Development of an Underground Automated Thin-Seam Mining Method

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(ABSTRACT)

It is predicted that coal mining in Southwest Virginia, and the economic stability that it brings to the area, will continue to decline over the next decade unless an environmentally sound, and economically viable means can be found to extract seams of high quality coal in the thickness range of 14 to 28 inches. Research into autonomous machine guidance, coupled with developments of thin-seam mining equipment, offer new opportunities for devising mining layouts suitable for extracting these thin seams in a cost effective manner. These layouts must involve well-planned transportation and ventilation routes that will allow safe conditions for personnel. This implies that the mining face, where coal is extracted, will be completely automated, ensuring the safety of the workers.

This thesis presents a brief overview of current technologies utilized for underground coal mining in the United States. This is followed by a review of developments in highwall mining that are potentially applicable in underground mining of thin seams. Some past attempts at thin seam mining are discussed, and evaluated for their short comings. An overview of the more recent advances in the guidance systems for use in autonomous mining machines is also presented. The new advances that several manufacturers are developing to address the integration of mining and continuous haulage systems are also investigated. That background is employed in devising a conceptual mining system for the underground mining of coal seams in the 14 to 28 inch range of thickness. This thesis proves that adapting new technologies and concepts from existing ones can lead to meaningful advances in the field of natural resources recovery. This system utilizes a newly designed panel layout that takes into account haulage, supplying, ventilation, equipment, and machine guidance. This system is proposed to

show that new ways can be developed to take advantage of the reserves in the 14 to 28 inch range of thickness. This shows that new technology and design innovation can turn currently uneconomic resources, into economic reserves. This kind of innovation is what is needed to keep this region of Southwest Virginia economically viable.

This system is a huge step in the direction that thin-seam research needs to take. Most of the equipment suggested for this proposed system is already available.

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Chapter 1

Background

Coal mining has been in progress in Southwest Virginia since before the time of the Civil War. During the period 1880 to 1930, the region changed from one of subsistence farming to economic dependence on the coal mining industry. As in many other mining regions, the following sixty years saw coal production being maintained or increased, using fewer employees, through the introduction of mechanized methods of mining. However, Table 1.1 shows that from a peak of 46.5 million tons in 1990, Virginia's coal output has declined. This has occurred for a variety of reasons including competition from both domestic and overseas sources, environmental concerns, geological considerations and exhaustion of many of the thicker coal seams. (Holman, McPherson, and Loomis, 1999b)

Table 1.1 Virginia coal production and numbers of production-related employees since 1990[+]

Year	1990	1991	1992	1993	1994	1995	1996	1997
Output (million tons)	46.5	42.3	42.5	40.1	38.8	35.9	36.8	36.9
No. of Employees	10,265	9,756	9,009	9,146	8,318	6,975	6,093	6,534
Productivity (tons/man hr)[#]	2.61	2.49	2.67	2.75	2.62	2.96	3.19	3.21

In 1951 a detailed survey was conducted to determine the state of Virginia's coal reserves. It was estimated that Virginia had 10.776 billion tons of bituminous coal in seams 14 inches and thicker. (Brown, 1952) Between 1951 and 1996, 1.577 billion tons were mined with estimated losses of 40%. (VCCER, Virginia Coal Directory, 1998) This effectively removed 2.208 billion tons from the resource leaving 8.568 billion tons[1].

The original estimate for measured, indicated, and inferred reserves between seam thicknesses of 14 and 28 inches was 5.041 billion tons[2]. (U.S.G.S., 1952) Subtracting

[+] Source: Virginia Department of Mines, Minerals and Energy
Total output divided by total man-hours.
[1] Effects on seam interaction not taken into account

this from the 8.568 billion tons of remaining resource yields slightly more than3.5 billion tons of coal in seams greater than 28 inches. (Holman, McPherson, and Loomis, 1999b)

Using estimated values from Bureau of Mines studies on Appalachian coal the amount of economically mineable coal can be obtained. From the remaining 3.5 billion tons in seams thicker than 28 inches, 34% will be recoverable but only a little less than 9 percent can be considered as an economic reserve. (Rohrbacher, 1993). The latter represents 321 million economic tons. With an annual production of 36 million tons, this yields some 9 years of mining for the region. As reserves diminish, production will decrease, slightly extending this life.

It is the inevitable consequence of any mining region that the fuel or mineral resources that can be mined economically will eventually be depleted. If the region is to continue as economically viable then other industries and opportunities for employment must be developed well before mining ceases.[*] This may be encouraged and funded by utilizing some of the taxation revenues generated by the mining, as has occurred in Southwest Virginia through the Coalfields Economic Development Authority (CEDA).

In the case of Virginia's Coalfields, it is estimated that a resource of over 5 billion tons of high quality coal remains in the ground in seams of between 14 and 28 inches thickness. The life of coal mining in Virginia would be extended greatly if techniques are developed to enable this valuable resource to be extracted in a manner that is both competitive and environmentally benign. An extension in the life of coal mining in Southwest Virginia would also extend the funds and time available to promote the development of other avenues of sustainable economic endeavor. However, it should not be expected that there will, ever again, be large numbers of persons employed in the coal mining industry of Virginia. In order to meet the challenges of the market, coal mining companies have little choice but to utilize increasingly sophisticated and automated equipment. If thin seams are to be extracted then those working places (faces) where the coal is actually won will be entirely manless. (Holman, McPherson, and Loomis, 1999b)

In 1996, the Virginia State Government enacted legislation that provided a tax credit to coal mining companies operating in the State. This was an enticement for those companies to remain in Virginia and to assist them in extracting some of the thinner seams. This has had the result of temporarily inhibiting the continuous decline in coal production. However, it can be no more than a stopgap. The only way that Virginian coal mining can be extended well into the next century, when coal will continue to be needed as a primary fuel source[+] is through the development of thin seam mining technology.

Unfortunately, there are significant barriers to the research and development that is necessary for the introduction of new mining technologies. The selection of an underground mining method is influenced by numerous factors, including tradition. A pervasive idea is that what has worked in the past, will continue to work in the future. This is a mode of operation that maintains the comfort of familiarity. Hard won experience with tried and true methods allows for sound decision making for as long as that method continues to be employed and remains viable. (Holman, McPherson, and Loomis, 1999b)

Many (but not all) American mining companies operate with a conservative outlook, and are reluctant to try something new until they see it working successfully elsewhere. The primary reason given for this unwillingness to innovate is low profit margins. This reluctance has resulted in a traditional lack of research and development by the American mining industries. Another contributing factor to this trend is that market trends often force coal companies to operate on a boom-bust cycle. While business is booming, many companies indicate that they have little need for research and development. Whereas when business is down they say they cannot afford it.

A further influence on this problem is that throughout most of the 20th century mining research and development has been conducted by government agencies in the U.S. and elsewhere. This effectively came to an end in 1996, in the United States, with

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[*] McPherson, M.J.(1996).The Tax Credit has Given Us a Few More Years, But What Then? Energy Outlook Vol XVI, No. 3. Fall, 1996.
[+] Coal. Energy for the Future (1995) National Research Council. National Academy Press

the dismemberment of the US Bureau of Mines, except for matters of safety and health that are now addressed by the National Institute for Safety and Health (NIOSH). The problem we are left with is that it is difficult to introduce new technologies and ideas that have potential to move the coal industry forward into the next century. New pathways that could yield safe, clean, and high production are not being explored, simply because they are new. For an innovative idea to bear fruit, it must first be tried. (Holman, McPherson, and Loomis, 1999b)

Within is presented a brief overview of current technologies utilized for underground coal mining in the United States. This is followed by a review of developments in highwall mining that show promise of having an application in the underground mining of thin seams. Some past attempts at thin seam mining are discussed. The more recent advances in the guidance systems used in autonomous mining machines are introduced. The state-of-the-art products of several manufacturers in addressing the integration of mining and continuous haulage systems are also highlighted. All of that background is employed in outlining a conceptual mining system for the underground mining of coal seams in the 14 to 28 inch range of thickness. Most of the equipment suggested for this proposed system is currently available. Furthermore, the mining and manufacturing expertise necessary for the cooperative development of such a system is already available in Virginia and its neighboring states.

Underground thin-seam mining in Virginia cannot hope to achieve the productivity (in tons per man-hour) of western surface mines extracting much thicker seams. However, there are distinct incentives to continuing coal mining in Virginia where the coal is not only low in sulfur content, but has a calorific value some 20 percent higher than western coals. About 50 percent of Virginia coal is of metallurgical quality while proximity to the coal terminals in Hampton Roads allows a ready route to the export market. About 37 percent of Virginia coal is shipped overseas. Furthermore, the environmental impacts of surface extraction are much greater than those of underground mining, a matter of increasing concern throughout the nation. These are some of the reasons that research into the feasibility of underground coal mining in Virginia is a worthwhile endeavor. (Holman, McPherson, and Loomis, 1999b)

Chapter 2

Current Underground Coal Mining Technologies

2.1 Introduction

This chapter presents an overview of the current methods of mining that are used in southwest Virginia and how these technologies are selected. These methods are all currently producing coal but, by themselves, do not have the flexibility to extend the life of the Virginia coalfields.

A vital factor in the selection of a mining method is the prevailing geology of the area. The properties of the confining strata and the coal itself, dictate which methods can and cannot be successfully employed in each seam. For example, in a longwall operation, an ideal strata sequence would have coal of a uniform thickness covering a large areal extent and a roof that is strong enough for adequate ground control, but not so competent that it inhibits controlled caving. By contrast, a room and pillar operation benefits from a strong roof, and this method allows greater flexibility for variations in the seam thickness and extent. Consideration must also be given to floor conditions, faulting, water, and depth. Depth is a very important factor, because as depth increases the stress on the support pillars increases, requiring the pillars to be larger. This means that less coal is extracted, causing the extraction ratio to decrease and results in room and pillar operations becoming uneconomical in deeper workings. Longwall operations may be preferred under those conditions.

Economics play a very important role in almost every decision made in the mining industry. Longwall mining systems are very capital intensive, requiring large initial investments and a long mine life to recover that investment. This limits the use of the longwall system to the larger mining companies that also have significant reserves. On the other hand, room and pillar operations can deal with more variable geology and require less investment, which can be recovered over a shorter mine life. Consequently, this method is amenable to smaller reserve tracts.

Environmental conditions in the mine also tilt the scale in favor of one method or the other. A longwall face has more concentrated releases of gas and dust because the rate of coal extraction from a single face is greater. Room and pillar operations have less immediate emissions of pollutants, but the ventilation is more difficult to control.

Another factor that comes into play is component interdependence of a mining method. In a longwall setup, numerous conveyors, the shearer, and the hydraulic roof supports are all interdependent. If one component goes down, the entire system is off line. With such a high capital investment, downtime causes serious financial losses. In room and pillar mining, the system components are not so interdependent. If an element in the system fails the consequences are less severe.

2.2 Elements of the Longwall System

In conditions where roof control is difficult, the coal of significant lateral extent, and of sufficient thickness, longwall mining is preferred. Longwall mining offers the benefits of enhanced safety due to its system of face supports that cover the entire working face. This method also allows higher extraction ratios, conserving valuable coal reserves. Some other advantages of this system are its flexibility in dealing with greater mining depths, multiple seams, and a significant reduction in roof bolting. (Bibb, 1992)

There are also significant disadvantages to longwall mining, such as the high capital cost of the required equipment. It follows that interruptions to production can have a serious economic impact whether they are short term such as starting and stopping the shearer, or longer term as longwall equipment is moved from a depleted section to a new panel. There may be problems with gas well location, seam thickness, and in soft floor and roof. There is also the possibility of this system being impractical beneath thick strong roof beds, due the size and cost of the required roof support as well as difficulties in controlled caving. Because of the areal extent required by a longwall section, variability in the seam thickness, roof and floor conditions, local faulting and the presence of wells can also limit the potential of this method.

There are two distinct types of longwall mining that are employed, advancing and retreating. While the advancing system is sometimes used in other countries, American companies prefer the retreating system. In the advancing method, the development entries progress slightly ahead of the advancing face and away from the main entries, while in the retreating system the entire section is developed prior to commencement of production from the longwall face at the inby end, and mines back towards the main entries. The focus of this chapter will be on the retreating system of longwall mining.

The development that is carried out for longwall pillars utilizes a continuous miner and the room and pillar method (Fig. 2.1). These entries are called the headgate and tailgate entries, with the ventilation return being the tailgate entry. When one panel is mined out, the headgate entry of the last adjoining panel often becomes the tailgate entry of the next.

The coal is removed from the extraction panel by a machine called a coal shearer that moves back and forth along the working face, riding on the sides of the armored face conveyor, fragmenting coal from the face, and dropping it onto the face conveyor (Fig. 2.2). This conveyor leads down the face and onto a main haulage conveyor in an entry of neutral ventilation. The roof above the working face is supported by automated hydraulic supports that are advanced as the mining progresses. Controlled subsidence is achieved by allowing the top to collapse behind the row of supports. The conveyor is snaked over, after passage of the shearer, by horizontal jacks attached to the self-advancing hydraulic supports.

Moving a longwall set up is a complicated, time consuming process, that entails dismantling, refurbishment and reconstruction of the computer control system, shearer unit, armored-chain conveyor and the support shields,. This process may involve the majority of the underground workforce for a period of one to two weeks at a cost of up to one million dollars. (Suboleski, 1998)

A method of longwall mining that was developed, primarily, in Germany was the coal plough (Fig 2.3). The coal plough replaces the shearer in a typical longwall setup. The coal plough can mine coal as thin as 18 inches, but beneath 30 inches currently available powered supports cannot be employed, greatly reducing the productivity of the system. The main mechanism of the plough is the armored face conveyor that has two main functions, to transport coal away from the face and to guide the plough unit on the face during mining. There are many problems with the plough system, such as the plough cutting into softer floors and deviating from the desired horizon. The plough also has difficulties in seams where the coal hardness is not reasonably constant across the face. The feasibility of this system is questionable in seams less than 24 inches in thickness.

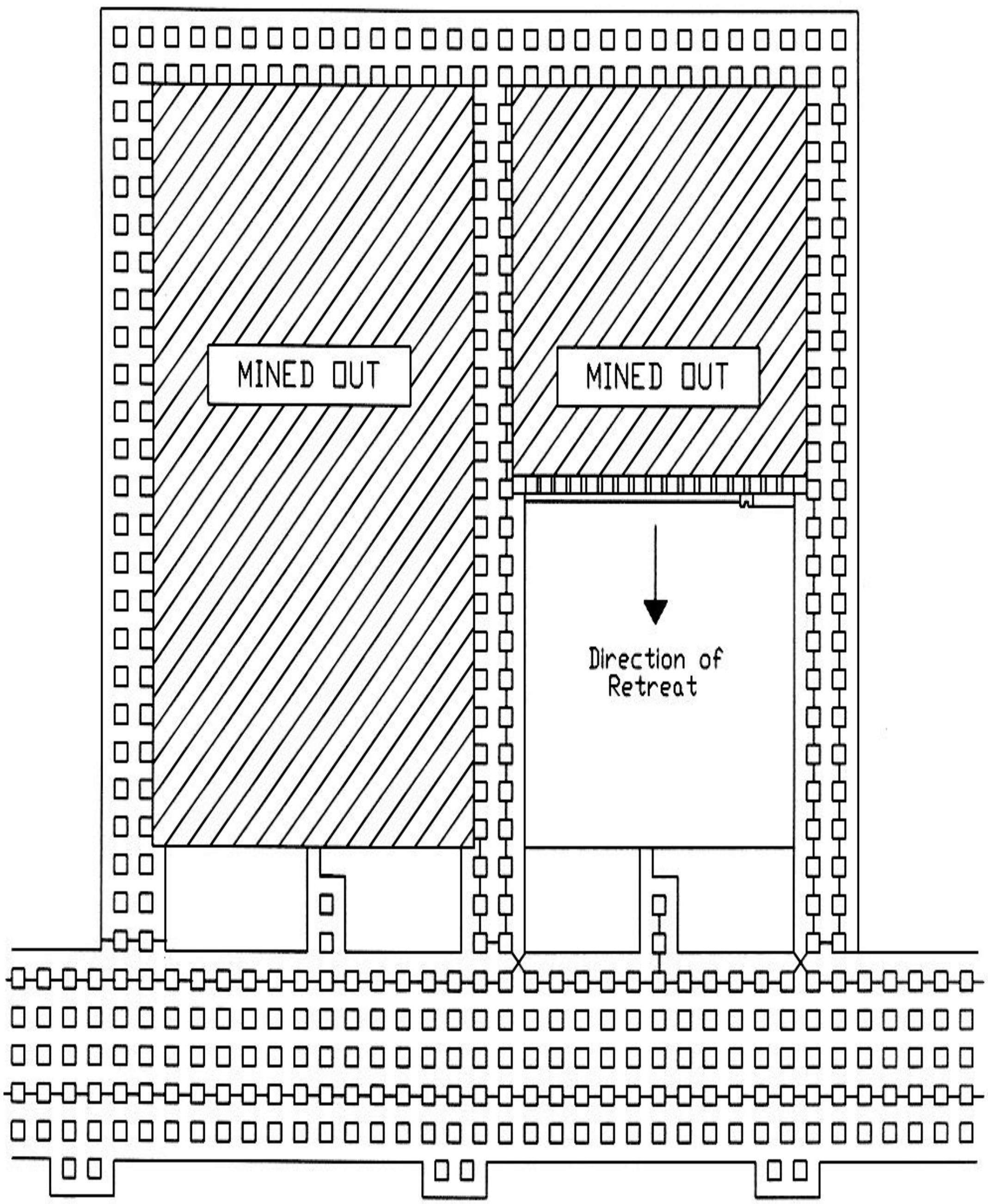

Figure 2.1 Typical Longwall Panel Layout (not to scale)

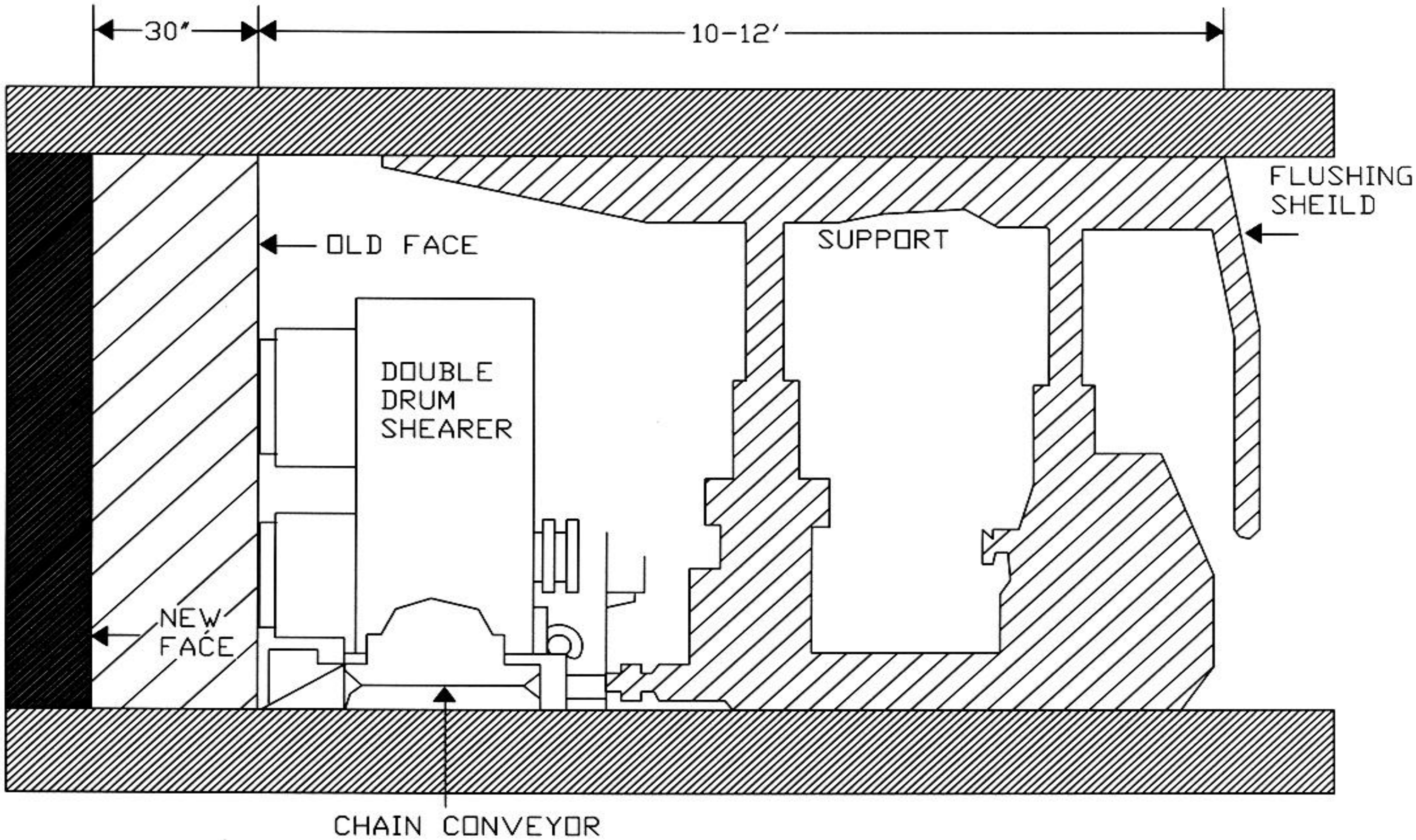

Figure 2.2 Longwall face section with Conventional Double-Drum Shearer

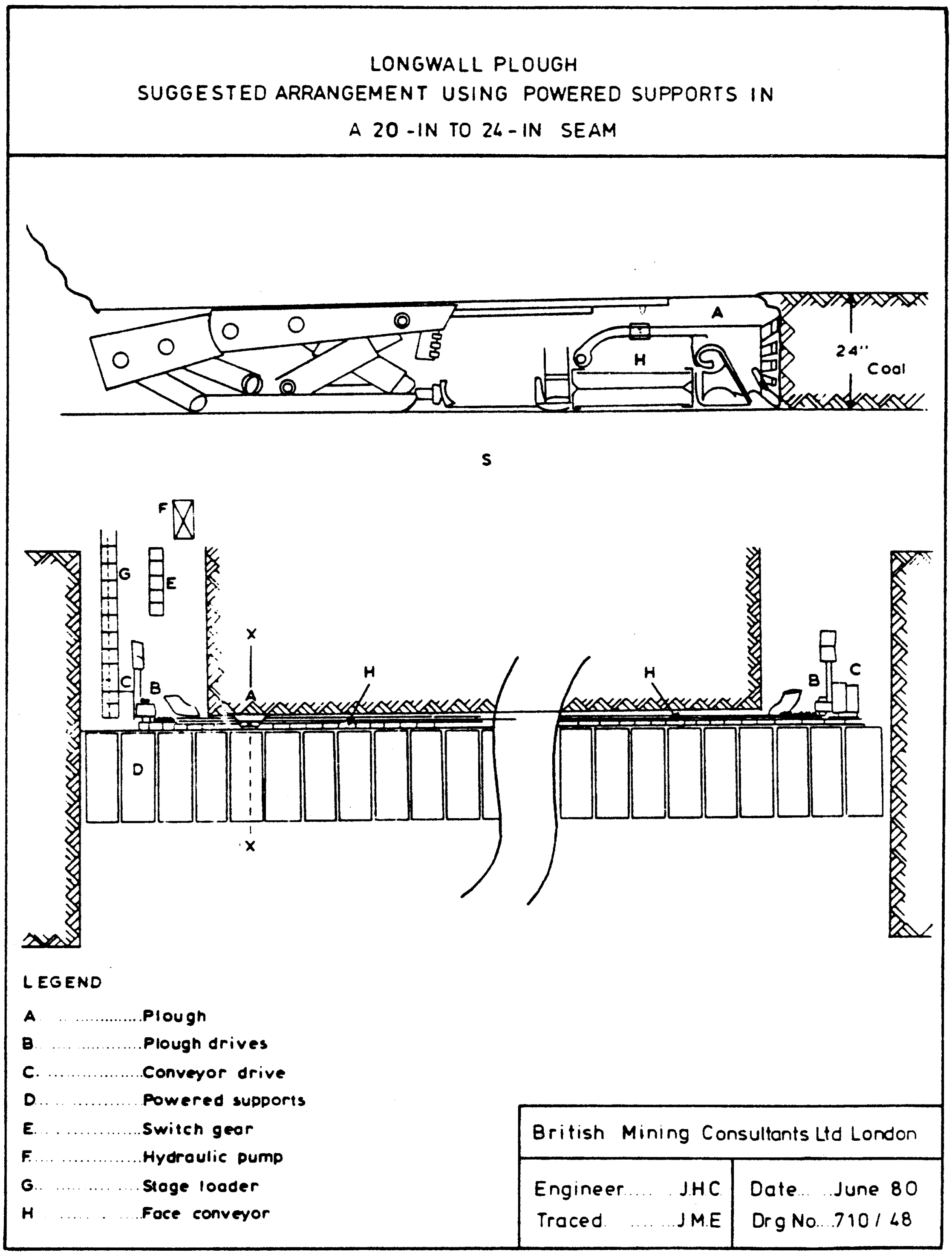

Figure 2.3 Longwall Plough Arrangement
(Clark, Cauldon, and Curth, 1982) reproduced with permission

2.3 Elements of the Room and Pillar System

The room and pillar system is the older and traditional method of underground coal mining in the United States. A typical layout is illustrated on Figure 2.5. Coal is removed from the working faces as the rooms are advanced. Cross-cuts, connecting the rooms are also mined leaving pillars of coal for support. The rooms and cross-cuts are typically about 20 feet wide and of a height consistent with that of the seam.

Prior to the development of continuous mining technology, the conventional room and pillar method was composed of undercutting the coal, drilling, blasting, and loading. These steps have largely been supplanted by continuous mining machines that combine the processes of fragmenting the coal from the face and loading it on to haulage equipment for movement to the main coal transportation system. This has greatly reduced the equipment needed to conduct room and pillar mining. The process is now simplified, using only a continuous miner, a means of haulage, and a roofbolter. The latter inserts steel, cable or fiber bolts into the roof, pinning together the immediate overlying strata to provide roof support.

The process is cyclic in the majority of current room and pillar operations. The continuous miner advances on one half of the face width, then the other for a distance such that the operator remains under supported roof. The continuous miner is then trammed to mine in an adjoining room while the roof is bolted. Mining and roofbolting operations alternate in any given room.

Several designs of continuous miners have been manufactured, such as the Joy Ripper Miner, boring miner, milling-head miner, auger miner, Jeffrey 101 MC Helimatic, and the boom-type miner. The most commonly employed type of continuous miner is the milling head miner. (Fig 2.4) This type of miner has a rotating drum that rips the coal from the seam. The fallen coal is then collected by gathering arms and moved toward the back of the miner by an onboard chain conveyor. This conveyor runs up an adjustable boom that is used to load a shuttle car, which carries the coal to the main belt. A typical system uses two shuttle cars traveling sequentially between the continuous miner and the loading point on to a belt conveyor.

In most current room and pillar systems, the coal is transported by shuttle cars from the continuous miner to a feeder breaker which, in turn, loads on to a section conveyor. A shuttle car is effectively a mobile bunker with a built in steel conveyor for the purpose of loading and unloading. (Fig 2.5) Shuttle car traffic is controlled to increase both loading time and mining efficiency. Scheduling the positions of all mobile equipment is very important to the efficient operation of a miner section. The roofbolter must be bolting roof, while the miner and shuttle cars are mining the next cut.

The production section is usually composed of either five or seven entries, employing split ventilation (Fig. 2.6). This means that fresh air is brought in through the central entry or entries and is split and directed across the working face. The return air is taken down the outermost entries and out of the mine. The entries are usually of the order of twenty feet in width. The depth of the overburden determines the size of the support pillars. The cut sequence is determined by pillar size, cut length, and number of entries.

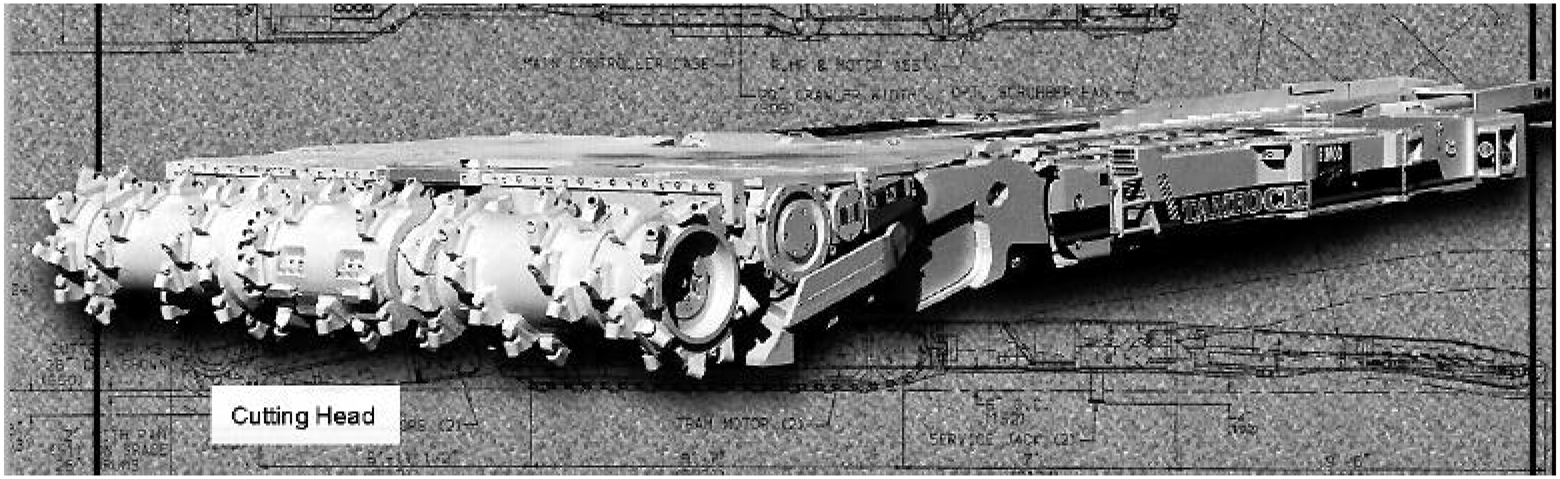

Figure 2.4 Continuous Miner (Tamrock Inc. 1998) reproduced with permission

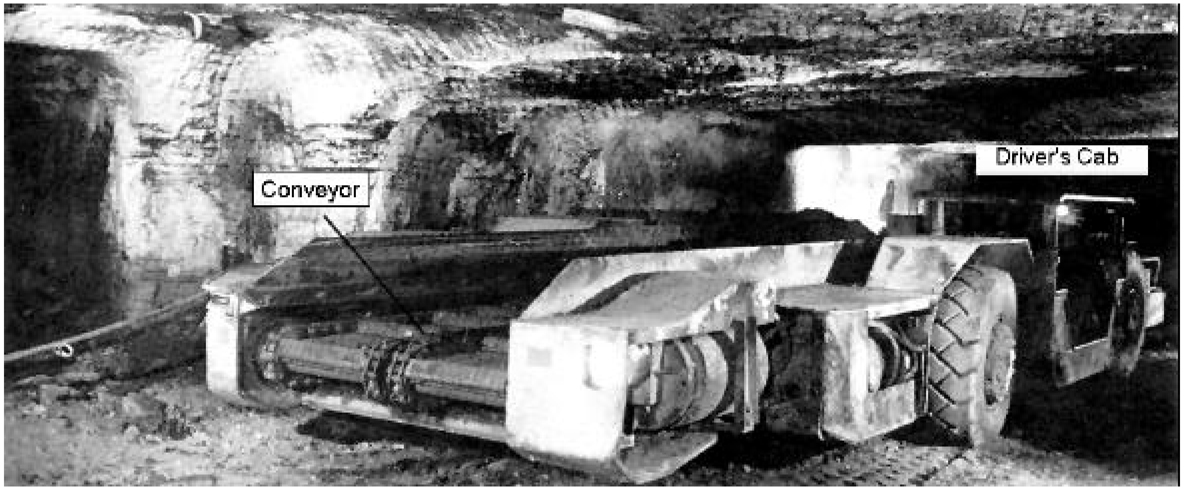

Figure 2.5 Shuttle Car (Stefanko, 1983) reproduced with permission

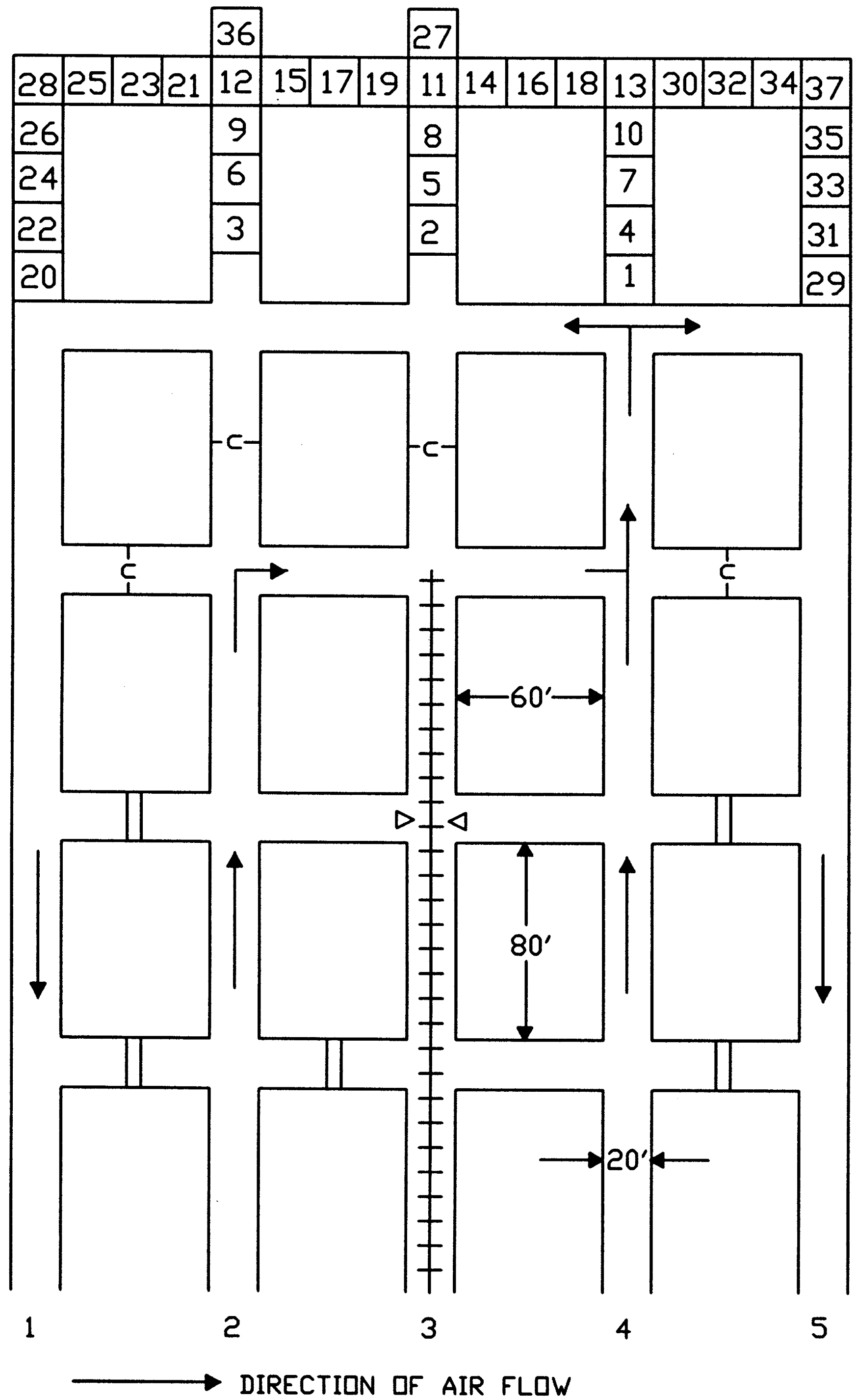

Figure 2.6 Room and Pillar Layout (Stefanko,R. 1983)

Chapter 3

Highwall Mining

Highwall mining is the practice of mining coal by tunneling into the exposed highwall face of a surface mine and removing the coal. This is accomplished in a number of ways including auguring, addcar systems, and Archveyor (Arch Technology Corporation) systems. Highwall mining allows the recovery of coal that would otherwise be lost, and offers high productivity measured in tonnage per man-day. (Walker, 1997). While highwall mining has been developed to extend the recovery of coal from surface mines, it does, additionally, provide an opportunity of examining how such methods may be adapted for the underground mining of thin seams.

Highwall auger systems are composed of three main components: the cutting head or heads, flights for moving the coal and the motorized drive. (Fig 3.1) The cutting head cuts into the face at right angles to the exposed face, while being pushed by the flights and turned by the drive unit. (Fig 3.2) As the head cuts coal, the flights carry the coal back out to the base of the highwall. When the flight reaches the extent of its length, the flight is uncoupled from the drive and another flight is added to the string. This is repeated until the drill string reaches the desired cutting length. Then a new hole is drilled further down the face, leaving a small pillar to support the overburden. (Clark, 1982).

The main weakness of this method is difficulty in maintaining straight holes over long distances. New radar technologies are being developed to maintain hole and to increase the auger range to over 600 feet. (Mowrey, Ganoe, and Monaghan, 1995).

Any curvature of the highwall also has serious effect on the attainable recovery. Concave highwalls cause auger holes to fan out, while convex highwalls cause auger holes to intersect. Both of these situations result in lower recoveries, and intersecting holes also pose a risk of collapse. The highest possible recovery comes from having a straight highwall face. (Fig 3.3) (McCarter, 1992).

The addcar system (Figure 3.4) was also designed for highwall mining. The current system is not applicable for thin-seam application, because it can only be used in seams of more than 90 cm or 35.4 inches in thickness. This system recovers up to 60% of reserves, using 12.5 m-long individually powered Addcars. The standard system depth is 365m, but the new upgraded 'Highwall Hog' has an extended range to 500m. The Addcar system utilizes a continuous miner and Addcars that utilize chain conveyors to remove coal from the entry. The advantages claimed for this system over its closest competitors are: 150% more penetration; 90% more annual production capacity; 82% more installed horsepower; and it can produce the same tonnage in a reduced length of highwall. (Walker, 1997)

The newest innovation in highwall mining comes in the form of the Archveyor. (Fig 3.5) This mining system receives its cutting power from a highly modified Joy 12CM continuous miner capable of cutting a 3.8m-wide from 1.8 to 4.9m thick. The Archveyor itself follows the miner into the heading, transporting the coal out. The Archveyor has drive units every 7.5m, enhancing both vertical and horizontal flexibility. A chain conveyor is used for coal transportation and when lowered and reversed, to move the system forward. When mining is underway, hydraulic jacks lift the Archveyor clear of the ground allowing the conveyor to transport coal. (Walker, 1997)

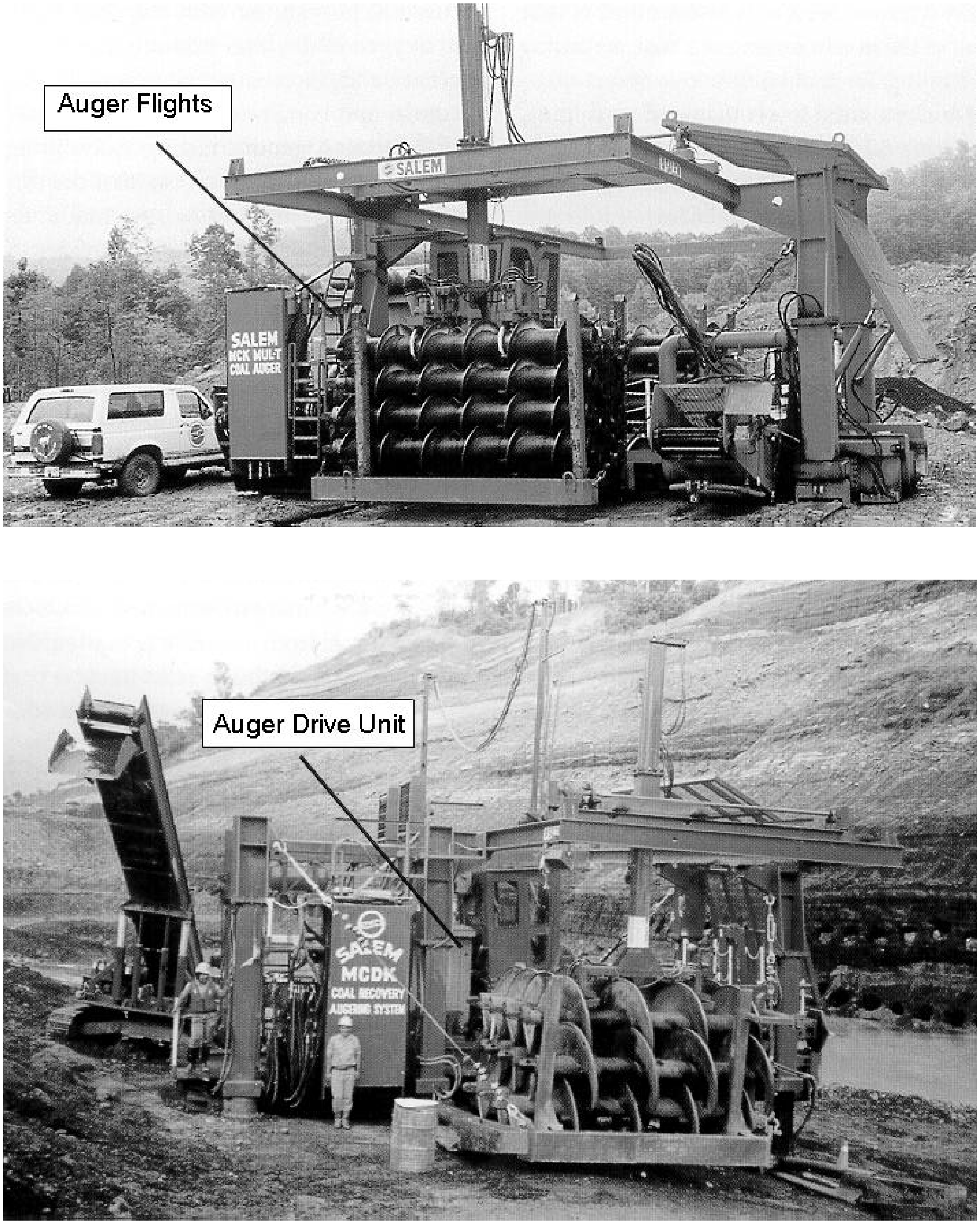

Figure 3.1 Highwall Auger Unit
(Walker, 1997) reproduced with permission

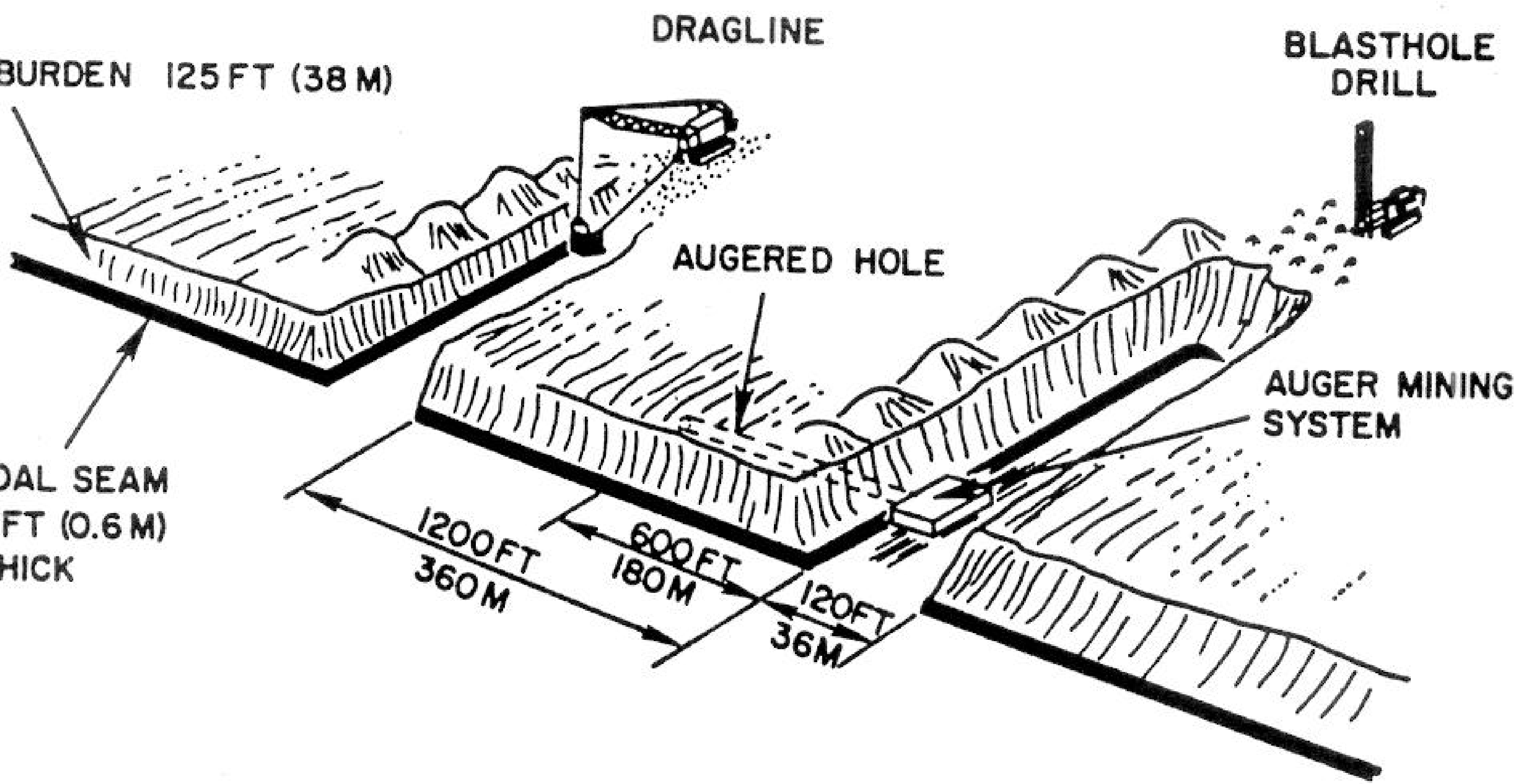

Figure 3.2 Overburden Removal
(McCarter, 1992) reproduced with permission

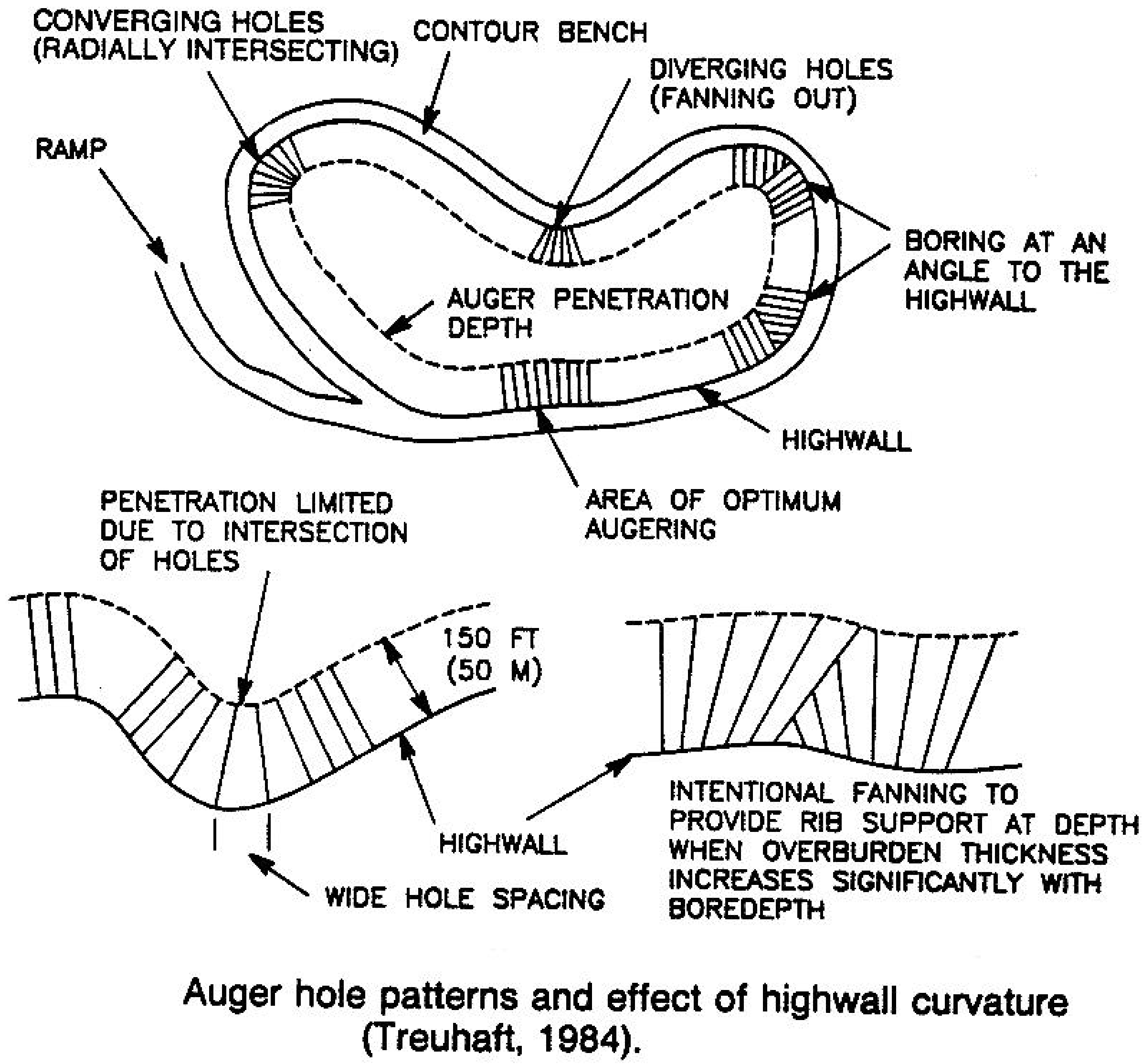

Figure 3.3 Auger Hole Pattern
(McCarter, 1992) reproduced with permission

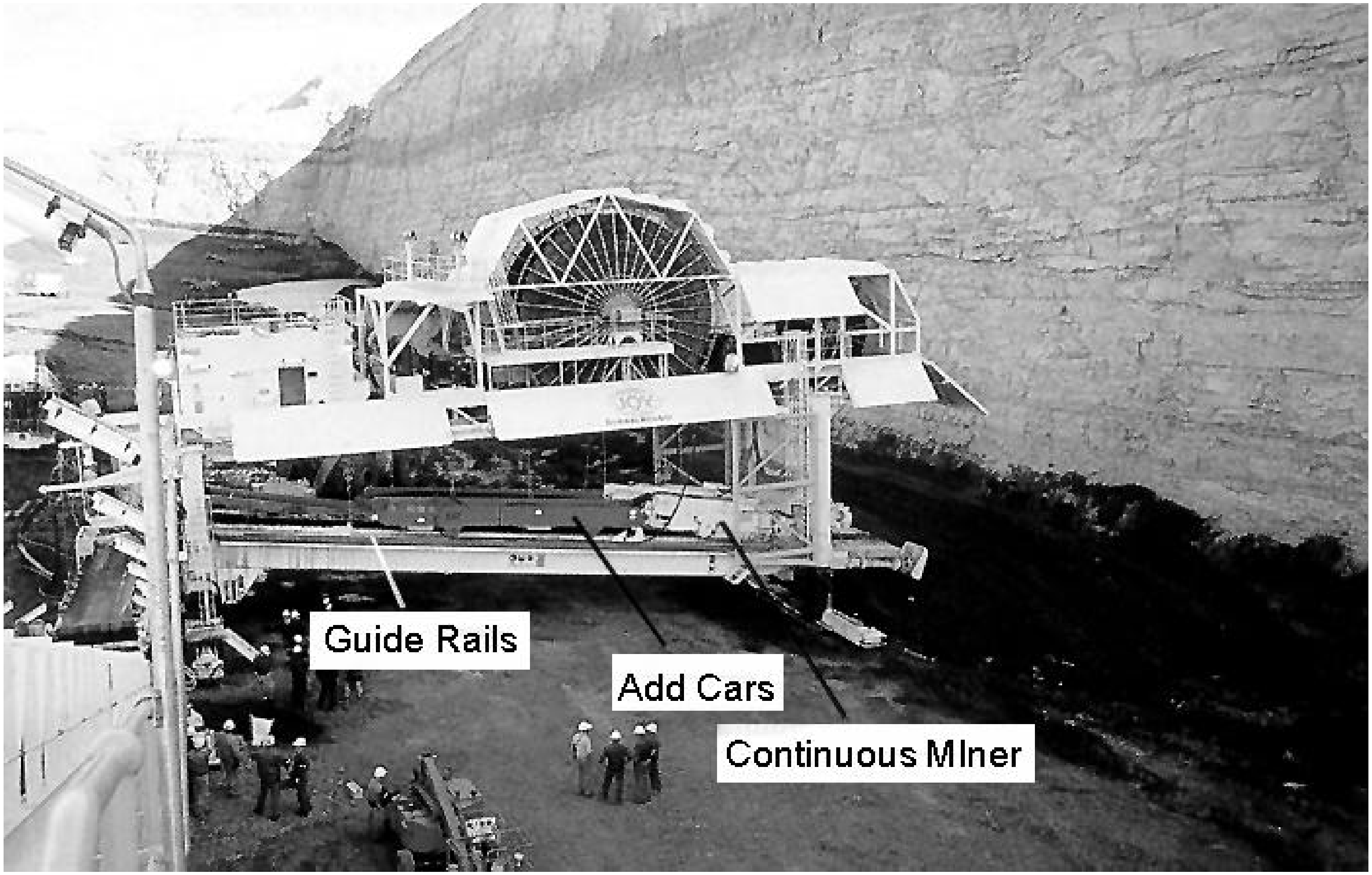

Figure 3.4 Highwall Addcar System
(Walker, 1997) reproduced with permission

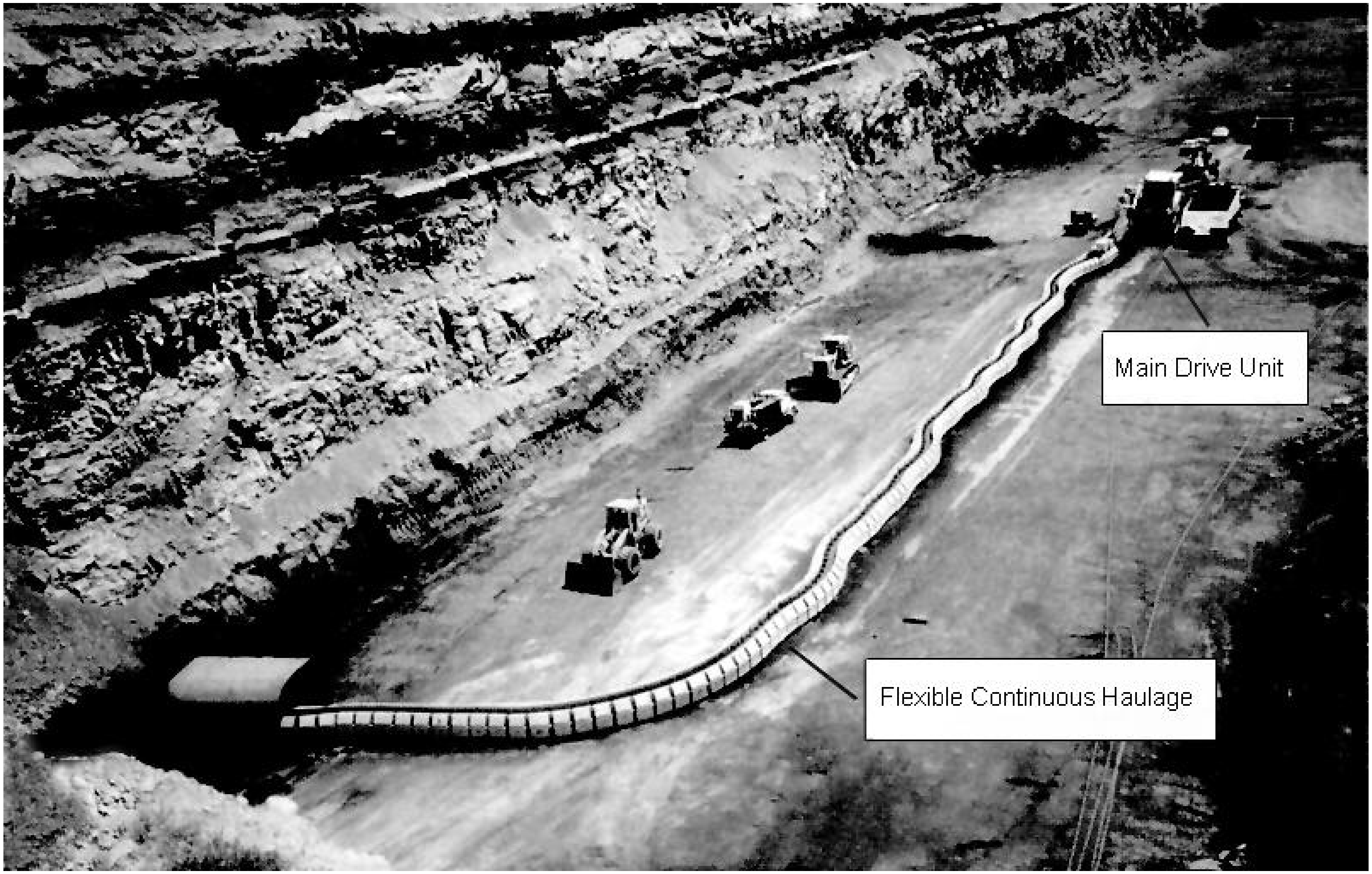

Figure 3.5 Archveyor System
(Arch Technology Corporation) reproduced with permission

Chapter 4

Past Attempts at Underground Mining of Thin Seams

4.1 Mole Miners

A mole miner is a machine that can cut a "blind" narrow face entry while being remotely-operated. This technology is composed of several sub-systems; haulage for coal removal, ventilation, monitoring, and a control and pushing system which advances the miner into the face. Mole miners have been employed in the USA, UK, and the former USSR with less than outstanding results. This method has become more feasible for thin-seam mining with the development of autonomous mining machines and has the advantage that personnel do not enter the extraction zone.

The first such miner developed in America was by the Union Carbide Company to mine coal outcrops exposed in the highwalls of strip operations, once stripping was no longer economic. (Clark, 1982) This miner was self-propelled, moving its haulage system behind it. The cutting height was kept to no more than 5 ft. The miner maintained its position in the seam utilizing pick force sensing to detect differences in rock hardness.

The most recent application of this technology in the former USSR has been in steep seams that would not require a separate haulage system to remove coal from the working face. Other attempts in more level seams have been discarded. (Clark, 1982)

In the United Kingdom a machine called the Collins miner was created specifically to mine thin seams. (Fig 4.1) Several versions were designed and tested. Many promising results were produced but due to economic factors, including the growing availability of natural gas as an alternate fuel source, the research was not continued.

The Collins miner was intended to cut a 300 ft long entry that had dimensions of 6 ft 3 in wide, from development sections of dimensions 12 ft wide by 7ft 6 in high.

Numerous consecutive parallel entries would be driven to obtain the desired production. The width of the rib pillars between the entries was dependent on the thickness of overburden and rock strength. (Clark, 1982)

The cutting head of the Collins miner was composed of three overlapping augers. These were driven from a water-cooled gearbox by a single 120-hp electric motor. Un-powered cutting blades that squared the three overlapping circles removed the upper and lower cusps between the auger holes. A small flight conveyor moved the cut coal to a belt conveyor behind the machine.

The entire machine was mounted on skid plates that were connected to the main frame, by hinges in the front and by lifting jacks at the rear. The jacks controlled the angle of the cutting head. Side jacks were also included in the design to help facilitate horizontal steering. The launching platform was mounted on rails and carried the miner from hole to hole. The platform included jacks for positioning the miner, and pushing cylinders and a pawl mechanism for driving the miner. (Holman, McPherson, and Loomis, 1999b)

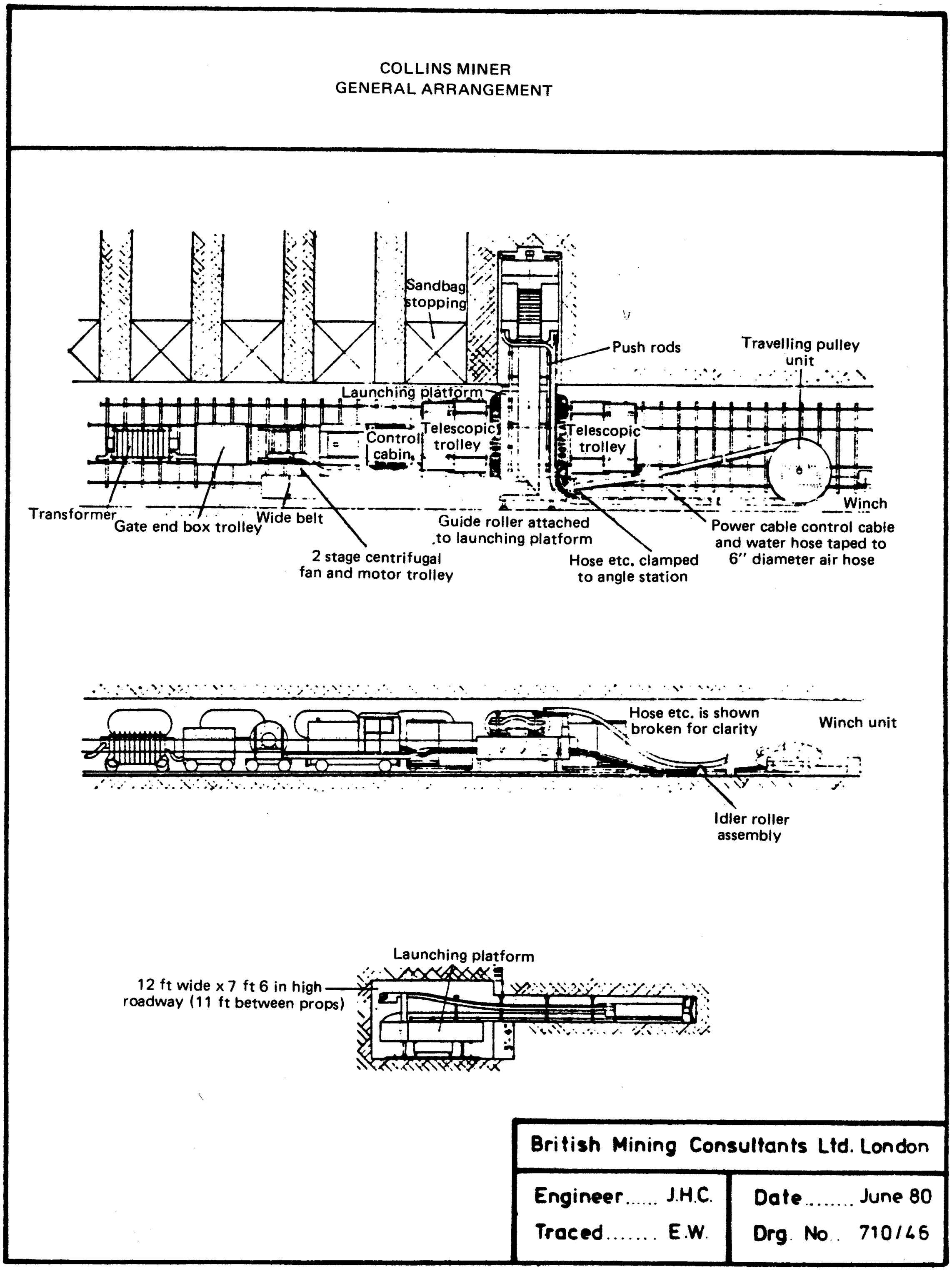

Figure 4.1 Collins Miner Layout
(Clark, Cauldon, and Curth, 1982) reproduced with permission

4.2 A Brief Overview of Full-Face Miners

Full-face miners are mining machines that span the entire width of the mining face and are advanced or retreated along the entire width at one time. Due to the fact that these systems utilize so much equipment, there must be personnel on the working face for the purpose of maintenance. The size of the equipment has, to this time,limited the application of full-face miners to seam dimensions of no less than 16 inches.

Examples of full-face miners include the Miniwall (Fig 4.2), the In-Seam Miner (Fig 4.3), and the Yarmak Miner (Fig 4.4). All of these systems remove coal from the face by means of a laterally moving, cutting device. (Holman, McPherson, and Loomis, 1999b)

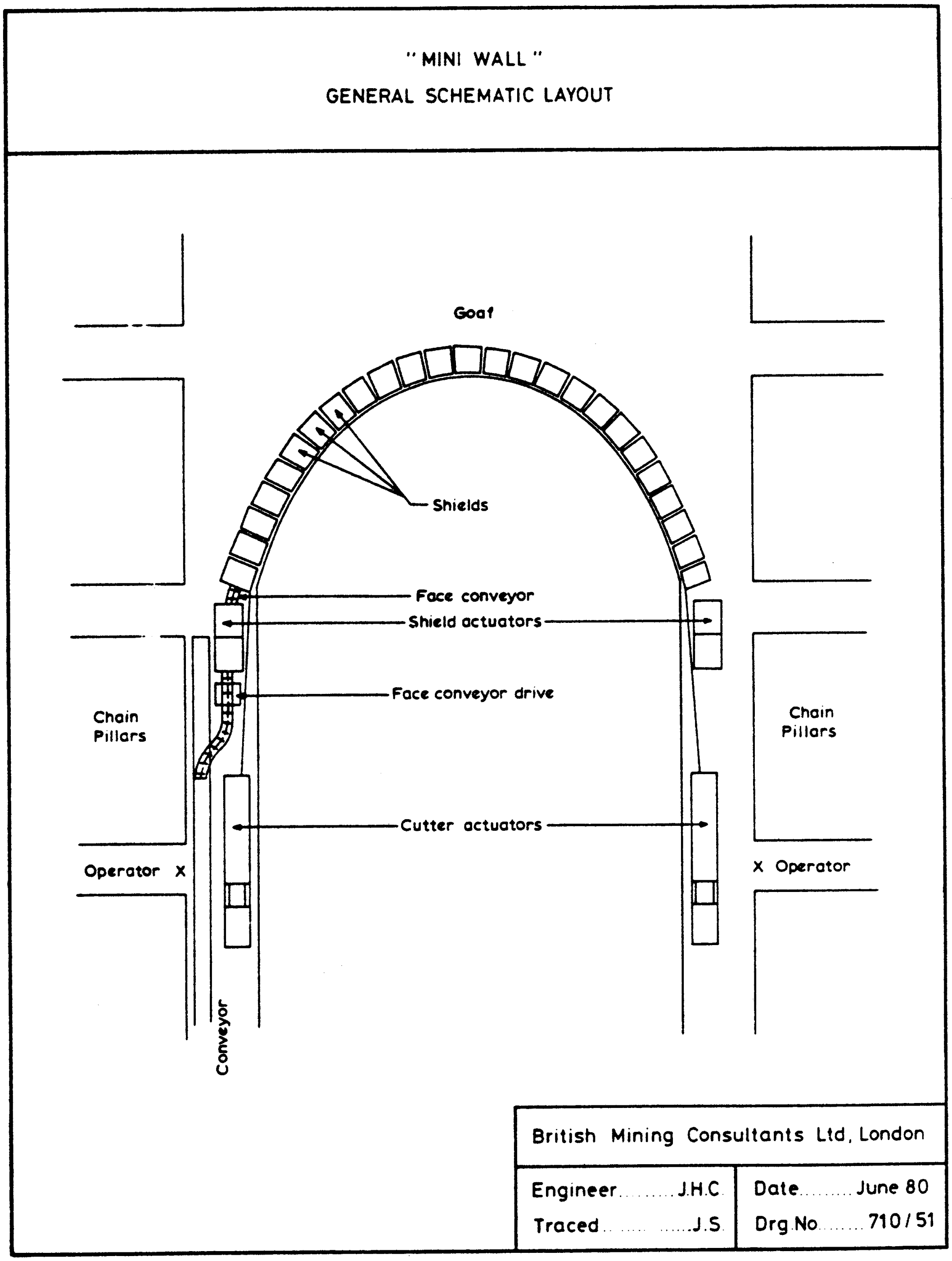

Figure 4.2 "Mini-wall"
(Clark,Cauldon, and Curth, 1982) reproduced with permission

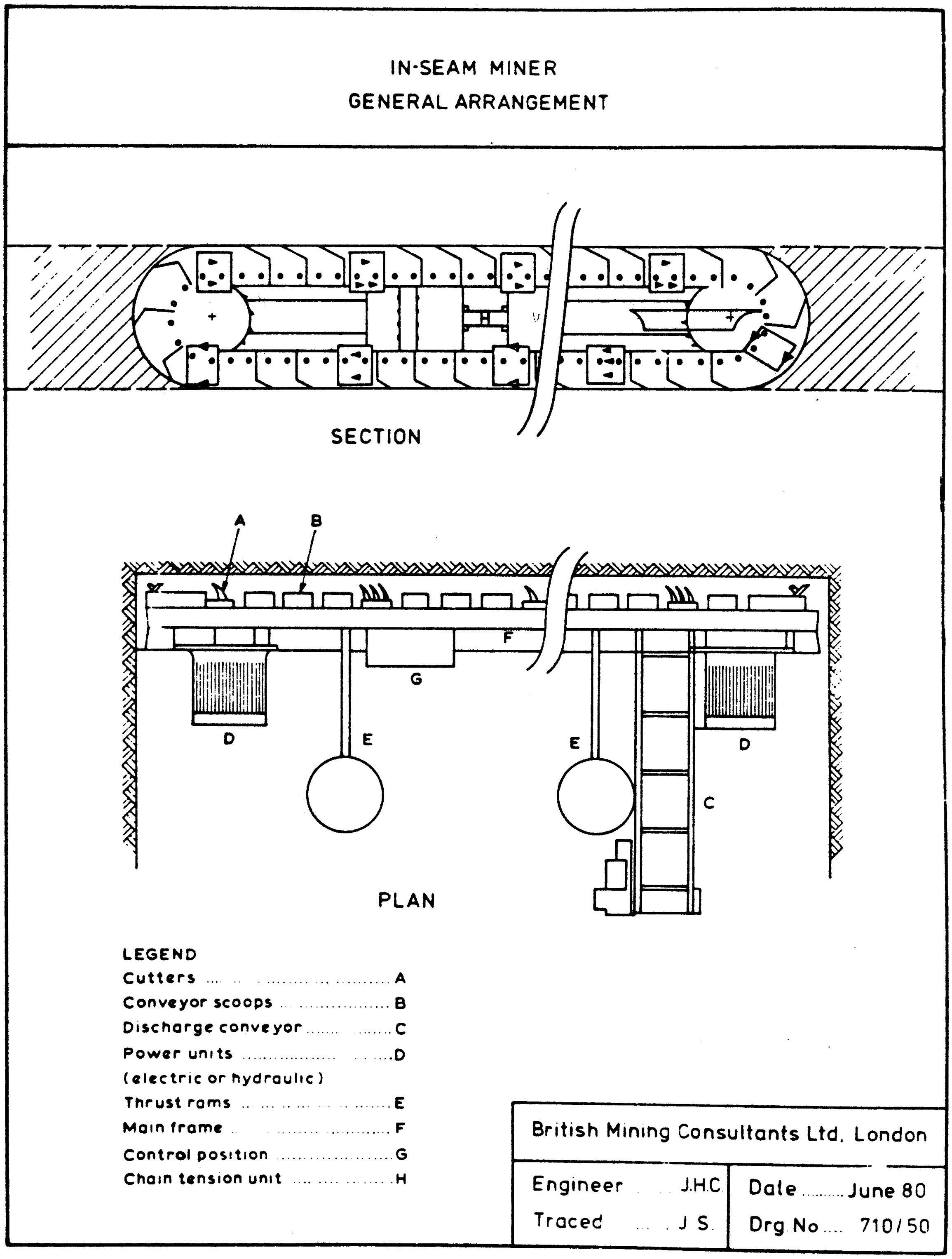

Figure 4.3 In-Seam Miner
(Clark, Cauldon, and Curth, 1982) reproduced with permission

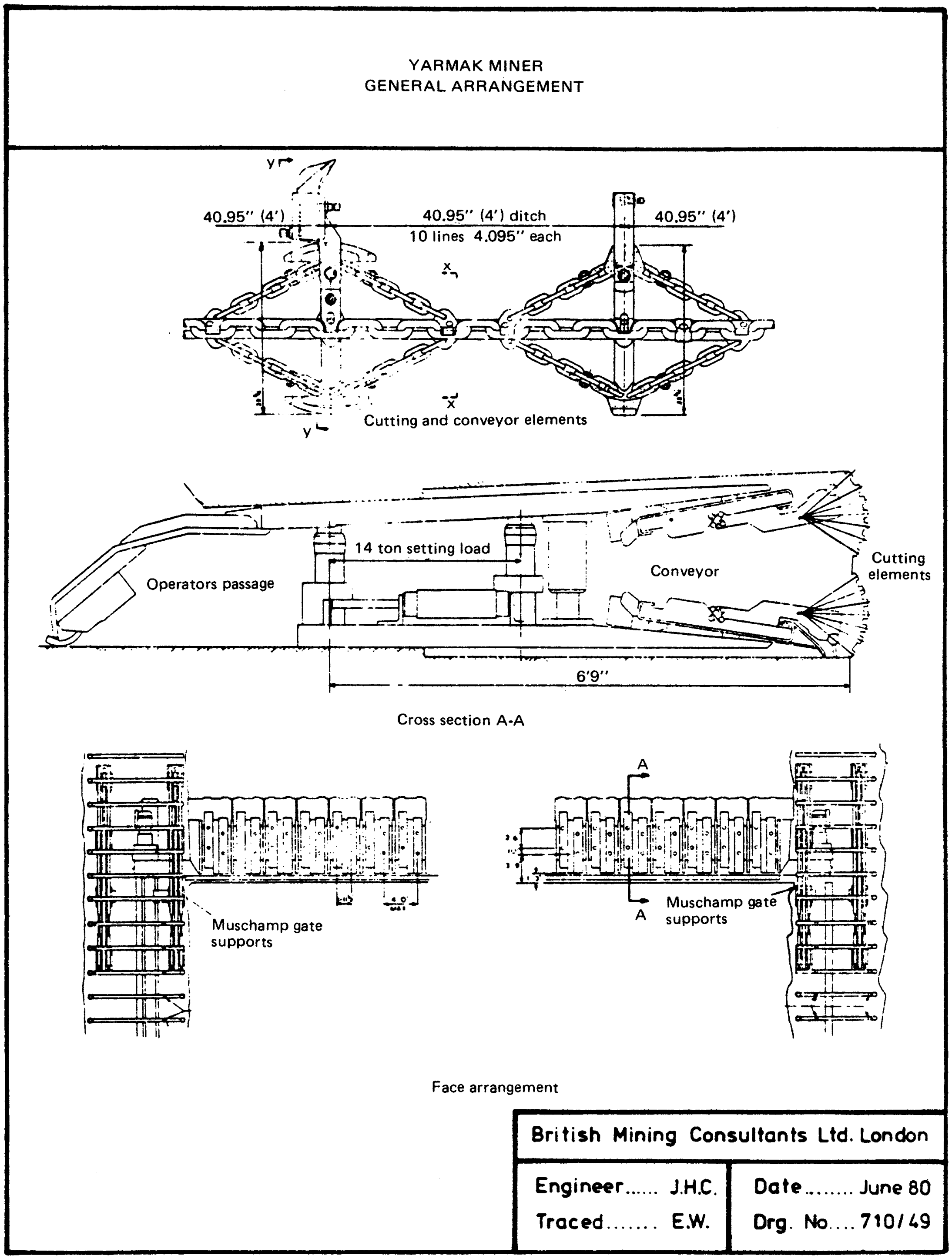

Figure 4.4 Yarmak Miner
(Clark, Cauldon, and Curth, 1982) reproduced with permission

4.3 Scraper Boxes

The scraper box was one of the simplest longwall-type systems. Originally scraper boxes were used simply as haulage units on hand-worked longwalls in moderately thin seams. The scraper box was made up of a box that is open at the top, front, and bottom. A scraper blade was hinged into the rear of the box. When drawn forward the blade took on a closed position preventing the contents from leaving the box. When the box was drawn backwards the blade would rotate into an open position allowing the box to pass over any objects in its path. To give the system more reach, multiple scrapers were usually employed on the same wall. Each scraper carried its contents to the end of its section of wall, dumping it where the next scraper could pick it up and pass it along to the head gate of the wall.

One of the premier scraper box systems was the Haarman scraper box, which utilized a heavy skid board to press the scraper box against the face. The caused the box to take shallow cuts off the face each time it passed over it. In later systems the skid board was removed and a heavy-duty chain that ran the length of the wall took its place. In these methods the ends of the walls were kept slightly ahead of the center, in a bow shape, to facilitate the movement of the chain. Since the tension in the chain kept the box against the face cutting coal, this system came to be known as the "chain tension scraper box" (Fig 4.5). By removing the skid board, the job of constantly moving the board was also eliminated, hence greatly improving efficiency. The only job that remained for personnel was the installation of roof supports. (Clark, 1982)

This system, despite all of its advances, still required the use of personnel on the working face, to install roof supports. This gave the system a minimum seam height of 16 inches. The pulling forces needed to move the boxes on the face required a large winch to generate them. Skid boards prevented easy access to the scraper boxes.

In Germany this system produced five tonnes per man shift. The tension used in the chain varied from 4,000 to 8,000 pounds. This system had many problems that kept it from being a widely used system. The shape of the wall coupled with the use of the chain did not provide a sufficient normal force to generate an adequate rate of advance. Also the bow shaped wall had extra stress accumulated on the lagging section of the wall. This set up also limited the lengths of the wall and greatly hindered the economic potential of this system. (Clark, 1982)

In any system where caving is desired behind the working section, it is important to have a straight break line along which the roof can fail. Curved faces do not provide this straight break line and make it more difficult for the roof to fail. If the roof fails to break in a timely fashion, excessive stresses can accumulate on the supports. Keeping the working section straight, or on-line, maintains responsive, predictable caving and facilitates smooth operation of the system.

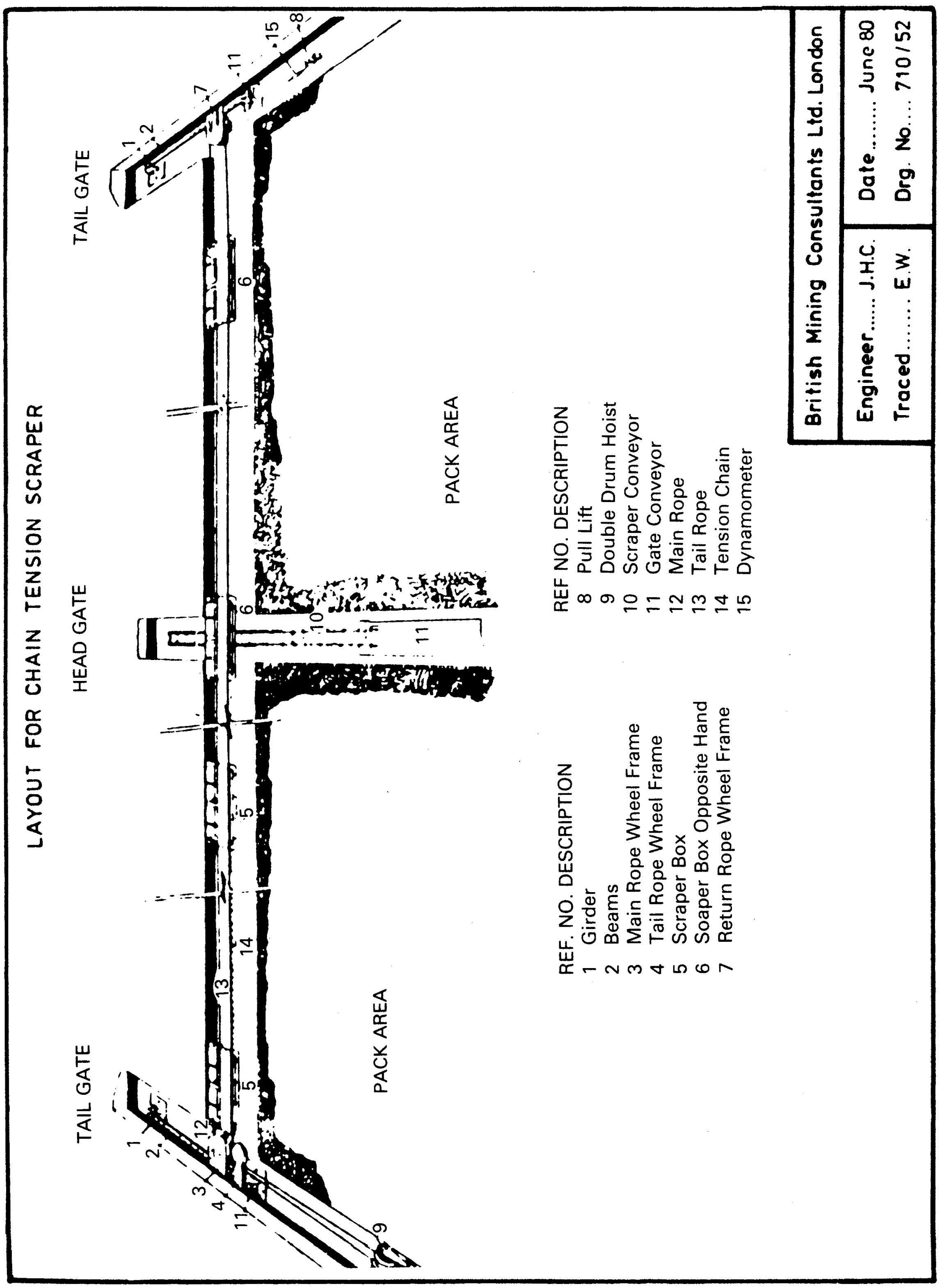

Figure 4.5 Chain Tension Scraper
(Clark, Cauldon, and Curth, 1982) reproduced with permission

Chapter 5

Autonomous Mining Machines

5.1 Introduction

Any future system of underground mining of thin seams must be completely automated and operate without the presence of personnel actually on the working face. This is a prerequisite, not only because of the limited height of the faces but also to maintain the costs of mining at a competitive level. It is also an unfortunate fact of geology that coal seams seldom lie in a flat uniform plane of constant thickness. For these reasons, mining equipment designed to cut and remove coal from thin seams must be autonomous; that is, it must have a significant degree of artificial intelligence in order to navigate minor anomalies in geology or obstacles along the coal haulage routes.

There are three types of mine navigation; face, local, and global. Face navigation is the process of positioning the mining equipment within the area of the working face. Local navigation is the guidance of mining equipment in the immediate area, excluding the face, encompassing guidance around corners and obstacle avoidance. Global navigation is utilized when maneuvering mobile equipment between various points in the mine. Local and global tasks are very similarly performed, while face navigation is specific to the equipment and the task being performed. (Anderson, 1989)

The required objectives of any guidance system are to align the equipment at the face, achieve proper guidance during cutting, adequately tram through entries, and turn corners. Other requirements include the ability to generate a path by means of a computer model of the mine, and be able to follow that path. This model must be constantly updated to keep up with the changing configuration of the mine. (Anderson, 1989)

For the purpose of this method, face navigation only will be considered. The basic assumption is that personnel will guide equipment through the non-face areas. The two most critical aspects of face navigation are the alignment of the continuous miner and guidance of the machine during cutting. It is also important that the miner have the ability to detect rock-coal interfaces, and to stay within the coal seam Coal-rock Interface Detection (CID) technologies are employed for this purpose. (Anderson, 1989)

5.2 Coal-Rock Interface Detection (CID)Technologies

One necessary component of any automated coal mining method is seam-following technology. This is usually achieved through detecting the coal-rock interface at the boundary of the seam. This has been abbreviated to CID for coal-rock interface detection. There are numerous types of CID technology that are being employed and tested around the world. These include natural gamma radiation (NGR), vibration, infrared, optical / video sensors, radar, and pick force. Each of these methods has its strengths and weaknesses that determine whether it can be utilized for fully automated mining in the underground mining environment. (Mowrey, 1992)

Natural Gamma Radiation (NGR) CID Technology

The first of these methods, natural gamma radiation, is the only method already being employed in commercial mining applications, with over one hundred and fifty units around the world. This method works on the principle that shale, clay, silt, and mud have higher levels of naturally occurring radioactivity than coal. This is due to their content of minute quantities of radioactive potassium (K-40), uranium, and thorium. The attenuation of the NGR by the coal can be used to measure the thickness of the coal between the sensor and the rock interface. The measured NGR decreases exponentially as a function of coal thickness.

This method has many strong features that make it a viable option for use in automated mining operations. It can measure coal thickness readings from 2 to 50 centimeters. The indication is easily read from a display panel. This compact unit can be mounted on the miner itself, where it will not be in the way, and it is applicable in most seams. The most prevalent applications to date have been on longwall units.

There are a few inherent weaknesses in this system that arise from the distribution of radioactive material in the seam. For example, NGR levels vary from seam to seam, requiring the units to be calibrated for each seam in which they will be used. Another minor problem is that the NGR levels can vary throughout a seam, depending on the levels of radioactive constituents that were present at the time of geologic deposition. Another problem that presents itself, at times, is that of rock partings, layers of rock that sometimes intrude into a continuous coal seam. These can show false seam boundaries to the unit by indicating a coal-rock interface within the seam. (Mowrey, 1991)

Vibration Based CID

As coal or rock is being cut, it induces different patterns of vibration. By interpreting the change in vibrations produced, the sensor can detect when the machine has started cutting boundary rock instead of coal. The three types of vibrations that are studied are machine vibration, in-seam seismic, and acoustic vibrations. The strengths and weaknesses of this method vary depending on which type of vibration is being examined. When studying machine vibration the sensors can be mounted on the machine itself so that the sensors are out of the way and need not be remounted as mining progresses. This method has good potential when adaptive signal discrimination technologies are used to help interpret the vibrations. This system also gives immediate feed back when rock starts to be cut, so mining can proceed up to the roof.

In-seam seismic and acoustic sensors must be attached to the coal itself, requiring the sensors to be remounted as mining progresses. This is inefficient, necessitates personnel to be at the working face to mount the sensors, and contradicts one of the main reasons for having an automated mining method, i.e. to eliminate workers from the production face. Hence the in-seam seismic, or acoustic sensing methods are not practical in thin seams. (Mowrey, 1991)

Infrared CID Technology

Different types of strata release different amount of infrared radiation while being cut. This is primarily a factor of their physical characteristics. Infrared sensing devices can measure the values of infrared radiation emitted from the cutting zone. Changes in the intensity of emission can be attributed to changes in the strata being cut. This informs the computer when the miner is leaving the seam, so that corrections can be made.

This method has distinct advantages that merit its further development. The radiation readings can be taken from a location behind the cutting drum, from a remotely mounted sensor, even when the drum is obscured by dust and water sprays. This method can be used under any type of roof, allows coal to be mined up to the roof, and yields an instantaneous response time. (Mowrey, 1991)

Optical / Video CID

The theory that is applied here is that different types of strata have different reflectivities, meaning that they reflect different amounts of light from a similar source. This physical attribute can be exploited to discern the difference in two materials using a reflected light source. This technology is not very accurate but is greatly improved by the addition of video cameras and image analysis equipment.

These sensors, like the infrared sensors, can be remotely mounted and, with the appropriate video cameras, can see through moderate dust and water sprays. Heavy dust and water can cause problems. Another benefit of this system is that data obtained from the video systems can also be employed for guidance purposes. (Mowrey, 1992)

Radar Based CID

Radar based CID utilizes a single antenna, which transmits and receives Doppler radar pulses. A network analyzer is utilized to control frequency, and for signal analysis. The signals are attenuated as they pass through coal and bounce off the density interface of the confining rock. The attenuation of these waves can be interpreted to find the distance to that interface.

This system has reliable accuracy and operates well under most roof conditions. Another application of this technology is that it may be used to measure the thickness of the ribs, to ensure straight holes in highwall-auger mining. Some inherent problems with this system are that it does not work well in coals with wave dispersing properties, and it requires the transmitter to be within 10 cm of the coal. (Mowrey, 1995)

Pick Force CID

This CID method measures changes in the force exerted on one or more of the picks on a continuous miner. The energy required to break differing types of rock results in varying forces being applied to any given pick. This phenomenon can be used to determine when the mining machine cuts into a different type of strata.

This system could be conveniently integrated into the mining machine, keeping all of its components compact and protected. This system also gives instantaneous feedback when the miner leaves the seam. No system of this type is currently developed for advanced testing. (Mowrey, 1992)

5.3 Continuous Miner Guidance Technologies

There are four main types of continuous miner guidance systems that appear at the forefront of this technology. These systems are Laser Based Miner Guidance, Ultrasonic Continuous Miner Guidance, Modular Azimuth Positioning (MAPS) and Angular Position Sensing Systems (APSS). Ultrasonic sensors, scanning laser arrays, and ring laser-gyroscopes are all employed in these miner guidance systems.

Laser Based Miner Guidance

The laser system is composed of four laser-scanning sensors that scan for two retro-reflective targets, and report their angular coordinates. This is accomplished by panning the laser beams in the horizontal plane and recording the angles of the beams when they encounter the targets on the rear of the continuous miner. This information is used to triangulate the position and heading of the miner with respect to the known position of the laser arrays. The computer that processes this information may be programmed and linked to drive the continuous miner using these data. (Anderson, 1989)

This system has acceptable accuracy, but is limited to a range of 100 feet and a 110° field of view. Problems with uneven floor have also caused problems by moving the targets out of the plane of laser scanning. This problem can be corrected with longer targets or more scanners. Another problem with this system is that it can only be used for face navigation, and the lasers have to be moved and re-installed as mining advances. This installation requires workers at the face, making this method inappropriate for remote mining in thin-seam applications. (Anderson, 1989)

Ultrasonic Continuous Miner guidance

Ultrasonic sensors have been utilized for experimental miner guidance. In one application, ultrasonic ranging sensors were arranged in formation on a 27-inch diameter fiberglass ring, the Denning ring, at 15° intervals. This ring was mounted on top of the continuous mining machine. These sensors send out ultrasonic pulses and interpret the reflected waves. The data give the computer the coordinates of ribs, corners, and obstructions that are necessary for miner guidance. The computer is also set up to drive the miner through mine workings and to mine coal. (Strickland and King, 1993)

This system stands out as one of the most promising of those reviewed. The sensors are inexpensive and have few moving parts, or lenses to clean. Measurements can be taken through dust and smoke, and the system is integrated directly into the continuous miner so that no accompanying workers are needed.

A few problems have been encountered such as differences in the reflective surface characteristics, reducing accuracy. For example, some surfaces absorb the sound energy, instead of reflecting it back. This causes those surfaces to appear to be much farther away. The sensors, though relatively tough, can be damaged requiring their replacement. (Strickland and King, 1993)

Modular Azimuth Positioning System

The Modular Azimuth Positioning System (MAPS) employs a ring laser optical gyroscope in cooperation with a Zero-Velocity Update (ZUPT) system. When the miner stops moving the translation and rotation data are fed into the dynamic reference unit and processed, giving the new location and heading of the miner. (Sammarco, 1993)

This system is also integrated into the miner, so that personnel are not needed at the mining face. The ring laser gyroscope has low power consumption, is fast leveling, and requires little re-calibration. This very expensive piece of equipment has to be started from a known location, facing in a known direction. The gyroscope is also sensitive to vibration. The ZUPT unit requires that the miner often make frequent stops that can last over a minute. These time delays are the major drawback of this system. (Sammarco, 1993)

Angular Position Sensing Miner Guidance

This method utilizes a mobile computer framework developed by the Bureau of Mines. This system is basically the same as the laser based guidance system, except that the lasers are mounted on this framework. They must, therefore, be driven to the location where they need to be set up again, as mining progresses. (Anderson, 1989)

Chapter 6

Integrated Mining and Haulage Systems

6.1 Introduction

The most widely used method of transporting coal away from a continuous mining machine at the present time is the shuttle car (Section 2.3). There are typically two or more shuttle cars operating in a room and pillar operation. This introduces problems of traffic control, cable handling and delays. Each shuttle car requires an operator for guidance and control. The intermittent nature of the operation inherent in separating the mining and haulage units, and the necessary involvement of personnel renders this system unsuitable for autonomous mining in a thin seam environment. It becomes necessary to combine the mining and coal clearance processes into an integrated system. Manufacturers of mining equipment have already developed such systems. This chapter outlines three examples that show promise for thin seam mining.

6.2 The "Archveyor" Automated Mining and Continuous Haulage Unit

This system was introduced in Chapter 3 as one of the techniques of highwall mining in surface operations. The Archveyor is a long flexible chain conveyor that was developed by Arch Technology, a subsidiary of Arch Coal, Inc. (Figs 6.1, 6.2) It has been used in conjunction with a continuous miner to mine in both highwall and underground environments. Arch's coal mining complex in Wyoming has operated an Archveyor high wall system since 1992 in the Hanna Basin. (Walker, 1997) These automated machines require only two employees to mine coal from highwall type environments. Since the operators are not located near the dangerous high wall area or the coal mining face, the Archveyor has been operated, since its introduction, for the last six years without a lost time accident. (Walker, 1997)

The mining sequence for this type of setup is relatively simple. The continuous miner begins making a cut and off loads onto the Archveyor. At this time the Archveyor is in conveyor mode, with its hydraulic jacks lifting it off the ground allowing the conveyor to turn. As the miner advances to the point where its boom is almost out of range of the Archveyor, the miner begins to cut upward in the seam allowing coal to collect at the foot of the face until the Archveyor is advanced forward. While this is occurring, the Archveyor moves its load farther down its length towards the tail end to free up a length of belt. Then the hydraulic jacks are retracted and the conveyor rests on its return flights. By running the conveyor in the opposite direction, the Archveyor is propelled forward. The jacks are extended again, and the cycle recommences. This process is repeated for the entire length of the heading, and is utilized for every production cut. (Walker, 1997)

This mining cycle presents problems for thin-seam applications. In thin seams there is no room for an upward cut by a milling head of a continuous miner. This eliminates the possibility of using the foot of the face for storage of fragmented coal while the conveyor repositions itself. In any automated thin-seam mining method, difficulties also arise in finding a way for a milling head continuous miner to pass coal back from the head to the miner's conveyor. For this reason an auger head continuous miner that swings side to side is more likely to be successful. (Figure 6.3)

This system also has to deal with the extra space needed to raise and lower the conveyor on the hydraulic jacks. Sufficient headroom may not be available in an underground thin-seam coal operation. New advances on the Archveyor may remedy some of these problems. The newly designed Archveyor will be able to mine coal seams as thin as 28 inches (Stickel, 1998). Automation of these machines has been achieved by combining programmable logic controls with inclinometers and ring-laser gyroscopes. Guidance reliability has been established and proven with success in multiple pass mining. User friendly controls minimize the need for specially trained technical personnel. (Walker, 1997)

In thin-seam underground mining, the infrastructure of development openings is where personnel are required to work or travel. These entries must, therefore, provide greater headroom than the height of the seam itself. The method of excavating development sections for thin-seam underground mining depends primarily on the type of overlying and underlying rock. If the roof is composed of soft shale or similar rock, then a continuous miner may be adequate for the job. If the roof is composed of a harder type of rock, such as sandstone, a road-header would probably be a better choice.

In highwall mining no roof support is employed. Future underground thin-seam mining will require that only the development sections, where personnel work, will be supported. This is possible because of the coal pillars that are left in between each cut, and the small width of each cut. This ensures good natural roof in the heading. (Donovan, 1998)

The flexible design of the Archveyor allows it to mine traverse through undulating seams, and to turn at ninety degrees, even off narrow entries. This is accomplished using drive motors every 7.5 meters to run small sections of the chain conveyor and provides both horizontal and vertical flexibility. This powerful system is currently mining seams that dip at up to 30 percent, while operating at well under maximum available power. If necessary the modular design allows for an increase in power per unit length for steeper pitched seams. (Walker, 1997)

Built-in diagnostic and trouble shooting routines aid the operator in monitoring system performance, ensuring more operational time and facilitating predictive and preventive maintenance. Scheduled maintenance can be programmed and alarmed automatically, and production data can be collected through real time data sampling.

This adaptable system was constructed with the ability to incorporate a number of subsystems including atmospheric monitoring and ventilation. Air ducts can be installed along the length of the conveyor to deliver air directly to the face. (Fig 6.2) (Walker, 1997)

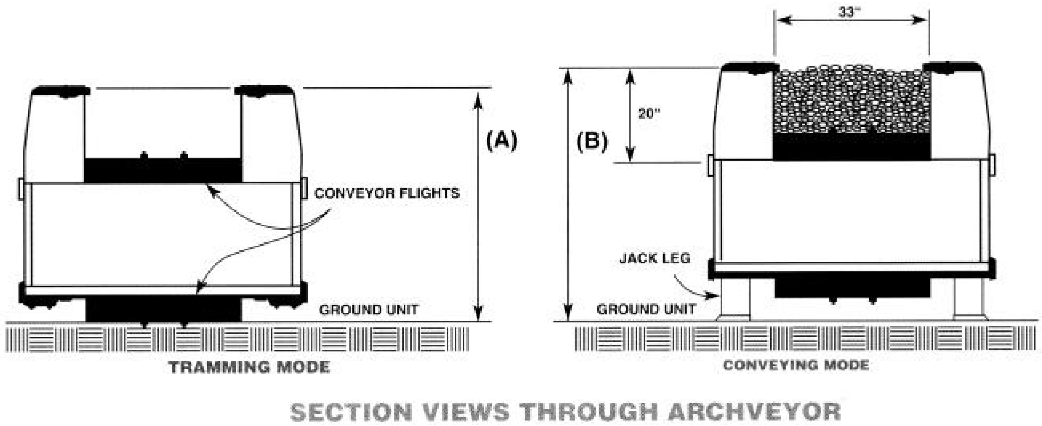

Figure 6.1 Archveyor Section
(Arch Technology Corporation) reproduced with permission

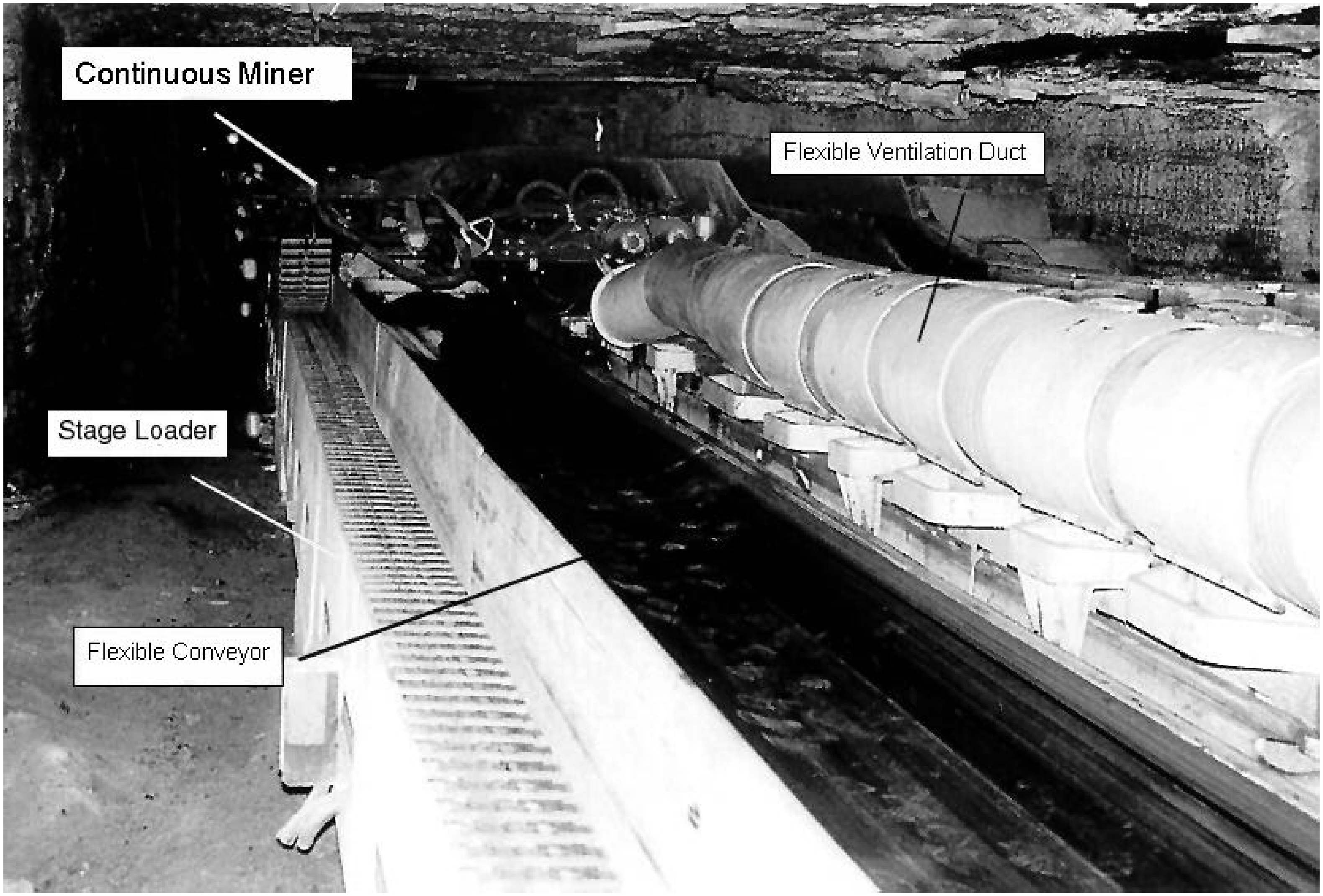

Figure 6.2 Archveyor Underground
(Arch Technology Corporation) reproduced with permission

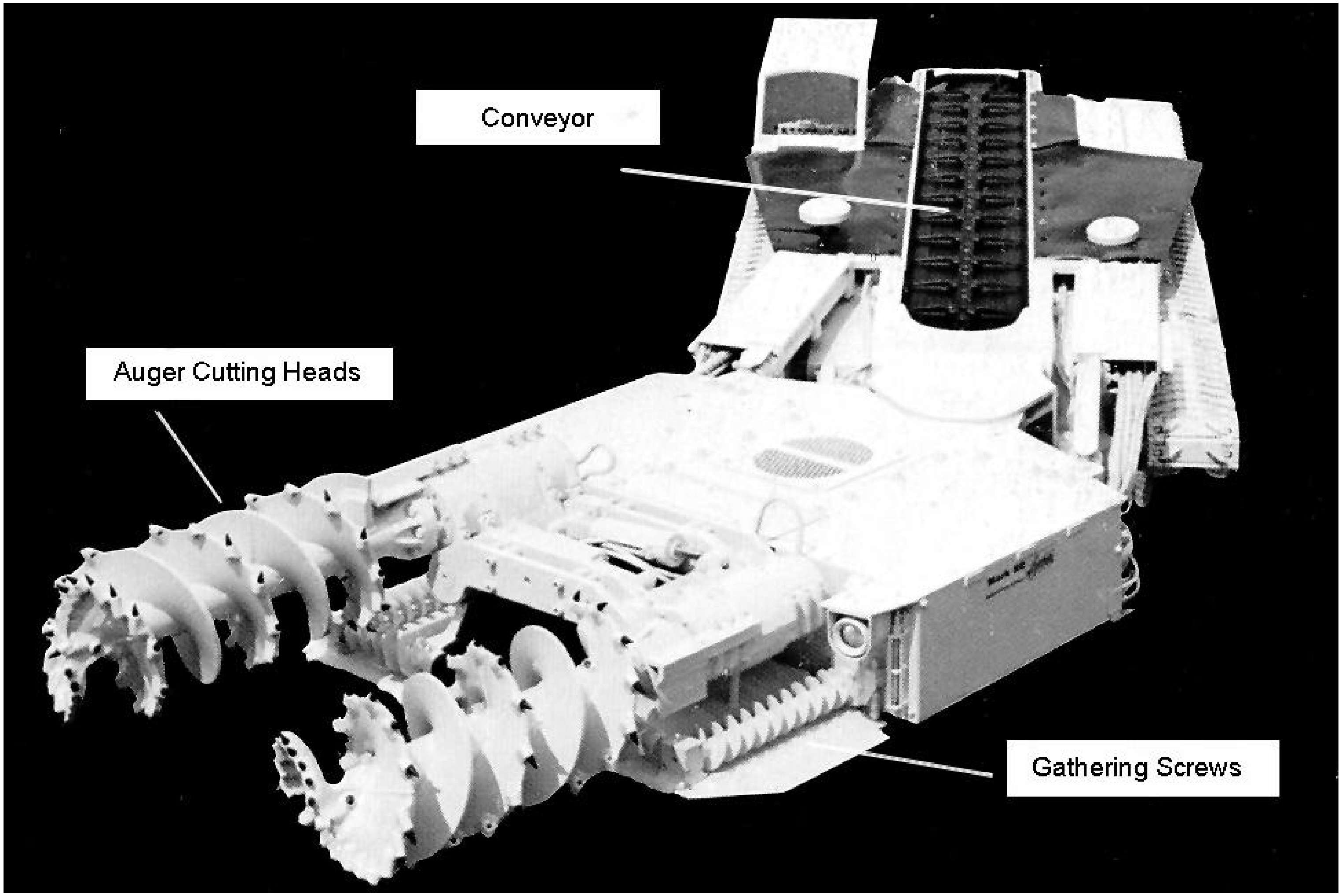

Figure 6.3 Fairchild Continuous Miner
(Fairchild Incorporated) reproduced with permission

6.3 The Long-Airdox Full Dimension Continuous Haulage System

A new form of face haulage has been created and marketed by Long-Airdox. This has evolved from a system that was conceived, initially, in 1958, but has undergone constant updates and improvements since that time. Some of these improvements include the introduction of custom designed dual, extended life conveyor chains and many others. (Long-Airdox)

The system contains three main components that are utilized together to transport the coal. These components include the inby mobile bridge carrier (Fig 6.4), the intermediate mobile bridge carrier (Fig 6.5), and the piggyback bridge conveyor. (Fig 6.6) In a standard set-up as shown on Figure 6.7, the inby mobile bridge carrier is fed by a continuous miner and dumps into the first piggyback bridge conveyor. This conveyor dumps into the intermediate mobile bridge carrier, which off-loads into the second piggyback bridge conveyor. This bridge conveyor empties onto the haulage belt of the section. Currently all of the mobile bridge conveyors require operators. The flexibility of the system allows the configuration to be adapted to specific layouts. (Long-Airdox)

This system has consistently set productivity and reliability records with over 125 installations operating across the country. These systems are used in seams ranging in thickness from 30 inches to 13 feet. With haulage capacities of over 30 tons per minute and tram rates up to 85 feet per minute this system will fill the haulage requirements of the majority of continuous miner sections in operation. (Long-Airdox)

These systems are not yet fully automated but the design lends itself to automation. The manufacturer is looking into automating this already versatile mining tool.

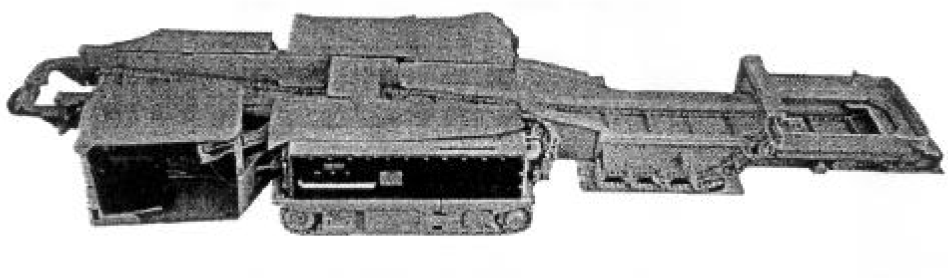

Figure 6.4 Inby Mobile Bridge Conveyor

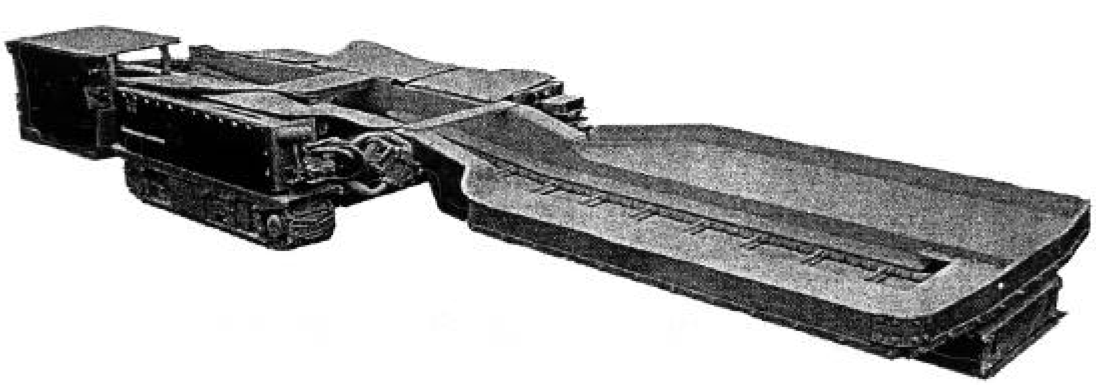

Figure 6.5 Intermediate Mobile Bridge Conveyor

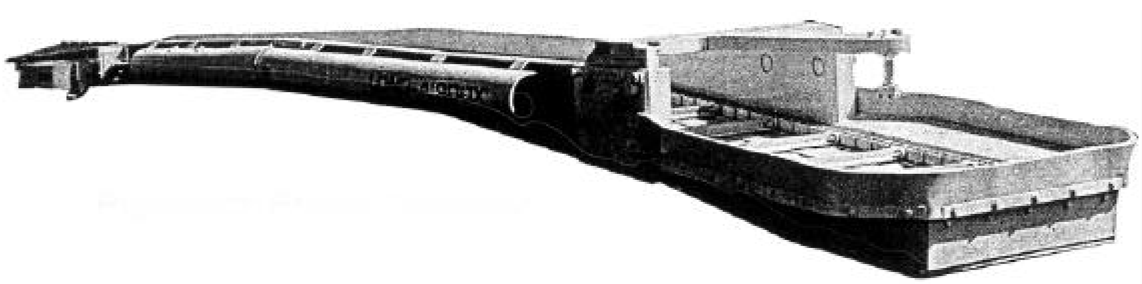

Figure 6.6 Piggyback Bridge Conveyor
(Long-Airdox) reproduced with permission

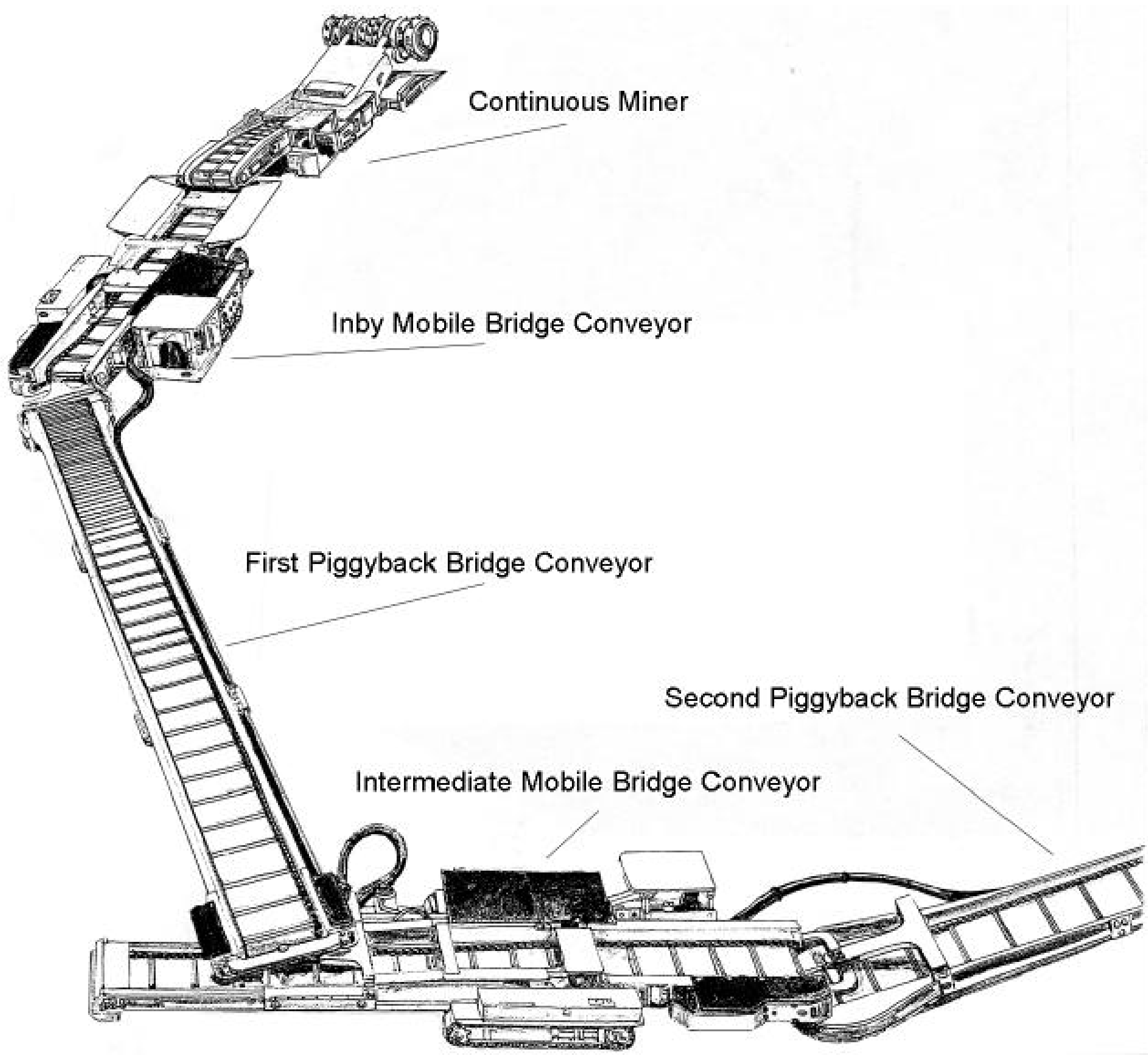

Figure 6.7 Assembled Continuous Haulage System (Long-Airdox) reproduced with permission

Chapter 7

A Proposed Mining System for Thin seam Mining

7.1 Introduction

This chapter outlines the concept of a system for the underground mining of thin seams. Its development has taken into account the experience of the past, as discussed in previous chapters, the more recent developments in mining machine guidance and system integration, and the industrial and commercial environment within which modern mining takes place in the United States. (Holman, McPherson, and Loomis, 1999b)

7.2 Overview of the Mining Layout

This new mining layout was designed taking into account ventilation, development cost, haulage, equipment maneuverability, materials transportation, and percentage of coal recovered. In this layout large blocks of coal are developed, similar to longwall panels. (Fig 7.1) The nominal dimensions of these blocks are 1000 feet wide by 5000 feet in length. In this design the longer the panel, the higher the recovery that is attained.

The layout of the complete section encompasses a block with two entries on each side. Three longitudinal entries bisect the block into two panels, with the middle entry used for the section belt conveyor. This conveyor entry is shared between the two panels. Three parallel entries run along the shorter ends of the panel blocks. The first entry on the outby side of the panel is used for transportation of supplies. All of these entries serve as main intakes for the ventilation system. The three entries on the inby end of the panel serve as ventilation main returns. Each of the long parallel entries that run the length of the panel, connecting main intakes to main returns, are equipped with regulators to control the air flow distribution through the panel. (Holman, McPherson, and Loomis, 1999b)

The production stalls are cut parallel to each other in a herring bone pattern. These are 500-foot long production cuts that are roughly 13 feet in width, consistent with the capacity of the continuous miner employed, with support pillars between each stall. The width of the support pillars is dependent on the competency of the country rock, coal strength, depth of overburden, and strength of backfill, if used. An average value for a pillar width at 1000-feet of overburden with moderately intact rock, strong coal, and a 30 inch mining height would be 11 feet wide with no backfill, and 5.9 feet wide with a 1000-psi strength backfill. (Donovan, 1998)

The herring bone pattern was selected for length of cut and maneuverability of equipment. Production cuts that angle at 45 degrees from the adjacent entries are accessible by continuous haulage units of the type discussed in Chapter 6. This allows the haulage units to remain in one continuous string, instead of having to be disassembled and put back together each time a production cut is completed.

Haulage in this system is accomplished using a series of chain conveyors. The continuous miner fragments the coal from the solid at the face of the stall and loads it onto the receiving unit of the stall conveyor train. This in turn is connected and loads on to the secondary conveyor train, which is located in the adjoining access entry. The coal travels down the secondary conveyor, which turns into a 45-degree angled, open crosscut to the section belt conveyor in the center entry. This belt conveyor loads on to the main conveyor at the outby end of the panel.

At the outby end of block is a pumping station. A backfill slurry is pumped through a pipe range to fill into the mined out stalls. By sealing these mined-out stalls, leakage pathways are inhibited, requiring the air to travel in the pathways that have been designated as the ventilation network. (Holman, McPherson, and Loomis, 1999b)

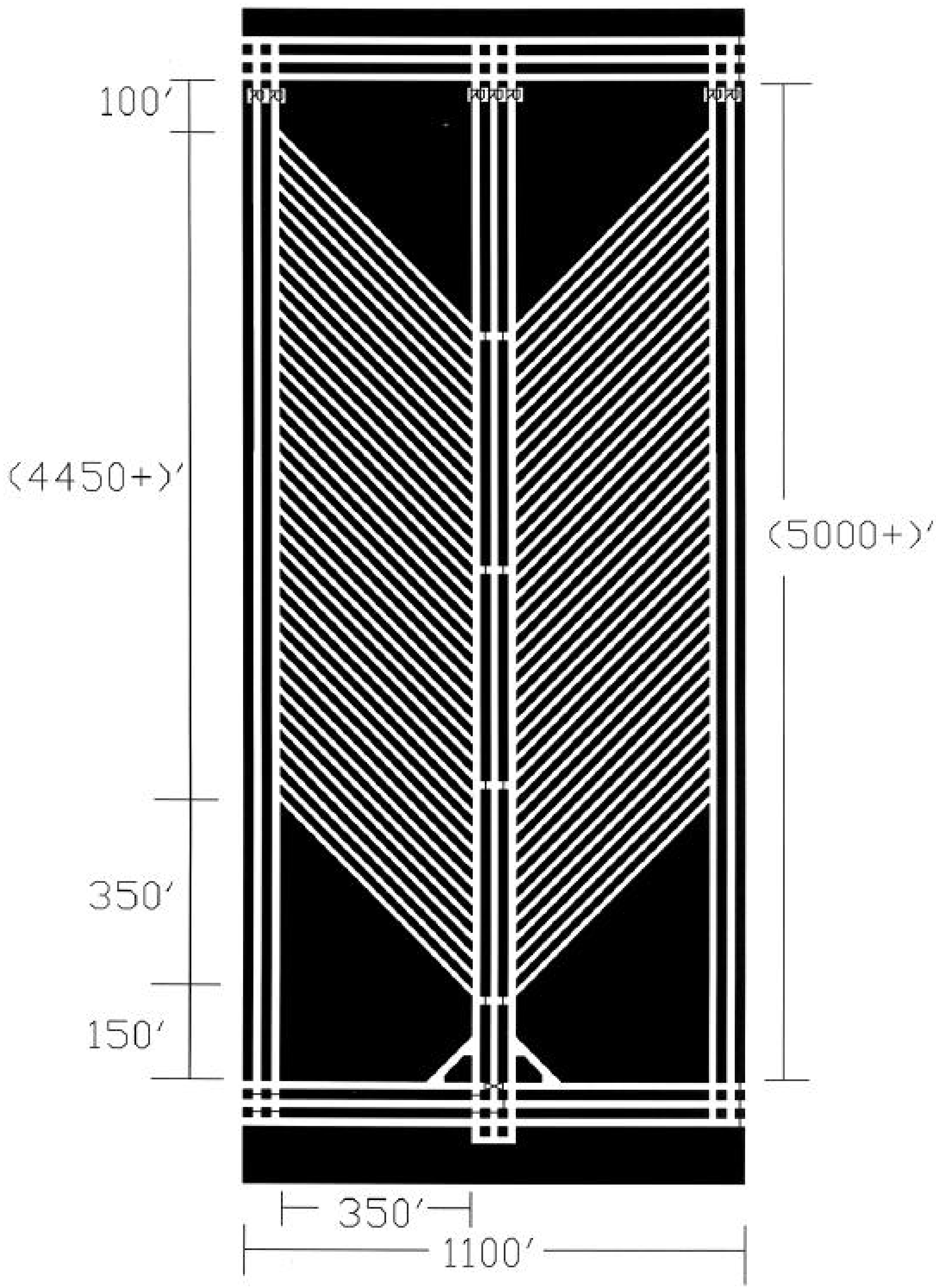

Figure 7.1 Mining Layout

7.3 Equipment

For mining of this kind, many pieces of equipment will be needed for the functions of development, roof support, mining, haulage, and back filling. Strata conditions will determine the type of equipment needed for developing the main entries where personnel will be working, but otherwise the equipment for this type of mining is essentially the same.

For purposes of development, continuous miners can be used to cut coal and rock in soft strata. For harder country rock a road header might need to be employed. In these open entries, roof support will be achieved through the use of roof bolts. Roof support in the stalls will not be provided. For the purpose of roof bolting in the entries, a twin boomed roofbolter is recommended (Fig 7.2). A scoop for material transport and general utility should be maintained on each section.

The mining in the stalls may be carried out by a Fairchild type of auger continuous miner that cuts from side to side. (Fig 6.3) This miner off loads onto a stall conveyor train such as the Archveyor or the system produced by Long-Airdox Inc. (Fig 7.3) This haulage unit is composed of repeated components and stretches to a length of 520 feet. The secondary haulage system in the adjoining entry is also envisioned to be a version of the Long Airdox Full Dimension unit with elongated bridge sections. (Holman, McPherson, and Loomis, 1999b)

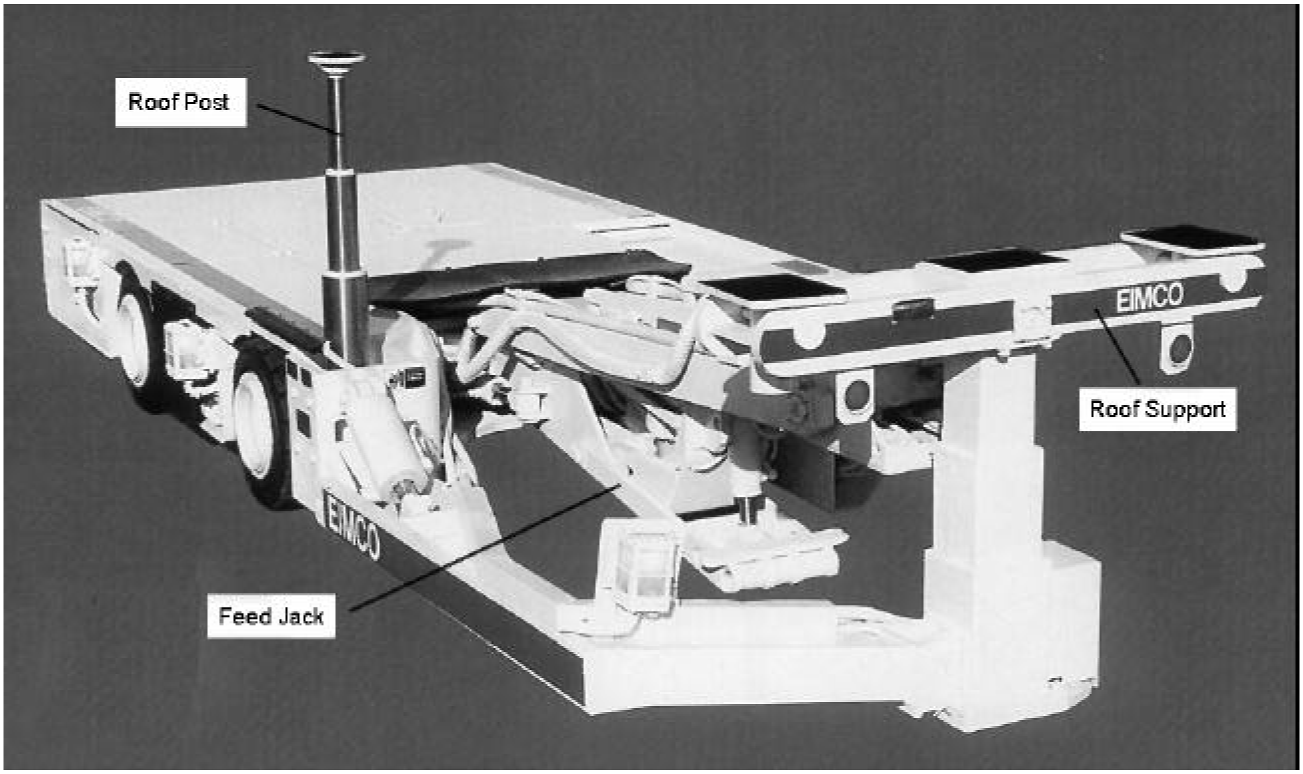

Figure 7.2 Roof Bolter (Tamrock Inc., 1995) reproduced with permission

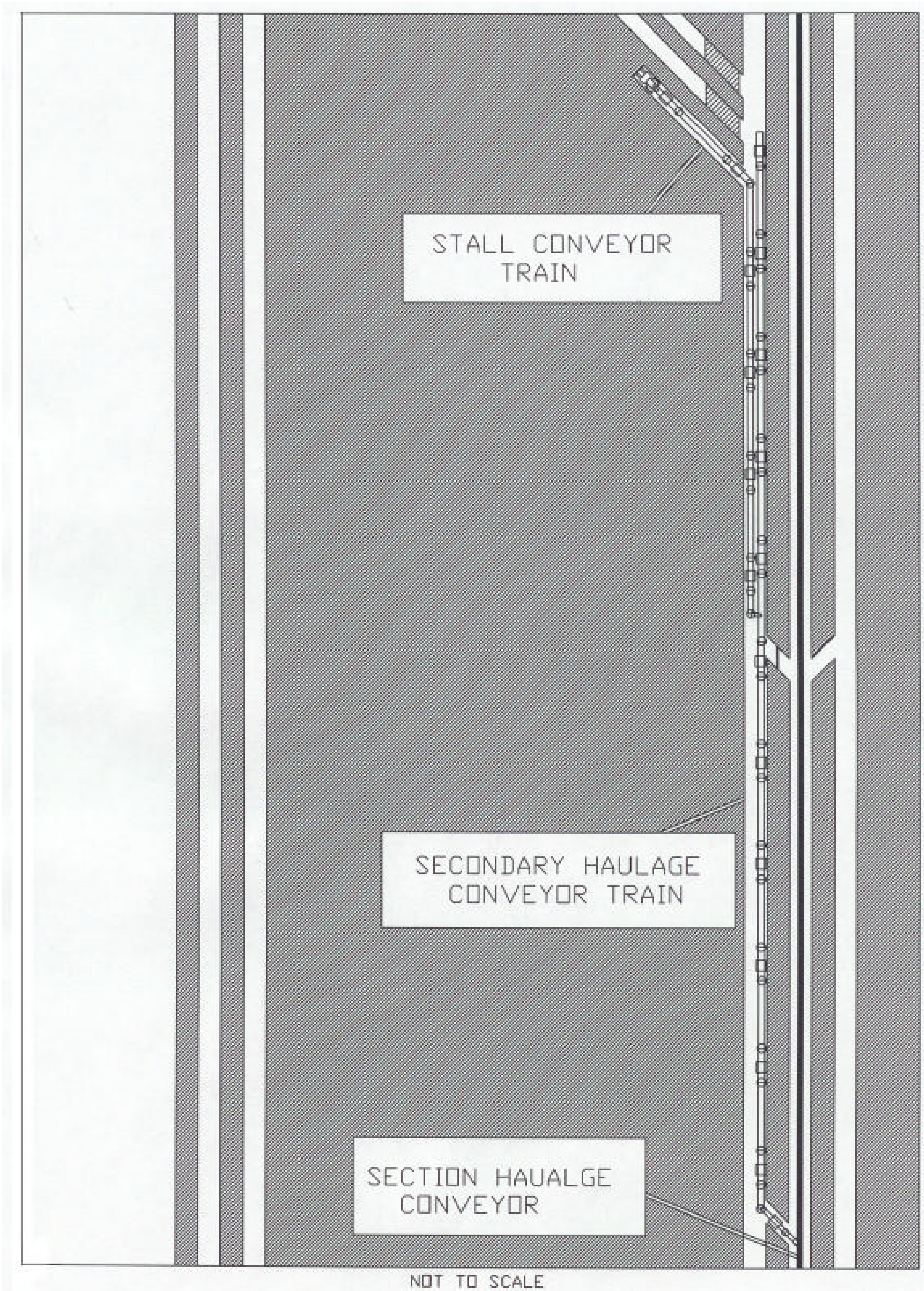

Figure 7.3 Conveyor Placement

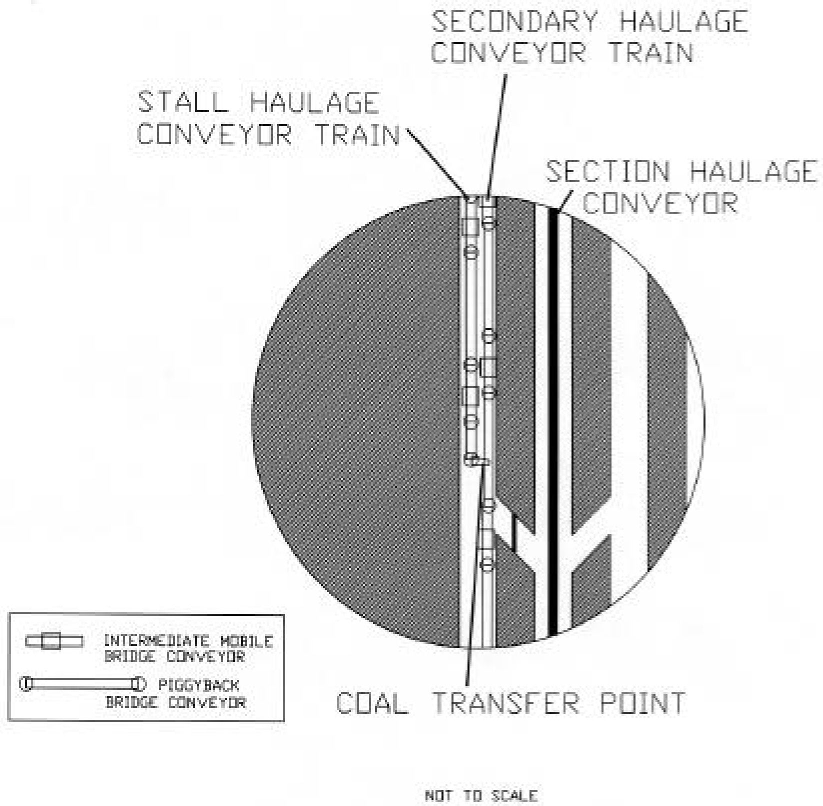

Figure 7.4 Detail of chute transfer point from the stall conveyor train to the secondary conveyor

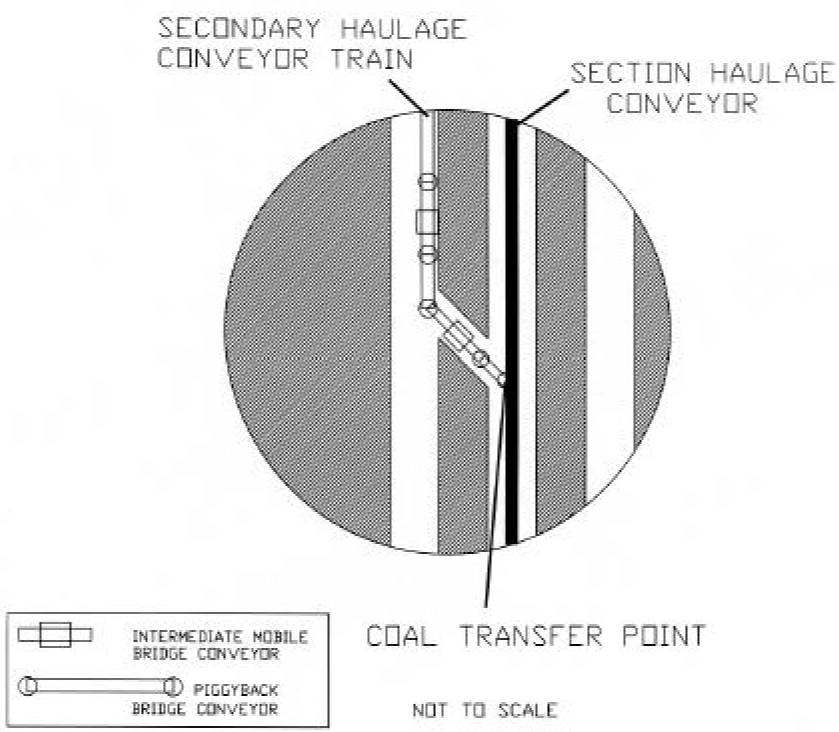

Figure 7.5 Detail of Transfer from secondary conveyor on to section belt conveyor

7.4 Coal Transportation

The coal is transported from the continuous miner within the stall to the mains by means of three conveyor systems, the stall conveyor train, the secondary conveyor unit in the adjoining entry, and the section belt conveyor in the center entry.

At the commencement of a new stall, the nearly 500 ft of the articulated series of chain units that comprise the stall conveyor train will lie alongside the secondary conveyor unit. As the continuous miner advances into the stall, the stall conveyor unit will follow, and retract as the miner is withdrawn at the completion of a 500 ft cut. The out bye end of the stall conveyor train will comprise a curved-jib or chute type of unit attached to a slide rail running the length of the secondary conveyor and loading on to that conveyor (Figure 7.3).

The secondary conveyor will be a little more than 1000ft long but will comprise a series of alternating intermediate mobile bridge carriers and elongated forms of piggyback bridge conveyors (Figures 7.4 and 7.5). This flexible arrangement allows the outby sections to turn into a 45 degree angled crosscut and to load on to the belt conveyor in the center entry. Individual units of the secondary haulage system may be disconnected and stored inby temporarily as the stalls are completed and the panel is mined in retreat. In addition to progressively reducing the active length of the secondary conveyor, this may also be necessary to provide the space required to initiate each new stall.

The initial 1000 ft length of secondary conveyor is suggested to allow the panel to retreat for 500 ft before the complete secondary conveyor has to be moved back, under its own power, to the next angled cross-cut leading to the center entry. Hence, in this arrangement, those crosscuts would be 500 ft apart. Shortening its length to an intermediate value between 500 and 100 feet could reduce the capital cost of the secondary conveyor. This would necessitate more frequent moves of that conveyor, and angled crosscuts that were closer together. In order to facilitate movements of both the stall conveyor train and the secondary conveyor, it is important that excessive spillage is not allowed to accumulate. **(Holman, McPherson, and Loomis, 1999b)**

7.5 Guidance System for this Layout

The guidance system that will control the continuous miner will utilize a 120° scanning laser array, angular transducers, a micro processor with signal transmission capabilities, a Natural Gamma Radiation CID unit, and a radar based rib thickness monitoring unit.

The Natural Gamma Radiation CID unit will be used to inform the miner when it begins cutting out of seam material. A signal will be sent to the microprocessor, which will, in turn, signal the miner to lower its cutting height.

The radar based rib thickness monitor will come into use starting with the second production stall. It will send a radar signal perpendicular to the stall to measure the distance between the miner and the previous stall. The unit will send its results to the microprocessor who will adjust the miner heading to maintain alignment, and proper support pillar width.

The scanning laser array will be able to read bar codes on machine reflectors up to 50 meters away. The bar codes will allow the processor to identify the machines and better keep track of machine location and process location data. These data will be used to maneuver the haulage conveyor around the 45° turn into the production stall.

The scanning laser array will be mounted on a mobile frame unit. (see Figure 5.1) This frame unit will have three legs. Two of these legs will be located long the rib, framing in the active production cut, while the third will be located in the middle of the access entry way. The scanning laser array will be mounted on the central leg. The miner and haulage train will pass underneath the frame unit. The legs will be equipped with hydraulic cylinders, and computerized leveling for the laser array. The legs will also have wheels on their bases to allow for easy movement the frame from one cut to the next.

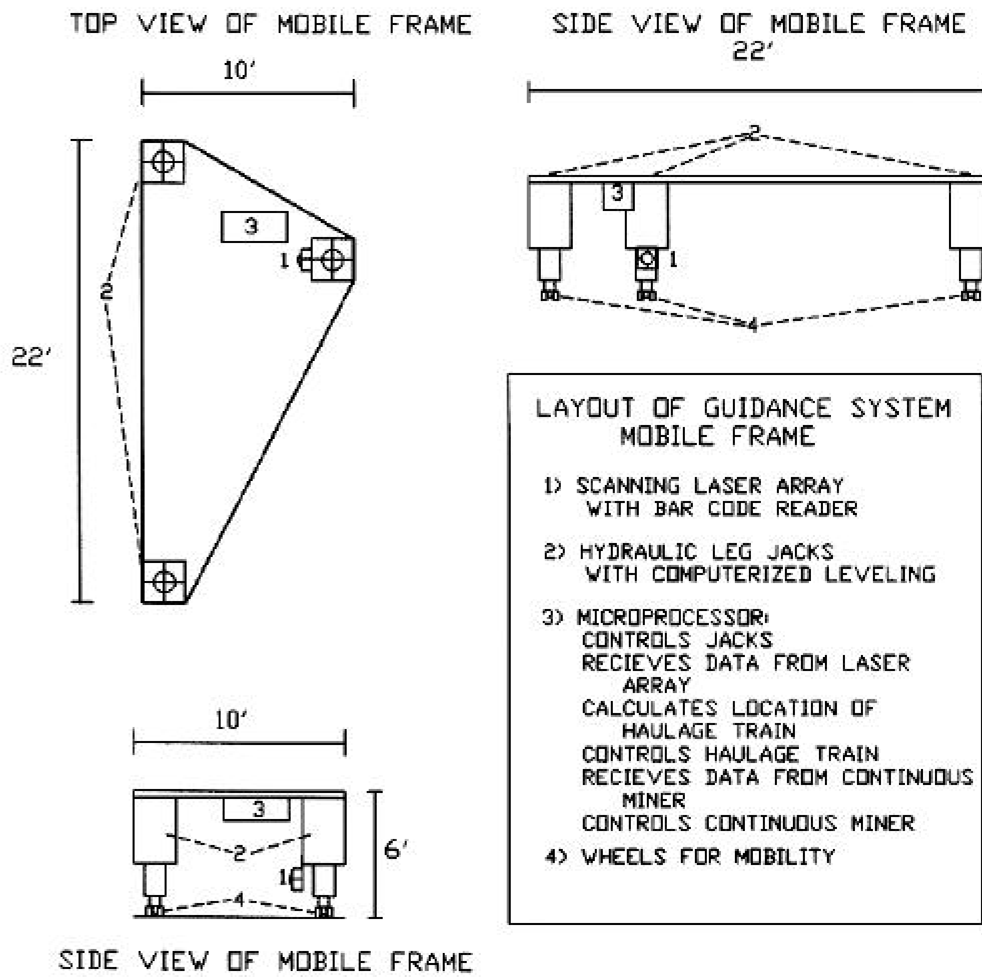

Figure 7.6: Mobile Guidance Frame Unit

The two legs that frame in the production cut are located in positions that are known to the microprocessor. The computer also knows that the production cuts are to be made 45° to the access entry. The computer can adjust the miner to ensure that it is cutting a 45° cut. The computer knows the locations of the corners of the ribs, the angle at which the cut is made, and the length that the miner has progressed. This allows the computer to mathematically model the ribs of the cut, and check for collisions with the equipment. (Figure 5.2)

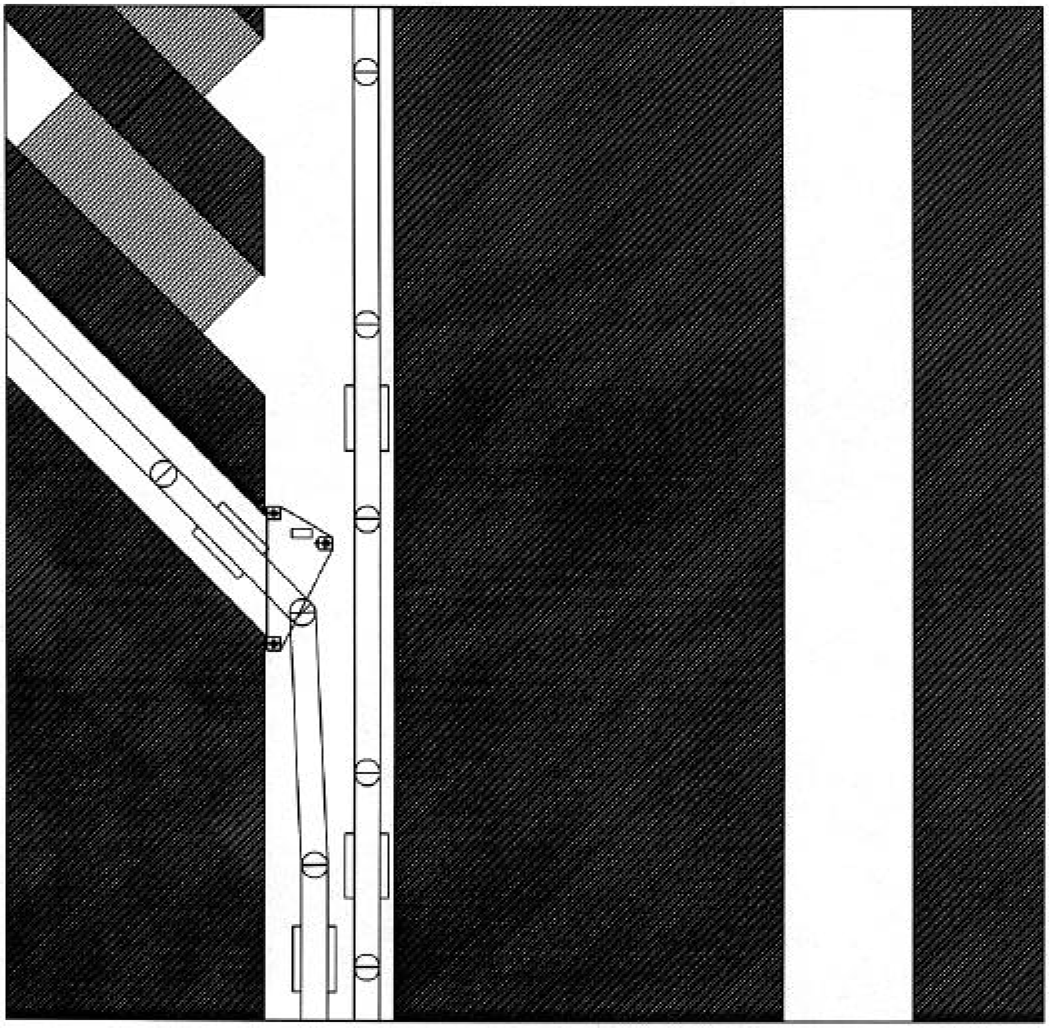

Figure 7.7: Mobile Guidance Frame Unit in use

The angular transducers will be installed at all of the pivot points of the haulage train. The scanning laser array will provide the identity of at least one unit in the train at all times. The position of the entire production train can be calculated in reference to this unit and its surroundings. This is possible because the dimensions of all of the units are known quantities, the angles between the units are reported by the angular transducers, and the position of at least one units in reference to its surroundings will be provided by the laser array.

There will be an operator interface box where an operator can stop the sequence at any time and manually guide the system. Safety cut off lines to stop the system will be located on all pieces of machinery and on the transmission box.

Series of events during automated mining

First the continuous miner and the haulage train are backed up away from the next production cut. The miners position the framework at the location of the next cut. The secondary conveyor train must be partially disassembled to have enough room to face up the continuous miner. The position of the production train will be computed by the microprocessor utilizing the data obtained by the laser array, the angular transducers, and the know dimensions of the units of the train. This position calculation will be constantly updated anytime one of the variables changes. While constantly checking for intersection between the safety buffer around the units and the objects in their environment (ribs, other equipment, etc.), the microprocessor guides the production train through its production cycle utilizing the data from the CID unit, and rib monitor. Once the miner has mined a few feet into the face. The secondary haulage train is reassembled.

Mining progresses until the production stall is mined out. The system automatically shuts down, and awaits the mine personnel to give it the command to back the haulage train and miner out of the cut. The order is then repeated for the next cut. When the system has mined down 500 feet of panel length, the secondary haulage train must be advanced. The secondary haulage conveyor is then moved down 500 feet to the next open crosscut.

Steps in the guidance algorithm for the continuous miner

(See figure 7.8)

1) Receive input value for mining height (CH) and pillar width (PW).

2) Check alignment (rib thickness vs. PW).

   If not aligned, then correct alignment.

3) Check haulage position.

   If not in position, then signal for haulage advance.

4) Check CID interface.

   If not correct, then adjust miner.

5) Check distance from beginning of cut.

   If not at the end, then advance miner.

   If at the end, then stop and wait for personnel.

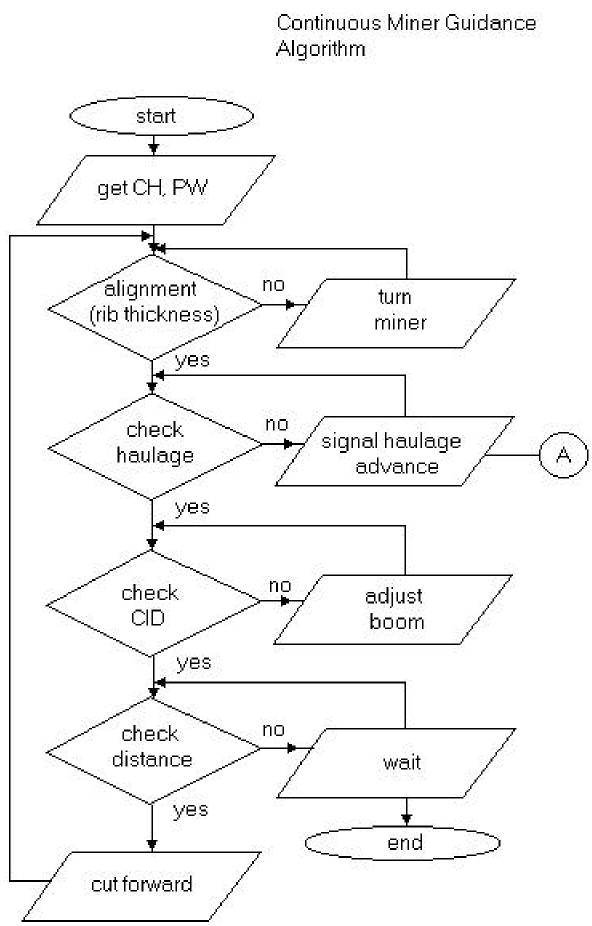

Figure 7.8: General Algorithm for continuous miner guidance system

Steps in the guidance algorithm for the primary haulage conveyor train (See figure 7.9)

1) Assign coordinates to rib corners (X1 and X2.)
2) Find placement of all visible machines.

a) Find the equation of the centerline of each unit. The equipment locations can be obtained utilizing the length of the unit, the data from the angular transducers, and the known position of one of the units in the train.

b) Add to and subtract from the equations of each unit to produce a rectangle encompassing the machine, to act as a collision safety buffer.

3) Check for intercept between the area of the rectangles and the ribs and other hazards.
4) Send commands to appropriate machines to maneuver the conveyor train around the corner and down the stall.
5) When last machine gets to final position the system goes on hold and waits for an operator to initiate the extraction sequence.

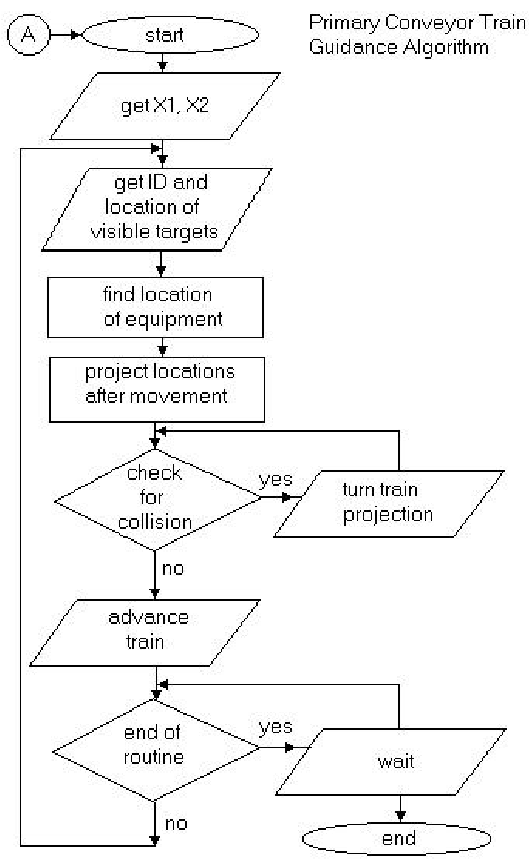

Figure 7.9: General Algorithm for continuous haulage train guidance system

7.6 Ventilation

Bleederless Ventilation System

This ventilation system is employed when suitable backfill material is available, because it utilizes backfill plugs or full backfilling in the mined out stalls. These provisions eliminate the need to take special action to control the methane emissions by direct ventilation controls. The plugs or seals isolate the methane that is released from ignition sources and cause the concentrations to rise well above the explosive range of the gas. (Holman, McPherson, and Loomis, 1999)

To prevent leakage from the isolated stalls to the back entries and main returns, a slight pressure drop must be maintained from the central entries to the outer entries of each panel.

Leakage is countered because the entries on both sides of the sealed stalls are kept at a higher pressure than the trapped pockets between the seals. Under normal circumstances, the air moving in the outer drifts will flow past the seals carrying away any leakage. This air will carry the methane directly to the main returns at the inby end of the panel. Intake air in the central entries will be used to ventilate the mining equipment and personnel in the panel. A split from this air will be directed to the remote face, continuous miner, and stall conveyor train. Air that passes over equipment will then be routed to the main returns at the in by end of the block. When conditions are conducive to a neutral belt entry, leakage will move to the section belt entry from the front entries. Air flowing across the section belt will be directed towards the main returns.

After the current stall is completed air will flow through the recently completed stall, towards the outer drifts, until the stall can be effectively sealed with fill material. The outer drifts must be kept at a lower pressure than the center entries, to insure that airflow proceeds in that direction. (Holman, McPherson, and Loomis 1999)

Separate splits of air will be employed to ventilate the back, front, and section belt entries. The basic ventilation configuration dictates that the section belt entry be kept as a neutral airway, to prevent leakage from that entry to the front entries. This scheme is illustrated in figure 7.6. (Holman, McPherson, and Loomis, 1999)

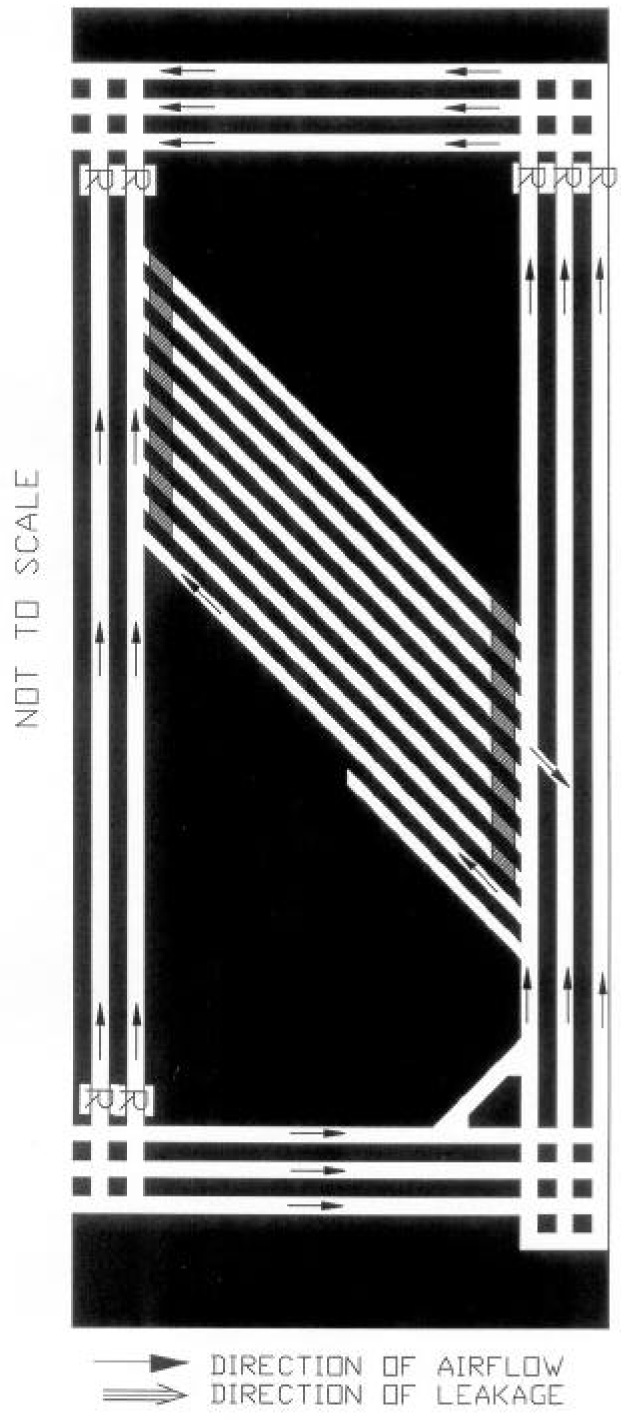

Figure 7.10: Generalized Bleederless System for Proposed Thin-Seam Panel (Holman, McPherson, and Loomis, 1999)

Bleeder Ventilation System

A bleeder type ventilation system is employed when backfill material is not available. Airflow can be controlled through the extracted area of the mined out stalls by means of brattice curtains. This leaves the concern of controlling the methane that is liberated from the exposed ribs in the mined stalls. It is best if the methane enriched air enters the general mine return as late in the ventilation process as possible.

A schematic of this ventilation system can be seen in figure 7.7. This figure is presented in semi-schematic, meaning that some entries are shown as single entries when in reality multiple parallel entries may be needed to accomplish the same functionality.

This system utilizes the inby entries as bleeder entries. The most recently completed stall is left open for use as a general section return. This system utilizes the outer pairs of entries on either side of the block for dual purposes. They are utilized as panel returns, outby the current working stall. They are also utilized as bleeder entries inby the current stall. (Holman, McPherson, and Loomis, 1999)

This dual nature causes a fragile balance between the air movements of the air in the ventilation stall and the air moving through the barricaded stalls. It is more desirable if the airflow moves slightly toward the bleeder side of the panel, instead of toward the mains. This keeps the bleeders well flushed, and delays the entry of methane enriched air into the general mine returns.

This system places the main ventilation design into the U-tube configuration, causing the related problem of leakage between intake, return, and belt air that are flowing in parallel. This system could also lead to excessive development costs to drive the necessary parallel entries needed to efficiently move the required amounts of air. Another problem arises from the complex arrangement of overcasts and stoppings needed for the intake air to cross the returns in the area of the center entries of the active block. (Holman, McPherson, and Loomis, 1999)

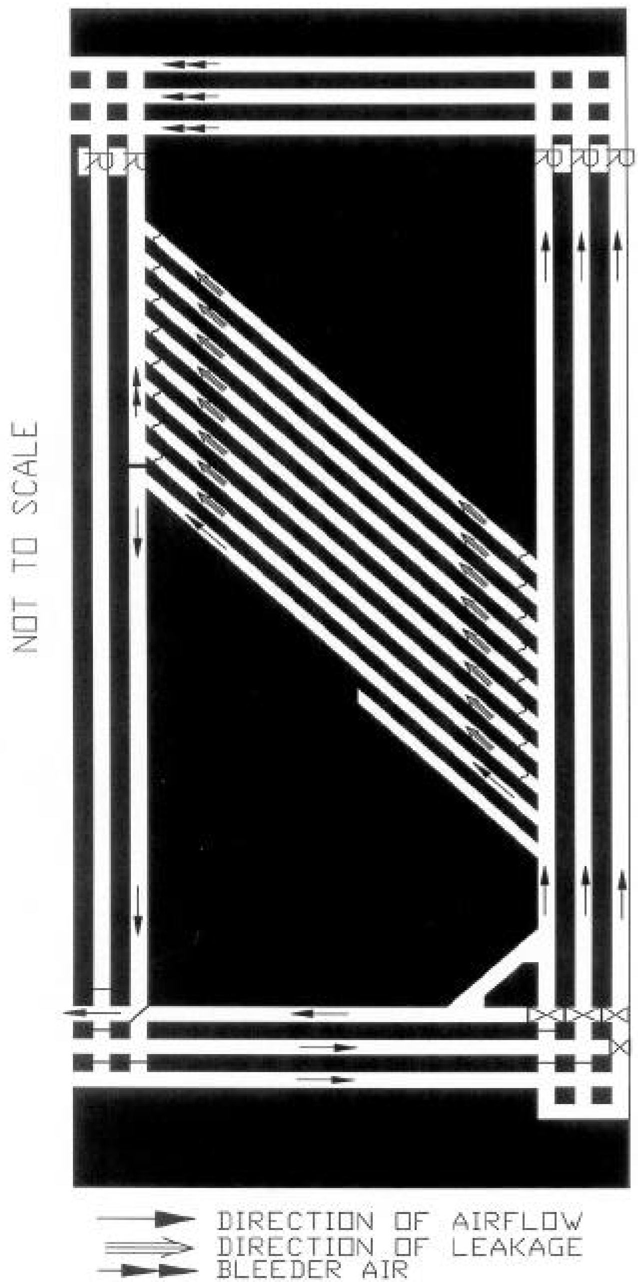

Figure 7.11: Generalized Bleeder System for Proposed Thin-Seam Panel (Holman, McPherson, and Loomis, 1999)

Ventilation Modeling

The panel layout is the main difference between this ventilation system and those currently being used in the mining industry. For this purpose the ventilation modeling will focus only on the intricacies of modeling the ventilation of the panel. The remainder of the ventilation will be carried out in a manner that is consistent with a typical underground coal mine in similar conditions.

VnetPC for Windows version 1.0a was utilized to construct and test the ventilation model. This software was developed by Mine Ventilation Services, Inc. to tackle ventilation-modeling tasks of this type. (Holman, McPherson, and Loomis, 1999)

The first step in the modeling process was to design a schematic to represent the entries of the panel. In the schematic parallel entries that carry air in the same direction were sometimes modeled as single branches of the schematic. An observable example of this is the dual entries that run along the outside edge of the panel. The mined out stalls were also treated in this manner. The resistances used in these special branches are consistent with what they would be for the multiple entries. Another thing to take note of is that crosscuts between dissimilar, parallel entries were treated as equivalent resistances.

Figure 7.8 shows an example of the basic network utilized in this model. The ventilation fans have been modeled as fixed quantity type fans. The airways located to the left of the block are adjunct branches, needed to complete the model. These branches have no resistance, and so do not affect the model. Another consideration that was taken into account was that the airways inby the active block have been modeled to allow for airflow to deeper parts of the mine. Their resistances have been assigned values that are consistent with their intended purpose. This plan of modeling allows us to investigate the ventilation of the panel without neglecting the effect of the rest of the mine workings. (Holman, McPherson, and Loomis, 1999)

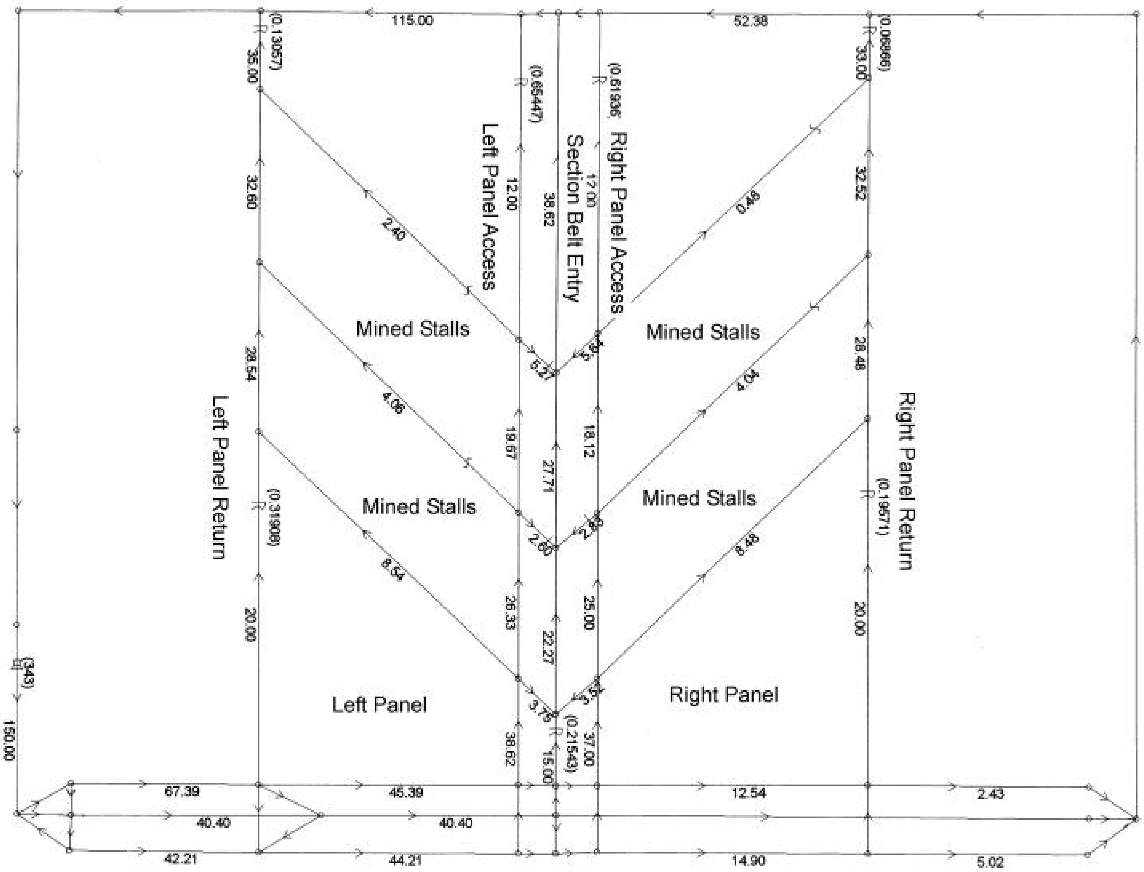

Figure 7.12: Ventilation Schematic of Suggested Ventilation Plan (Holman, McPherson, and Loomis, 1999)

The nature of mining in this method requires that the model be able to handle an increasing number of parallel production stalls. To accomplish this task the model was designed to be conducted in three stages, simulating the effect on ventilation as retreat mining progresses. The first stage has two stalls mined out, one located in each block, simulating the completion of the first production stalls in each panel.

The second stage shows two open production stalls. This required the schematic to have two more branches to model leakage of air through sealed stalls. This is modeled with the equivalent resistance of the panel being mined out half way to completion. The last stage tackles the worst case scenario, an open stall coupled with two equal parallel resistance branches in each panel. This situation would arise when the final cuts of the block are completed. (Holman, McPherson, and Loomis, 1999)

Figure 7.9 shows a typical airflow model of the favored ventilation layout. The airflow in the section belt entry of the block is kept under negative pressure, with respect to the panel access, entries to the left and right. This is done using regulators on both ends of the center entry, and on the inby end of the panel access entries.

The exhaust entries, on the left and right of the panel, are equipped with regulators on both ends closing in the herringbone mining stalls. The outby regulator limits the amount of fresh air to flow into these entries, while the inby control the amount of flow to the completed panel stalls. These ensure that flow will move from the central panel accesses to the outer returns. Airflow through the stalls can be maintained at a reasonable level with prudent control between these regulators. (Holman, McPherson, and Loomis, 1999)

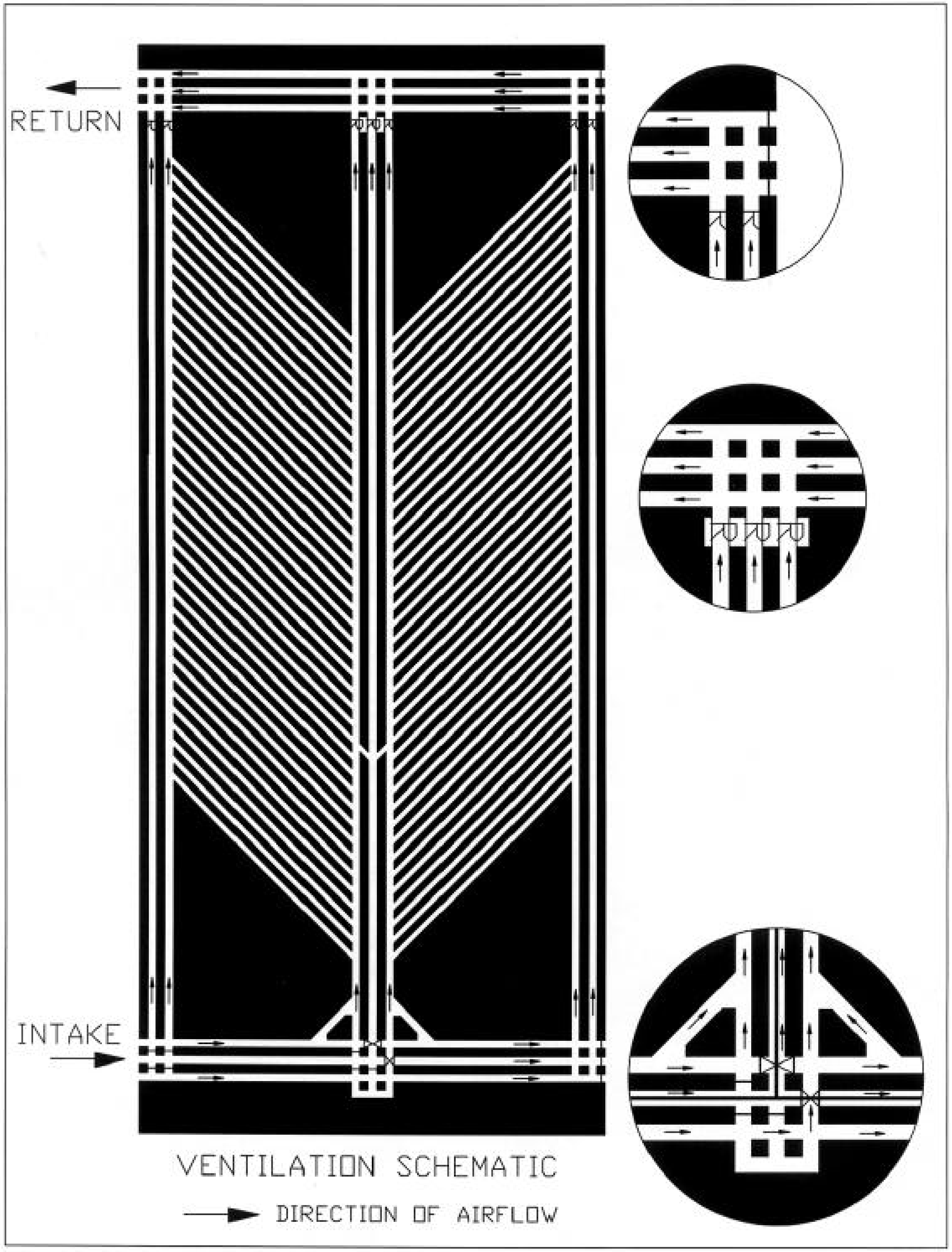

Figure 7.13 General Airflow Layout

Suggested Ventilation Plan

This system utilizes through-flow ventilation. Regulators are placed on each of the entries to control airflow. (Fig 7.8) The stall conveyor train can be equipped with an auxiliary fan and flexible ducting. This directs airflow directly to the mining face to clear methane and dust. The air returning from the mining cut is then picked up by the through-flow air and carried out via the returns. An air velocity of 100 ft/min in a 13 ft wide and 28 inch high opening can be provided by an airflow of 3,120 ft^3/min even when no allowance is made for the area of cross-section filled by equipment. If the ventilation duct is smooth lined, one foot in diameter and 500 ft long this will produce an air velocity in the duct of some 4,000 ft/min and will require a total applied pressure of 10 inches of water gauge from the auxiliary fans. (Holman, McPherson, and Loomis, 1999)

Fresh intake air enters the section at the outby end and all entries, except for the face openings, act as intakes. Air is directed to the working face by the auxiliary fans mounted on the continuous haulage system (Fig 7.10). The air then exhausts back into the entry and flows to the returns at the inby end of the panel.

The open mined-out stalls will be back filled, or partially back-filled, for ventilation control. The number of open entries requiring through-flow ventilation in this layout is similar to that in a longwall system, and considerably less than in a room and pillar section. This will promote better control of the ventilation, less leakage, and improved environmental conditions in areas where personnel work or travel

The amount of backfill depends on the support needs of the mine and availability of waste material for backfill. If stability is an issue then consideration may be given to backfilling the entire length of the stall to provide confining pressure for the pillars. However, if such additional support is not an issue then 40-foot plugs will be filled on each end to prevent air leakage through the stall. In this study it is assumed that total backfilling is not required and that the 40 ft plugs are all that are required. (Holman, McPherson, and Loomis, 1999)

The air trapped between the plugs will have suppressed oxygen levels as a result of oxidation processes and increasing methane levels. This keeps the gas mixture out of the explosive range of 5 to 15 percent methane and sealed away from all sources of ignition. With 40-ft seals, there will be very little leakage at the seals. The regulators will be adjusted such that the air pressure in the three central main entries is slightly higher than that in the outer entries of the block. This will ensure that no emissions from sealed areas can occur into the conveyor entries. (Holman, McPherson, and Loomis, 1999)

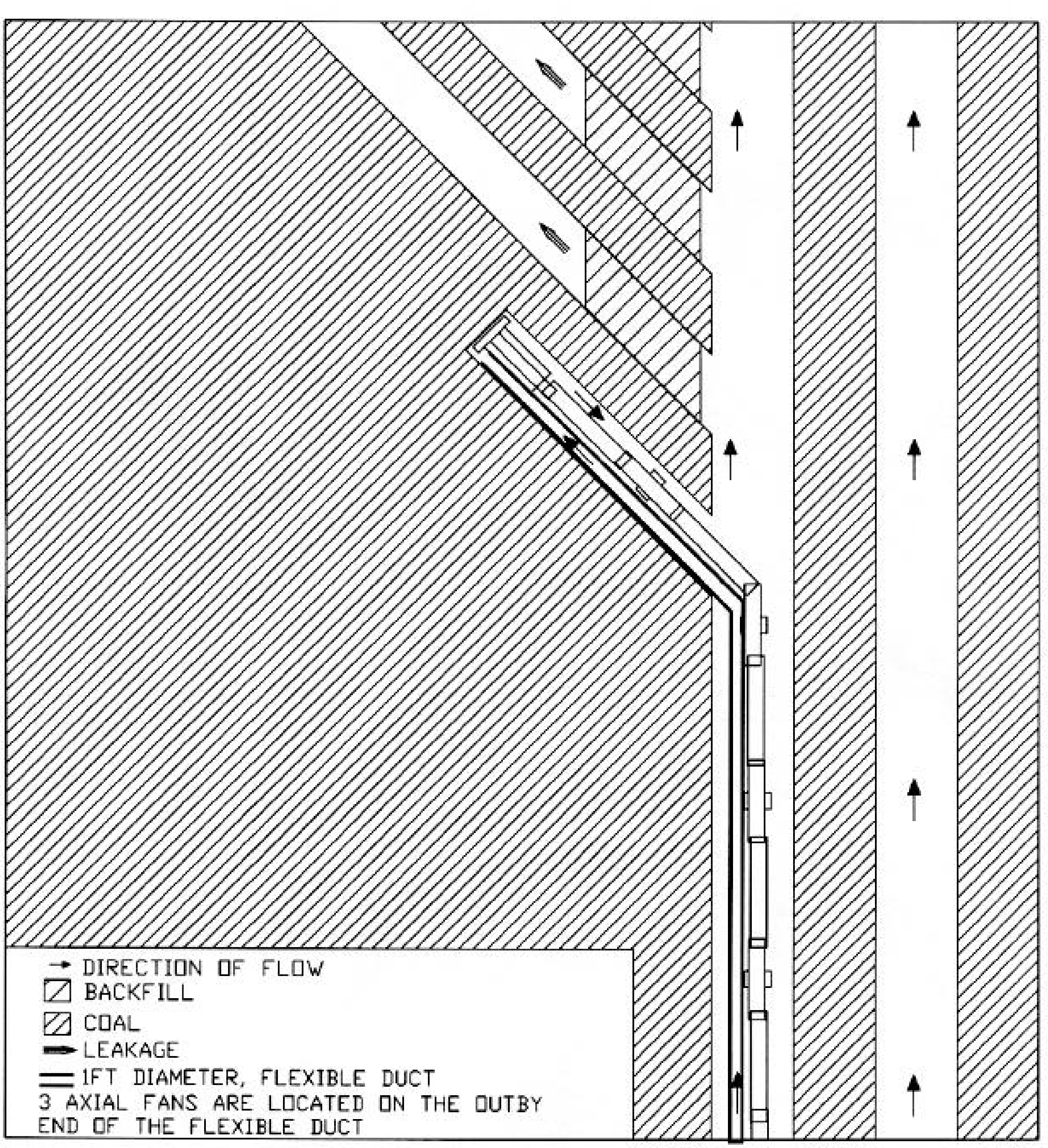

Figure 7.14 Face Ventilation

7.7 Rate of Production

Production for One Panel

• **Shift time**	**8 hours**
• **Production time**	**7.5 hours**
• **Production shifts**	**2 shifts/day**
• **Production days**	**244 days**
• **Depth of cut**	**500 feet (152.4 m)**
• **Mining height**	**28 inches (0.71 m)**
• **Location change**	**30 min**
• **Width of cut**	**13 feet (3.96 m)**
• **Cross-sectional area of cut**	**30.3 feet2 (2.8 m^2)**
• **Rate of Advance**	**3 feet/min (0.015 m/s)**

A 500-ft long stall, 13 ft wide and 28 inches wide has a volume of 15, 167 cubic feet. At an average coal density of 80 lb/ft^3, this equates to 607 short tons. If the continuous miner advances at an average rate of 3 ft per minute, the stall will take 167 minutes to mine. Allowing 30 minutes to move to the next stall gives 197 minutes, or 3.3 hours effective time to mine one stall. Hence, it is estimated that during each production shift, two stalls or 1,214 tons can be mined in each panel.

The annual production rate can be estimated for a production schedule of two eight hour production shifts, six days a week, operating 48 weeks a year. This gives a production of 2,428 tons per day. The mine operates 244 days per year, giving an annual production of roughly 592,432 tons per panel per year. Two panels can be operated on each section doubling this production figure. Multiple sections can be operated in the mine further increasing production.

The average price for coal at the mine mouth in 1998 was $28.45. (Loomis, 1998) At this price, revenue of $16,855,000 will be generated each year per panel. The total annual cost for owning and operating this mine is $8,710,000 not counting the cost of development, ventilation, and main haulage. It is apparent that the profitability of this method is in question, and highly depends on the price of coal. With estimations of taxes, the cost of labor, development, ventilation, and main haulage this system comes out fairly close to the breakeven mark.

If this system can be employed to mine high-grade metallurgical coal, which sells for as high as $40.00 per ton, revenues would increase to $23,697,000. Since many of the unmined seams in this area are metallurgical coals this is a strong possibility. Profitability could also be increased with tax breaks for low sulfur coal producers. This region's coal is high in Btu's (British thermal units) and low in sulfur making it more attractive to consumers who have to watch their sulfur emissions.

7.8 Ownership and Operating Costs

The ownership and operating costs for one section using a FAIRCHILD F330 continuous miner and the two custom Long-Airdox continuous haulage conveyor trains (without guidance systems) are presented below using straight line depreciation and no salvage value.

Ownership Cost

Continuous Miner	$700,000
Primary Continuous Haulage Train	$3,000,000
Secondary Continuous Haulage Train	$6,000,000
Total Cost	**$9,700,000**

Average Investment

Continuous Miner (7 years at 0.57)	$399,000
Primary Continuous Haulage Train (7 years @ 0.57)	$1,710,000
Secondary Continuous Haulage Train (7 years @ 0.57)	$3,420,000
Total Cost	**$5,529,000**

Annual Fixed Cost

Depreciation (7 years)	$1,386,000
Interest etc. (16% @ $5,529,000)	$885,000
Total Fixed Cost	**$2,271,000**

Annual Operating Cost

Labor (2 shift @ 3 workers @ $50,000)	$300,000
Maintenance, repairs and supplies (25% dep)	$346,000
Power (1568 kW @ 65% @ 4,320 hrs @ $0.06/kWhr)	$264,177
Total Annual Operating Cost	**$891,000**

Total Annual Cost	$8,710,000

7.9 Backfilling

Reasons for Backfilling

In addition to the control of ventilation, backfilling the mined-out stalls has other important benefits. By placing backfill into the mined out stalls, structural integrity of the pillars is greatly increased. The fill material becomes compacted and exerts a confining force on the remaining coal pillars, increasing their strength. This is particularly important close to the entries where personnel are located. If there is sufficient waste material to allow total backfilling then the pillars between face openings may have a smaller width. This will give higher recoveries and improved profit margins for the mine. Additionally, the minimization of displacements in the overlying strata will eliminate, or reduce greatly, the occurrence of subsidence, disturbance of groundwater movement and potential problems of acid mine drainage and long-term spontaneous combustion.

Another reason to consider full backfilling falls directly under the wider economic considerations of this plan, disposal of solid wastes. If imported solid wastes can be mixed into the fill to provide a rapid-setting and low permeability material, this provides a means of generating additional revenue for the mining company. However, for the present it is assumed that the fill material required for seals will be provided from the non-coal waste produced from development drivages. (Holman, McPherson, and Loomis, 1999b)

Backfilling System Requirements:

The waste materials will be crushed to a specific size and mixed with a consolidating compound, such as a gypsum-based plaster, into a pumpable slurry. The requirements of the pumping system are that it must be able to pump slurrified backfill material, over level conditions, for an estimated distance of some five thousand feet. The Grud mine in West Germany pumped high density fill in the form of a highly consistent paste over a distance of 6500ft with a pipe velocity of 137 fpm (Crandell, 1992) demonstrating that this plan of action is feasible. High volume paste fill has also been used in the silver mines of Idaho.

Assuming a 13 foot wide stall cut in a 28 inch seam, with 40 foot long seals pumped into each end of every mined-out stall, 2,426 cubic feet of fill per stall will be required. With an average mining advance rate of 3ft/min on a 500 ft long stall, and allowing some 60 minutes for withdrawal of equipment from a completed stall to the commencement of the next stall, the seals in each stall will have to be placed in some 230 minutes. If the two ends of the stall are sealed simultaneously, this equates to a slurry pumping rate of less than 11 cubic feet per minute.

There are a number of types of pump that can accomplish this task. However, several site-specific factors should be considered when selecting the pumps and pipe sizes. These include allowable pressures, maximum particle size, pump wear, power, and efficiency. Pump specifications for slurry material are available from manufacturers. (Zhou, 1993)

Another factor that is important in slurry pumping is pipe wear. Abrasion resistant steels and epoxy-coated pipes are available but very expensive. (Zhou, 1993). This design minimizes bends in the pipe in order to minimize wear and head losses. Two pipe strings are necessary for each miner in operation, one for each end of the stall. (Fig 7.11) The pipe sections will be connected in lengths that will minimize the amount of pipe alteration between filling sessions. A flexible section of pipe will allow the operator to seal two stalls before shortening the pipe string on the retreat. When a pipe string is finished filling it will be flushed with water to avoid consolidation in the pipe string.

This system will employ a hopper where the fill material can be mixed and stored. This hopper will be top loaded either by a conveyor or from a supply train. The hopper will be emptied and flushed out when not in use. Two pumps may be required when running two sections.

If the stalls are to be backfilled along their entire length then waste material must be imported into the mine from the surface. Dependent on its source, such imported waste may require pre-treatment before it can be converted into a pumpable slurry. Furthermore, the duties of the pumps and section pipelines will be increased significantly. For complete backfilling of a mined-out stall a remotely controlled vehicle may be employed (Figures 7.12 and 7.13). Another concept that has been considered involves attaching the inby end of the flexible string to the continuous miner and backfilling during its withdrawal. This would entail an even greater rate of pumping if mining is not to be unduly inhibited. (Holman, McPherson, and Loomis, 1999b)

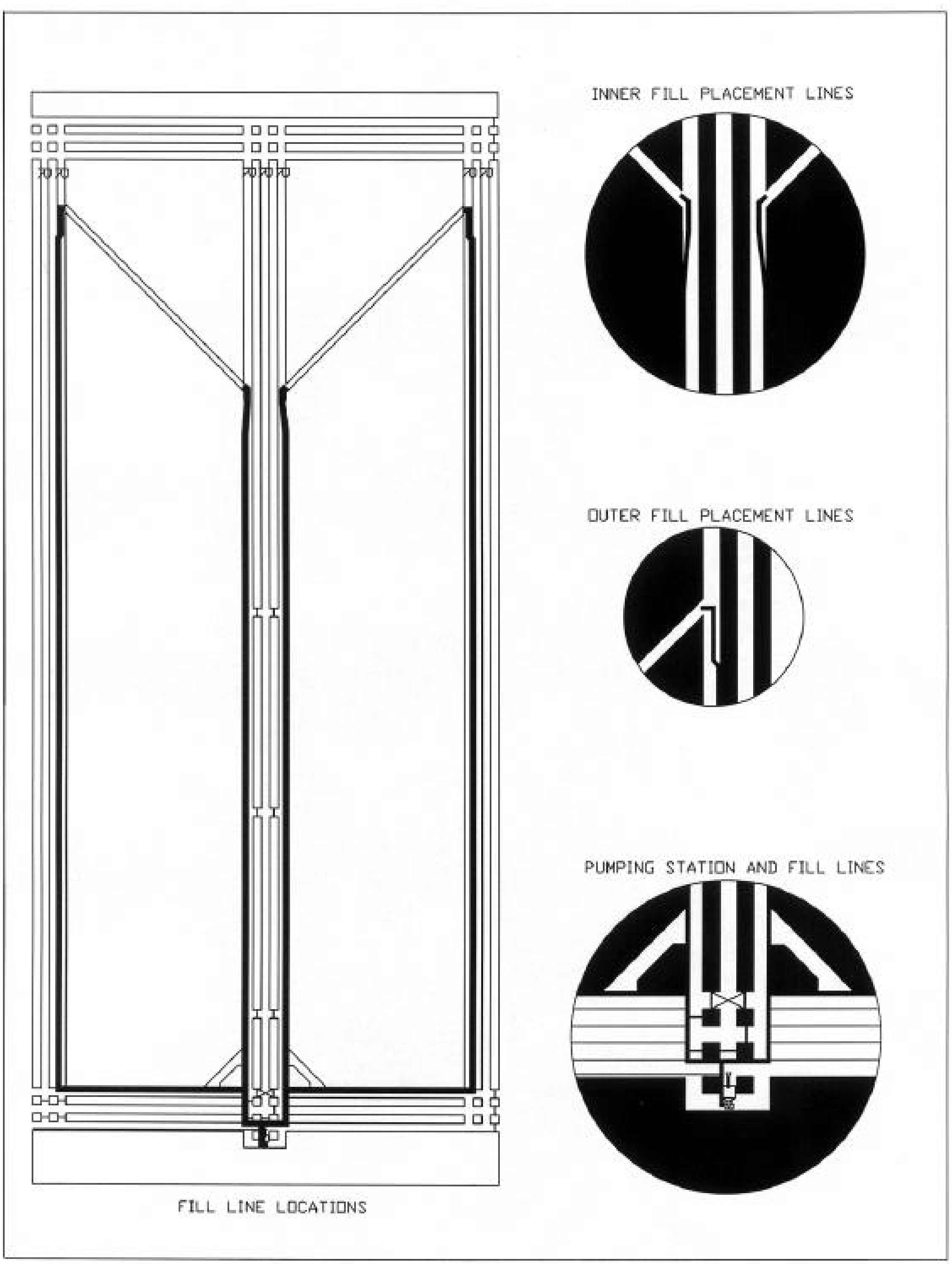

Figure 7.15 Pipe Location

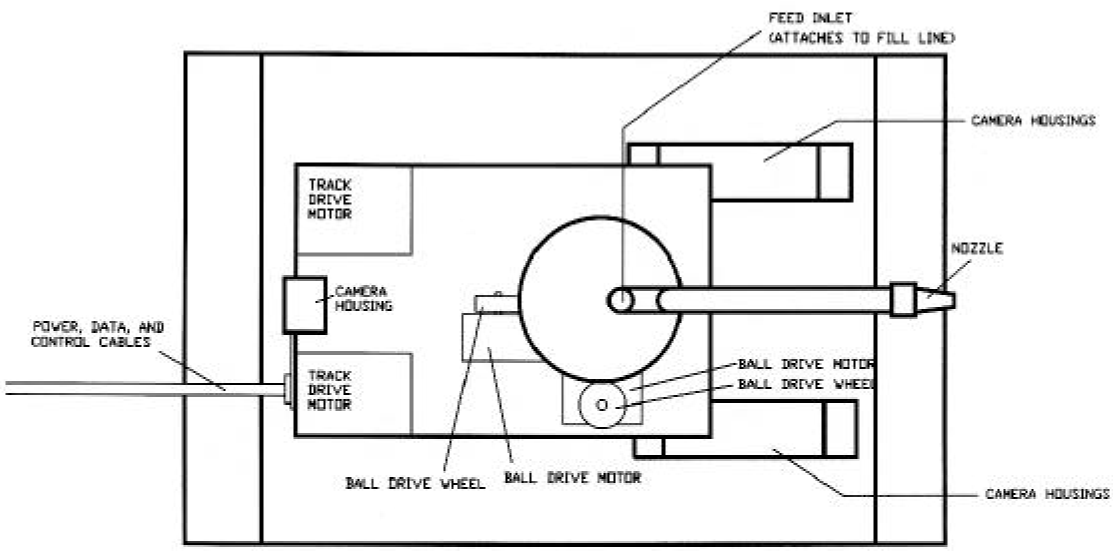

Figure 7.16 Plan View of Fill Vehicle

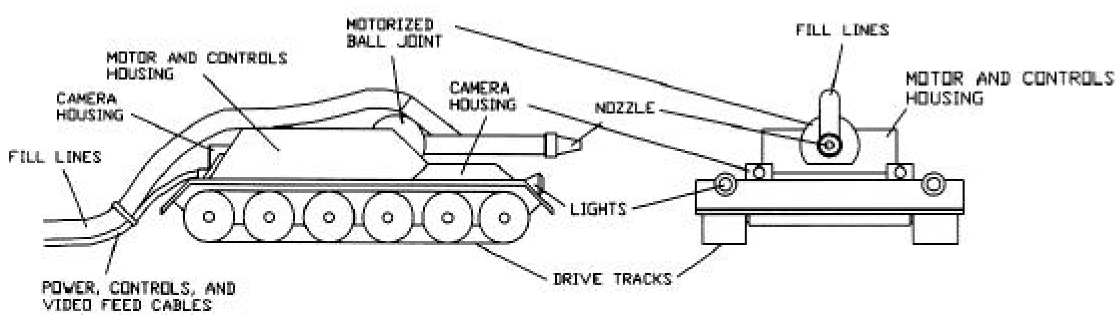

Figure 7.17 Section View of Fill Vehicle

Conclusions

The research done throughout the preparation of this thesis has pointed out many things. First off, there is a large, valuable reserve of coal between 14 and 28 inches of thickness in Southwest Virginia. Numerous methods have been employed in the past to extract seams in this range, but failed due to insufficient technological advancement. They were unable to eliminate workers in the extraction zone. New developments in autonomous vehicle guidance technologies, coupled with advances in thin-seam mining equipment and other mining technologies are making new methods of extracting this coal possible. These methods show great potential for economically mining seams in this range.

Coal is still a necessary resource, and the need for the energy it produces will continue to be in great demand into the next century. These Appalachian coal fields contain large amounts of valuable reserves between 14 and 28 inches of thickness. The need for a method to extract these resources will continue to grow as the Thicker seams are depleted.

For this system and other of its kind to be successful, the manufacturers must continue to show interest , and seek partnerships with:

- **Mining companies** to test and evaluate the systems and equipment that are developed at their sites and facilities.
- The **Mine Safety and Health Administration** (MSHA) and **the National Institute for Occupational Safety and Health** (NIOSH), for cooperation in researching ways to provide higher levels of health and safety in these systems. They will also be needed to grant exemptions in existing regulations as needed to conduct field trials.

- The **State Government of Virginia** to provide financial incentives to operators, research institutions, and equipment developers to develop new ways to economically extract these valuable resources.

Areas for Future Research

This thesis shows that development of new ways to extract these reserves is possible, and that there is great potential in mining seams between 14 and 28 inches of thickness. When industry leaders examined this method, a few areas were highlighted for future research. The first comment was that the cost of developing the main entries was going to be excessive, so attempts to lessen the amount of required development must be undertaken. Another concern was the recovery of the method. This can be improved by cutting the production cuts at angles closer to perpendicular to the access entries. This will greatly reduce the triangular pillars left at the ends of each panel. Concern was voiced about developing a way to extract or access the miner in the event of breakdowns or roof falls. Development of a means of equipment recovery is at the top of the list of concerns. The last point of interest was in the possibility of developing a vacuum haulage technology to eliminate the expensive conveyor trains. This will require much research and development, but could be combined with the face ventilation system, vacuuming the dust and methane out of the stalls.

Other areas that need to be investigated are improving backfill technology, and more work on autonomous vehicle navigation. These areas will yields many great innovations that the mining industry can utilize in future mining systems similar to this one.

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