left to protect the bleeder system and then, if the seam was less than 10 ft (3m) in thickness, two rooms were developed by a single-pass method (Figure 36). As one room was on retreat, the next outby room was on advance. Where the thickness of coal to be extracted was greater than 10 ft, a benching method was followed in the rooms (Figure 37). The use of continuous miners proved to be successful in applications of up to 20 ft (6m). Here again, two rooms were worked at the same time; one would be on advance in the upper bench and other, on retreat in the lower bench. When the lower entry of the previously mined section was reached by a developing room, the crew backed out approximately 25 ft (7.6m) and drove a notch in the inby pillar, also in the upper bench. The crew then dropped further back down the room, ramped into the floor coal, and extracted the bottom bench of the room and notch, as well as any of the stump coal. Hand-held air drills were required for those parts of the stumps which were beyond the range of the miners. Shooting the stumps was beneficial. Even though all the stump coal could not be recovered, the crew retreated further out of the room and started the cycle over again (Huntsman, 1974).

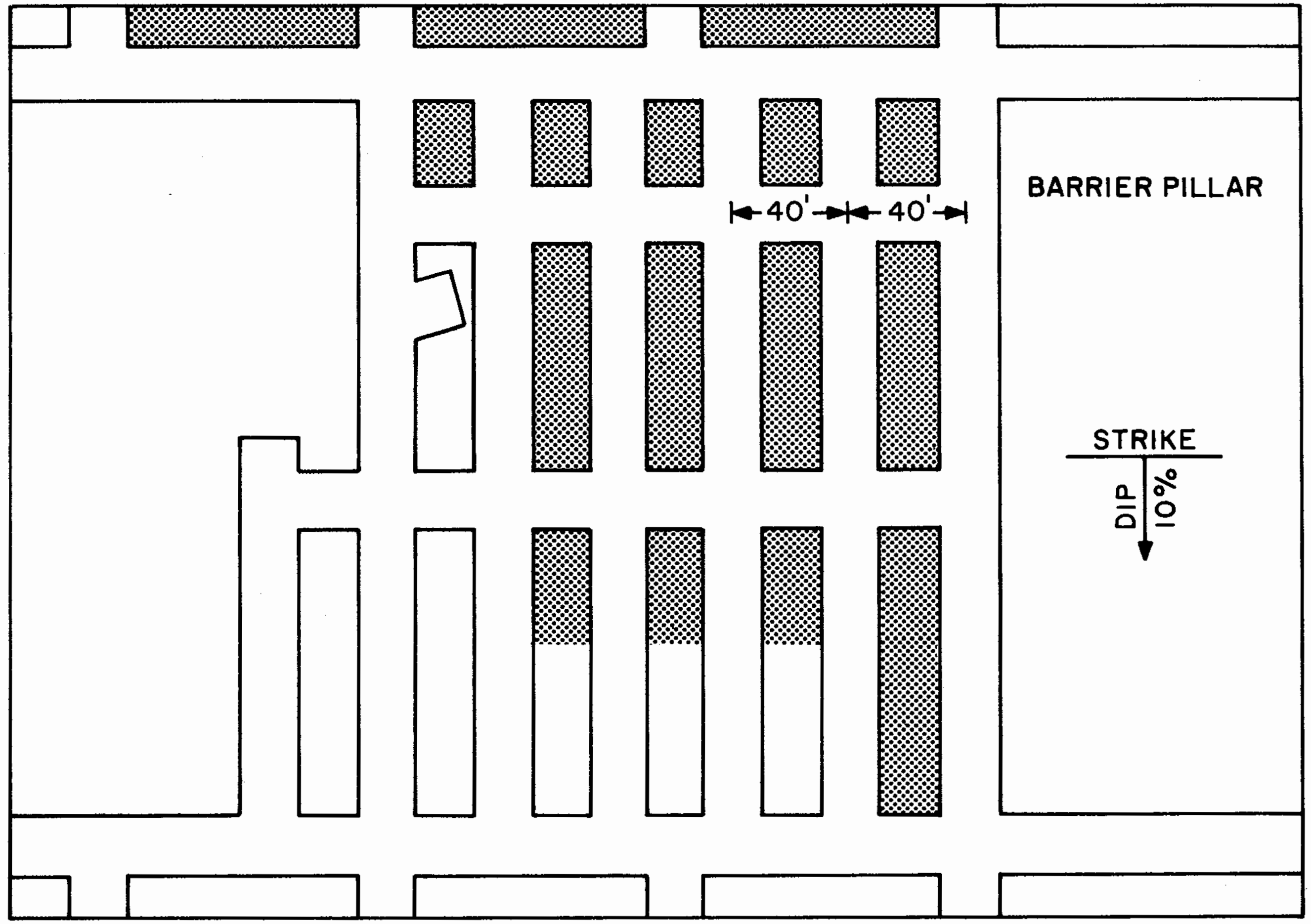
A typical continuous miner section consisted of the following equipment and men (Huntsman, 1974):

Equipment	Units
Trolley Locomotive and Mine Cars	1
10SC Shuttle Car	1



1 Foot = 0.3048 Meters

FIG. 36 MINING OF ROOMS BY A SINGLE-PASS METHOD, SUNNYSIDE NO. 1 MINE
(Huntsman, 1974)



1 Foot = 0.3048 Meters

FIG. 37 MINING OF ROOMS BY A BENCHING METHOD, SUNNYSIDE NO. 1 MINE (Huntsman, 1974)

11BU Loader	1
1CM2 Continuous Miner	1
R48 Stoper and Bolting Equipment	1

Category	Men
Locomotive Motorman	1
Shuttle Car Operator	1
Loader Operator	1
Miner Operator	1
Bolters	2
Supervisor	1

In the conventional section, the development of the panel entries was identical to that in the continuous section, except that three or more entries were driven to accomodate the additional face units. The procedures followed for room mining, in either the single-pass or benching situations, were identical to those used in the continuous sections except that three or more rooms had to be worked at the same time. The conventional section was better equipped to handle areas where the rock parting between both seams was over 1 ft (0.3m); the major disadvantage, however, was that a large number of working places had to be opened in the rooms to accomodate all the pieces of machinery. This slowed the recovery process and hindered full recovery (Huntsman, 1974).

A typical conventional section consisted of the following equipment and men:

Equipment	Units
Trolley Locomotive and Mine Cars	1
10SC Shuttle Car	1

11BU Loader	1
10RU Cutter with CD40 Drill	1
DM8 Dual-boom Bolting Machine	1

Category	Men
Locomotive Motorman	1
Shuttle Car Operator	1
Loader Operator and Helper	2
Cutter Operator and Helper	2
Bolter and Helper	2
Shotfirer	1
Supervisor	1

In 1961, as the thick-seam areas at the No. 1 Mine became exhausted, Kaiser adopted the longwall method (Ross, 1974). During the mine visit, two of the four longwall panels in operation were seen. The two longwall faces were 6 and 9 ft (1.8 and 2.7m) in height.

The first face visited included the demonstration project on the single entry longwall concept. The panel was driven 5000 ft (1515m) deep. The first and last third of the headgate were driven as a single entry (Figure 38). The middle third of the headgate development, due to a few local ground control problems, contains two entries. The seam does not pitch in this area. Some of the major reasons for studying the single-entry concept are to minimize bumps and roof stresses in virgin coal areas (Ross, 1974). Since chain pillars have been identified as contributing to bump occurrences, their elimination through single-entry development may eliminate bump conditions. There are several other advantages of the single-entry development. Better roof

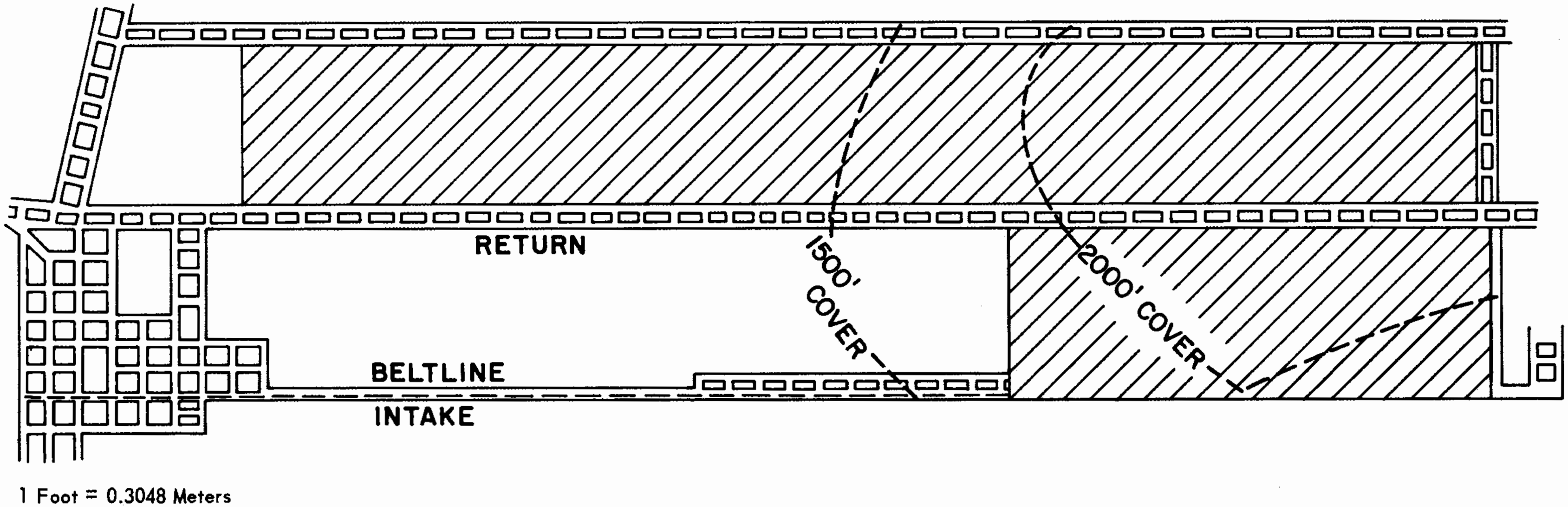


FIG. 38 PLAN VIEW OF THE SINGLE ENTRY LONGWALL SECTION, SUNNYSIDE NO. 1 MINE

conditions can be achieved by reducing exposed roof area through the elimination of crosscuts and multiple entries. Increased recovery from a longwall panel is also facilitated with no coal lost in the chain pillars. Additionally, the absence of chain pillars should lead to a more uniform subsidence.

The single entry is driven 26 ft (7.9m) wide and is supported with roof bolts, mats, and cribs placed along the center of the entry (Figures 39 and 40). Attached to the cribs is a line of galvanized metal panels which are sprayed with an air-tight sealant. This cribline separates the single entry into an intake and a return-beltline.

The equipment used in driving the single entry include a continuous miner, a shuttle car, a roof bolter,a 36-in.(914mm) beltline, a locomotive and mine cars. Face ventilation is accomplished by a travel curtain and line brattice which are extended from the end of the cribline. The cribline is advanced so that it is no more than 150 ft (45.5m) from the face (Ross, 1974).

Since the longwall was already in operation during the visit, the belt entry was no longer used as a return. Although the single entry seemed to be holding up well, there was some concern as to whether or not it would be in good condition to serve as the tailgate for the next panel.

With the seam thickness of only 6 ft (1.8m), the 500-ft (152m) longwall face consisted of a standard double drum shearer installation with chock supports. Where the headgate becomes two entries, the pillar coal is extracted by heavily cribbing the entry nearest the face and extracting the pillar with the shearer, also.

The second longwall face visited was laid out on a two-entry

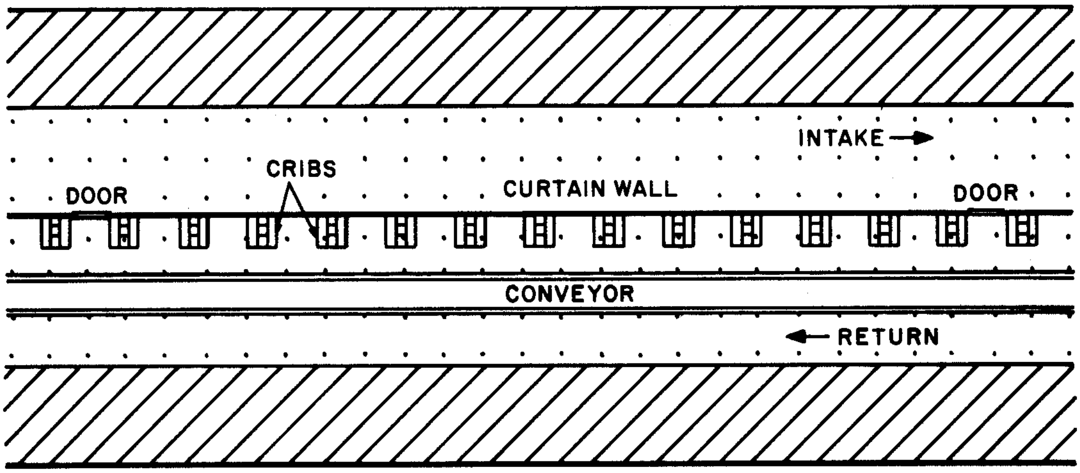


FIG. 39 PLAN VIEW OF THE SINGLE ENTRY LONGWALL DEVELOPMENT, SUNNYSIDE NO. 1 MINE
(Ross, 1974)

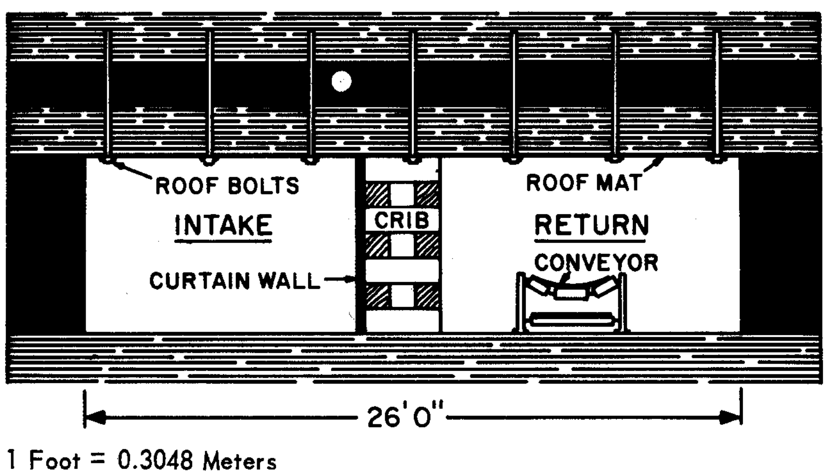


FIG. 40 END VIEW OF THE SINGLE ENTRY LONGWALL DEVELOPMENT, SUNNYSIDE NO. 1 MINE
(Ross, 1974)

system. The face pitches 5° with the two lower headings serving as headgate and haulage access and the two upper headings serving as the tailgates. The face, which is 9 ft (2.7m) in height, is also 500 ft (152m) long. The entries were 22 ft (6.7m) wide. Roof bolts, on 4-ft (1.2m) centers, and mats, as required, provide the roof supports.

A double-drum shearer, in conjunction with chock supports, is used in the longwall. As the face is retreated, cribs are installed on 8-ft (2.4m) centers in the lower entry, as this entry will be used as the tailgate for the next panel.

Fortunately, Kaiser has shown an awareness of, and attention to, the mining problems confronted under difficult conditions and have published their experience extensively in technical meetings and industry magazines.

Beehive Mine

The Beehive Mine of American Coal Co. is located six miles (9.7km) west of Huntingdon, Utah. The mine operates in the tabular, 14-ft (4.2m) Blind Canyon seam and access is by a drift. Similar to other operations in the area, the mine has to contend with a variable overburden, due to the mountainous terrain. The high-quality steam coal is soft. Sloughing of the ribs is common. The coal is underlain by a fireclay and the immediate roof is a competent sandstone. Bolting is generally not required for roof support.

Five-entry panels, with 16-ft-wide (4.8m-wide) entries on 80-ft (24m) centers, are driven off the mains by Jeffrey or Joy narrow-head milling miners. On retreat, the pillars are split into quarters. Where possible, the stumps are reduced. In both development and retreat work, only 12 ft (3.6m) of the seam is extracted. Face haulage is provided by Joy shuttle cars, while 36-in.(914mm) belt conveyors are used for secondary and mainline haulage. Diesel transportation is used for men and materials. Production averages 500 tons (454 metric tons) per machine-shift. Eight daily machine-shifts are operated in the four panels.

York Canyon Mine

Kaiser Steel's York Canyon Mine is located forty miles (64.4km) west of Raton, New Mexico. The York Canyon coalbed decreases in thickness from 13 to 4 ft (4m to 1.2m), with the maximum thickness occurring at the outcrop. The seam is basically tabular and the pitch rarely exceeds 3°. The overburden ranges from 30 ft (9m) at the outcrop to a maximum of 700 ft (212m). The seam is intersected by numerous faults which create displacements of up to 12 ft (3.6m) in the areas visited. The roof is a black, sandy shale and the floor, a 6-in.(152.4mm) layer of fireclay, is underlain by a more competent shale (Jackson, 1976).

Transportation for men and supplies is provided by rubber-tired battery-powered tractors and golf carts. The coal is transported out of the mine by 48-in.(1219mm) main belts which are fed by 36-in.(914mm) panel belts. York Canyon employs nearly 250 men and the daily tonnage, based on two production shifts, averages 4500 tons (4082 metric tons).

The 3 North Panel, which contains the 10-ft-high (3m-high) shield-supported longwall, was visited. The general layout of the panel is shown in Figure 41. In the past, chock supports with wire mesh, used to provide protection against rock falling between the chocks, were employed in the longwall panels.

At the present time, the face equipment consists of an Anderson-Mavor double-drum shearer with a Dowty underframe, an Eichoff face

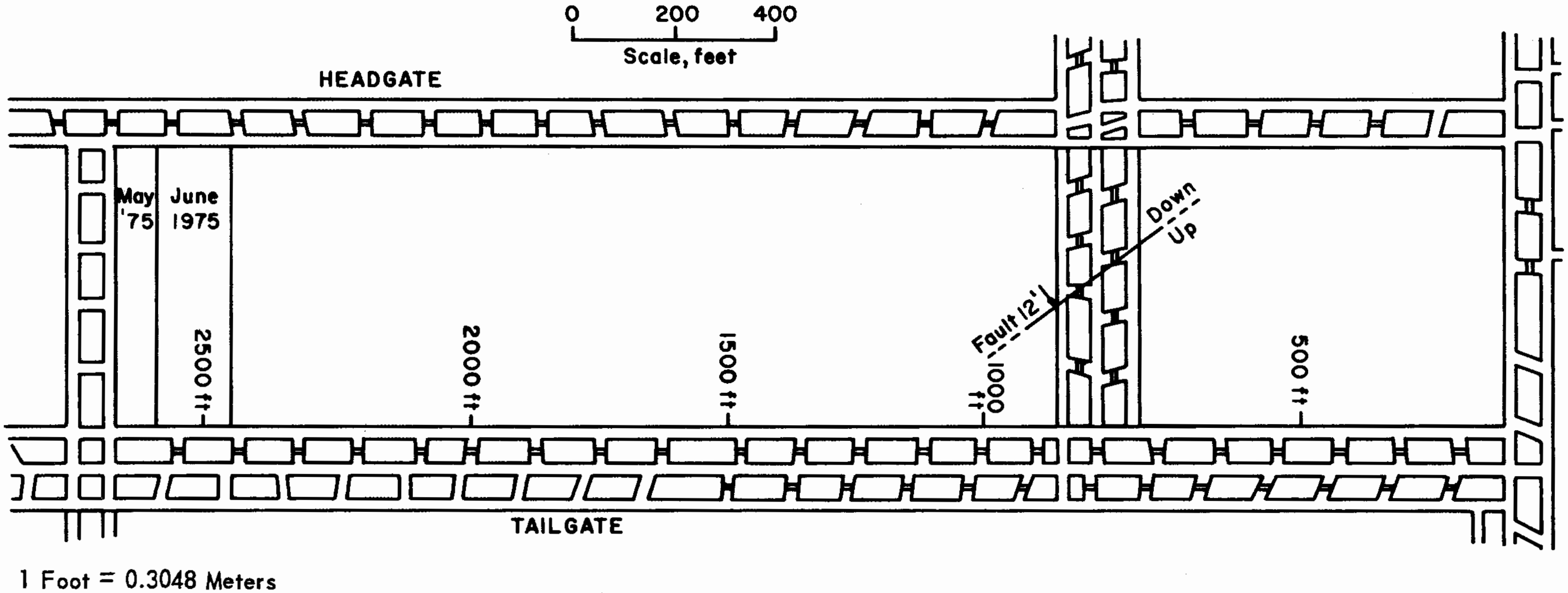


FIG. 41 3 NORTH PANEL, YORK CANYON MINE

conveyor, a Long-Airdox feederbreaker, a Dowty-Meco stage loader, 115 Hemscheidt Sheilds, and Ocean Energy lights. The shields are set on 5-ft (1.5m) centers. The face went into production in May, 1975, on a two-shift basis. In the following month, just over 35,000 tons (31,745 metric tons) were mined. The average shift production was 840 tons (762 metric tons). At the time of the mine visit, the face had advanced only 250 ft (76m).

A definite mining sequence, which is strictly adhered to, is required to extract the 10-ft (3m) seam. The coal from a face of this height tends to spall, thus clogging the face conveyor system. To counteract this, the shearer initially sumps into the middle of the face and advances toward the headgate. After the shearer has passed a shield, face guards on the shields are dropped to prevent the top portion of the face from spalling (Figure 42). After the shearer reaches the headgate, it is deadheaded back to the middle. The same procedure is followed for the other half of the face as the shearer advances toward the tailgate.

Although the longwall was not in operation at the time of the mine visit, the steel under the roof, both provided by the shields, were most impressive.

Moss No. 3 Mine

The Pittston Company's Moss No. 3 Mine, located in the southwestern corner of Virginia, has been in operation since 1958, and can be classified as a thick-seam operation. In an area where the Jawbone and Tiller seams come together, the coal thickness ranges from 10 to 18 ft (3 to 5.5m). The Tiller part of the seam is of metallurgical quality and the Jawbone part is of steam grade. Utilizing a full-face

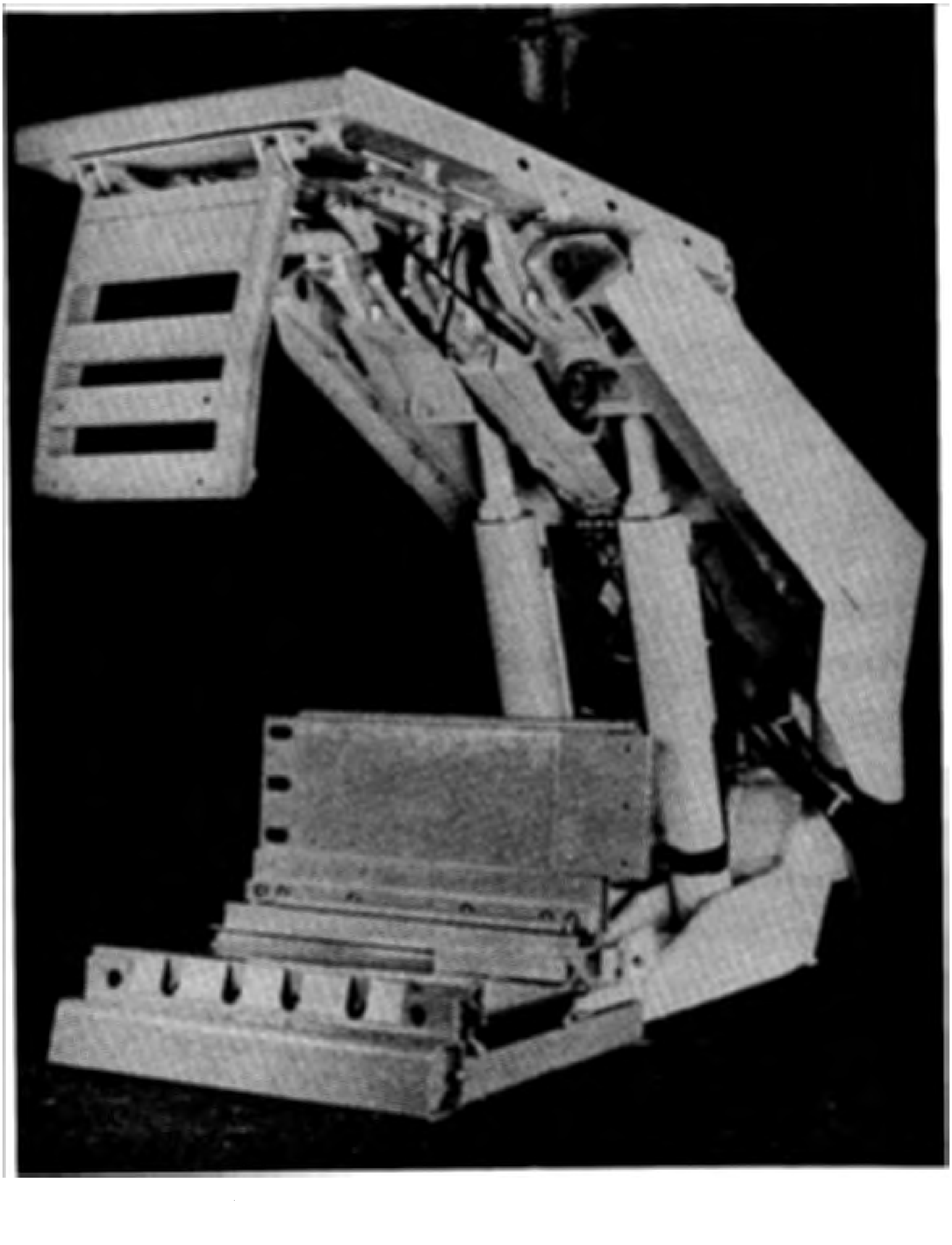


FIG. 42 SHIELD SUPPORT WITH LOWERED FACE GUARD

mining method, annual recovery has been over three million tons (2.72 million metric tons). Efficient and extensive preparation procedures are, therefore, practiced to produce different products. The bulk of the prepared coal is sold to the metallurgical market while the intermediate grade product is sent to the 450,000 kw Clinch River generating station ("A Mine of Tomorrow," 1957).

When the decision was made, in 1958, to mine the coal by the conventional method, the equipment on the market at that time was not designed for this type of application. Among the equipment specially designed for the mine were high-capacity loading machines, shuttle cars with rated capacities in excess of 15 tons (13.6 metric tons), cutting machines with 12-ft (3.6m) bars, and twin-boom coal drills. A hydraulic boom and platform was also developed which enabled the bolter operator and the stoper to secure the roof at these high workings (Figure 43). A timber setter was also developed to aid in handling the roof support material (Figure 44). It handles timber of up to 2 ft (0.6m) in diameter and permits two men to do the work of six. As the desirability of continuous miners became evident,Lee-Norse designed the largest milling-type continuous miner ever manufactured in the United States: a 606LN (Figure 45). This miner has a 10.5-ft (3.2m) head and can mine up to 15 ft (4.5m).

In the mining areas visited, the seam thickness varies between 14 ft (4.2m) and 16 ft (4.8m). The immediate roof consists of five ft (1.5m) of slate, overlain by a poor-quality sandstone. Where the slate runs out, and the sandstone becomes the immediate roof, the roof conditions deteriorate. The depth of workings from the surface varies between 500 ft (152m) and 700 ft (212m).

FIG. 43 THICK-SEAM ROOF BOLTER

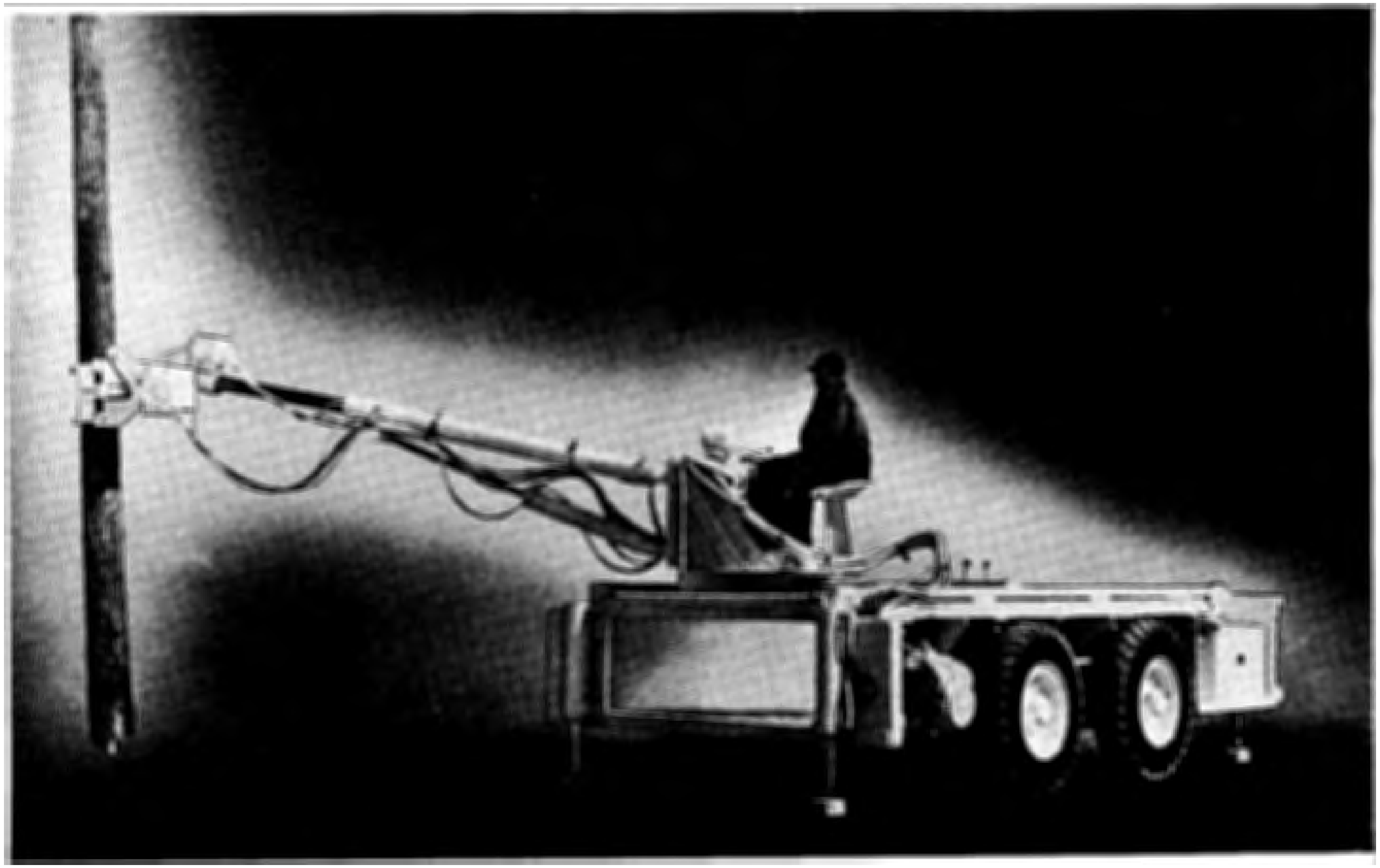


FIG. 44 THICK-SEAM TIMBER SETTER

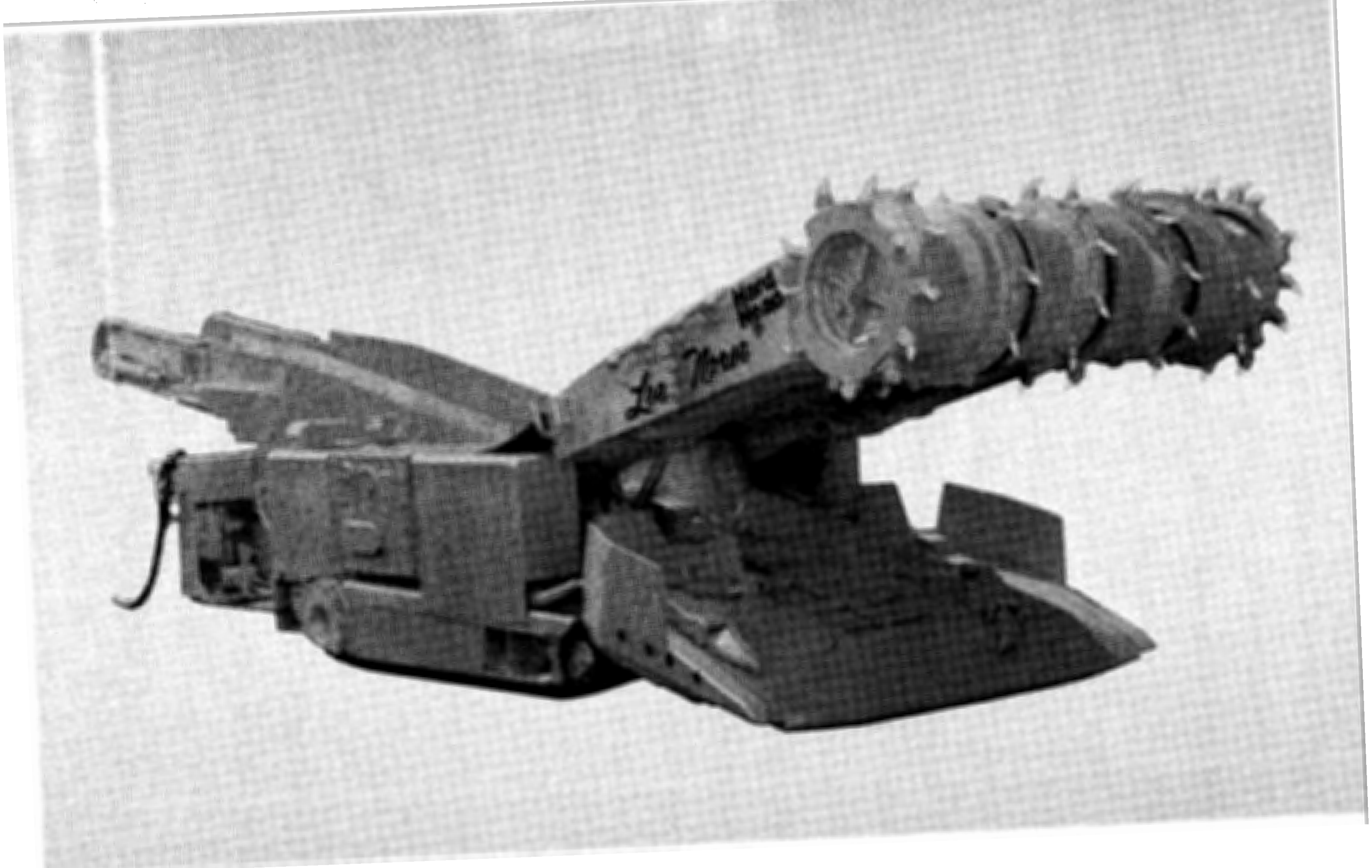


FIG. 45 606LN CONTINUOUS MINER

With a work force of 160 men and three operating sections, the daily production, based on seven machine-shifts, averages 2200 tons (1995 metric tons). Two of the sections are on development and are located at a distance from the portal which requires a long travel time for the crews. This may account for the seemingly low level of productivity when compared to that of earlier years. The one section on retreat averages 600 tons (544 metric tons) per shift.

At the present time, 20-ft-wide (6m-wide) entries are driven on 100-ft (30m) centers. These dimensions were determined by experience. When the entries were spaced on 60-ft (18m) centers, and then later on 80-ft (24m) centers, numerous ground control problems were encountered. With the present mining dimensions, many of the problems are not so severe and are under control. It is felt that total mine recovery would have increased greatly if larger pillars were left on development, in the earlier panels. At present, the retreating sections recover up to 70% of the coal in each panel. This is a very good figure for full-face thick-seam mining.

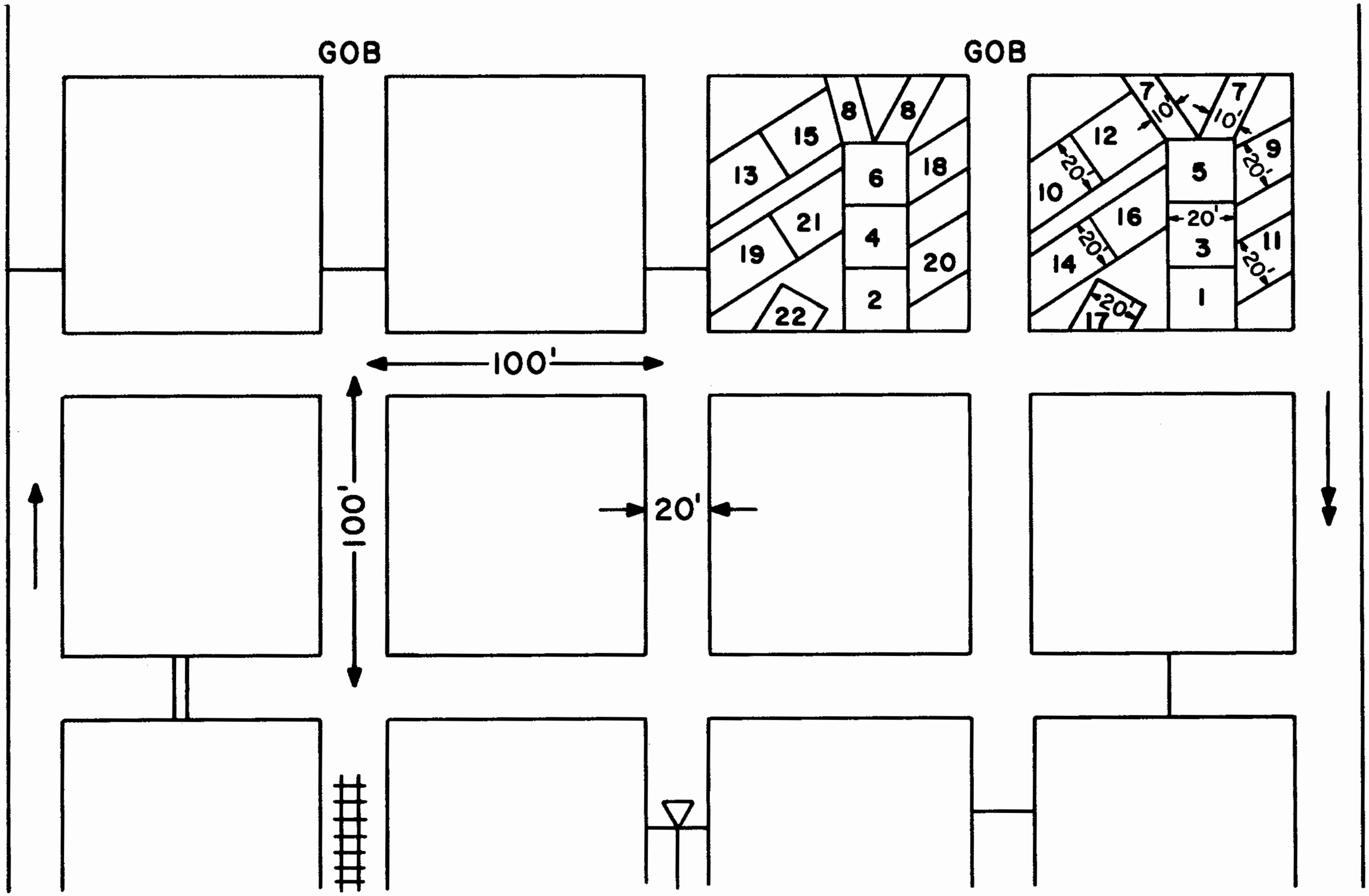
In the retreating section, pillar mining is done with the 606LN continuous miner. Locally, from the top to the bottom, the cross section of the seam consists of 8 ft (2.4m) of Jawbone coal, 1 in. (25.4mm) of rash, and 6 ft (1.8m) of Tiller coal. From left to right, the entries are designated intake, track, beltline, and two returns. Face haulage is provided by two Joy 15SC shuttle cars. A 36-in.(914m) panel belt is used to discharge the mined product onto the 48-in.(1219m) mainline belt. Roof support is established with 5-ft (1.5m) bolts set on 5-ft centers. Where necessary, additional support is provided by timber props. The production crew consisted of the following workers:

Category	Men
Continuous Miner Operator	1
Miner Helper	1
Section Mechanic	1
Shuttle Car Operators	2
Roof Bolters	2
Timber Car Operator	1
Timber Car Helper	1
Utilityman	1
Section Supervisor	1

During retreat, two pillars are extracted at the same time. This is dictated by the location of the return entries (Figure 46). When the returns are located on the right side of the panel, the pillaring cycle begins with the two pillars on that side. The first cut in each pillar is taken such that a fender of less than 20 ft (6m) is on the return side of the pillar. The first three cuts of each pillar are driven 20 ft (6m) wide. Following the three cuts,two cuts are taken 10.5 ft (3.2m) wide in a V-shaped pattern at the gobside end of the pillar. These cuts are not bolted, as well as the two cuts driven to the return through the fender. Other cuts are taken only where the coal can be safely extracted with good ground control. Proper posting is followed throughout. If the returns are located on the opposite side of the panel, the procedure is merely reversed.

Summary

The field trips were very instructive in gaining an insight into the performance of the single-entry longwalls, the use of shields for



1 Foot = 0.3048 Meters

FIG. 46 PILLARING PLAN, MOSS NO. 3 MINE

roof support in longwall faces, hydraulic mining and transport of coal, benching in thick coal with continuous miners,and the use of slushers in room-and-pillar extraction. It was also very useful in identifying several important operational constraints of the current methods and equipment for thick-seam mining. In the next chapter, therefore, the applicability and limitations of the equipment are discussed in great detail.

V. EQUIPMENT, GROUND CONTROL, AND SAFETY CONSIDERATIONS

Equipment

It is necessary to evaluate the merits, as well as the limitations, of the mining and haulage equipment presently on the market for use in thick seams. The conditions in thick seams may dictate a closer examination of machines that do not dominate the industry today. It has already been emphasized in this thesis that, for gains in the immediate future, existing equipment must be adapted for thick-seam operations. A list of some of the currently available equipment with their ranges is presented in Table 6. This information is provided only to illustrate that there is equipment that may find application in thick-seam mining and, therefore, is not all encompassing.

Application of continuous miners, in seams of thin to medium height, have been quite successful in the United States. The inherent versatility of continuous miners was responsible for the increased productivity that helped the industry through the severe decline in demand and the increased competition from oil and gas during the 1950's. However, during the last five years, productivity increases have not only been halted, but have taken a sharp downward trend, from a peak of 15.6 tons (14.1 metric tons) per manshift in 1969, to 11.3 tons (10.2 metric tons) per manshift in 1974. Despite the success of longwall mining, and its great potential, only four percent of the nation's annual deep production comes from longwalls.

In the area of face transportation, shuttle cars have met with overwhelming approval in eastern coalfields, despite their intermittent style of operation. Shuttle cars may not be applicable in seams which

Table 6. Thick-seam Equipment, Manufacturers, Models, and Operating Ranges

Equipment	Manufacturers	Models	Operating Ranges
Boom-type Mining Machines	Alpine Equip. Corp.	AM50	Cutting Range: 12.0 ft
	Anderson Mavor, Ltd.	Boom Miner	Cutting Range: 11.5 ft
		Roadheader	Cutting Range: 18.0 ft
	The Dosco Corp.	Mk2A	Cutting Range: 12.0 ft
Shields	Hemscheidt America	G500-25/50	Support Height: 16.4 ft
	Klockner-Ferromatik	Type IV	Support Height: 14.8 ft
	Westfalia-Luenen	WS1.7,BS2.1	Support Height: 17.0 ft
Shearers	Eickhoff-National Mining Co.	EDW340-LH	Cutting Range: 16.4 ft
	Sagem	Super DTS-300(600)	Cutting Height: 16.4 ft
Continuous Ripper Miner	Joy Manufacturing Co.	1CM-3	Cutting Range: 10.0 ft
Continuous Milling Miners	Lee-Norse Co.	HH 606	Cutting Range: 14.0 ft
		CM 60H	Cutting Range: 12.6 ft
Extensible Belts	Lee-Norse Co.	MC 36 Drive Storage Unit	
		TC 36 Tram Car	
	Long-Airdox Co.	Full Dimension System	

1 ft = 0.3048m

pitch at an angle greater than 12°. As an alternative to shuttle cars, extensible conveyors have been on the market for nearly two decades but, like longwalls, have not received much support from the coal industry. Therefore, a discussion of each equipment category is provided, with an evaluation of their possible application to thick-seam mining.

Boom-Type Mining Machines

Boom-type mining machines will probably find acceptance for the development of roadways in thick seams (Figure 47). This judgement is based primarily on the conditions prevalent in many of the seams. Where seams are split by rock bands, a boom-type miner is quite capable of cutting rock with an unconfined compressive strength of up to 18,000 psi ($12,411 \times 10^4 Nm^{-2}$) (Auer, 1976). Also, the seams with greater thicknesses could experience heavy ground movement during second mining. If yieldable arches are used to control the roof and ribs, a boom-type miner is ideally suited for developing the arched entry. These machines are capable of working on gradients up to 19° (Auer, 1976).

Although boom-type mining machines have been marketed in Europe for over 20 years, the first application in the United States did not occur until 1969. At present, there are over 50 units in operation in this country and almost all of them have a ripper-type cutting head (Kogelmann, 1974).

The flexibility of these machines is due to their relatively compact design. These machines are slightly over 5 ft (1.5m) in height and usually less than 10 ft (3m) in width. A boom-type miner can drive a place 12 ft (3.6m) high and 15 ft (4.5m) wide without repositioning.

FIG. 47 BOOM-TYPE MINING MACHINE

Shields

Shield-type roof supports are a recent improvement over the chock-type supports found at most longwall sections. Shield supports were first applied in eastern Europe over 15 years ago. However, these supports were more like sealing units between the face and the gob (Vorobjev and Deshmukh, 1966), rather than roof support units. Mechanized shields did not appear in western Europe until 1970, and they finally came to the United States in 1975 (Welzel, 1975). The major advantages of shields are their ability to operate under less than favorable conditions and their extended range. Their design creates less upsetting moment in thick seams, when compared to chocks.

The shields, designed for 15-ft-high (4.5m high) longwall faces, consist of a floor beam, a waste shield, a caving shield, a roof shield, and hydraulic props (Figure 48). The floor beam is quite resistant to bending. Because of the large contact area of the shield with the floor, it has increased effectiveness in soft bottom. The waste shield covers the joint between the caving shield and the floor beam. The caving shield provides skin-to-skin protection between adjacent shields against gob flushing. Thus, complete cover is possible in the face area, even during the advance of individual supports. The roof shield, linked to the caving shield, completely covers the working area. It is usually fitted with a face guard which can be lowered to prevent spalling of the upper part of the face and potentially clogging the conveyor system (Barnard, 1976). There are usually two hydraulic props in each unit, with a yield load of approximately 550 tons (499 metric tons) (Simpson, 1976). Most shields marketed today are set on 5-ft (1.5m) centers.

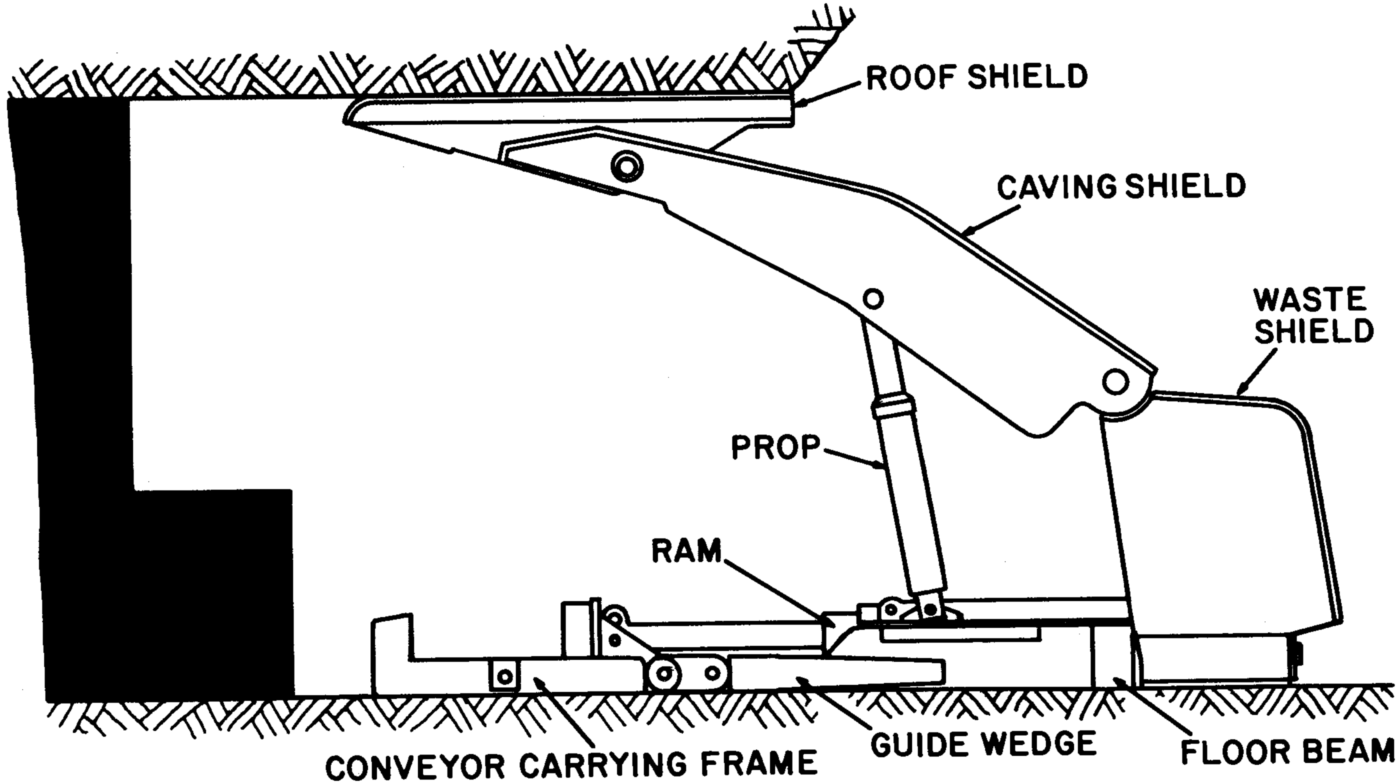


FIG. 48 SHIELD COMPONENTS

There are many design aspects of shield supports that make them superior to chock supports. Since a shield is basically a three-joint unit, collapse of the unit must be accompanied by deformation. Conversely, the four-jointed design of chocks makes them susceptable to collapse without deformation. Kinematically, therefore, shields are much more stable than chocks. The skin-to-skin design of shields not only protects the workers from falling rock, but also keeps dust generated due to caving from entering the face area. Since most shearers can travel at 20 ft (6m) per minute, shields, due to their simplified hydraulic system, can be advanced at a rate of four units per minute - also 20 ft (6m). Because chocks can only achieve half that rate, shields can help increase production from a longwall panel, without any decrease in safety (Barnard, 1976).

Though a relatively new roof support system, shields may well become an important segment in future longwall installations. Their use for multi-lift longwall panels should gain acceptance over the use of chocks and wire mesh.

Shearers

The trend in European mining of thick coal seams has been directed toward longwall applications with double-drum ranging shearers. Although most slicing and sublevel caving methods employ shearers with a range of 8 ft (2.4m), applications of shearers which range up to 16.4 ft (5m) can be found in the full-face operations of West Germany. This increase in range seems to have been stimulated by the development of thick-seam shield supports to the point that greater thicknesses can now be extracted in one pass whereas, heretofore, chock applications required two passes and the use of wire mesh.

Thick-seam shearers are beginning to appear in the United States. West German equipment is the most noticeable (Barnard, 1976). The shearers are basically 12-ft (3.6m) machines that have been extended to 15 ft (4.5m) with no major change in design (Figure 49). They are also capable of working on a gradient of up to 25°. The drums are usually radio-controlled and can cut to a depth of nearly 3 ft (1m) (Barnard, 1976).

Continuous Miners

Ripper and milling-head continuous miners are flexible enough to be adapted for thick-seam applications. Both machines have potential for the drivage of development entries. As primary recovery machines, however, each will be limited insofar as their ranges are concerned. Borer-type miners will have little or no application.

Although it was the forerunner of today's continuous miners, the ripper still finds application, as was observed in the mine visits. Basically, a ripper's cutting head consists of five multi-bit chains (Stefanko, 1976) (Figure 50). The ripper cuts the coal by taking 42-in.-wide (1067mm-wide) cuts, to a depth of 24 in. (610mm), across the face. Without repositioning the chasis, a ripper can advance an entry up to 17 ft (5.2m) wide and 10 ft (3m) high in 2-ft (610mm) increments.

The two aspects of the ripper miner that increase its flexibility are its stationary chassis with extendable sumping head, which permits the mounting of roof bolters to its sides for concommitant roof support, and the small cut width, which permits drivage of any entry width by varying the number of cuts taken across the face. The latter aspect is important because variable entry widths are necessary for good rib control. If a ripper is used for bench mining, the lower bench can be driven narrower for proper sloping of the ribs.

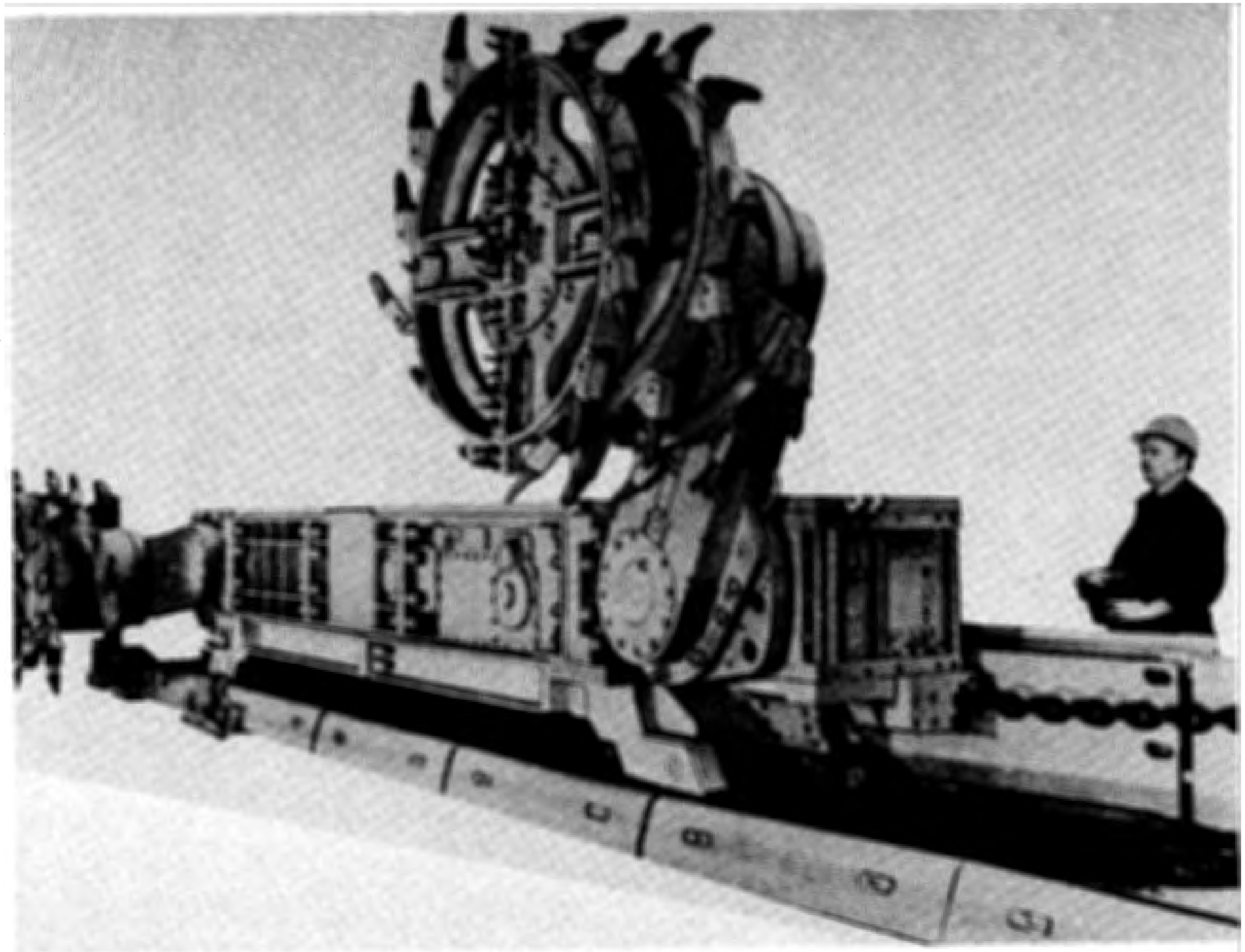


FIG. 49 THICK SEAM SHEARER

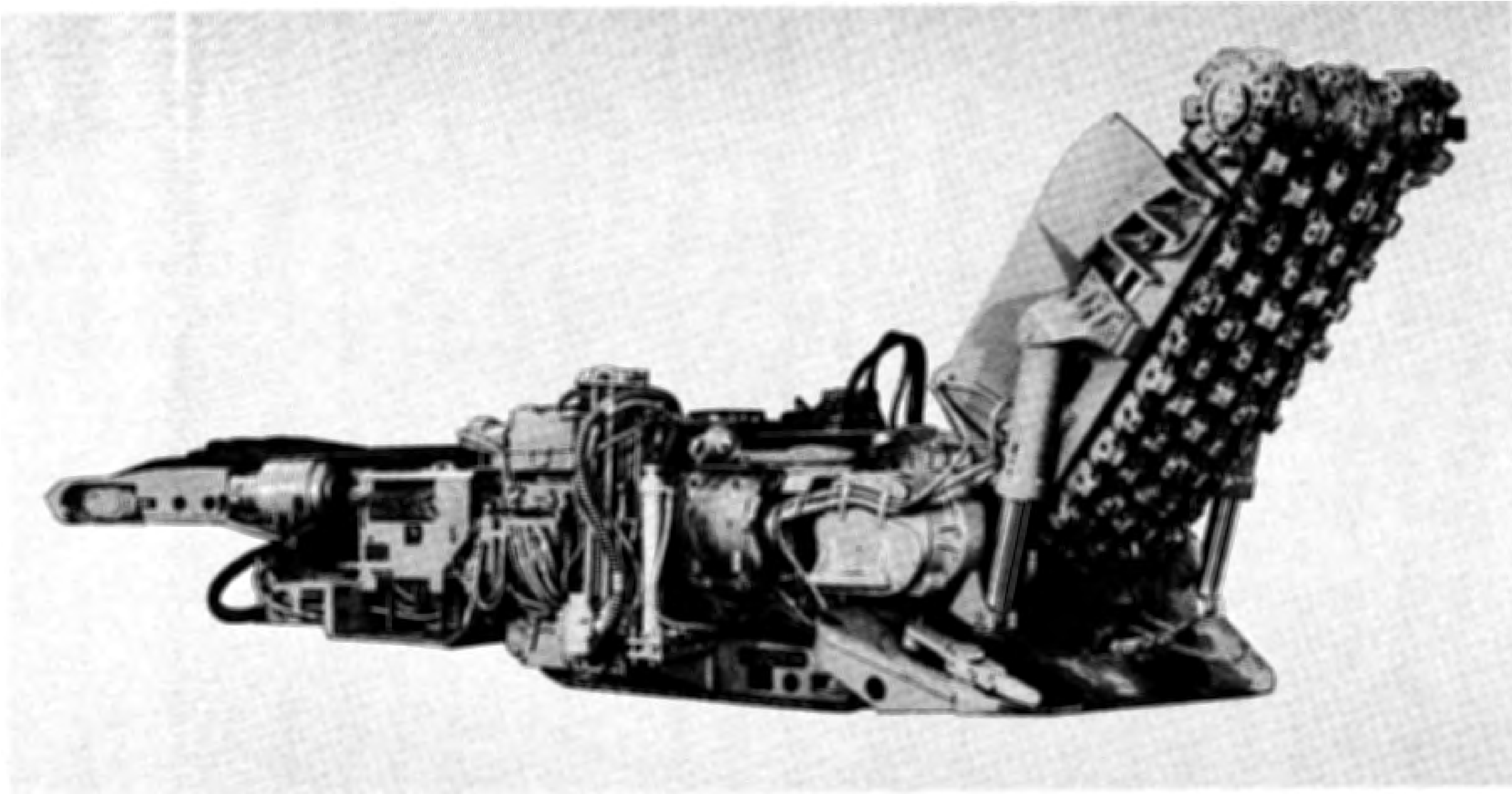


FIG. 50 RIPPER MINER WITH SIDE-MOUNTED ROOF BOLTERS

Milling-type continuous miners, in general, have a greater cutting range [12 ft (3.6m) although it was noted that Moss No. 3 uses a specially built miner with a range of 15 ft (4.5m)] than a ripper, but lack the latter's flexibility in varying entry widths. Since these miners have a fixed head, there are two common cutting widths: 9 ft (2.7m) and 15 ft (4.5m). These miners can also be fitted with side-mounted roof bolters. However, roof bolters mounted on a milling miner can greatly affect the machine's availability, since it has to halt its sumping and shearing actions during the roof support operation.

The U.S. Bureau of Mines has recently awarded several contracts to develop continuous mining systems to improve safety and productivity. One such system is the Automated Extraction System where the extraction operations of a milling miner can function simultaneously with the roof-support operations of side-mounted roof bolters (National Mine Service Company, 1976). Milling miners should also find application in thick-seam mining, for the development of sublevels and longwall gateroads and for the benching of tabular seams which may range up to a thickness of 20 ft (6m).

Seam gradients will, of course, impose some operational limitations on the ripper and milling miners. Though they can negotiate a maximum grade of 20°, their efficiency is seriously affected by tractional problems, particularly on wet floors. Therefore, they are not well applied on grades over 15°. Fitting these miners with side-mounted roof bolters can aid in maintaining better footing and prevent additional degradation of the floor, which can result from frequent place changes.

Shuttle Cars

Of all the commonly used pieces of mining equipment, shuttle cars will be difficult to effectively use in many thick-seam mining methods. The application limitations are related to performance in pitching roadways, intermittent method of operation, and a need for a reduction in the pieces of equipment used in confined working areas.

Performance of shuttle cars in pitching seams is rather poor. Although they are used at pitches of up to 12° (Balmer North Mine), only one car is used behind the miner, thereby limiting the section performance. Their tram rates are not high and, therefore, may not be considered for face haulage at grades more than 12°.

In the development of a sublevel entry or a wide entry for single-entry longwalls, it is better to use some other forms of face haulage. This recommendation is made not only to avoid the intermittent nature of shuttle car haulage but also to avoid, from a safety point of view, as much mobile equipment as possible from confined working places.

Extensible Conveyors

Cognizance of the intermittent nature of shuttle car haulage behind a continuous miner has stimulated the development of a continuous face haulage system. The bridge conveyor was one of the earliest pieces of equipment. It consisted of a long chain conveyor, extending from the panel belt, along which rode a chain-conveyor bridge attached to the boom of a continuous miner (Stefanko, 1976). Since the bridge was approximately 30 ft (9m) in length, this distance could be driven before new sections of pan and chain had to be added to the chain conveyor. Crosscuts could also be turned with this type of system; however the maximum depth of penetration was limited to 40 ft (12.1m). Since place changing was restricted by the conveyor, this system was used primarily where roof support was concurrent with mining (Stefanko, 1976).

Extensible belts for face haulage evolved from the bridge conveyor concept. Essentially, the fixed chain conveyor was replaced by a headpiece containing up to 1000 ft (305m) of belting which would extend as the tailpiece, attached to the miner, advanced. Idler stands were installed under the belt as it was extended. This system, though an advancement over the bridge conveyor, had the same limitation of necessary, concurrent roof support (Stefanko, 1976).

For extensible conveyor usage in thick-seam mining, a few design modifications may be necessary. The belt system must be flexible to adapt to minor variations in a mining plan. It must not take much room in the entry; the miner must be able to change to other working places as rapidly as possible. An analysis of the extensible conveyors marketed today shows that there are two basic styles: 1) modular extensible belt systems and 2) bridge carrier systems.

The modular extensible belt systems are a variation of the extensible conveyor belts. However, the belt is modular, as the name indicates, and each module is self-tramming (Figure 51). Several modular units can form one complete face transportation system (Marsh, 1975). A modular unit, when collapsed, is 30 ft (9m) long and can extend up to 150 ft (45.5m). Their extensible distance of 150 ft (45.5m) can provide the flexibility required for different thick-seam mine plans where larger pillars are warranted. When a tram car is sued behind the continuous miner, the entire system replaces the shuttle cars and belt feeder on the section (Figure 52). Also, the belts are capable of functioning on a grade which is greater than the efficient range of a shuttle car (12°) and equivalent to that of a continuous miner (15°).

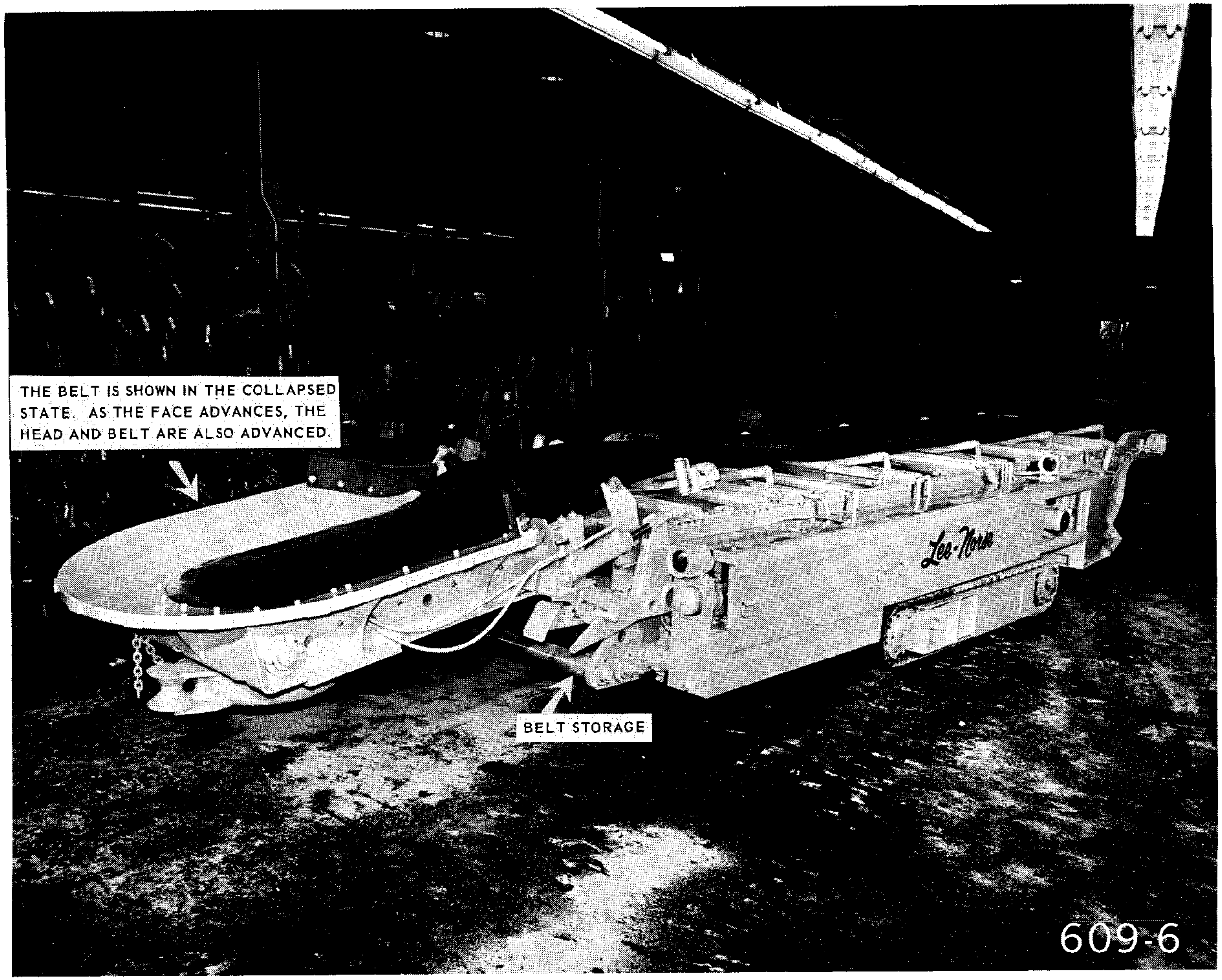


FIG. 51 MODULAR EXTENSIBLE BELT SYSTEM

FIG. 52 TRAM CAR

For a typical section with three belt units, the manpower consists of the following (Marsh, 1975):

Category	Men
Miner Operator	1
Tram Car Operator	1
Miner Helper	1
Utilitymen	3
Bolters	2
Mechanic	1
Supervisor	1

The utilitymen also work with the belts.

Although this system is a relatively new development, it has several aspects that should be advantageous in thick-seam mining. The belts work better than shuttle cars in bad bottom. Where the miner is mounted with roof bolters, the belts will permit faster development. Finally, neither the tram car [8.33 ft (2.54m) wide] nor the modular belt units [7.0 ft (2.1m) wide] take up much space in the entries (Marsh, 1975).

The bridge carriers are the most common form of extensible conveyor system in use today. Basically, these units link the panel belt and the continuous miner with a series of conveyor bridges and carriers (Stefanko, 1976). The tram speed of the bridge carriers can be geared to that of the mining machine. There is only a limited selection of conveyor bridge lengths [34 ft (10.4m), 36 ft (11m), 46 ft (14m), 50 ft (15.2m)]. Because of these lengths and the limited turning radii of the bridge conveyor components, crosscuts may have to be turned at angles of 60° or less (Stefanko, 1976). Therefore, under certain situations where equipment flexibility is not critical, bridge carriers may find applications in thick seams.

Ground Control

The western coal seams, when compared to their eastern counterparts, create many unique problems for ground control in underground mining. This section will discuss the ground control problems encountered in thick-seam mining with a review of some of the preventive actions used worldwide.

Roof and Rib Control

The support of the immediate roof in thick seams is dependent upon the method of extraction. Shields, flexible steel matting, steel beams, conventional wooden props and roof bolts have been used either alone or in combination. Wooden props, though more commonly used, are heavy, difficult to erect vertically, and are labor intensive. Steel props are usually not considered, due to the weight problem and loss. The installation of steel matting with chocks is common where the seam is extracted in slices (Bise and Ramani, 1975).

While the development of the coal in the top section of a thick seam is useful for securing the roof with bolts, special equipment may be required ("A Mine of Tomorrow",1957). The height of the workings creates a serious hazard from rib rolls. Constant vigilance is required to keep the entries properly dressed of overhangs. Development in the top section, here again, can be advantageous. In addition to the bolting of the roof, the sides can be widened and dressed of all overhangs. Then, during the second extraction when the lower bench is mined, the entries can be deepened to a width less than that of the top section. The narrowing of the entries in the bottom section can be so controlled as to provide a favorable slope for the workmen below (Bise and Ramani, 1975).

It has been recommended that, where roof and rib control poses serious safety hazards, the number and width of entires in coal be limited to a minimum, and the majority of the development work be done in the rock (Borecki and Dziumikowski, 1964).

Coal Bumps

Violent instantaneous failures of coal pillars, due to applied stresses, are termed bumps. These are usually accompanied by fragmentation and displacement of large volumes of coal, clouds of dust, high methane emission, and considerable damage to equipment and workings (Holland and Thomas, 1954).

Mountain terrain, steeply dipping beds, massive roof, strong floor, geological disturbances, depth of cover, size of support pillars, degree of recovery, and the nature of packing, if any, all play a major role in the frequency of occurrence and the accompanying violence of bumps (Olsen, 1963).

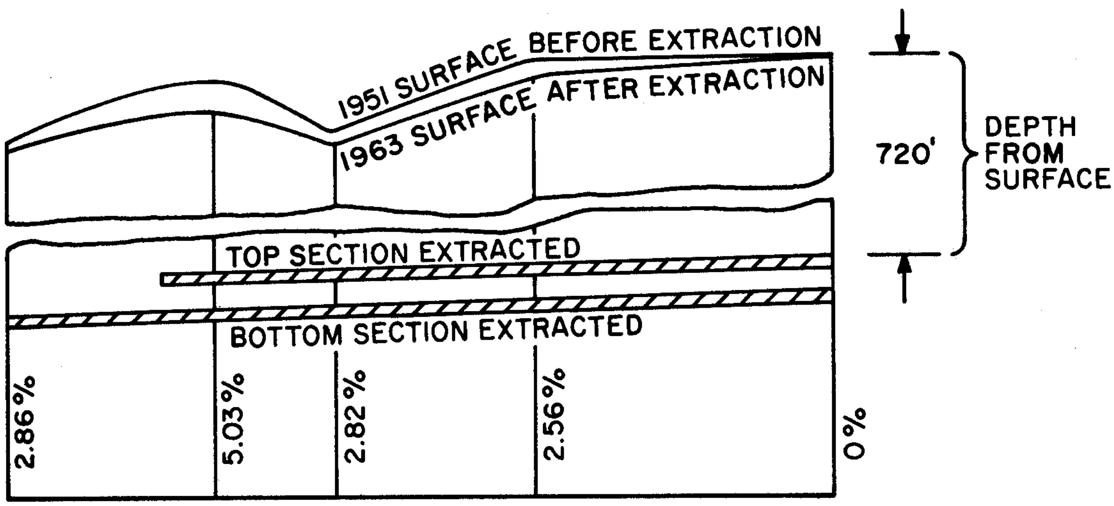
Bump control is mainly achieved through designing a mining system such that excessive stresses and strains do not accumulate or are otherwise relieved. Longwall mining, quick and complete extraction, a small number of entries, complete or adequate packing, and extraction which avoids pillar line points are some of the preventive measures that have been successfully practiced. It has been recommended that, in seams liable to bumps, a slice adjacent to the roof should be taken before extracting slices in the ascending order (Ramlu, 1964). Since the western thick seams are located in mountainous areas where bump conditions have already been encountered (Olsen, 1963 and Peperakis, 1958). provisions for the protection of men and equipment from this hazard should be of particular concern.

Subsidence

Thick-seam extraction has been achieved with full caving, partial mechanical or manual stowing, as well as complete pneumatic or hydraulic packing. Even with full caving methods, fenders or inferior bands are left behind, thereby making subsidence experience, in one instance, not transferable to other sites. Subsidence with hand and mechanical stowing is reported to be in the order of 50% to 70% of the seam thickness. With hydraulic stowing of the gob, a maximum subsidence of 10% to 25% of the seam thickness has been observed (Fritzsche, 1964). Early and tight packing with sand is reported to have minimized subsidence to less than 5% of the extracted thickness (Figure 53). With pneumatic stowing, the packing is not so tight, and the subsidence may go up to 50% of the seam thickness.

Packing of the gobs not only minimizes subsidence, but has favorable overall effects. With complete packing, bump and spontaneous combustion-prone thick seams are more safely extracted with greater conservation of the natural resource. It also aids in roof control and ventilation.

Stowing of underground workings is an ancillary operation requiring experienced planning. Greatest attention to details is necessary for efficient extraction. It is an operation parallel to the production



1 Foot = 0.3048 Meters

FIG. 53 SURFACE SUBSIDENCE WITH STOWING
(Guin and Koshi, 1964)

operation, with its own organization and logistics. The availability of stowing material is important. Also the availability of labor, since the process is labor intensive, is essential. The coordination between the extraction and stowing phases, and the maintenance of the time sequence, is rather critical since production bottlenecks from stowing delays cannot be ruled out. It is necessary to ensure that the packing is tight and up to the roof because openings between the roof and the pack lead to methane accumulations. These openings are also potential heating sites (Bise and Ramani, 1975). A recent study estimated that it would add \$1 to \$4 to the cost of a ton (0.907 metric tons) of coal to stow coal mine waste underground, even when the mining conditions are unusually favorable and an average amount of waste is produced (National Academy of Sciences, 1975). This cost figure for refuse disposal has already been challenged, and a figure of \$5 per ton (.907 metric tons)of coal has been advanced as more representative, even when waste material is readily available (Poundstone, 1974).

Safety Considerations

Besides hazards related to the ground control, there are other safety factors which, although not unique to thick seams, are aggravated by the physical conditions. Among these considerations are spontaneous combustion and ventilation.

Spontaneous Combustion

The danger of underground fires from spontaneous combustion of coal is ever present in many coal mining countries. Fortunately, in the United States, incidents of spontaneous combustion in coal workings have not been widely reported. Thick seams are considered to be more prone to spontaneous combustion. Some of the reasons for the high risks are:

1. Certain sections of thick coal seams contain inferior quality coal or bands, and these are usually left behind.

2. Complete extraction of thick coal seams is not practical. Coal is usually left in the roof, floor, or in fenders for support. In time, these are crushed and accumulate as broken coal in the gob. Exploitation losses, therefore, are usually high with thick coal.

3. Heat accumulation under ambient temperature is the main cause for ignition of coal. The wide and high roadways and gob areas lead to low air velocities which may facilitate such heat accumulations.

4. Low-rank and high-pyritic coals are more liable to spontaneous combustion than high rank coals. Since the inferior quality coals or bands in a thick seam are advantageously used to extract the seam in sections or slices, favorable conditions for spontaneous combustion are created.

It has been suggested that, in seams liable to spontaneous combustion, the development within the coal seam should be kept to a minimum (Ramlu, 1964, and Harris and Walker, 1964). Where the problem is acute, the coal seam is extracted in subpanels, such that the coal in the subpanel is recovered within the incubation period. The incubation period, which is defined as the time between the start of pillaring operations and the first sign of heating, usually varies between 3 and 18 months, depending upon the rank of coal. The longer period is associated with high-ranking coals. Feng, *et al.* (1973) have estimated that, for the mountain coals of Canada, this period is between 9 and 18 months. Obviously, with an efficient extraction method the amount of coal removed during the incubation period can be greatly increased.

Ventilation

In thick-seam workings, particularly during the phase when the whole seam is extracted, the wide and high roadways create conditions which may lead to low air velocities. In gassy seams, methane layering along the roof may result, in addition to the fire hazard from spontaneous combustion.

Summary

In the chapters heretofore, the various aspects of thick-seam mining - methods, equipment, and safety - were reviewed with the objective of identifying methods and equipment for a selected set of conditions. In the next chapter, therefore, methods are proposed for four sets of conditions.

VI. RECOMMENDED MINING METHODS

For thick seams to contribute to the solution of the energy crisis in the short run, adaptation of existing equipment and methods, as opposed to the development of new equipment and methods, can hardly be over-emphasized. As previously mentioned, most equipment is limited in its direct use. It is necessary to design mining layouts which combine the advantages of various pieces of equipment with the natural conditions of thick-seam occurrences to maximize productivity and recovery.

One of the major aspects to be concerned with is ground control. The use of stowing, to ensure good ground control, may not be readily applied due to the cost of personnel. Added to this problem will be the cost of securing and transporting backfill materials. In short, stowing may not allow deep mining to be economically competitive. In the methods that are proposed, therefore, the use of stowing is not recommended.

Roof and rib control have a great influence on the selection of mining methods. Highside ribs in thick seams, such as those found at the Vicary Creek Mine, should be avoided primarily for safety reasons. Therefore, the proposed methods call for development entries at working heights of 10 ft (3m) or less. Where the entries are eventually heightened, the ribs are stepped to provide a favorable workplace. By keeping the entries under 10 ft in height on development, roof control practices are facilitated. The roof is easily accessible for bolting and cribbing. The overhangs are also within reach for dressing. Conventional supports, such as bolts and cribs, are generally recommended and, where heavy ground movement is anticipated, use of yieldable arches is recommended.

Ventilation, also a major consideration for any mining method, is given special attention. Any proposed method should be designed to comply with the ventilation requirements of the Health and Safety Act. However, in the proposed methods situations exist where variations from the provisions of the Act are warranted, and these are discussed.

Mining methods are proposed for the thick-seam conditions shown in Table 7. The underlying reasons for selecting these varied conditions for methods development is that these conditions represent those that will be attacked in the near future. In fact, mines are presently operating under some of these seam conditions. Mining methods for panel recovery are proposed for seams with these specified conditions, within the constraints of available technology.

Table 7. Conditions for the Proposed Methods

	Upper Limits	
Recommended Methods	Thickness	Dip
Method A	20 ft (6m)	15^0
Method B	30 ft (9m)	0^0
Method C	20 ft (6m)	45^0
Method D	50 ft (15m)	45^0

Method A

Seams which are 20 ft (6m) thick and pitching up to 15° can be found in the coalfields of southern Utah, southern Wyoming, and western Colorado. Use of shuttle cars in seams which pitch at, or more than, 12° is not particularly suited for high production. Though continuous miners can negotiate this pitch, a layout with excessive place changes is also not particularly suitable. To limit the number of place changes, the mining machine could be fitted with side-mounted roof bolters, permitting it to advance more than 20 ft (6m) in one place. Preferably, ripper miners should be employed for this purpose. Mounting roof bolters on milling miners creates a problem because the penetration of the machine must be halted while the roof is bolted. On the other hand, a ripper miner's main chassis is stationary during mining, thus concomitant bolting is possible. Also, rippers are more flexible in varying the entry widths. Since machine cutting is unhindered by bolting with bolters mounted on rippers, continuous haulage systems could permit rapid face advance. Unlike shuttle cars, many of the extensible conveyor systems marketed today can operate on this pitch. From the roof and rib control standpoint, the initial development should be in the top, rather than in the bottom, of the seam.

Development

On advance, four 17.5-ft.-wide (5.3m-wide) entries are driven 2000 ft (606m) along the strike on 90-ft (27.3m) centers with breakthroughs every 105 ft (31.8m) (Figure 54). These dimensions have been chosen because this width is mineable without repositioning the ripper. Further, the maximum distance allowed by law for breakthrough centers is 125 ft (37.9m) and the full width of the panel is within reach of two modular belt units. The entries, which are 10 ft (3m)high, are

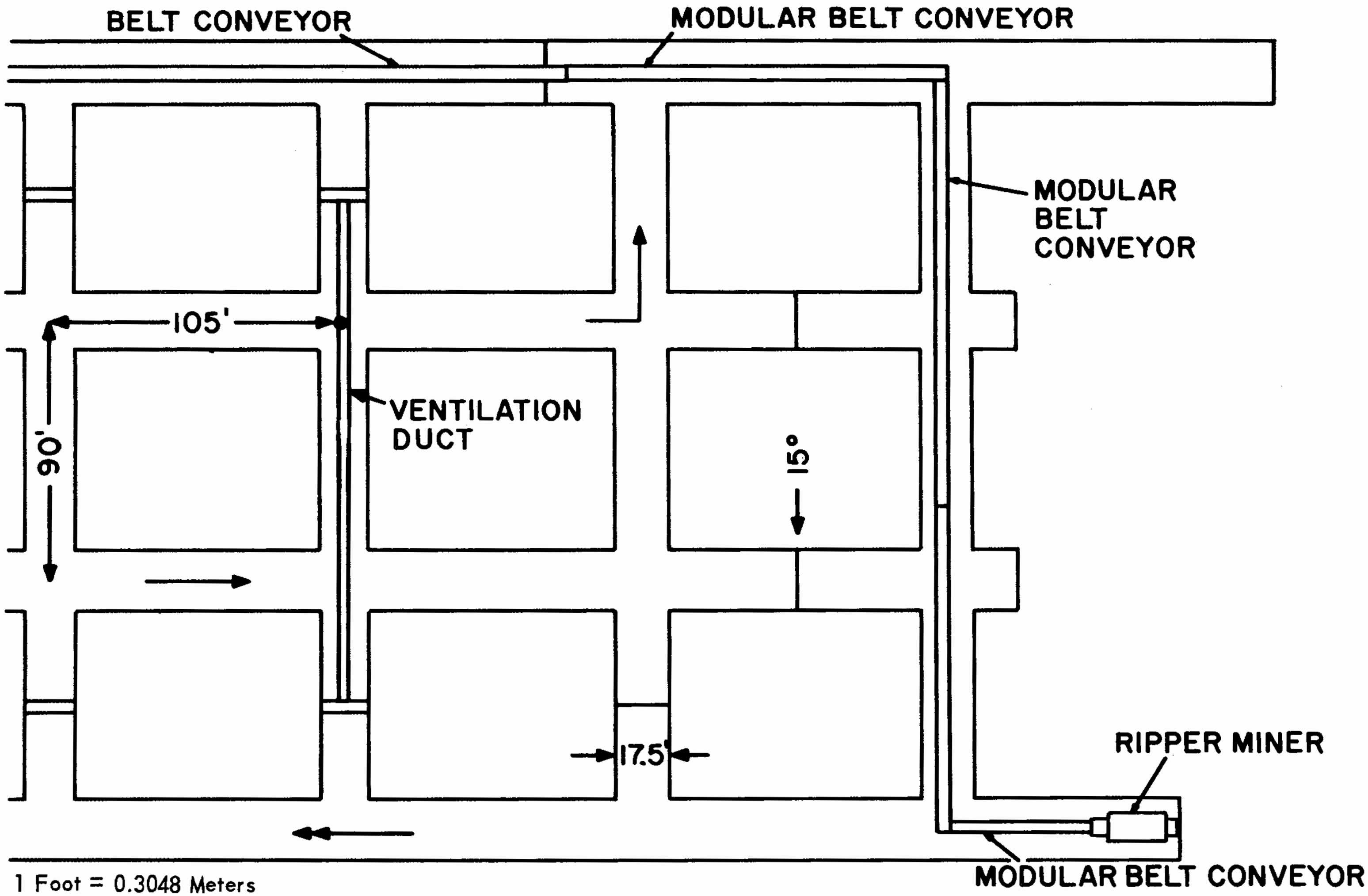


FIG. 54 PLAN VIEW OF DEVELOPMENT, METHOD A

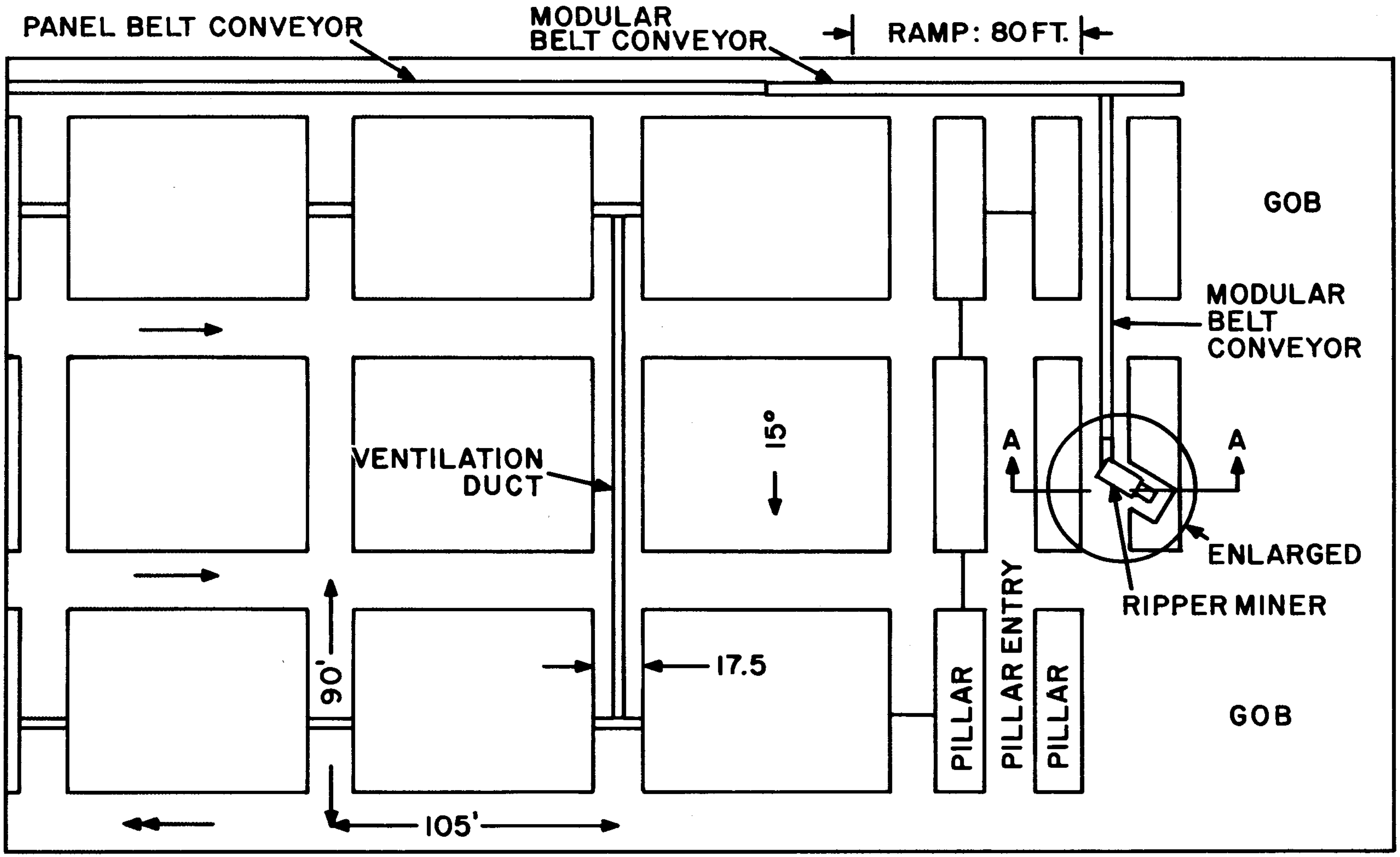
driven in the top portion of the seam. By placing the beltline in the highest outer entry, the use of the extensible belt conveyors for the drivage of the breakthroughs is facilitated.

Pillaring

On retreat, a row of pillars is split by two entries, also 10 ft (3m) high, driven all the way from the belt entry to the outer return entry. These pillar entries are also in the top part of the seam (Figure 55). When this is completed, the miner is backed out along the belt entry to a point approximately 15 ft (4.5m) outby the last breakthrough. The miner then proceeds to ramp down into the lower 10 ft (3m) of the floor coal. This ramp, driven at a grade of 7°, will reach the floor at the start of the inby pillar entry. The bench driven into the floor coal is 14 ft (4.2m) wide, for better rib control. The entire operation is facilitated because the roof is adequately bolted.

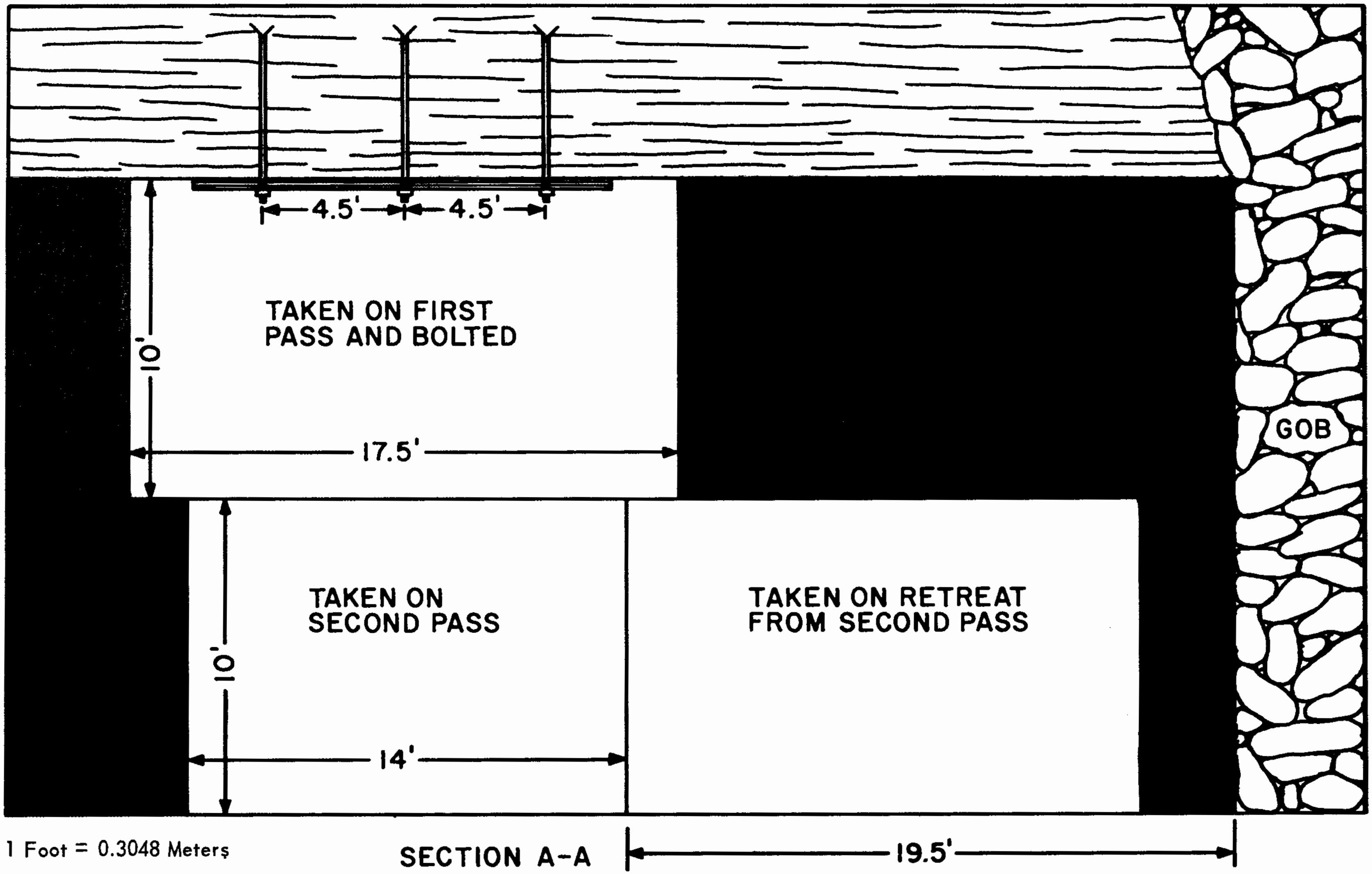
The floor coal of the inby pillar entry is then extracted for the width of the panel. When retreating out of the pillar entries, the inby pillar splits can be recovered (Figure 56). For each pillar split, there are three options: 1) take only two cuts out of the bottom coal and leave the roof coal, 2) take two cuts out of the bottom coal, and attempt to recover some of the roof coal above one cut by blasting it down and using the miner as a loading machine, or 3) take two cuts out of the bottom coal and attempt to recover some of the roof coal above both cuts by blasting it down and using the miner as a loading machine.

For this analysis, the first alternative is chosen for three reasons. Panel recovery is only improved by approximately 5% when four cuts are taken instead of two cuts. Blasting of roof coal, from a safety standpoint, is hazardous. Finally, halting the miner to go



1 Foot = 0.3048 Meters

FIG. 55 PLAN VIEW OF RETREAT, METHOD A



1 Foot = 0.3048 Meters

SECTION A-A

FIG. 56 ENLARGED FRONT VIEW OF PILLARING, METHOD A through the time-consuming process of blasting the roof coal will affect the rapid recovery of the coal. However, if ripper miners with operating ranges greater than those presently available are developed, blasting of roof coal could be eliminated from consideration. Thus, two cuts, driven at a 60° angle for a width of 14 ft (4.2m), are taken out of each pillar split (Figure 57). This operation is advantageous because the coal can be recovered without the installation of more roof support.

After the cuts are taken in this pillar entry, the same sequence can be followed in the other pillar entry, and then in the outby break-through.

Roof Support

Roof support in the upper bench is provided by rows of mats, with three 6-ft (1.8m) bolts per row, installed every 4 ft (1.2m). The side-mounted bolters on the ripper miner can install the outer two bolts per row, while a stoper can be used to install the center bolt at a later time. This form of roof support has been used successfully in the eastern United States.

Ventilation

A conventional ventilation plan, using brattice cloth for coursing fresh air through the section and tubing used in conjunction with a 25-hp self-propelled auxillary fan for face ventilation, is envisioned. Both rows of stoppings are constructed of galvanized metal panels which are sprayed to give an air-tight seal (Figure 58 and 59). Flexible tubing is used to direct any beltline air directly into the return at the last set of stoppings. When the panel is driven 2000 ft (606m), a bleeder system is established to permit the pillaring on retreat.

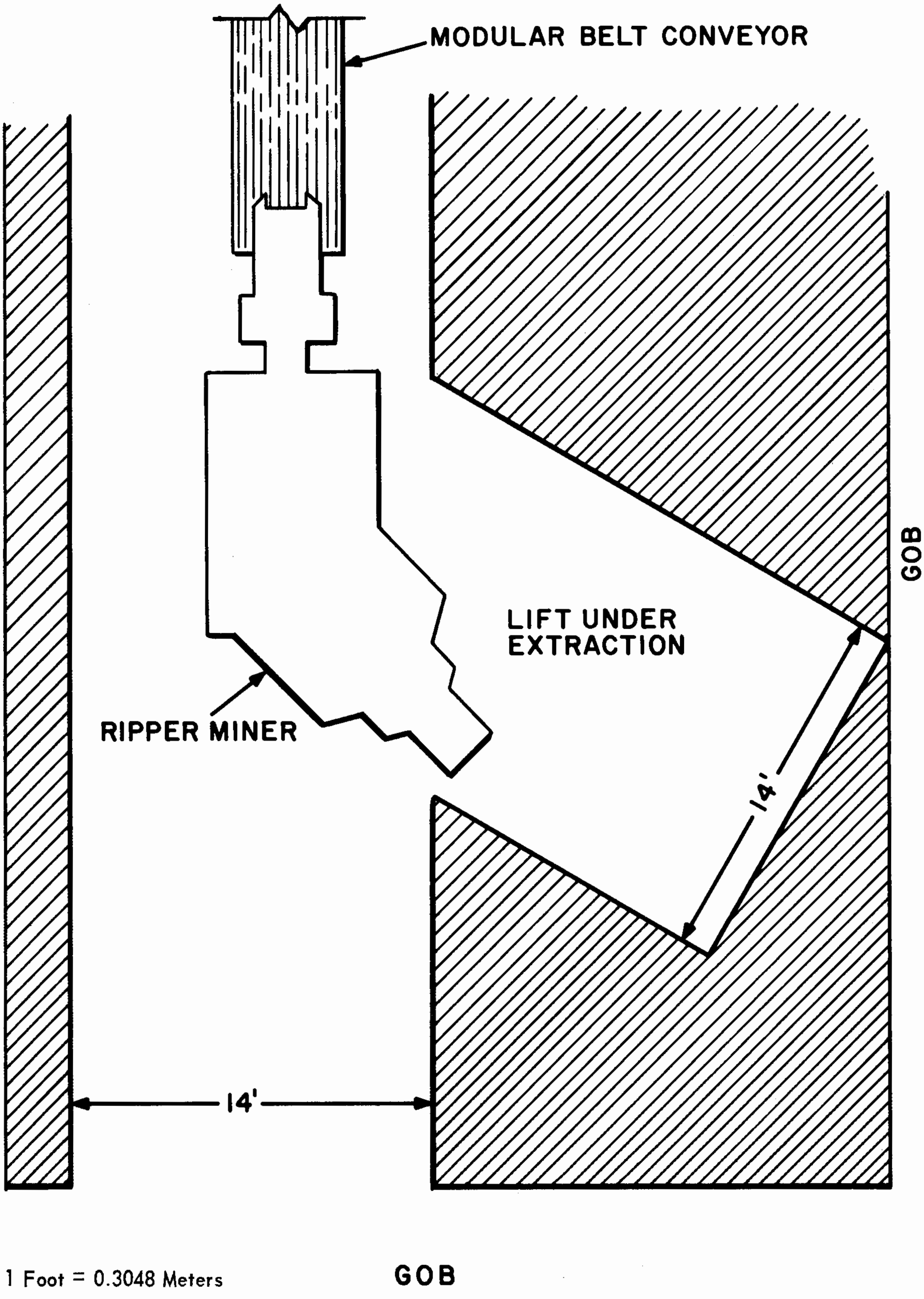


FIG. 57 ENLARGED PLAN VIEW OF PILLARING, METHOD A

FIG. 58 INSTALLATION OF METAL STOPPING PANELS

FIG. 59 SPRAYING A STOPPING WITH SEALANT

Equipment

The following capital equipment will be required for the mining of one panel:

Equipment	Units
Ripper Miner with Side-mounted Roof Bolters	1
Complete Stoper Unit	1
Extensible Belt System	1
2000-ft (610m) 36-in.(914mm) Beltline	1
2000-ft (610m) 7.2 kv Cable	1
2000-ft (610m) 2-in (51mm) Waterline	1
750-kva Load Center	1
25-hp Auxillary Fan	1
Sealant Spray Unit	1

Manpower

The following face personnel will be required:

Category	Men
Miner Operator	1
Roof Bolters	2
Utilitymen	4
Section Mechanic	1
Section Supervisor	1

The utilitymen also take charge of the extensible belt system.

Production Calculations

During development, 87,000 tons (78,909 metric tons) of coal can be extracted. Assuming a production rate of 600 tons (544 metric tons) per machine shift, it would take 145 shifts to develop the section to its projected limits.

Similarly, on retreat, 211,000 tons (191,377 metric tons) of coal can be extracted. This figure includes the pillar entries, and the two cuts in each pillar split. At a rate of 800 tons (726 metric tons) per machine shift (33% increase over development), it would take 265 shifts to retreat out of the section.

In all, the total recovery for the panel is 65%.

Health and Safety

The mining methods, as described, should not need any variance from the provisions of the Coal Mine Health and Safety Act. The roof control plan is straightforward and the ribs have been designed to provide a safe workplace. However, noncompliance of the ventilation provisions of the Health and Safety Act is an ever present possibility in the high and wide entries of this example. These entries could very easily deliver the quantity of air required for face ventilation, but the velocity may not, in itself, satisfy the conditions encountered. For example, the Health and Safety Act stipulates that the minimum mean entry air velocity shall be 60 feet per minute (18.3 meters per minute). To deliver 9000 cubic feet per minute (4248×10^3 cubic centimeters per second), the minimum quantity allowed by law at the last open breakthrough, a velocity of 103 feet per minute (31.4 meters per minute) is required in an entry 5 ft (1.5m) high and 17.5 ft (5.3m) wide. To deliver the same quantity, when the entry height in this example is increased, a velocity of only 29 feet per minute (8.8 meters per minute) is required. Clearly, this figure is not in keeping with the requirements of the law. Thus, it should be mentioned in passing that the normal ventilation requirements for a panel in a moderately thick seam may not have enough velocity to properly ventilate a thick-seam panel.

Although the miner operator in this example is always under supported roof, it is further recommended that the miner should be operable by a remote control unit during pillaring. The coal in the pillar splits may undergo crushing during second mining and the removal of the operator from the face area should enhance the safety of the system.

Method B

Occurrences of 30-ft-thick (9m-thick) tabular seams are quite common in the West. For example, production from the Kaiparowits field is expected to be from seams of this nature. Although a tabular seam is favorable for the application of continuous miners, recovery in seams in excess of 30 ft (9m) in thickness is rather low. The intergrated method, as practiced in France, may be applicable; however the mining costs and productivity, required for applications in the United States, must be lower and higher, respectively, than those achieved in France. If face advance is not rapid, due to the non-productive operations associated with the integrated method (e.g. wire mesh) conditions for spontaneous combustion are created. Longwall slicing has considerable promise for application, but will require modifications to eliminate some of the labor-intensive and costly operational features described before. Therefore, a longwall slicing method, which will attempt to avoid some of these limitations, is proposed.

As an alternative to the installation of wire mesh, a layer of coal can be left between slices to achieve the same purpose. The thickness of the layer will have to be determined for individual application. In the U.S.S.R., for example, the use of a layer of coal between the slices, varying in thickness between 1 ft (0.3m) and 2 ft (0.6m), is common (United Nations, 1968). Obviously, the lost coal will affect the recovery of the in situ reserves, and can create favorable conditions for spontaneous combustion. However, the rapidity with which the faces can advance should offset these shortcomings.

The location of the gateroads for each slice is important. There are three methods of locating the lower gateroads: inside, outside, or directly below the upper gateroads. In several practices abroad, the lower entries are either within or directly below the upper entries. These may also be the most acceptable options from rock mechanics considerations. However, in most these practices, the interval between the slices is much larger than that envisioned in the present application. With the small thickness of the parting between the lifts, the ensuing caves, as the upper lift is extracted, may cause the floor coal to crack. Since the floor coal is also designated as the immediate roof for the lower lift, this could make the maintenance of the lower-lift gateroads very difficult. On the other hand, if the lower gateroads are placed wider apart than those in the upper lift, they may be subjected to the abutment pressures caused by the extraction of the upper lift. Improved roof conditions may be achieved by driving the lower entries on centers that place them beyond the zone of abutment pressures. Particularly, single-entry longwall development in the top and bottom slices is preferable to multiple-entry longwalls. In the latter case, too much coal will be lost, not only between the entries but also in the barriers to be left between the top and bottom longwalls.

The final major consideration is the sequencing of the faces. In each panel, the extraction of the upper lift can be fully completed before the extraction of the lower lift; i.e., allow the subsidence to take place completely before returning for the second slice. It is also possible to simultaneously retreat the faces. In the latter case, the face of the lower slice should not be located within the active caving zone of the upper slice or in the vicinity of the rear abutment; i.e., the advance interval between the two faces must be carefully determined, and maintained.

Development of the Upper Slice

The coal seam is extracted in two slices (Figures 60 and 61). Initially, a headgate and a tailgate are driven on 320-ft (97m) centers (A, Figure 60) in the upper 13 ft (3.9m) of the seam for a distance of 4200 ft (1273m). Each gateroad is 28 ft (8.5m) wide and 9 ft (2.7m) high, leaving four ft (1.2m) of roof coal. The entries are partitioned into a 15 ft wide (4.5m wide) intake and a 10-ft-wide (3m-wide) belt-return by 3-ft (1m) cribs, installed every 8 ft (2.4m), upon which are attached sealant-sprayed galvanized metal panels (Figures 62 and 63). Development is by a continuous miner with side-mounted roof bolters while face haulage is provided by a modular belt system (Figure 64). The miner, which has a 15.5-ft (4.5m) head, drives the face in a two-step pattern.

Recovery of the Upper Slice

After the top bench is developed and the bleeder is established, the entire 13-ft (3.9m) thickness of the 292-ft (88.5m) longwall face is mined with a double-drum shearer and shield roof supports.

Development of the Lower Slice

The development of the lower slice, proceeding simultaneously with that of the upper slice, is also by the single-entry system. As already mentioned, the lower gateroads are driven in an area where good

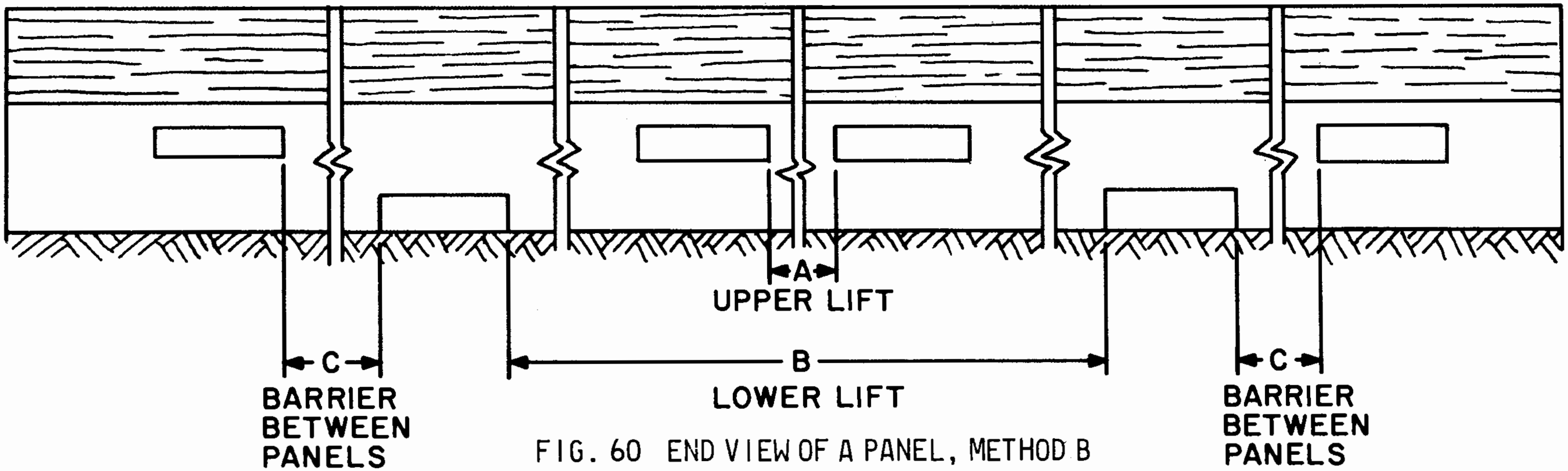
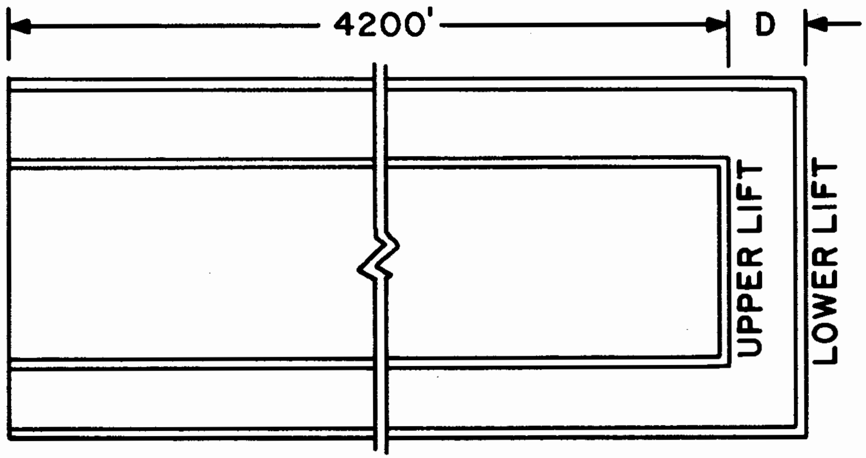
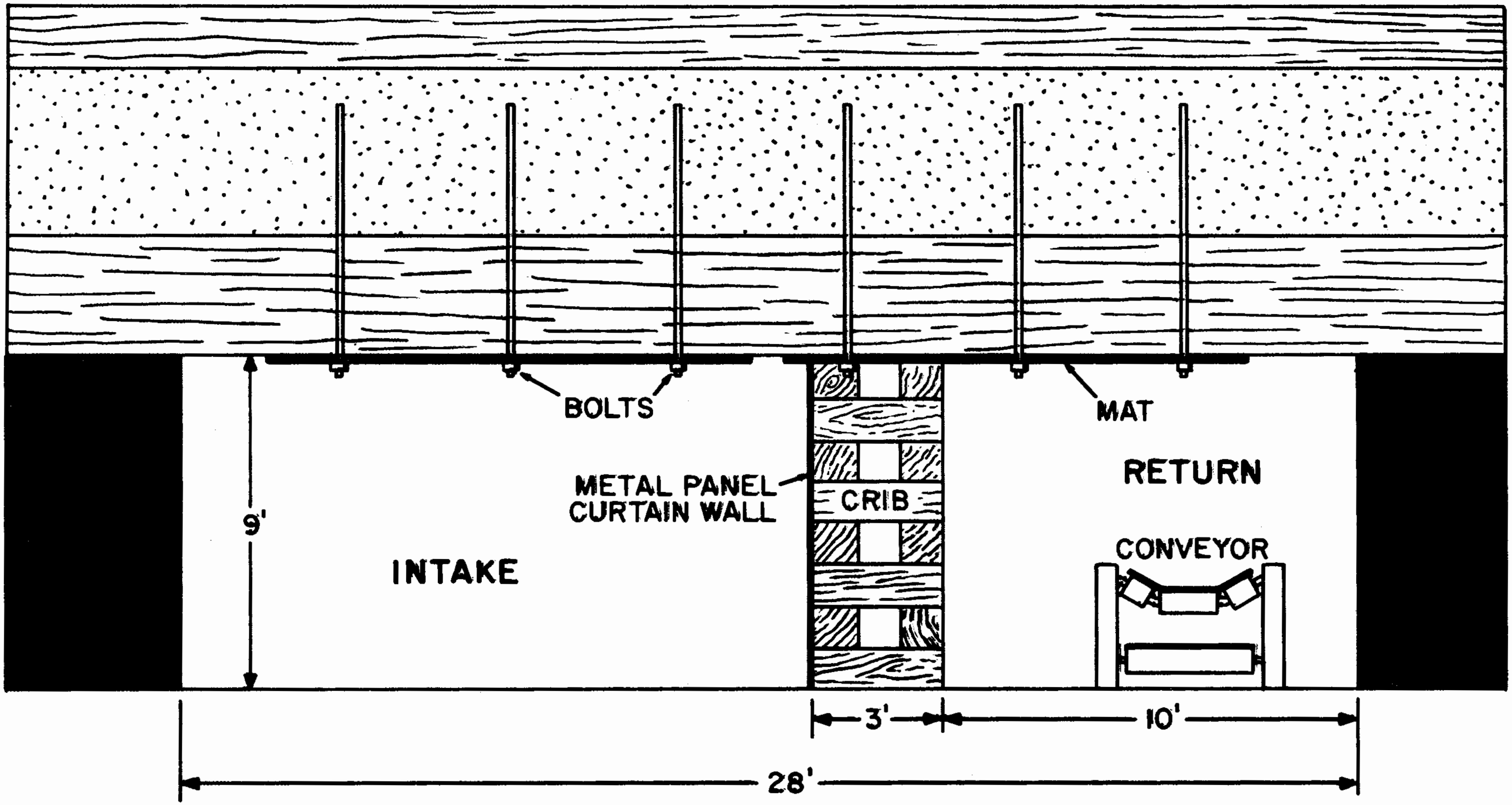


FIG. 60 END VIEW OF A PANEL, METHOD B



1 Foot = 0.3048 Meters

FIG. 61 PLAN VIEW OF A PANEL, METHOD B



1 Foot = 0.3048 Meters

FIG. 62 END VIEW OF SINGLE ENTRY DEVELOPMENT, METHOD B

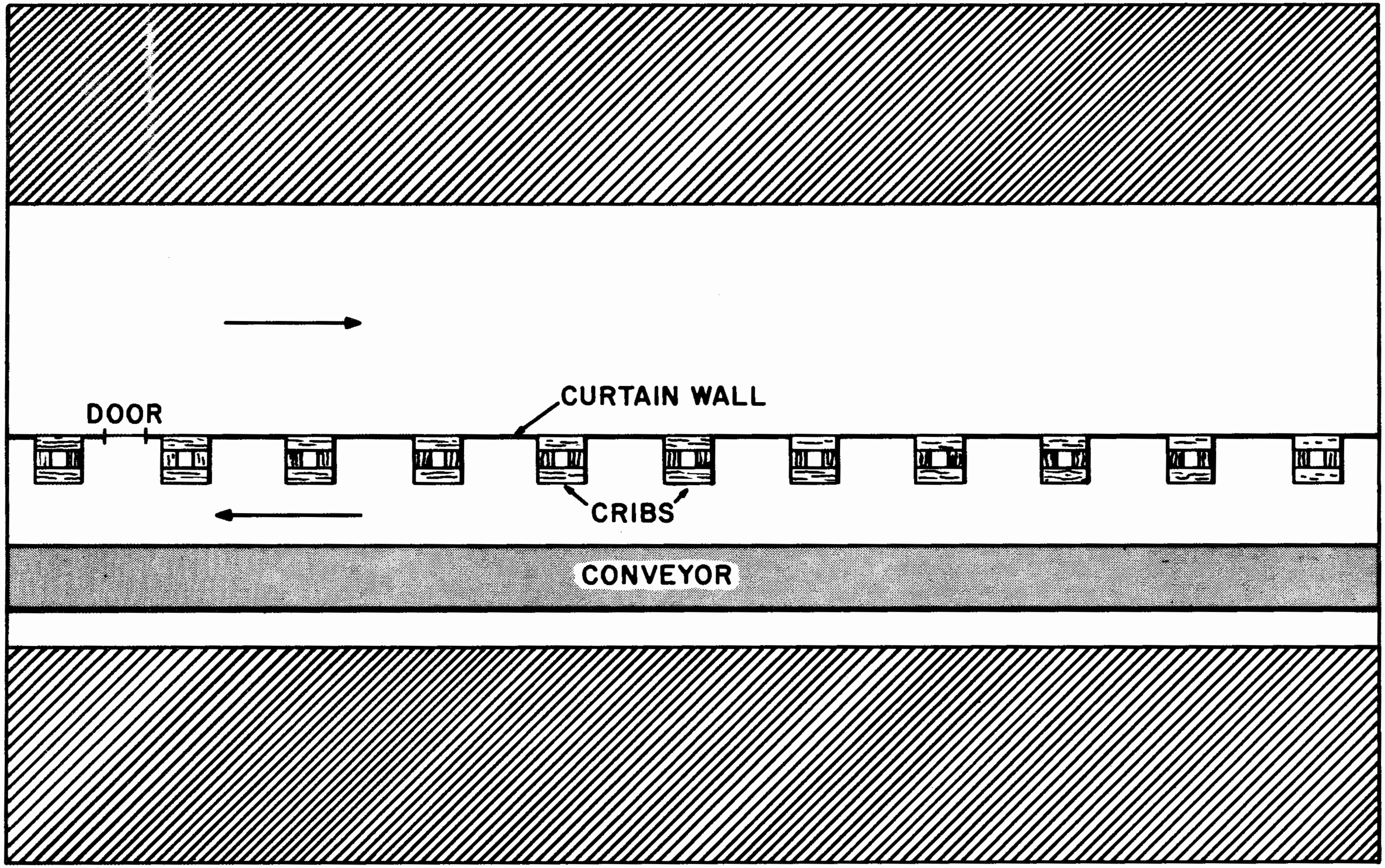


FIG. 63 PLAN VIEW OF SINGLE ENTRY DEVELOPMENT, METHOD B

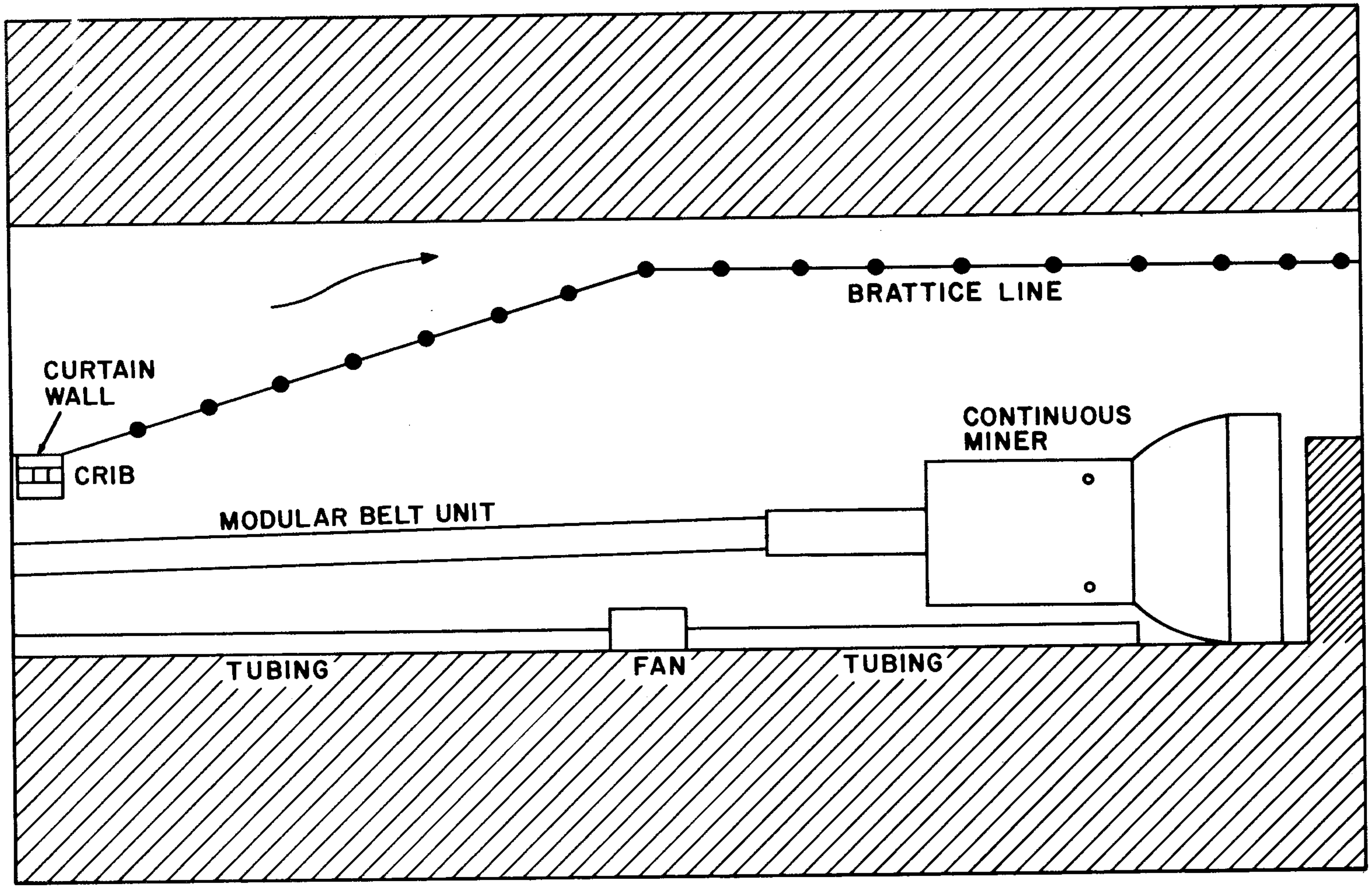


FIG. 64 PLAN VIEW OF THE FACE AREA DURING THE DEVELOPMENT OF THE SINGLE ENTRY

roof conditions can be maintained. Here, they are spaced further apart than those in the upper slice. These entries, driven along the floor of the seam, are also 9 ft (2.7m) high. To minimize any side abutment pressures on the entries from the extraction of the upper slice, a barrier of approximately 110 ft (33.3m) is maintained between the upper and lower gateroads, as well as between the lower lift gateroads in adjacent panels (C-28 ft, Figure 60). The 110-ft pillar should be sufficiently large enough to withstand the side abutment pressures. Thus, the face for the lower slice is 568 ft (172m) wide (B, Figure 60). The panel depth is 4300 ft (1303m) (D + 4200 ft, Figure 61).

Recovery of the Lower Slice

On retreat, the lower 13-ft (3.9m) longwall face will eventually be under gob so that the thickness of coal, between the slices, is 4 ft (1.2m). In many instances, partings or inferior coal exist in thick seams which can be conveniently left instead of good coal. If the lower longwall also uses shield supports and a double-drum shearer, both of which can range up to 16 ft (4.8m), recovery can be increased by ranging up to 16 ft (4.8m) as soon as the face is outside the confines of the upper slice. Manufacturers indicate that shields will soon be available for heights up to 19 ft (5.8m) (Simpson, 1976). Therefore, the recovery of future panels can be further improved. In any case, the recovery by the method proposed here is significantly larger than the average recovery of in situ coal in the United States.

Roof Support

Roof support in the gateroads is provided by rows of two mats, installed every 4 ft (1.2m). The side-mounted bolters on the continuous miner can install the outer two bolts per mat, while a stoper can be used to install the center bolts at a later time. Additional roof support is provided by the 3-ft (1m) cribs, installed every 8 ft (2.4m).

Roof support for the longwall faces is provided by 16-ft (4.8m) shields, installed on 5-ft (1.5m) centers.

Ventilation

Ventilation for the gateroads is provided by partitioning the entries with galvanized metal panels sprayed with sealant and attached to the cribline. Since the return contains the beltline, sufficient clearance is provided. Mandoors are installed every 200 ft (61m) along the metal partition.

Face ventilation is provided by brattice cloth and an auxillary fan with PVC tubing on the intake and exhaust sides. The tubing on the exhaust side permits the face workers to function inby the panel belt in a less dusty environment.

When the gateroads are driven to their specified distances, the face is then opened and the bleeder entries are connected to the returns.

Equipment

The following capital equipment are required for the mining of one panel. It has been assumed that both longwall faces are in operation at any one time. As such, four complete development sections are required. Further, two modular belt units are used for face haulage in each development section.

Equipment	Units
Continuous Miners with Side-mounted Roof Bolters	4
Complete Stoper Units	4
Extensible Belt Systems	4
750-kva Load Centers	4
1000-kva Load Centers	2

25-hp Auxillary Fans	4
Sealant Spray Units	4
17,000 ft (5182m) 42-in. (1066.8mm) Beltline	1
17,000 ft (5182m) 7.2 kv Cable	1
17,000 ft (5182m) 4-in. (101.5mm) Waterline	1
Shield Supports	185
Shearers	2
Face Conveyors	2
Stage Loaders	2

Manpower

The face personnel for the four development sections are as follows:

Category	Men
Miner Operators	4
Roof Bolters	8
Utilitymen	8
Section Mechanics	4
Section Supervisors	4

The utilitymen also work with the extensible belt system. The face personnel for the two longwall faces are as follows:

Category	Men
Shearer Operators	2
Operator Helpers	2
Shield Operators	5
Tailgate Men	4
Headgate Men	4
Section Mechanics	2
Section Supervisors	2

Production Calculations

The continuous miners, developing the gateroads for the longwall faces, will extract 88,000 tons (79,816 metric tons) in the upper slice and 92,000 tons (83,444 metric tons) in the lower slice. Production from bleeder-entry development has not been included in these figures. Assuming that each entry can advance 75 ft (22.7m) per day (two shifts for production and catch-up cribbing and one shift solely for cribbing), it takes slightly under 70 days to develop the panel.

Leaving a 200-ft (61m) barrier at the neck of the panel, 610,000 tons (553,270 metric tons) of coal can be extracted from the upper longwall face and 1,200,000 tons (1,088,400 metric tons) of coal can be extracted from the lower longwall face. At a production rate of 1000 tons (907 metric tons) per shift, it would take 305 days to mine the upper lift and 600 days to mine the lower lift, based on two daily production shifts per face.

In all, 2,000,000 tons (1,814,000 metric tons) of coal can be extracted from the panel, accounting for a recovery of 60%.

Health and Safety

Since the method incorporates the single-entry longwall system presently under study by the U.S. Bureau of Mines, its acceptability is contingent upon the findings of the Bureau. Since each longwall has its own gateroads, the gateroads can be abandoned after the face is mined. Therefore, no major problems with single-entry longwalling is anticipated.

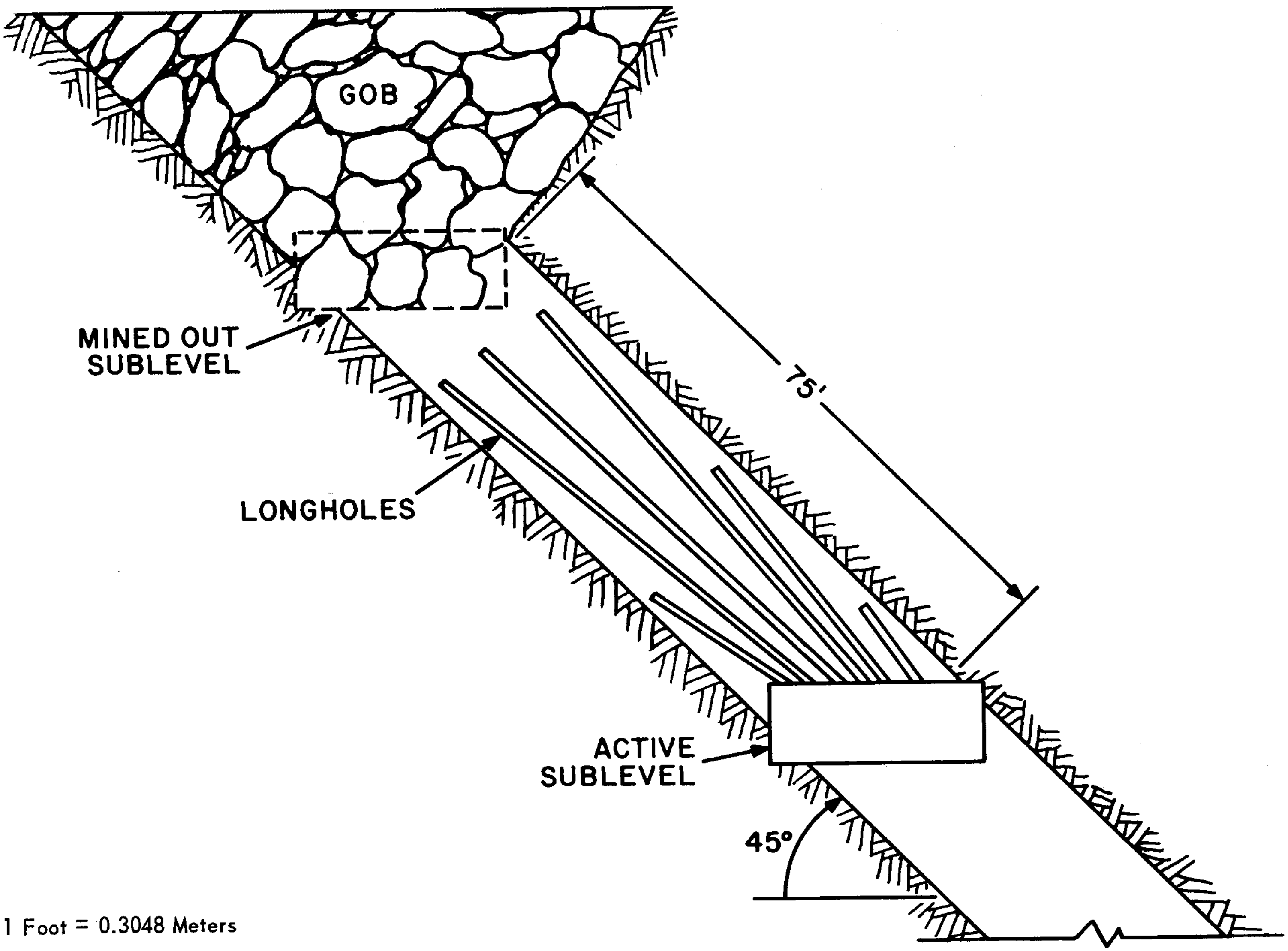
Longwalling with shield supports, though relatively new to American coal mines, should find wider acceptance because of the many safety aspects.

Method C

Coal seams which are approximately 20 ft (6m) thick and pitch at 45° pose many unique problems. Firstly, the equipment manufactured today is not designed to operate at such an extreme pitch. Secondly, the equipment can operate along, or slightly off, the strike but the thickness in the strike direction, from footwall to hangingwall, is not great enough to permit the development of a shortwall face. The pitch, however, should be used to an advantage in the design of the haulage medium. Since these conditions are prevalent in many of the fields located in the mountain regions and have been encountered in areas such as the Vicary Creek Mine, the proposed method incorporates a single-entry development. On retreat, longhole blasting and caving are recommended.

Development

The panel is developed with a single sublevel, driven 2000 ft (606m) long, 28 ft (8.5m) wide, and 10 ft (3m) high (Figure 65). The increase in height of the single entry, in comparison to that used in the two-lift longwall proposal, is due to the large working area required by the drilling equipment to be used on retreat. The sublevel is driven 5° updip to accommodate a flume similar to that reported in Kaiser's Hydraulic Mine. The entry is partitioned into an intake and a return by cribs and sealant-sprayed metal panels. The bolts are on 4-ft (1.2m) centers and the cribs are on 8-ft (2.4m) centers. This partition, similar to that found at the Sunnyside No. 1 Mine, would permit a return air escapeway, which also contains the flume. Thus, if a fire occurred at the entry neck, the workers at the face will not be trapped inby. This type of arrangement should make the method compatible with the health and safety standards of the United States.



1 Foot = 0.3048 Meters

FIG. 65 END VIEW OF A PANEL, METHOD C

The entry is driven with a continuous miner fitted with side-mounted roof bolters. The cut sequence at the face consists of a step system where the left side of the entry is advanced to a prescribed distance. The miner, after cleaning up the face, backs out of the cut to begin the extraction of the right side. Face ventilation is provided by a 25-hp self-propelled auxillary fan. The coal is transported from the face by a modular belt unit to a feederbreaker. The coal is then sized, mixed with water, and flumed out of the mine. At the end of the panel, a bleeder is established.

Roof Support

Roof support in the sublevel is provided by rows of two mats, installed every 4 ft (12.m). The side-mounted bolters on the continuous miner install the outer two bolts per mat, while a stoper is used to install the center bolts at a later time. Additional roof support is provided by the 3-ft (1m) cribs, which are installed every 8 ft (2.4m).

To further support the coal roof, particularly during retreat blasting, yieldable leg sets will be placed on the intake side of the cribline as the sublevel is advanced. Two sets will be positioned between each pair of cribs, on approximately 3.5-ft (1m) centers.

On retreat, the cribs and leg sets are left in place while the sublevel barrier is drilled. Where needed, roof jacks will also be used. Before the lift is blasted, the cribs and leg sets are removed.

Ventilation

The dual compartment sublevel is the main ventilation system for the section. Face ventilation is provided by brattice cloth and an auxillary fan with PVC tubing on the intake and exhaust sides. Tubing on the exhaust side permits the face workers to maneuver inby the feeder-breaker in a less dusty environment.

At the top end of the section, a bleeder is established to permit pillaring on retreat.

Retreat

On retreat, the coal left above the sublevel entry is extracted in 20-ft (6m) lifts with longhole blasting (Figure 66). Jacks are set across the sublevel before the coal is drilled. All the longholes should be kept under 60 ft (18.2m) in length to limit hole deviation and maintain an acceptable penetration rate. This, therefore, limits the dimension of the barrier pillar to 75 ft (22.7m). For drilling, 1.75-in.-diameter (44.5mm-diameter) augers are used. After drilling, the coal is blasted with a low-density permissible explosive and the broken coal is flushed to the feederbreaker before fluming. When all of the coal is extracted from the lift, the feederbreaker is retracted and the cycle is repeated. Figures 65 and 66 show a possible layout for the ring drilling pattern, with the rings spaced on 4-ft (1.2m) centers.

Equipment

The following capital equipment are required for the recovery of one panel:

Equipment	Units
Continuous Miner with Side-mounted Roof Bolters	1
Complete Stoper Unit	1
Extensible Belt System	1
2000-ft (610m) Flume	1
2000-ft (610m) 7.2-kv Cable	1
750-kva Load Center	1
25-hp Auxillary Fan	1

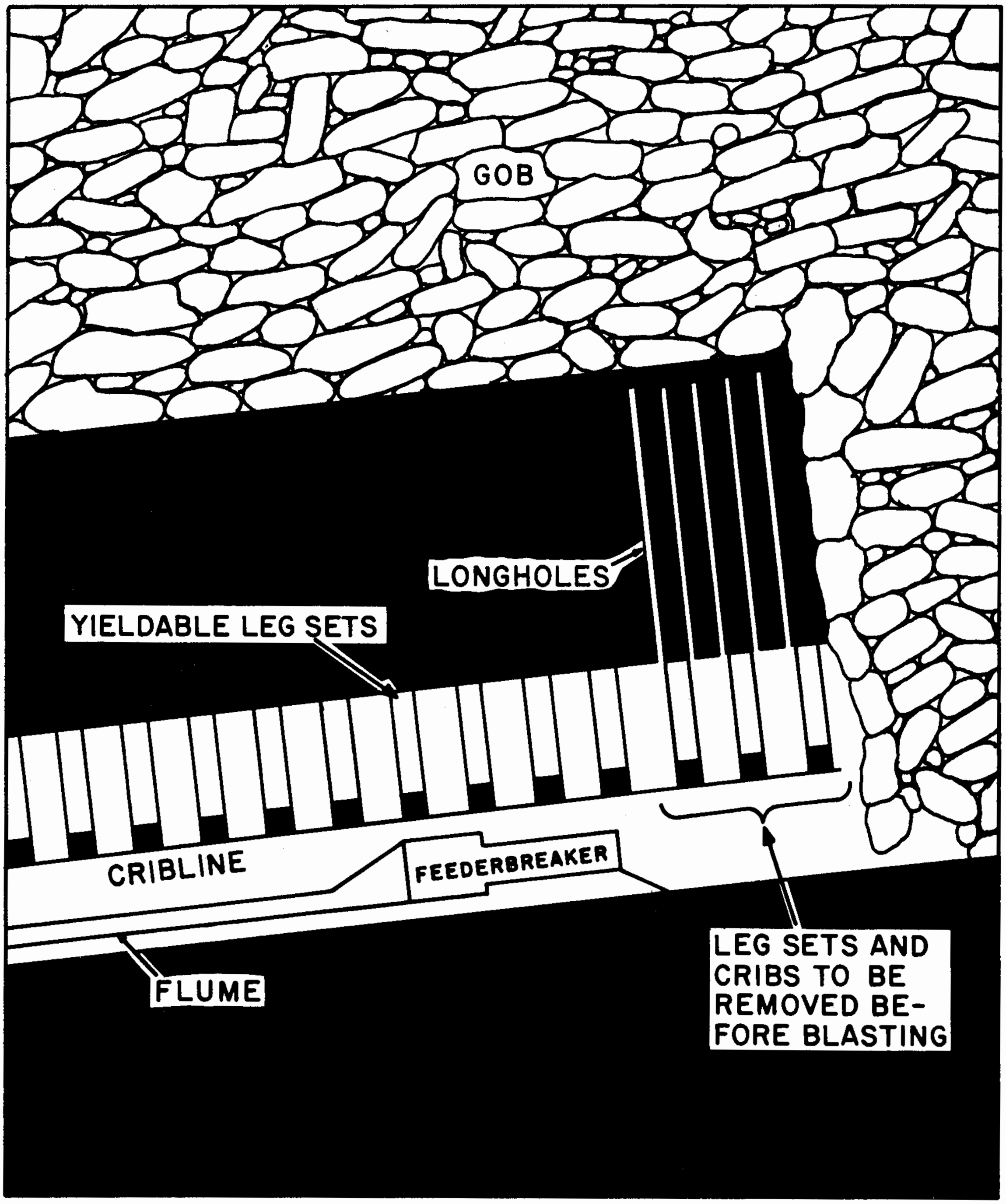


FIG. 66 PLAN VIEW OF THE FACE AREA DURING RETREAT, METHOD C

Sealant Spray Unit	1
600-cfm Compressors	2
Drill Units	2
Feederbreaker	1
Monitor and Pump	1
Yieldable Leg Sets	500

Manpower

The following face personnel are required during development:

Category	Men
Miner Operator	1
Utilitymen	2
Roof Bolters	2
Section Mechanic	1
Section Supervisor	1

The utilitymen also take charge of the extensible belt system. During the retreat phase, the number of men required is only four:

Category	Men
Driller	1
Shooter	1
Monitor Operator	1
Section Supervisor	1

Production Calculations

On advance, 23,000 tons (20,861 metric tons) of coal will be extracted from the sublevel entry. Assuming that the entry can advance 75 ft (22.7m) per day (two shifts for production and catch-up cribbing and one shift solely for cribbing), it would take approximately 30 days to develop the section.

On retreat, assuming a 70% recovery (similar to that reported by Kaiser's Hydraulic Mine) 85,000 tons (77,095 metric tons) of coal will be extracted. Initial calculations suggest that it would take two shifts to drill and blast the coal, leaving only one shift for coal loading per day. If the coal loading rate averaged 1000 tons (907 metric tons) per shift, it would take 85 days to retreat out of the panel.

Health and Safety

Since the acceptability of single-entry development in coal mines is presently under study by the U.S. Bureau of Mines, this is one aspect of the mining method that is dependent upon federal approval.

By combining the mining and bolting equipment into one machine, the working place is safer, due to the elimination of one piece of mobile machinery. Also, men will not be required to work under temporary support since the roof is bolted, except in the immediate face area. This facilitates the extension of the brattice line.

Sufficient cross-sectional area is provided for ventilation. The use of tubing in conjunction with an auxillary fan should keep the men, inby the feederbreaker, free of an excessively dusty environment.

As previously mentioned, the fire-resistant partition of the entry can provide an adequate escapeway in case of a fire in any part of the section.

Method D

Seams which are approximately 50 ft (15m) thick and pitch in the range of 45° can be found in the coalfields of northern Colorado, southern Wyoming, and the Canadian Rockies. Under these conditions, more strike development is possible than in seams only 20 ft (6m) thick. However, shortwalling is still difficult since the seam is only 70 ft (21.2m) thick across the strike. However, with a pitching seam of this type, gravity should be used for transportation of the mined coal. Though the proposed method is somewhat similar to that reported at Kaiser's Hydraulic Mine, a few variations have been incorporated to satisfy the requirements of the U.S. Coal Mine Health and Safety Act.

Development

The panel is developed with two arched sublevels which are driven 2000 ft (606m) long, 16 ft (4.8m) at the floor, and 11 ft (3.3m) high along the center line (Figure 67). They are driven 5° updip by a boom-type continuous miner to accomodate the flume, which is used to transport the coal. The flume is in the fresh air sublevel, which is driven along the footwall. The return is driven along the hangingwall. The entries are on 36-ft (10.9m) centers. Crosscuts, every 200 ft (61m), are driven from the intake to the return at a 60° angle, providing an acceptable operating gradient for the equipment (13.5°). A shuttle car transports the mined coal from the face to the feederbreaker where it is sized before fluming.

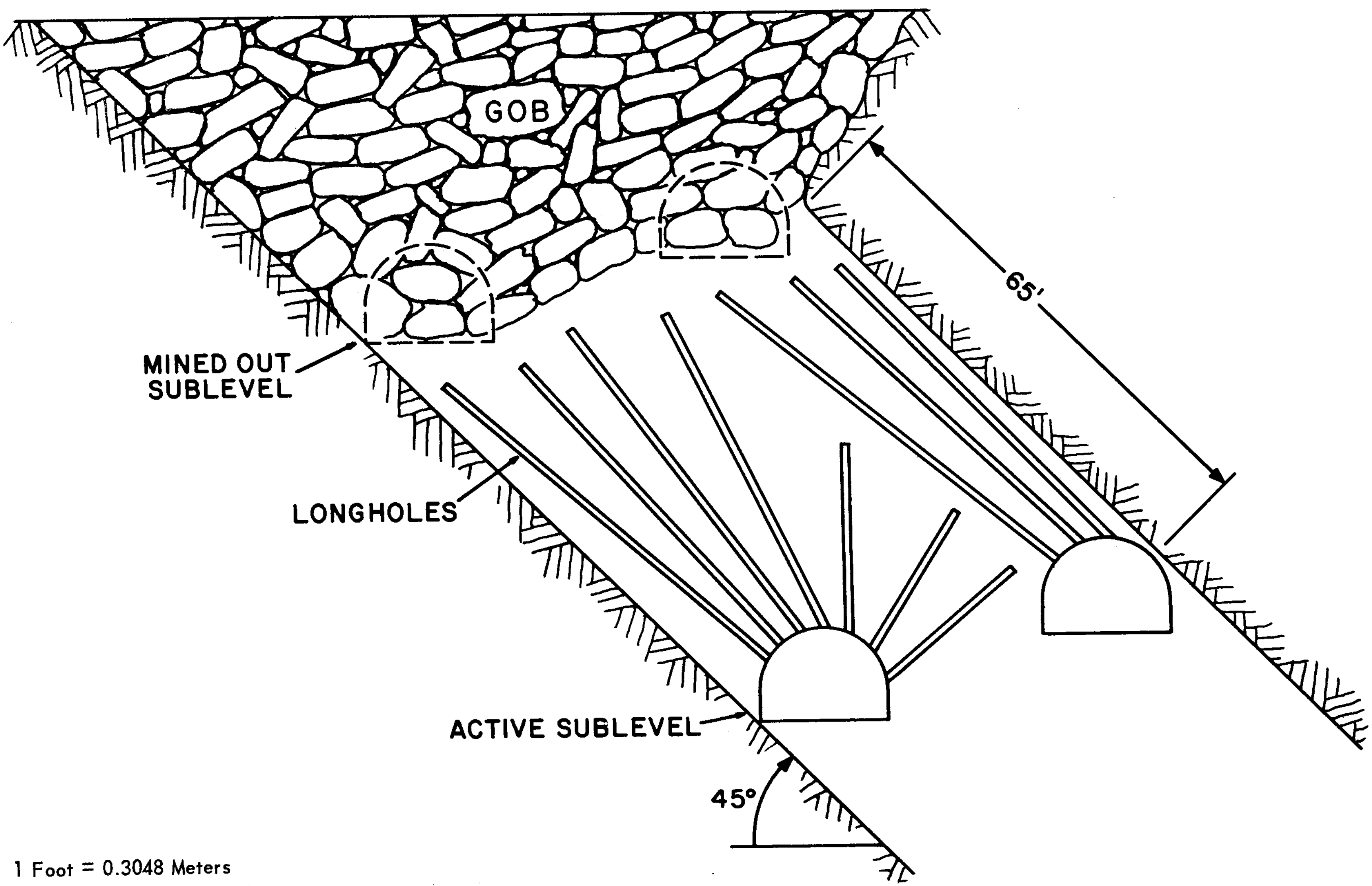
When the section is fully developed, a bleeder system is established.

Roof Support

Roof support, in both the entries and crosscuts, is provided by yieldable arches. This form of roof support has been chosen due to the heavy ground movement noted in areas with these conditions. Arches are placed on 4-ft (1.2m) centers and are fully lagged (Figure 68).

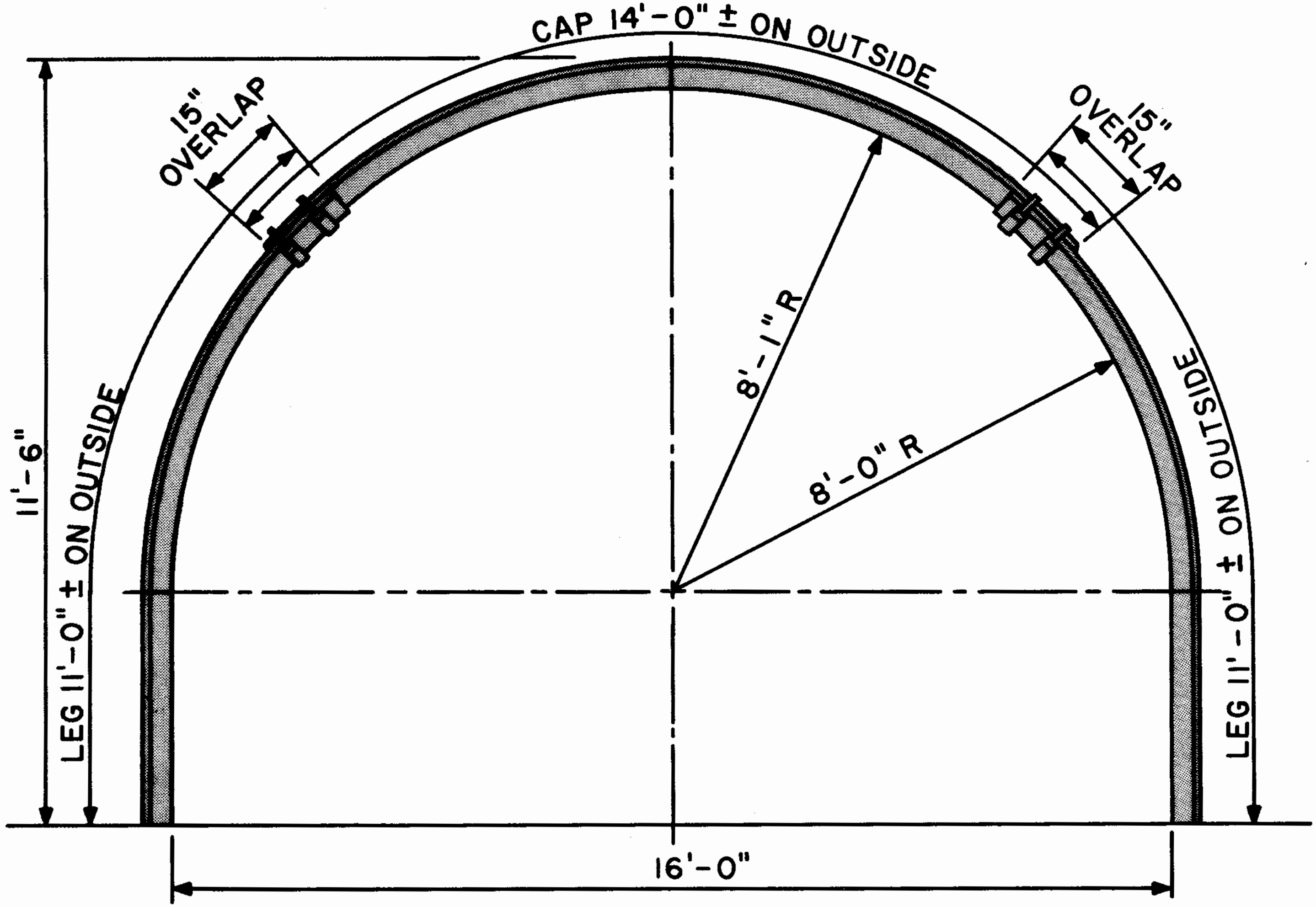
Ventilation

With two separate entries driven, one is used for intake air and



1 Foot = 0.3048 Meters

FIG. 67 END VIEW OF A PANEL, METHOD D



1 Foot = 0.3048 Meters

FIG. 68 YIELDABLE ARCH DIMENSIONS

the other is used for return air. The stoppings are constructed of metal panels sprayed with sealant. Face ventilation is provided by brattice cloth and an auxillary fan with PVC tubing. A bleeder is established prior to the recovery of the pillar coal.

Retreat

On retreat, the coal left above the panel is extracted by long-hole ring drilling with two rubber-tired mobile drills, blasting and caving. Five arches are pulled back from both entries to expose the drilled coal to be blasted with a low-density permissible explosive. The rings are spaced on 4-ft (1.2m) centers (Figure 69).

After the coal is shot, it is flushed into the feederbreaker prior to fluming. When all the coal is recovered, the coal for the next lift is drilled through the lagging of the arches. After drilling, five arches are pulled back and the cycle is resumed.

Equipment

The following capital equipment are required for the recovery of one panel:

Equipment	Units
Boom-type Mining Machine	1
25-hp Auxillary Fan	1
750-kva Load Center	1
Feederbreaker	1
2000-ft (610m) Flume	1
2000-ft (610m) 7.2-kv Cable	1
2000-ft (610m) 4-in. (101.6mm) Waterline	1
Monitor and Pump	1
Sealant Spray Unit	1

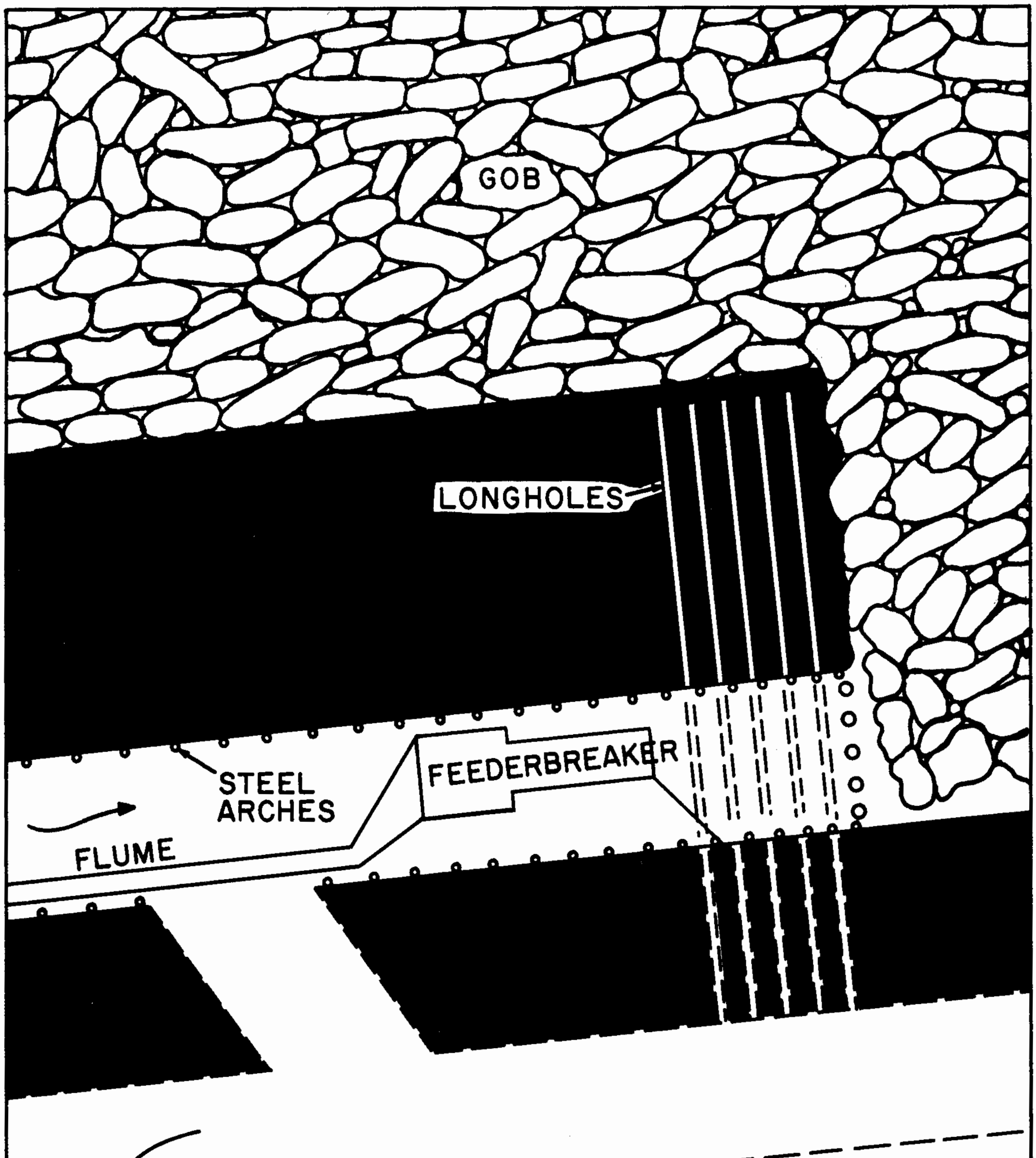


FIG. 69 PLAN VIEW OF THE FACE AREA DURING RETREAT, METHOD D

Mobile Drill Units	2
600-cfm Compressors	2
Shuttle Car	1
Yieldable Arches	1100

Manpower

During the development phase, the following face personnel are required:

Category	Men
Miner Operator	1
Shuttle Car Operator	1
Miner Helpers	2
Utilitymen	1
Section Mechanic	1
Section Supervisor	1

On retreat, the number of personnel required is only six:

Category	Men
Drillers	2
Shooters	2
Monitor Operator	1
Section Supervisor	1

Production Calculations

On advance, 28,000 tons (25,396 metric tons) of coal will be extracted from the panel. Assuming that the entries can be driven 20 ft (6m) per shift (accounting, also, for the installation of the arches), it would take almost 75 days to develop the section.

On retreat, assuming a 70% recovery rate (similar to that reported by Kaiser's Hydraulic Mine), 185,000 tons (167,795 metric tons) of of coal will be extracted. With two daily shifts dedicated to drilling and blasting, a loading rate of 2000 tons (1814 metric tons) per shift implys that it would take 93 days to retreat out of the panel.

Health and Safety

Although the conditions, as described, are quite different from the typical conditions upon which the Coal Mine Health and Safety Act is based, the only deviation from the law in this proposal is the interval between crosscuts. The law stipulates that crosscuts may not be driven more than a center line distance of 125 ft (37.9m) apart. An interval of 200 ft (61m) was selected because it was felt that 125 ft was too close for an application of this nature. Closer breakthroughs would unnecessarily hamper an already slow penetration rate. Further, by putting a mandoor in each crosscut, the proposed plan could satisfy the escapeway regulations.

Summary

In this chapter, four mining methods were proposed. The equipment, manpower and the operational features were discussed in detail. The analysis of any proposed mining method must include an economic assessment. It is important that at least an order-of-magnitude cost estimate be provided for comparison with existing methods and costs. Therefore, in the next chapter, the economics of the proposed methods are developed and comparisons are made between the proposed methods and continuous and conventional methods.

VII. COMPARATIVE ANALYSIS

Deep-mined coal must compete in the marketplace with other energy sources, including surface-mined coal. It has been shown that a difference in mining costs between deep-mined bituminous coal and surface-mined subbituminous coal gives a competitive edge to the latter, on a cost-per-Btu basis, by permitting its transportation to far-away markets (Table 8) (Yancik, 1975). The transportation costs for deep-mined western coal can be substantially reduced by building mine-mouth power plants though the recent court decision against the building of the Kaiparowits power plant complex is a major setback for the development of western underground reserves.

Scope of Analysis

For the purpose of this thesis, the economic analysis is limited to the evaluation of mining costs at the panel. In this study, mine-wide costing is difficult since overall mine-design considerations will include factors which are site specific. For example, the shape and size of the property will influence mine design requirements, such as the number and location of entries, haulage, and ventilation.

The comparative analysis tabulates the labor, capital, maintenance, overhaul, and supply costs for the four proposed mining plans listed in the previous chapter. These costs are then compared with the averages for the conventional and continuous methods calculated for a specific set of conditions. A brief comparative analysis on safety aspects is also included.

Table 8. The Effect of Mining Costs on Transporatation Distances
(after Yancik, 1975)

Coal A - Subbituminous coal. Surfaced mined. 8500 Btu. Similar to that mined in the Powder River Basin, Wyoming. Mining cost = \$3.80/ton.

Coal B - Bituminous coal. Underground mining. 12,000 Btu. Similar to the coal of the Kaiparowits Plateau, Utah. Mining cost = \$10.00/ton.

Coal A = 17.6 MBtu/ton, 21.59¢/MBtu (B-A) = 20.08¢/MBtu
Coal B = 24.0 MBtu/ton, 41.67¢/MBtu

If the coal is shipped by unit trains at a rate of 0.6¢/ton mile, the transportation costs for Coal A are .034¢/MBtu/mi and the costs for Coal B are .025¢/MBtu/mi

$$\frac{20.08\text{¢/MBtu}}{.034\text{¢/MBtu/mi}} = 591 \text{ mi}$$

This is the additional distance Coal A can be transported for the same delivered cost per million Btu as Coal B.

1 short ton = 0.907 metric tons
1 mile = 1.609 km

Assumptions

Several assumptions have been made in performing this economic analysis. All the capital cost items are based on a 10-year life, and the time value of the money is not considered. Since the economic viability of a project to a company is based on its capital structure, the cost of capital has not been included in this analysis. Therefore, the number of panels which could be extracted during that period of time is calculated and the corresponding percentage of the capital cost is prorated to determine the cost per panel. The mine is assumed to be in operation for 228 days per year. Methods A and B are figured on two production shifts and one maintenance shift per day. Method C is projected in a similar manner on development, only. During retreat in Method C and the development and retreat of Method D, three daily production shifts are incorporated. Thirty-five percent of the personnel costs were added for fringe benefits.

Maintenance costs were calculated at 50% of the capital cost for primary cutting equipment (miners, shearers, cutters, etc.) and 20% for other face and haulage equipment operation. Therefore, to figure the equipment maintenance cost per panel, the following equation is used:

$$\text{Equipment maintenance cost per panel} = \frac{(\text{Initial Capital Cost}) \times (50\% \text{ or } 20\%) \times (\text{Life of equipment in years})}{(\text{Number of panels extracted with the equipment})}$$

For example, the shearer maintenance cost per panel is calculated below, assuming an initial investment of $540,000.00, and a life of 10 years. This machine, on the average will serve 3.6 panels in the 10 years while working in the bottom slice of Method B.

$$\text{Shearer maintenance cost per panel} = \frac{\$540{,}000.00 \times 0.50 \times 10}{3.6} = \$750{,}000.00$$

Overhaul costs, assumed to be incurred once during the life of all face and haulage equipment, are figured at 65% of the original capital cost of the equipment. To appropriate this cost to a panel, the following formula is used:

$$\text{Overhaul cost per panel} = \frac{\text{(Initial Capital Cost)} \times (65\%)}{\text{(Number of panels extracted with the equipment)}}$$

For example, the overhaul cost for the abovementioned shearer is $97,500, as calculated below:

$$\text{Shearer overhaul cost per panel} = \frac{\$540{,}000.00 \times 0.65}{3.6} = \$97{,}500.00$$

Although roof bolts, mats, stoppings, and cribs are considered unrecoverable, yieldable arches, ventilation tubing, and drill rods are costed on the basis of projected recoveries. On the average, a 10% to 25% loss of these supplies is assumed for each panel. Table 9 lists the job performance rates which are used for figuring labor costs. Table 10 lists the assumptions associated with the supply cost calculations. A 10% contingency factor is included in the tabulations of the capital cost figures and the grand totals are rounded off to the nearest $1000.

Economic Analysis of Method A

In the previous chapter it was shown that, with this method, 87,000 tons (78,909 metric tons) of coal is extracted during the development of a panel and 211,000 tons (191,377 metric tons) is extracted on retreat for a total panel production of 298,000 tons (270,286 metric tons). It was also shown that it takes approximately one year to mine one panel.[1]

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[1] $$\text{Panel Factor} = \frac{\text{Life of the panel in years}}{\text{10 years}}$$

Table 9. Job Performance Rates

Job Classification	Quantity	Manshifts
Stopping Installation (10 ft by 18 ft)	2	1
Belt Advance	125 ft	4
Belt Reduction	125 ft	4
2 in. Pipe Installation	600 ft	2
2 in. Pipe Reduction	600 ft	2
4 in. Pipe Installation	400 ft	2
4 in. Pipe Reduction	400 ft	2
Crib Installation (10 ft high)	2	1
Curtain Wall Installation	75 ft	4
Flume Advance	75 ft	4
Flume Reduction	75 ft	4
Yieldable Leg Set Installation (10 ft by 16 ft)	6	4

1 in. = 25.4mm 1 ft = 0.3048m

Based on these production figures and the above assumptions, the panel categories are calculated in Tables 11, 12, 13, and 14, respectively. The unit cost at the panel is \$3.24 per ton, as shown in Table 27.

Economic Analysis of Method B

During the development of a complete panel, 180,000 tons(163,260 metric tons) of coal is extracted. Based on the production figures derived in the previous chapter, 28 panels can be mined in a 10-year period. As such, 3.5% of the development capital cost is applied to the panel.

It takes 1.4 years to mine the upper longwall slice and 2.75 years to mine the lower slice. The total recovery from the two slices is 1,810,000 tons (1,641,670 metric tons). Therefore, 14% of the capital cost for the top lift and 28% of the capital cost for the lower lift are charged to the panel. The panel cost for the method is calculated in Tables 15, 16, 17, and 18. The panel cost per ton is \$6.85 (Table 27).

Economic Analysis of Method C

In developing the sublevel entry for this method, 23,000 tons (20,861 metric tons) of coal is extracted. Based on the performance rates in Table 19 and the production assumptions, 50 sublevels could be driven in a 10-year period. Thus, two percent of the development capital cost is applied to each panel.

Based on the assumed production rates in the previous chapter, 85,000 tons (77,095 metric tons) of coal is extracted from each panel and 20 panels can be mined in 10 years. Each panel, therefore is charged five percent of the retreat capital cost. Tables 19, 20, 21, and 22 show the cost calculations for this method. The cost per ton, on a panel basis, for Method C is \$5.85 (Table 27).

Table 10. Supply Cost Assumptions

Item	Cost/Item	Quantity Required
Stopping Sealant	$11.70/60 lb can	1 can/64 sq ft
Lagging for Yieldable Arches	$20/linear ft of entry	
Hydraulic Oil	$0.05/ton	
Grease	$0.015/ton	
Rockdust	$0.12/continuous miner ton	
Miner Bits	$0.17/ton	
Roof Bolter Bits	$4.00 each	1 bit/10 holes
Shearer Bits	$0.13/ton	
Permissible Explosives	$0.40/lb	2 lbs/cu yd of materials to be blasted
Drill Bits	$40.00 each	1 bit/1000 ft to be drilled
Miscellaneous (Water, Power, etc.)	$0.20/ton	
Curtain Wall	$10/linear ft (9' high) $11/linear ft (10' high)	

1 ft = 0.3048m 1 lb = 2.2kg 1 short ton = 0.907 metric tons

Table 11. Personnel Costs for Method A

Classification	No.	Wages/Shift	Shifts	Total
Miner Operator	1	$57.20	456	$ 26,083.20
Roof Bolters	2	$57.20	456	$ 52,166.40
Utilitymen	4	$51.20	456	$ 93,388.80
Mechanic	1	$57.20	456	$ 26.083.20
Supervisor	1	$80.00	456	$ 36,480.00
Bratticemen	2	$48.91	10	$ 978.20
Beltman	2	$48.91	30	$ 5,869.20
Pipemen	2	$48.91	7	$ 684.74
Fringe Benefits				$ 84,606.81
			Subtotal:	$326,340.55

Table 12. Capital Costs for Method A

Item	Cost/Item	Quantity	Panel Factor	Total
Ripper Miner with Side-mounted Roof Bolters	$318,122.00	1	.1	$ 31,812.20
Complete Stoper Unit	$ 30,000.00	1	.1	$ 3,000.00
Extensible Belt System	$365,000.00	1	.1	$ 36,500.00
36-in. Beltline	$100,000.00	1	.1	$ 10,000.00
7.2-kv Cable	$12.873.00/1000 ft	2000 ft	.1	$ 2,574.60
2-in. Waterline	$1.50/ft	2000 ft	.1	$ 300.00
750-kva Load Center	$ 22,855.00	1	.1	$ 2,285.50
25-hp Auxillary Fan	$ 8,300.00	1	.1	$ 830.00
Sealant Spray Unit	$ 1,935.00	1	.1	$ 193.50
Ventilation Tubing	$5.00/ft	150 ft	.1	$ 75.00
Replacement Vent Tubing	$5.00/ft	50 ft	1.0	$ 250.00
Contingencies				$ 8,782.08
			Subtotal:	$ 96,602.88

1 in. = 25.4mm 1 ft = 0.3048m

Table 13. Maintenance and Overhaul Costs for Method A

Item	Total Cost	Maintenance Factor	Overhaul Factor	Maintenance Cost/Panel	Overhaul Cost/Panel
Ripper Miner with Side-mounted Roof Bolters	$318,122.00	0.50	0.65	$159,061.00	$20,677.93
Complete Stoper Unit	$ 30,000.00	0.20	0.65	$ 6,000.00	$ 1,950.00
Extensible Belt System	$365,000.00	0.20	0.65	$ 73,000.00	$23.725.00
36-in. Beltline	$100,000.00	0.20	0.65	$ 20,000.00	$ 6,500.00
			Subtotals:	$258,061.00	$52,852.93

1 in. = 25.4mm

Table 14. Supply Costs for Method A

Item	Cost/Item	Quantity	Total
Complete Stopping	$ 225.00	40	$ 9,000.00
Mandoors	$ 58.00	14	$ 812.00
6 ft Bolts	$ 2.30	16000	$36,800.00
Roof Mats	$ 2.25	5400	$12,150.00
Roof Bolter Bits	$ 4.00	1600	$ 6,400.00
Hydraulic Oil			$14,900.00
Grease			$ 4,470.00
Rockdust			$35,760.00
Miner Bits			$50,660.00
Miscellaneous			$59,600.00
		Subtotal:	$230,552.00

1 ft = 0.3048m

Table 15. Personnel Costs for Method B

Classification	No.	Wages/Shift	Shifts	Total
Miner Operators	4	$57.20	160	$ 36,608.00
Roof Bolters	8	$57.20	160	$ 73,216.00
Utilitymen	8	$51.20	160	$ 65,536.00
Mechanics	4	$57.20	160	$ 36,608.00
Supervisors	4	$80.00	160	$ 51,200.00
Shearer Operator	1	$57.20	650	$ 37,180.00
Operator Helper	1	$54.06	650	$ 35,139.00
Shield Operators	2	$54.06	650	$ 70,278.00
Tailgate Men	2	$54.06	650	$ 70,278.00
Headgate Men	2	$54.06	650	$ 70,278.00
Mechanic	1	$57.20	650	$ 37,180.00
Supervisor	1	$80.00	650	$ 52,000.00
Shearer Operator	1	$57.20	1240	$ 70,928.00
Operator Helper	1	$54.06	1240	$ 67,034.40

Table 15. (Continued)

Classification	No.	Wages/Shift	Shifts	Total
Shield Operators	3	$54.06	1240	$201,103.20
Tailgate Men	2	$54.06	1240	$134,068.80
Headgate Men	2	$54.06	1240	$134,068.80
Mechanic	1	$57.20	1240	$ 70,928.00
Supervisor	1	$80.00	1240	$ 99,200.00
Timbermen	20	$48.91	80	$ 78,256.00
Bratticemen	16	$48.91	80	$ 62,604.80
Pipemen	8	$48.91	22	$ 8,608.16
Beltmen	16	$48.91	80	$ 62,604.80
Fringe Benefits				$568,717.06
			Subtotal:	$2,193,622.96

Table 16. Capital Costs for Method B

Item	Cost/Item	Quantity	Panel Factor	Total
Continuous Miners with Side-mounted Roof Bolters	$ 35,651.00	4	.035	$ 49,231.14
Complete Stoper Units	$ 30,000.00	4	.035	$ 4,200.00
Extensible Belt Systems	$140,000.00	4	.035	$ 19,600.00
750-kva Load Centers	$ 22,855.00	4	.035	$ 3,199.70
1000-kva Load Center	$ 34,000.00	1	.140	$ 4,760.00
1000-kva Load Center	$ 34,000.00	1	.280	$ 9,520.00
25 hp. Auxillary Fans	$ 8,300.00	4	.035	$ 1,162.00
Sealant Spray Units	$ 1,935.00	4	.035	$ 270.90
42-in. Beltline	$803,692.00	1	.035	$ 28,129.22
42-in. Beltline	$198,930.00	1	.140	$ 27,850.20
42-in. Beltline	$202,916.00	1	.280	$ 56,816.48
7.2-kv Cable	$12,873.00/1000 ft	17000 ft	.035	$ 7,659.44
7.2-kv Cable	$12,873.00/1000 ft	4200 ft	.140	$ 7,569.33
7.2-kv Cable	$12,873.00/1000 ft	4300 ft	.280	$ 15,499.09
4-in. Waterline	$3.26/ft	17000 ft	.035	$ 1,939.70

Table 16. (Continued)

Item	Cost/Item	Quantity	Panel Factor	Total
4-in. Waterline	$3.26/ft	4200 ft	.140	$ 1,916.88
4-in. Waterline	$3.26/ft	4300 ft	.280	$ 3,925.04
Ventilation Tubing	$5.00/ft	600 ft	.035	$ 105.00
Replacement Vent Tubing	$5.00/ft	200 ft	1.000	$ 1,000.00
Shield Supports	$ 40,000.00	70	.140	$392,000.00
Shield Supports	$ 40,000.00	115	.280	$1,288,000.00
Shearer	$540,000.00	1	.140	$ 75,600.00
Shearer	$540,000.00	1	.280	$151,200.00
Face Conveyor	$330,000.00	1	.140	$ 46,200.00
Face Conveyor	$500,000.00	1	.280	$140,000.00
Stage Loader	$110,000.00	1	.140	$ 15,400.00
Stage Loader	$110,000.00	1	.280	$ 30,800.00
Contingencies				$238,355.41
			Subtotal:	$2,621,909.53

1 in. = 25.4mm   1 ft = 0.3048m

Table 17. Maintenance and Overhaul Costs for Method B

Item	Total Cost	Maintenance Factor	Overhaul Factor	Maintenance Cost/Panel	Overhaul Cost/Panel
Continuous Miners with Side-mounted Roof Bolters	$1,406,604.00	0.50	0.65	$ 251,179.28	$ 32,653.31
Complete Stoper Units	$ 120,000.00	0.20	0.65	$ 8,571.43	$ 2,785.71
Extensible Belt System	$ 560,000.00	0.20	0.65	$ 40,000.00	$ 13,000.00
42-in. Beltline	$ 803,692.00	0.20	0.65	$ 57,406.57	$ 18,657.14
42-in. Beltline	$ 198,930.00	0.20	0.65	$ 55,258.33	$ 17,958.96
42-in. Beltline	$ 202,916.00	0.20	0.65	$ 112,731.11	$ 36,637.61
Shield Supports	$2,800,000.00	0.20	0.65	$ 777,777.77	$505,555.55
Shield Supports	$4,600,000.00	0.20	0.65	$2,555,555.55	$830,555.55
Shearer	$ 540,000.00	0.50	0.65	$ 375,000.00	$ 48,750.00
Shearer	$ 540,000.00	0.50	0.65	$ 750,000.00	$ 97,500.00
Face Conveyor	$ 330,000.00	0.20	0.65	$ 91,666.67	$ 29,791.67
Face Conveyor	$ 500,000.00	0.20	0.65	$ 277,777.78	$ 90,277.78
Stage Loader	$ 110,000.00	0.20	0.65	$ 30,555.55	$ 9,930.55
Stage Loader	$ 110,000.00	0.20	0.65	$ 61,111.11	$ 19,861.11
1 in. = 25.4mm			Subtotals:	$5,444,591.15	$1,753,914.94

Table 18. Supply Costs for Method B

Item	Cost/Item	Quantity	Total
Cribs	$ 200.00	2550	$510,000.00
Curtain Wall	$10.00/ft	17,000 ft	$170,000.00
Sealant			$ 31,100.00
Mandoors	$ 58.00	85	$ 4,930.00
Roof Mats	$ 2.25	9000	$ 20,250.00
6-ft Bolts	$ 2.30	27,000	$ 62,100.00
Hydraulic Oil			$ 99,500.00
Grease			$ 29,850.00
Rockdust			$ 21,600.00
Miner Bits			$ 30,600.00
Roof Bolter Bits	$ 4.00	2700	$ 10,800.00
Shearer Bits			$235,300.00
Miscellaneous			$398,000.00
1 ft = 0.3048m			Subtotal: $1,624,030.00

Table 19. Personnel Costs for Method C

Classification	No.	Wages/Shift	Shifts	Total
Miner Operator	1	$57.20	80	$ 4,576.00
Roof Bolters	2	$57.20	80	$ 9,152.00
Utilitymen	2	$51.20	80	$ 8,192.00
Mechanic	1	$57.20	80	$ 4,576.00
Supervisor	1	$80.00	80	$ 6,400.00
Driller and Shooter	2	$51.20	285	$ 29,184.00
Monitor Operator	1	$51.20	285	$ 16,302.00
Supervisor	1	$80.00	285	$ 22,800.00
Bratticemen	9	$48.91	40	$ 17,607.60
Flumemen	4	$48.91	60	$ 11,738.40
Pipemen	2	$48.91	10	$ 978.20
Timbermen	4	$48.91	85	$ 16,629.40
Fringe Benefits				$ 51,847.46
			Subtotal:	$199,983.06

Table 20. Capital Costs for Method C

Item	Cost/Item	Quantity	Panel Factor	Total
Continuous Miner with Side-mounted Roof Bolters	$351,651.00	1	.02	$ 7,033.02
Complete Stoper Unit	$ 30,000.00	1	.02	$ 600.00
Extensible Belt System	$140,000.00	1	.02	$ 2,800.00
7.2-kv Cable	$ 12,873.00/1000 ft	2000 ft	.02	$ 514.92
7.2-kv Cable	$ 12,873.00/1000 ft	2000 ft	.05	$ 1,287.30
4-in. Waterline	$3.26/ft	2000 ft	.02	$ 130.40
4-in. Waterline	$3.26/ft	2000 ft	.05	$ 326.00
750-kva Load Center	$ 22,855.00	1	.02	$ 457.10
750-kva Load Center	$ 22,855.00	1	.05	$ 1,142.75
25-hp Auxillary Fan	$ 8,300.00	1	.02	$ 166.00
Sealant Spray Unit	$ 1,935.00	1	.02	$ 38.70
Ventilation Tubing	$5.00/ft	150 ft	.02	$ 15.00
Replacement Vent Tubing	$5.00/ft	50 ft	1.00	$ 250.00
600-cfm Compressors	$ 30,000.00	2	.05	$ 3,000.00
Drill Units	$ 50,000.00	2	.05	$ 5,000.00

Table 20. (Continued)

Item	Cost/Item	Quantity	Panel Factor	Total
Feederbreaker	$65,500.00	1	.02	$ 1,310.00
Feederbreaker	$65,500.00	1	.05	$ 3,275.00
Flume	$35.00/ft	2000 ft	.02	$ 1,400.00
Flume	$35.00/ft	2000 ft	.05	$ 3,500.00
Yieldable Leg Sets	$ 180.00	500	.02	$ 1,800.00
Yieldable Leg Sets	$ 180.00	100	1.00	$18,000.00
Roof Jacks	$ 240.00	5	.05	$ 60.00
Roof Jacks	$ 240.00	2	1.00	$ 480.00
Monitor and Pump	$250,000.00	1	.05	$12,500.00
Drills Rods	$ 30.00	80	.05	$ 120.00
Drill Rods	$ 30.00	20	1.00	$ 600.00
Contingencies				$ 6,580.62
			Subtotal:	$72,386.81

1 in. - 25.4mm 1 ft = 0.3048m

Table 21. Maintenance and Overhaul Costs for Method C

Item	Total Cost	Maintenance Factor	Overhaul Factor	Maintenance Cost/Panel	Overhaul Cost/Panel
Continuous Miner with Side-mounted Roof Bolters	$351,651.00	0.50	0.65	$ 35,165.10	$ 4,571.47
Complete Stoper Unit	$ 30,000.00	0.20	0.65	$ 1,200.00	$ 390.00
Extensible Belt System	$140,000.00	0.20	0.65	$ 5,600.00	$ 1,820.00
600-cfm Compressors	$ 60,000.00	0.20	0.65	$ 6,000.00	$ 1,950.00
Drill Units	$100,000.00	0.20	0.65	$ 10,000.00	$ 3,250.00
Feederbreaker	$ 65,500.00	0.20	0.65	$ 2,620.00	$ 851.50
Feederbreaker	$ 65,500.00	0.20	0.65	$ 6,550.00	$ 2,128.75
Monitor and Pump	$250,000.00	0.50	0.65	$ 62,500.00	$ 8,125.00
			Subtotals:	$129,635.10	$23,086.72

Table 22. Supply Costs for Method C

Item	Cost/Item	Quantity	Total
Cribs	$200.00	250	$ 50,000.00
Curtain Wall	$11.00/ft	2000 ft	$ 22,000.00
Sealant			$ 3,700.00
Mandoors	$ 58.00	10	$ 580.00
Roof Mats	$ 2.25	1000	$ 2,250.00
6-ft Bolts	$ 2.30	3000	$ 6,900.00
Hydraulic Oil			$ 5,400.00
Grease			$ 1,620.00
Rockdust			$ 2,760.00
Miner Bits			$ 3,910.00
Roof Bolter Bits	$ 4.00	300	$ 1,200.00
Drill Bits	$ 40.00	125	$ 5,000.00
Permissible Powder	$0.40/lb	200000 lbs	$ 80,000.00
Miscellaneous			$ 21,600.00
		Subtotal:	$206,920.00

1 ft = 0.3048m 1 lb = 2.2kg

Economic Analysis of Method D

This method is projected to recover 28,000 tons (25,396 metric tons) of coal during development. As per the assumed production rates, 25 panels can be developed in a 10-year period. Thus, four percent of the capital cost is charged to each panel.

On retreat, 185,000 tons (167,795 metric tons) of coal is extracted from each panel. Since 20 panels can be mined in a 10-year period, five percent of the capital cost is applied to each panel. Tables 23, 24, 25, and 26 show the panel costs for this example, which total $4.61 per ton (Table 27).

Continuous and Conventional Mining

Continuous and conventional mining methods account for over 90% of the underground coal production in the United States. The methods are practiced under conditions which are extremely variable with regard to seam thickness, roof, floor, gas, depth, and management and labor efficiencies. Thus, the industry average is not a good indicator for comparative purposes. Therefore, for this study, continuous and conventional mining standards are developed for the following conditions:

Seam Thickness	5 ft (1.5m)
Depth of Cover	1000 ft (303m)
Roof Conditions	Competent (Bolting required)
Floor Conditions	Fireclay (no water)
Gas Emission	Moderate (95cfm, 448 x 10^2cm^{3}per sec)
Seam Gradient	Tabular

It is assumed that the average panel would be 3000 ft (909m) long and 300 ft (91m) wide. Since 50% recovery of the in-place tonnage in a

Table 23. Personnel Costs for Method D

Classification	No.	Wages/Shift	Shifts	Total
Miner Operator	1	$57.20	255	$ 14,586.00
Shuttle Car Operator	1	$51.20	255	$ 13,056.00
Utilitymen	3	$51.20	255	$ 39,168.00
Mechanic	1	$57.20	255	$ 14,586.00
Supervisor	1	$80.00	255	$ 20,400.00
Driller and Shooter	4	$51.20	315	$ 64,512.00
Monitor Operator	1	$57.20	315	$ 18,018.00
Supervisor	1	$80.00	315	$ 25,200.00
Bratticemen	2	$48.91	3	$ 293.46
Flumemen	4	$48.91	60	$ 11,738.40
Pipemen	2	$48.91	10	$ 978.20
Fringe Benefits				$ 77,887.62
			Subtotal:	$300,423.68

Table 24. Capital Costs for Method D

Item	Cost/Item	Quantity	Panel Factor	Total
Boom-type Mining Machine	$310,000.00	1	.04	$ 12,400.00
25-hp Auxillary Fan	$ 8,300.00	1	.04	$ 332.00
750-kva Load Center	$ 22,855.00	1	.04	$ 914.20
750-kva Load Center	$ 22,855.00	1	.05	$ 1,142.75
Feederbreaker	$ 65,500.00	1	.04	$ 2,620.00
Feederbreaker	$ 65,500.00	1	.05	$ 3,275.00
4-in. Waterline	$3.26/ft	2000 ft	.04	$ 260.80
4-in. Waterline	$3.26/ft	2000 ft	.05	$ 326.00
7.2-kv Cable	$ 12,873.00/1000 ft	2000 ft	.04	$ 1,029.84
7.2-kv Cable	$ 12,873.00/1000 ft	2000 ft	.05	$ 1,287.30
Flume	$35.00/ft	2000 ft	.04	$ 2,800.00
Flume	$35.00/ft	2000 ft	.05	$ 3,500.00
Monitor and Pump	$250,000.00	1	.05	$ 12,500.00
Sealant Spray Unit	$ 1,935.00	1	.04	$ 77.40
Mobile Drill Units	$ 80,000.00	2	.05	$ 8,000.00

Table 24. (Continued)

Item	Cost/Item	Quantity	Panel Factor	Total
600-cfm Compressors	$ 30,000.00	2	.05	$ 3,000.00
Ventilation Tubing	$5.00/ft	150 ft	.04	$ 30.00
Replacement Vent Tubing	$5.00/ft	50 ft	1.00	$ 250.00
Yieldable Arches	$ 260.00	1100	.04	$ 11,400.00
Yieldable Arches	$ 260.00	200	1.00	$ 52,000.00
Drill Rods	$ 30.00	80	.05	$ 120.00
Drill Rods	$ 30.00	20	1.00	$ 600.00
Shuttle Car	$ 74,500.00	1	.04	$ 2,980.00
Contingencies				$ 12,084.53
			Subtotal:	$132,929.82

1 in. = 25.4mm 1 ft = 0.3048m

Table 25. Maintenance and Overhaul Costs for Method D

Item	Total Cost	Maintenance Factor	Overhaul Factor	Maintenance Cost/Panel	Overhaul Cost/Panel
Boom-type Mining Machine	$310,000.00	0.50	0.65	$ 62,000.00	$ 8,060.00
Feederbreaker	$ 65,500.00	0.20	0.65	$ 5,240.00	$ 1,703.00
Feederbreaker	$ 65,500.00	0.20	0.65	$ 6,550.00	$ 2,128.75
Monitor and Pump	$250,000.00	0.50	0.65	$ 62,500.00	$ 8,125.00
Mobile Drill Units	$160,000.00	0.20	0.65	$ 16,000.00	$ 5,200.00
600-cfm Compressors	$ 60,000.00	0.20	0.65	$ 6,000.00	$ 1,950.00
Shuttle Car	$ 74,500.00	0.20	0.65	$ 5,960.00	$ 1,937.00
			Subtotals:	$164,250.00	$29,103.75

Table 26. Supply Costs for Method D

Item	Cost/Item	Quantity	Total
Complete Stopping	$200.00	10	$ 2,000.00
Mandoors	$ 58.00	10	$ 580.00
Lagging			$ 88,000.00
Hydraulic Oil			$ 10,650.00
Grease			$ 3,195.00
Rockdust			$ 3,360.00
Miner Bits			$ 4,760.00
Permissible Powder	$0.40/lb	480000 lbs	$192,000.00
Drill Bits	$ 40.00	210	$ 8,400.00
Miscellaneous			$ 42,600.00
		Subtotal:	$355,545.00

1 lb = 2.2kg

Table 27. Summary of Costs for the Proposed Methods

	Method A	Method B	Method C	Method D
Personnel Cost Subtotal	$326,340.55	$2,193,622.96	$199,983.06	$300,423.68
Capital Cost Subtotal	$ 96,602.88	$2,621,909.53	$ 72,386.81	$132,929.82
Maintenance Subtotal	$258,061.00	$5,444,591.15	$129,635.10	$164,250.00
Overhaul Subtotal	$ 52,852.93	$1,753,914.94	$ 23,086.72	$ 29,103.75
Supply Cost Subtotal	$230,552.00	$1,624,030.00	$206,920.00	$355,545.00
GRAND TOTAL	$964,000.00	$13,638,000.00	$632,000.00	$982,000.00
Section Tonnage	298,000	1,990,000	108,000	213,000
Cost per ton(at the panel)	$3.235	$6.853	$5.852	$4.610

1 short ton = 0.907 metric tons

panel is common for the coal industry, the production from this panel is 90,000 tons (81,630 metric tons).

For the purposes of estimating shift production from a panel under the above conditions, two sources of information were used. A recent study by NUS Corporation (1976) involved collection of production, manpower, and equipment data from operating mines. That study also included generation of production data from a computer-oriented simulation model, developed at The Pennsylvania State University (Manula, *et al.*, 1975). On the basis of the data from these two sources, it is estimated that the average tons per machine shift for continuous and conventional sections are 277 and 366 tons, respectively.

Manpower requirements for continuous and conventional sections are presented in Tables 28 and 29, respectively. For this analysis, 8 men for the continuous section and 13 for the conventional section are used. An additional 0.75 manshift is charged to each panel to account for the deadwork associated with such activities as ventilation and haulage, which are usually done by an outby crew.

With the manpower requirements established, an average labor cost can be calculated. For the continuous section, the average wage for hourly employees is $53.58 per shift. The average for the conventinal section is $52.72 per shift. The average salary for the section supervisor is the same as that assumed for the proposed methods, $80.00 per shift.

The NUS report (1976) has developed, from industry records, a relationship between the labor cost and the supplies and materials cost (Figure 70). This relationship is used to develop the supplies and materials cost for this analysis.

Table 28. Continuous Miner Production Section Manning Table (after NUS Corporation, 1976)

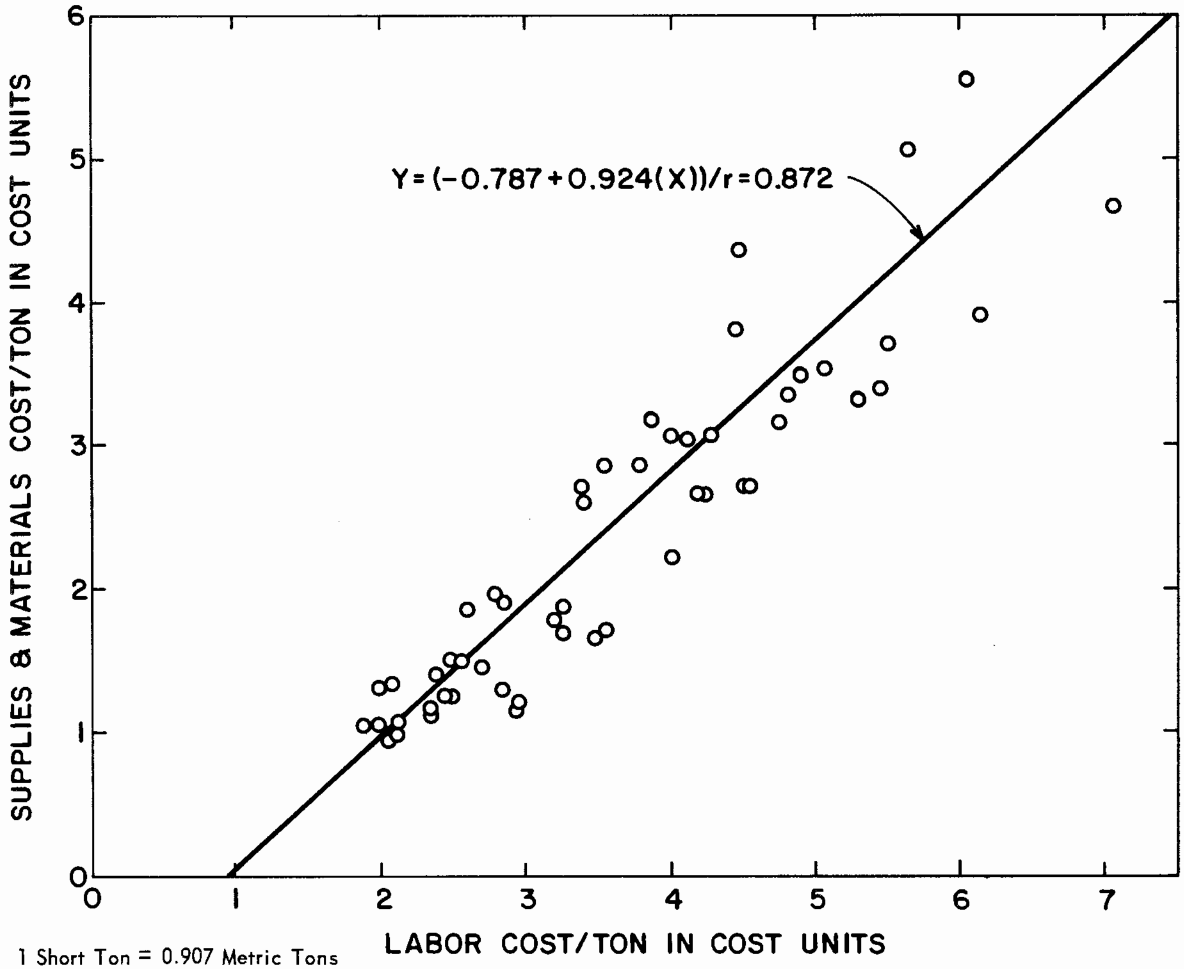
Job Title	Normal	Maximum
Miner Operator	1	1
Miner Helper	1	1
Shuttle Car Operator	2	2
Roof Bolter Operator	1	2*
Roof Bolter Helper	1	2*
Utility Man	1	2
Scoop Operator	0	1
Mechanic	1	1
	8	12

* Two Single Boom Bolters

Table 29. Conventional Section Manning Table (after NUS Corporation, 1976)

Job Title	Minimum	Maximum
Loader Operator	1	1
Shuttle Car Operator	2	2
Roof Bolter Operator	1	2*
Roof Bolter Helper	1	2*
Cutter Operator	1	1
Coal Drill Operator	1	1
Shot Fireman	1	2
Utility Man	2	4
Scoop Operator	0	1
Mechanic	1	1
Normal Range of Manpower	11	17
Most Common Manning	13	

* Two Single Boom Bolters



1 Short Ton = 0.907 Metric Tons

FIG. 70 SUPPLIES AND MATERIALS COST/TON VS. LABOR COST/TON
(NUS Corporation, 1976)

The capital cost estimates for continuous and conventional sections are shown in Tables 30 and 31, respectively. Tables 32 and 33 show the maintenance and overhaul cost calculations.

In Tables 34 and 35 are summarized the total costs at the panel for the continuous and conventional sections. Panel costs per ton for the continuous and conventional sections are $8.37 and $8.13, respectively.

Remarks

Although it has been shown that, on a panel basis, thick-seam mining can compete economically with mining conducted in moderately thick seams, final approval of any method is dependent upon the degree of safety that can be achieved. This safety requirement goes beyond the satisfaction of the problems peculiar to thick-seam mining, which have been discussed in Chapter 5. The methods must also be evaluated, on a comparative basis, as to their ability to control the problems which affect the coal industry as a whole. To do this, the 1970 safety statistics for the industry have been analyzed and a point-by-point discussion of the steps taken to alleviate similar problems in thick seams is included.

The United States Bureau of Mines has shown that nearly 50% of the disabling work-related injuries are attributed to haulage, falls of roof and rib, and machinery (Moyer and McNair, 1973). These catagories are also high by the severity standards. Although the accident severity associated with the handling of materials is much lower than any of the aforementioned categories, another 25% of the total number of injuries in attributed to this cause (Moyer and McNair, 1973).

Table 30. Face Equipment, Continuous Mining System (after NUS Corporation, 1976)

Item	Cost/Unit	Cost/Section
Continuous Miner	$339,900.00	$339,900.00
Shuttle Cars	$ 79,500.00	$159,000.00
Roof Bolter	$ 82,400.00	$ 82,400.00
Ratio Feeders	$ 70,200.00	$140,400.00
Auxillary Fans	$ 20,900.00	$ 41,800.00
Scoop Tram	$ 48,500.00	$ 48,500.00
Bantam Duster	$ 5,200.00	$ 5,200.00
Trickle Dusters	$ 3,300.00	$ 6,600.00
Section Power Center with Cables	$ 74,300.00	$ 74,300.00
Parts Car	$ 9,900.00	$ 9,900.00
Oil Storage Car	$ 4,400.00	$ 4,400.00
Section Tools	$ 6,700.00	$ 6,700.00
Ventilation Tubing	$ 1,700.00	$ 1,700.00
Section Welders	$ 700.00	$ 700.00
36-in. Section Haulage Belt (3000 ft.)	$142,700.00	$142,700.00
Fire Suppression System	$ 3,700.00	$ 3,700.00
TOTAL COST PER PRODUCTION SECTION		$1,067,900.00

1 in. = 25.4mm 1 ft = 0.3048m

Table 31. Face Equipment, Conventional Mining System (after NUS Corporation, 1976)

Item	Cost/Unit	Cost/Section
Cutting Machine	$156,600.00	$156,600.00
Coal Drill	$ 38,300.00	$ 38,300.00
Loading Machine	$190,800.00	$190,800.00
Shuttle Cars	$ 79,500.00	$159,000.00
Roof Bolter	$ 82,400.00	$ 82,400.00
Ratio Feeders	$ 70,200.00	$140,400.00
Auxillary Fans	$ 20,900.00	$ 41,800.00
Scoop Tram	$ 48,500.00	$ 48,500.00
Bantam Duster	$ 5,200.00	$ 5,200.00
Trickle Dusters	$ 3,300.00	$ 6,600.00
Section Power Center with Cables	$ 74,300.00	$ 74,300.00
Parts Car	$ 9,900.00	$ 9,900.00
Oil Storage Car	$ 4,400.00	$ 4,400.00
Section Tools	$ 6,700.00	$ 6,700.00
Ventilation Tubing	$ 1,700.00	$ 1,700.00
Section Welders	$ 700.00	$ 700.00
36-in. Section Haulage Belt (3000 ft)	$142,700.00	$142,700.00
Fire Suppression System	$ 3,700.00	$ 3,700.00
TOTAL COST PER PRODUCTION SECTION		$1,113,700.00

1 in. = 25.4mm 1 ft = 0.3048m

Table 32. Maintenance and Overhaul Costs, Continuous Mining System

Item	Total Cost	Maintenance Factor	Overhaul Factor	Maintenance Cost/Panel	Overhaul Cost/Panel
Continuous Miner	$339,900.00	0.50	0.65	$127,781.95	$16,611.65
Shuttle Cars	$159,000.00	0.20	0.65	$ 23,909.77	$ 7,770.68
Roof Bolters	$ 82,400.00	0.20	0.65	$ 12,390.98	$ 4,027.07
Ratio Feeders	$140,400.00	0.20	0.65	$ 21,112.78	$ 6,861.65
Scoop Tram	$ 48,500.00	0.20	0.65	$ 7,293.23	$ 2,370.30
36-in. Beltline	$142,700.00	0.20	0.65	$ 21,458.65	$ 6,974.06
			Subtotals:	$213,947.36	$44,615.41

1 in. = 25.4mm

Table 33. Maintenance and Overhaul Costs, Conventional Mining System

Item	Total Cost	Maintenance Factor	Overhaul Factor	Maintenance Cost/Panel	Overhaul Cost/Panel
Cutting Machine	$156,600.00	0.50	0.65	$ 42,324.32	$ 5,502.16
Coal Drill	$ 38,300.00	0.50	0.65	$ 10,351.35	$ 1,345.68
Loading Machine	$190,800.00	0.50	0.65	$ 51,567.57	$ 6,703.78
Shuttle Cars	$159,000.00	0.20	0.65	$ 17,189.19	$ 5,586.49
Roof Bolters	$ 82,400.00	0.20	0.65	$ 8,908.11	$ 2,895.14
Ratio Feeders	$140,400.00	0.20	0.65	$ 15,178.38	$ 4,932.97
Scoop Tram	$ 48,500.00	0.20	0.65	$ 5,243.24	$ 1,704.05
36-in. Beltline	$142,700.00	0.20	0.65	$ 15,427.03	$ 5,013.78
			Subtotals:	$166,189.19	$36,684.05

1 in. = 25.4mm

Table 34. Cost Estimate for a Continuous Section

Tons per shift		277
Shifts required (90,000/277)		325
Years per panel.		0.75
Panel factor		0.075

Labor:

Hourly: 8.75 x \$53.58 x 325 =	\$152,368.12	
Salary: 1.00 x \$80.00 x 325 =	26,000.00	
Fringe Benefits:	62,428.84	
	\$240,796.96	\$2.676/ton

Supplies:

$$\frac{-\$0.787 + (0.924)(\$2.676)}{0.875} = \$1.926/\text{ton}$$

Capital:

$$\frac{0.075 \times \$1,067,900.00}{90,000} = \$0.890/\text{ton}$$

Maintenance:

$$\frac{\$213,947.36}{90,000} = \$2.377/\text{ton}$$

Overhaul:

$$\frac{\$44,615.41}{90,000} = \$0.496/\text{ton}$$

TOTAL: \$8.365/ton

1 short ton = 0.907 metric tons

Table 35. Cost Estimate for a Conventional Section

Tons per shift			366
Shifts required (90,000/366)			246
Years per panel.			0.54
Panel Factor			0.054
Labor:			
Hourly: 13.75 x \$52.72 x 246	=	\$178,325.40	
Salary: 1.00 x \$80.00 x 246	=	19,680.00	
Fringe Benefits:	=	69,301.89	
		\$267,307.29	\$2.970/ton
Supplies			
$\frac{-\$0.787 + (0.924)(\$2.970)}{0.875}$	=		\$2.237/ton
Capital:			
$\frac{0.054 \times \$1,113,700.00}{90,000}$	=		\$0.668/ton
Maintenance:			
$\frac{\$166,189.19}{90,000}$	=		\$1.847/ton
Overhaul:			
$\frac{\$36,684.05}{90,000}$	=		\$0.408/ton
		TOTAL:	\$8.130/ton

1 short ton = 0.907 metric tons

Falls of roof, face, or rib caused 84 fatal and 1689 nonfatal injuries in underground coal mines in 1970. The largest number of injuries in this category, occurred during the roof bolting operation, 4 fatal and 445 nonfatal injuries. Although the thick-seam methods were designed to allow for the proper slope of the ribs, to limit roadway height and to permit the dressing of overhangs, the safety of the roof bolter was also considered. With concommitant bolting and mining, no place has to remain unbolted for long periods of time. This will minimize the initial separation of bedding planes, a principal cause for many roof falls in unbolted headings. The extensive use of cribs and yieldable arches can further help in thick-seam roof control.

Although accidents attributed to the handling of materials are very difficult to identify, due to the nature of the mine environment and the bulkiness of mine materials, the proposed methods have dealt with this problem in a two-fold approach. Firstly, any bulky items that could be eliminated, such as rails, were not incorporated in the panél plans. Secondly, sufficient manpower was projected for the deadwork. Many times, accidents are caused if operations are attempted by crews of insufficient size. Thus, the number of accidents caused by the handling of heavy materials can be reduced with proper planning and training.

Haulage accidents were the cause of 27 fatal and 1695 nonfatal injuries in 1970, with the largest segment of these (23 fatal, 342 nonfatal) occurring when an individual was squeezed between a shuttle car, or any other piece of mobile face equipment, and the roof, rib, or another object. This can justify the elimination of roof bolters and shuttle cars from the methods, particularly those that deal with a single entry, where freedom of movement is quite limited. As such, the attempt to limit mobile face equipment not only increases productive time but also, on the basis of these statistics, creates a safer workplace.

As the fourth-ranking cause of accidents that year, machinery accounted for 42 fatal and 1725 nonfatal injuries. The greatest number of accidents in this category were attributed to setting up and operating roof bolters (4 fatal, 474 nonfatal). Although roof bolters cannot, in many of the methods, be eliminated, the use of side-mounted roof bolters on continuous miners can reduce the number of accidents which occur during tramming and setting up.

There is one more safety aspect of the thick-seam methods which should be presented: less manshifts are required to reach high level of productivity. In most cases, one helper has been removed from the required panel manpower because of the design of the mining system. Even with the large backup crews involved in the installation of cribs, stoppings, and flumes, the number of men exposed at the face, particularly in the sublevel caving methods, has been greatly reduced.

In summary, thick-seam mining methods can be devised which are productive and economically competitive, as well as capable of enhancing the safety of the mine personnel.

VIII. SUMMARY AND CONCLUSIONS

Summary

The main objective of this research was to outline methods for the extraction of thick coal seams, particularly for the deep mineable reserves in the western United States. Other objectives of this study were to review the current state of the art in methods and equipment for thick-seam mining and to conduct a critical examination of the associated safety and economic factors.

Currently available equipment - mining machines, haulage equipment, and roof support systems - were evaluated with regard to their applicability to thick-seam mining. The safety and ground control considerations, which are peculiar to thick-seam mining, and thick-seam extraction techniques practiced abroad were studied. Proceedings of two international symposia on the subject were the main sources of information. Spontaneous combustion, bumps, and ventilation appear to be some of the more important safety considerations.

A general review of the western coal reserves on a state by state basis was conducted to determine the fraction of reserves that may have to be deep mined. Additionally, nine North American mines were visited to gain first-hand knowledge of the operating conditions. These mine visits provided an insight into applications of specialized equipment and methods not commonly employed in underground bituminous coal mines.

On the basis of this information, four mining methods were proposed for conditions most likely to be encountered in the West. The methods were designed for a 20 ft (6m) thick, gently pitching seam, a 30 ft (9m) thick, tabular seam, and two inclined seams which are 20 and

50 ft (6 and 15m) thick, respectively. In addition to the detailed development and pillaring plans, an economic evaluation at the panel level for each of the methods was done. A comparative analysis between these methods and the industry standards for a seam of average thickness mined by both continuous and conventional methods was also conducted.

Conclusions

On the basis of this study, several conclusions are drawn about the future of thick-seam mining in the United States. Production from thick-seam mining can compete economically on a panel basis, with production from seams of average thickness. The panel costs for the four proposed methods range from $3.24 to $6.85 per ton. These costs compare quite favorably with the projected panel costs per ton for continuous ($8.37) and conventional ($8.13) mining in a 5-ft (1.5m) seam. Thick-seam methods can also be designed to meet the provisions of the 1969 Health and Safety Act.

Other conclusions have been reached concerning equipment, applicability of methods practiced abroad, single entry development, hydraulic mining, and safety.

Some of the currently available mining machinery can be adapted to thick-seam mining;however, most equipment found in American coal mines will have little direct application. Continuous miners should find acceptance for the development of thick-seam panels. They should also find use as primary extraction machinery for benching operations. Side-mounted roof bolters on continuous miners will enhance these applications. The flexibility of ripper miners in allowing drivage of entries with varying widths is particularly advantageous for benching operations. Continuous miners should be limited to pitches under 15°. Shuttle cars will suffer as a primary face haulage medium in seams which pitch, and should not be considered if the pitch is greater than 12°.

Longwall mining with shield supports should become more common. With the availability of shields which range up to 19 ft (5.8m), many thick coal seams can be more efficiently recovered than with present methods.

Equipment, such as slushers, which are not commonly associated with coal mining are also applicable in certain circumstances.

Many of the foreign mining practices cannot be transferred directly to U.S. operations because of their dependence upon stowing procedures or their low productivity. Stowing has many benefits. However, non-availability of manpower, the difficulty of procuring enough packing materials, and the additional cost make the adoption of stowing for ground control unrealistic, at least in the near future.

Full-face methods will be limited to applications in seams less than 20 ft (6m) in thickness and tabular to slightly pitching. Slicing will be limited primarily by seam pitch and ground control requirements. Caving methods are the most flexible, but require skilled manpower for proper application.

Hydraulic jet mining can be advantageous, particularly in very thick, pitching seams. Conventional longhole blasting can be used in seams that are not friable.

Efficient extraction methods and rapid retreat mining can limit the possibility of spontaneous combustion. Higher air velocities than those encountered in seams of average thickness will be required to control methane layering in the high and wide entries of thick seam development.

Future Research

This study needs to be expanded in several directions. The areas that should be analyzed in the immediate future are as follows:

1. A site-specific detailed engineering and economic analysis should be conducted. The United States Bureau of Mines has recently awarded a contract to an operating company for this purpose and the information gathered should be helpful in the continuation of thick-seam research.

2. A thorough rock mechanics study should be conducted on the proper spacing of gateroads in slicing systems.

3. Continuous miners should be developed which are more effective at tramming on pitches up to 20° and can range up to a cutting height of 20 ft (6m). If remote-control units are employed with these machines, greater recovery is possible, in a large percentage of thick-seam reserves, than that presently achieved.

4. Roof support requirements and behavior in thick massive coal roofs should be analyzed.

5. Because of the many potential benefits of stowing, particularly in the areas of ground control and subsidence, a study should be initiated as to the future potential of stowing in the United States.

6. Because of its potential in pitching-seam applications, hydraulic transportation, either in open flumes or in pipes, should receive greater research emphasis.

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