FEASIBILITY OF AN INTEGRATED THIN SEAM COAL MINING AND WASTE DISPOSAL SYSTEM

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ABSTRACT

The depletion of more attractive thicker and easily accessible coal seams in the central Appalachia will direct attention towards the extraction of coal seams thinner than 28 in. This thesis investigates the feasibility of an integrated mining and backfilling system applicable to thin seams. Two conceptual mining systems, namely Auger mining and Self Advancing Miner, have been proposed for this purpose. Both these systems are designed to remotely mine coal from the seams. Several attempts were made in the past to mine coal in a similar fashion but were not very successful due to several problems inherent to thin seams. The lack of effective steering techniques, accurate coal/rock interface and pillar thickness detection techniques were the main shortcomings of the systems. These problems were addressed in the proposed conceptual mining systems. Several coal/rock interface and rib thickness detection techniques currently available in the market or in the prototype stage have been discussed. Recent developments in coal/rock interface detection and direction sensing techniques have good potential in alleviating the previously encountered problems.

Sensitivity analyses have been performed to assess the of effect critical mining parameters on the production potential of these systems. The self advancing miner has been found to be more promising than auger mining. Conceptual panels and face layouts for both systems have been included. Two types of filling methods namely pneumatic and hydraulic are considered applicable under thin seam conditions. A backfilling technique using rubber hoses for fill placement can be applied with both methods. Sensitivity analysis have been performed to establish the relationship between face operation cost, filling cost per ton and development cost per foot. Resulting analyses indicate that panel cost per short ton of coal is more sensitive to filling cost than on development cost.

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1. Introduction

Coal is widespread and abundant in most parts of the United States, and like petroleum and natural gas, has contributed to industrial and economic growth. Of the three fuels, coal is by far the most abundant. Recoverable coal resources contain about 10 times more heat value than the combined recoverable resources of petroleum and natural gas.

Although coal is abundant in the United States, resources have their limits. Active mining in the past has reduced the southern Appalachian coal reserve located in the thicker seams. In this extensively mined-out region, new areas containing thick beds of high rank and high quality coal are becoming increasingly difficult to locate. It is expected that active mining will deplete the more attractive thicker seams in about ten years. Therefore, in future one must focus attention on extracting thinner seams, in the range of 14 inches to 28 inches in order to maintain the economic prosperity of the region. The purpose of this thesis is to assess the feasibility of mining these thin seams. The first task undertaken was to establish the reserve base for the southern Appalachian region, which is comprised of Virginia, West Virginia and Kentucky. Past assessments vary widely, reporting dire shortage within a few years to no likely shortage with few hundred years.

The problem has arisen due to the fact that much of the basic technical fieldwork for evaluating reserve was done many years ago. Most of the estimates presently available are based on works done in the early part of the century without the advantage of sophisticated equipment and techniques. Due to this reason, the reserve estimations from various sources differ substantially. Moreover, the resource estimations done so many years ago have undergone substantial change as coal, present in the ground, cannot be considered recoverable due to environmental and legal reasons. These conditions were not present during the early reserve estimations. With time, the definition of resource has also undergone changes. To accurately describe the characteristics of coal in the United States, the whole classification system has to be divided into many hundred resource classes or categories.

According to the US Geological Survey [Coal Resource Classification System of the U.S. Geological Survey, 1983] , *Reserve* tonnage are estimated by summing the recoverable quantities of coal in the reserve base of each rank of coal and are assigned to the following categories 1) thickness of overburden - 0 to 500 feet and 500 to 1,000 feet; and 2) thickness of coal - 28 to 42 inches, 42 to 84 inches, 84 to 168 inches, and more than 168 inches for anthracite and bituminous coal; and 5 to 10 feet, 10 to 20 feet, 20 to 40 feet and more than 40 feet for subbituminous and lignite. Reserve must be considered as economically producible at the time of classification but facilities for production need not be in place and operative. *Resource* on the other hand is determined by summing the estimates for identified and undiscovered deposits of coal that are 14 inches or more thick for anthracite and bituminous coal and under 6000 feet of overburden, and 30 inches or more thick for lignite and subbituminous coal and under 6000 feet of overburden. The above definitions of *reserve* and *resource* were the results of revisions which modified the previous definitions. These modification were published by USGS in 1980.

Hence we can expect the recoverable reserve base to be much less than what is reflected by early works. To be considered for targets of development, coal seams must have desirable qualities like high heating value, low sulfur content, and most important of all, must occur in relatively thick uniform bed. The thickness of the coal seam is important as it directly controls the operating cost. This is evident from Figure 1.1, which depicts the distribution of mines and mining method across seam thickness. It can be seen that there is very less mining activity in seams less than 30 inches thick. Most of the thinner seams in West Virginia are untouched and hence offer a potential source of coal in the future.

Coal is found in Virginia in three widely different and dissimilar areas. They are South West Fields, Valley Fields and Eastern Fields. Of the total coal found in Virginia, South West fields account for 97%. It may be noted that Valley fields, concentrated largely in Montgomery, Pulaski and Wythe counties, were mined to some extent before 1960. There was continuous production since 1883. However, the production data is inadequate to permit

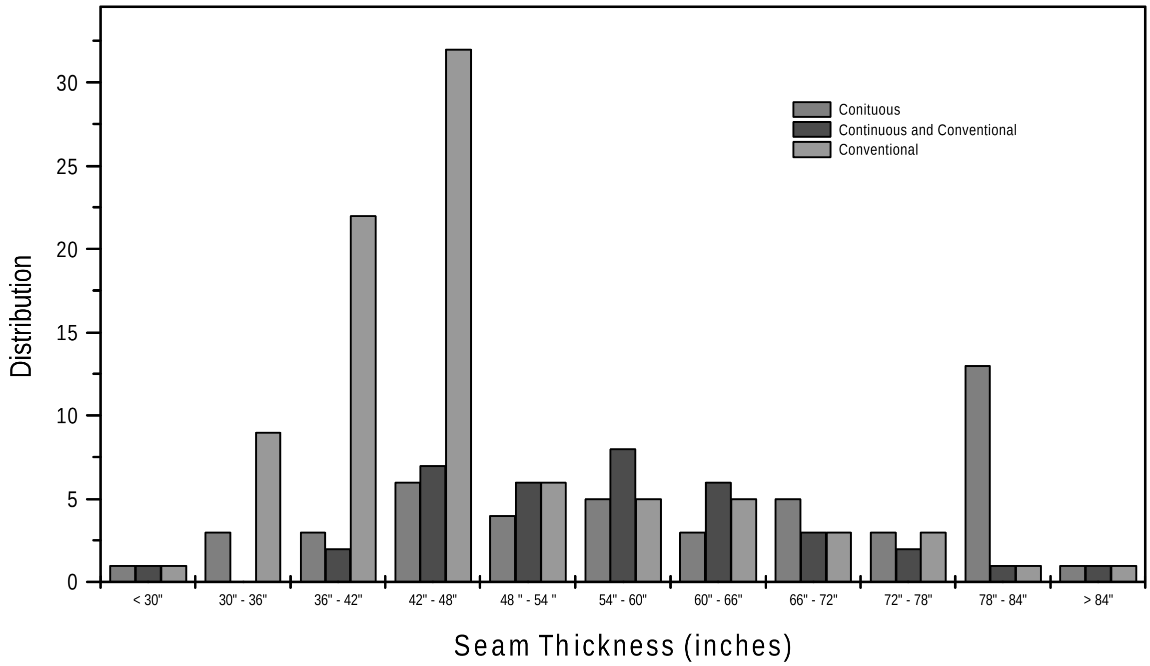


Figure 1.1 Mining method of 172 West Virginia mines by seam height (Schmidt, 1979)

any reserve calculation on its basis. Moreover Valley Fields contribution to the whole state's reserve is less than 3%. As a result, inaccuracy in estimation will not grossly under or over estimate reserve base of the state. The Southwest Virginia coal field occupies an area of about 110 miles long and 30 miles wide. It is mostly found in Buchanan and Dickenson Counties, and occupies part of Wise, Tazewell, Russell, Scott, and Lee Counties. Table 1.1 shown below is a compilation of the reserve as on January 1951 [Brown et. al., 1951].

It can be noted that this compilation of reserve consists only of South West coal fields as the other two regions are mostly depleted, and/or not much coal mining activity can be found there presently. As can be seen from the table, about 47% of the total reserve lie in seams 14" to 28" thick. This is a significant amount, and hence cannot be ignored. The same situation prevails in other surrounding states too. Table 1.2 shows the district wise coal reserve of Kentucky [Keystone Coal Mining Manual, 1996].

Table 1.1 Estimates remaining reserve of Bituminous coal in the SW Virginia Fields (in millions of short tons)

County	14”-28”	28”-42”	> 42”	County Total
Buchanan	2122.83	1307.64	343.75	3774.22
Dickenson	1275.94	920.48	474.99	2671.41
Lee	140.88	209.65	116.18	466.71
Russell	187.44	217.10	314.76	719.30
Scott	58.92	18.68	26.97	104.57
Tazewell	364.39	240.47	103.50	708.36
Wise	890.58	878.95	561.55	2331.08
Total	5040.98	3792.97	1941.70	10775.65

Table 1.2 Estimated remaining coal resource of Kentucky by thickness (in millions of short tons)

Districts	14” - 28”	28” - 42”	42” - 56”	> 56”	Total
Hazard	9907.66	5619.76	2619.15	1182.90	19400.47
Licking River	2698.38	717.20	143.08	31.37	3590.03
Princess	2660.15	609.77	63.34	28.93	3362.19
Southwestern	4889.59	2066.07	415.33	59.17	7430.16
Upper Cumberland	3948.62	4333.71	1196.42	515.13	9993.88
Big Sandy	9509.26	6321.17	2880.74	1587.97	20299.14
Total	33613.66	19667.68	7318.06	3405.47	64075.87

It can be noted that about 52% of reserve lies between 14” to 28” thick coal beds. In some districts, namely Licking River and Princess, about 75% of the total reserves lie in seams 14” to 28” thick. A more recent study undertaken by the Virginia Division of Mineral Resources of Appalachia 7.5 minute quadrangle [Sites & Hostettler, 1991] indicates a reserve of 203 million short tons or about 30% of the reserve in seams under 28 inches.

The above discussion indicate that attractive thicker seams are slowly but steadily being depleted in the southern Appalachia due to heavy mining. It is expected that in about 10 years, the whole area will be depleted of thick seams. Hence there is an immediate need to develop technology for extracting thin seams economically, otherwise coal mining companies of southern Appalachia will loose out in competition with the western counterparts, where presently there is no shortage of thicker seams. Though not much has been done in the past in United States regarding thin seam mechanization, considerable work were done in the British coal fields few decades back. The experience of mining thin seams in Britain can be utilized to our benefits in designing better and more efficient equipment for winning coal from thin seams. There seems to be no alleviating factors when considering the effect of reducing seam section on the design and application of mining equipment. With the thinning of coal seam comes numerous problems unheard of in mining thicker seams. Major problems are those of environment, control, and the ancillary effect that thinning seam section have on face operation. Since returns from thin seams cannot match with a standard longwall face, there is a possible limitation on capital as well as manpower investment. Difficult environmental conditions indicate that in addition to high productivity there must be an aim to reduce the total work force near the face. Powered supports usually consists of substantial top and bottom members. These members represent a constant height irrespective of seam thickness. Hence at very low seam height these supports may be useless. Since powered supports cannot be used, due to safety reasons, workers cannot be employed to work in the face. All these indicate that a completely remote operation of face machinery may represent the only feasible solution. Remote operation is very desirable as the restriction in the mobility of men working in thin seams is severely reduced and this reduction may limit the potential output from the face. The effect on economy of mining operations are also aggravated by thinning seam sections. It has been observed that cost of face equipment depends much more closely on the plan area covered rather than the seam section in which they are introduced. Indeed, due to miniaturization of equipment and its supply to a smaller market, it is possible the cost of such equipment is most likely to increase with reduction of seam thickness.

One thing however should be borne in mind that problems cannot be clear cut since solution must be individual for each installation. It is only possible to draw a general guide of the terms of reference within which face equipment design should be arranged in order to produce systems of mining suitable for thin seam work. A detailed discussion of already existing thin seam mining equipment is presented in Chapter 2. Detailed design features, problems faced during operation, and production achieved are discussed. Chapter 3 introduces the modified mining systems namely auger mining and self advancing miner to be used in thin seams. Detail production potential, cost and recovery for both the systems have been worked out. From these preliminary studies, self advancing miner is more favorable. In Chapter 4 several coal rock interface and rib thickness sensing techniques have been discussed. Currently numerous such techniques exist which can be used efficiently with both the proposed systems. It must be noted here that for successful implementation of thin seam mining, interface detection is of vital importance. Chapter 5 discusses in detail the filling method to be used. Blind filling the stalls is an important issue as most of the thinner coal seams lie deep in the ground where ground pressure is expected to be very high. Hence to improve recovery. Some preliminary calculations regarding the effect of filling cost on panel cost has been evaluated. It seems filling cost is substantial and care must taken to implement filling.

2. Review of Existing Thin Seam Mining Systems

Before conceiving the system for mining thin seams, a thorough literature review was done to draw ideas. A number of publications were found and subsequently referred to in this work. The earliest reference dates back to 1963 [Lansdown et. al., 1963] which discussed the development of Durham Miner and Collins Miner. This was the first major effort to develop a machinery for extracting coal from seams less than 30 inches. The development of Collins Miner was the brain child of Mr. H.E. Collins and developed by Crawley Industrial Products (England). The Miner underwent several field tests to evaluate its production potential and the problems associated with thin seam mining. Several modifications were incorporated into the final machine from the original design after field tests. These problems were taken into consideration before conceptualizing a new thin seam miner. Another reference [Charlton et. al., 1967] discussed in detail the performance of the Collins Miner at Rothwell Colliery. The paper discussed the problems associated with strata control, and machine characteristics experienced at the colliery. A detailed discussion of the design features, operational problems and performance of the Miner is presented in this chapter.

During the early 80’s major research and development work was undertaken in the United States to recover thin seams of southern Appalachia using highwall mining systems. Most of these projects were under contract of U.S. Department of Energy. Two detailed reports, [Skelly & Loy, 1981] and [Treuhaft, 1981], were referred in to the present work. Apart from these works, the Bureau of Mines also initiated some studies for extracting thin seam mountain top coal resources during late 80’s and went on till early 90’s. These methods were summarized in the paper by Mayercheck et. al. and has been referred in the current work.

2.1 The Collins Miner

The development of Collins miner was the first major step in the Mining Industry to exploit thin seams. The major factors governing the design of the machine were as follows:

- The cutting machine should be steerable and have a system of nucleonic coal sensing.
- It should have an independent system of ventilation and methanometer protection
- It should be electrically powered.
- It should avoid the use of cutting chains and should ensure adequate clearance of coal from cutting heads.
- The complete launching platform should be capable of accommodation within a roadway 12 ft wide.
- Control should be housed in a separate control cab.

The general concept of the cutting machine is based on the use of three auger type cutting heads with the outer heads counter rotating. They were driven by water cooled, flameproof electrical motors. Skids were attached to the cutting heads at the rear ends to facilitate ranging operation, thereby allowing steering in the vertical plane. All the equipment was housed in the working roadway. It consists of a telescopic bogie to store the push rods and belt structures that follow the machine up to the stalls, the launching platform with the machine, the control cab, the hydraulic power pack, the fan, and the transformers. Although the launching platform was similar in many respects to that used in the Durham Continuous Miner, it was found necessary, in order to house the equipment within 12 ft roadway, to reduce the stroke length of the push rods from 4 ft 6 in. to 2 ft 3 in.

The Miner was subjected to surface testing in order to test the robustness of the design. Certain modifications were incorporated in the miner after the testing was complete. Underground trials were carried out at the New Lount Colliery. The seam was 2 ft 7 in. thick and overlaid by roof which was not ideal. The trials began in late September 1962 and continued up to Feb 1963. During this period 98 stalls were drilled, accounting for 6168 tons. The longest stall has been 95 yards (285 ft) with the best output in a day of 220 tons. The miner is shown in Figure 2.1

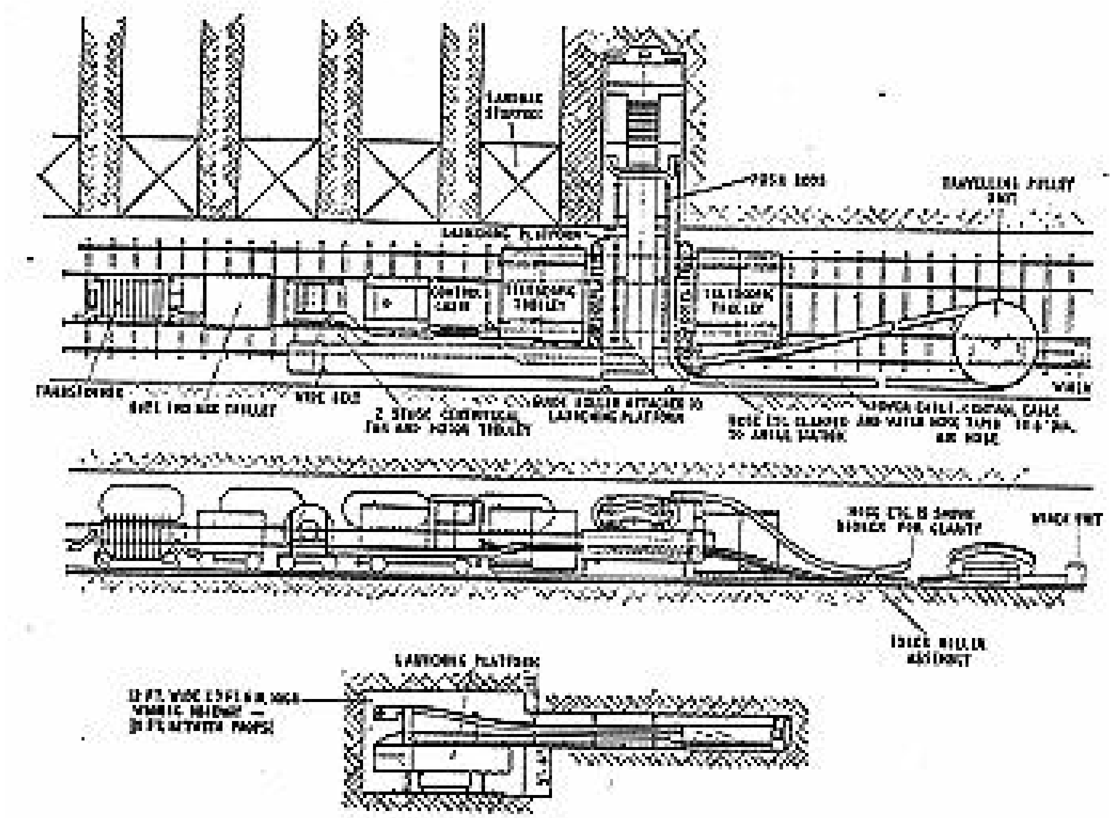


Figure 2.1 The arrangement of the Collins Miner (Lansdown, et. al., 1963)

2.1.1 Operational Problems

Since the whole operation of the Collins Miner was completely manless, number of problems were encountered previously unforeseen. The major problem faced by the crew has been that it was impossible to observe the operation of the miner in close detail. Consistent use of the machine showed that improvement was required in the following areas:

- Absorption of thrust.
- Absorption of power (a) on the cutting machine (b) on the main conveyor.
- Steering mechanism and indication.
- Mining Problems.

2.1.1.1 Absorption of Thrust

The cylinders of the launching platform had a maximum capacity of providing 33 tons of thrust. In spite of this apparently high value, there were occasions when this thrust was insufficient to move the machine. Absorption of thrust was mainly due to friction and weight of the pushing mechanism and the pulling force on the tail roller of the main conveyor. These problems are inherent to any system, where forward thrust is imparted by pushing. Though the designers of the Collins Miner improved upon the efficiency of the whole system by changing the design of the push rods they could not eliminate the problem completely. This was a very important fact that was taken care of while designing an improved version of the thin seam miner. As discussed in the subsequent sections, the push mechanism was completely replaced by a pull mechanism.

2.1.1.2 Absorption of power

It was noted that Collins Miner demanded 50% more power than a shearer. This was very obvious as the miner worked without the advantage of free face. This fact is very important while conceptualizing the improved thin seam miner, because it determines how far the equipment can penetrate into the coal seam. The higher the penetration, the better is its recovery and the higher the production potential. On the contrary the whole system is constrained by space and high power requires very bulky equipment. Hence a trade-off had to be achieved while designing the miner. In some occasions power requirements of the conveyor were insufficient for transporting coal. Experience in New Lount Colliery showed that this was due to three major factors:

- Fines and miscellaneous debris finding their way onto the bottom belt and thence into the tail roller.
- Material jamming between the bottom belt and cross member.
- Cross members being lowered by the push rod structure digging into the floor and jamming the bottom belt.

The problem of cross members digging into the floor can be eliminated by replacing the push mechanism with pull the mechanism. The main reason why the conveyors required excessive power was due to the fact that push rods eroded a groove into the floor while pushing the miner. This resulted in the cross members being lowered onto the bottom belt thus causing it to rub against the floor while transporting coal out of the stalls. The designers of Collins Miner met the situation by raising the level of the cross members.

2.1.1.3 Steering

It was very apparent that the success of the thin seam miner depended very much on the ability of maneuvering it in vertical as well as in horizontal plane. If long drivages were to be made in seams only marginally thicker than the height of the machine, and if consistent pillar thickness were to be maintained, a mechanism for steering was essential. It was noticed during the surface testing phase that the machine had the natural tendency to climb. This tendency of climbing is affected by factors, such as:

- shape of the cutters
- the effect of torque and thrust
- the ingression of fines beneath the base of the machine

The effect of torque and thrust creates a turning moment on the machine which tends to lift the front of the machine and effectively move back the center of gravity. This increases the tendency of the machine to climb and reduce the effective rate of dip, particularly when the torque or thrust on the machine is high. Presence of fines, and grooving of the floor by the rotating cutting heads, induce the machine to run on a bed of fines between 3/8 in. and 1 in. thick. This also increases the tendency of the machine to climb up.

These effects were considered while conceptualizing the thin seam miner. As discussed in the subsequent sections, the cutter head is quite different from the one used in Collins Miner. Horizontal steering of the machine is achieved by means of a pair of lateral jacks sited some little distance from the head of the machine and arranged to bear on the side of the drivage produced along the lateral center line. There were no complicating issues associated with the horizontal steering and this equipment functioned satisfactorily provided that adjustments are made in small increments.

2.1.1.4 Mining Problems

The operation of the machine underground at New Lount Colliery revealed a number of mining problems. Most severe of them was thinning of seam. It presented serious difficulty, because the steering of the machine is based on the floor horizon and the operator has no indication of thinning of the seam until machine is cutting roof stone. By the time the operator realizes that the machine is cutting roof by observing the material in the conveyor, severe damage to the roof has already been done. As a result the miner had to be pulled out before it could cut the full length it was planned for.

Problems associated with strata were sometimes encountered while the machine was in operation. Occasionally thin slabs of roof material had fallen on top of the machine or behind it. These occurrences were traced back to some point where the machine had cut into the roof and therefore broken one or more of the roof beds. The fallen roof materials had often caused serious problems in coal transportation. The thin slabs fell in such a way so as to scoop out material from the conveyor, thus seriously impeding smooth flow of coal. Apart from these, experience in New Lount and Fishburn colliery showed that soft floor and crumpling coal seriously hampered production. Experience indicated that the Collins Miner, as a partial extraction machine, with limitations in its applicability.

Experience in Rothwell Colliery, where 266 drivages were made, showed that 123 drivages, or roughly half, were terminated sooner than anticipated. The reasons for early termination, in most cases, were geological mainly due to spalling. However problems associated with the miner sometimes caused a premature termination of the stall. The more important problems encountered were:

- Speed of advance was less than anticipated because the machine climbed in the hole when pressure was applied to increase the rate of advance.
- The continuous push system of advancing the machine was a failure.
- There was an excessive build-up of pressure as the holes advanced because of high resistance in the transport system and resistance of the push rod structure.

- Spalling of the sides resulted in serious withdrawal problems.
- Old workings in adjacent seam resulted in small fall of stones which led to early withdrawal.

2.1.2 Indication of Position

Vertical and horizontal indication of the position of the miner is equally important as steering the machine. Without proper feedback about the position of the machine, the operator is unaware whether the machine is cutting coal or roof material. Since production from thin seams are paltry compared to longwall faces, mining waste rock under such conditions may have very drastic effect on the economics of the whole project. Hence to make the whole system economically competitive, an efficient mechanism had to be developed for sensing coal/roof boundary. Initially the Collins Miner had two nucleonic coal sensing devices arranged to report back to the driver in the control cab. In the early stages of operation, it became apparent that the coal sensing equipment was not giving reliable information. The reasons for this failure were attributed to the following:

- The coal sensing equipment was set in such a position as to read the thickness of coal beneath the machine. However a significant back clearance was provided between the cutters and the floor. This introduced an air gap between the machine and the floor. The coal sensing apparatus recorded this air gap as approximately double thickness of coal.
- The machine was originally designed to operate at a height of 2 ft. 7 in., but during the test, the seam in New Lount Colliery was found to fluctuate a good deal, down to as low as 2 ft. 2 in. As a result of these facts, it was necessary for the machine to operate with its lower cutting horizon. Sometimes, the machine had to cut 2-3 in. into the floor of the seam. Coal sensing equipment cannot detect how deep a machine is into the floor, but can only record the thickness of coal beneath it.

Consequently, visual examination of product on the conveyor was chosen as a mean to establish whether the machine was cutting coal or roof material. Though this method is far from ideal, it met the purpose. It may be noted that in the 60's, sophisticated technology was not available to design a more accurate coal/rock interface sensing device. Now, however numerous techniques are available which measures the boundary with high accuracy. A detailed review of the available techniques is discussed in Chapter 4.

In order to set the desired horizontal course of the machine and detect deviation, a system of optical sighting was deemed suitable. Though very simple in nature and reliable in operation, it had the following drawbacks:

- It necessitates stopping of the machine at predetermined intervals to observe the illuminated target on the rear of the machine.
- If the seam undulates by an amount approaching the height of the machine, the target disappears from the sight.

These were the fundamental weaknesses of the system. Hence an alternative for the system was being sought. The state of the art in 1960 was not advanced enough to provide an efficient solution to this problem. Currently numerous techniques are available to determine the horizontal bearing of the machine. These are discussed in detail in Chapter 4.

2.1.3 Operational Results from Rothwell Colliery

The operating results achieved with the Collins Miner are shown in Table 2.1 [Charlton et. al., 1967]

Table 2.1 Operational statistics of Collins Miner at Rothwell Colliery

	Heading No. 1	Heading No. 2	Total
Saleable output	15,472 tons	18,816 tons	34,288 tons
Length of heading	530 yd	490 yd	1020 yd
Extraction	27.3 %	32.1 %	29.7 %
OMS at face	8.5 tons	11.7 tons	10.0 tons

The highest weekly output obtained was 1573 tons of saleable coal at an Output per man shift (OMS) of 18.5 tons with the Miner producing on three shifts per day. The best face

OMS obtained during a week was 24.7 tons. It can be observed that results of No. 2 heading were much better than those obtained in No. 1. This improvement in production can be attributed to the experience gained in No. 1 heading. A comparative study between Collins Miner and Anderson low seam bi-directional shearer, working under almost similar condition of seam height indicated that Collins Miner was more economical and productive of the two system. This finding seems quite encouraging, indicating that projects like these may become feasible in the future.

2.2 Highwall Mining System

Highwall mining is a process of extracting coal reserve that are exposed in the highwall created during surface mining. The immediate advantage of highwall mining is that coal reserve can be extracted that would otherwise be uneconomic to mine by conventional surface mining techniques due to high stripping ratio. It can also be utilized to extract coal left as support or as waste during underground mining operations. Since mining of highwall entries is achieved by leaving overburden undisturbed, the systems' economics are independent of strip ratio. For greater recovery of coal from highwalls, South West Research Institute (SwRI) developed an advanced highwall mining system where an auger cutter head could be steered. This would allow the head to remain in the coal seam while penetrating more deeply into the highwall than an unguided auger. This type of mining system has the potential of increasing recoverable highwall coal reserve extensively. A successful implementation of the above technology would result in longer mine life, increased production rate, and reduced overall cost per ton. The key issue that needs to be answered for the success of the system is the development of the guided cutterhead assembly. Standard augers which are available in the market operate as blind boring and extraction systems. They remove coal from relatively horizontal seam which are exposed by removing overburden to form a bench or highwall. Auger mining techniques are primarily used to recover coal from surface where stripping operation or underground methods are not suitable. Though productivity is good under ideal conditions, only 30% to 40% of coal is recovered by this method. Large amount of coal is left above and below the auger hole and in webs between each hole. The highwall miner is shown in Figure 2.2.

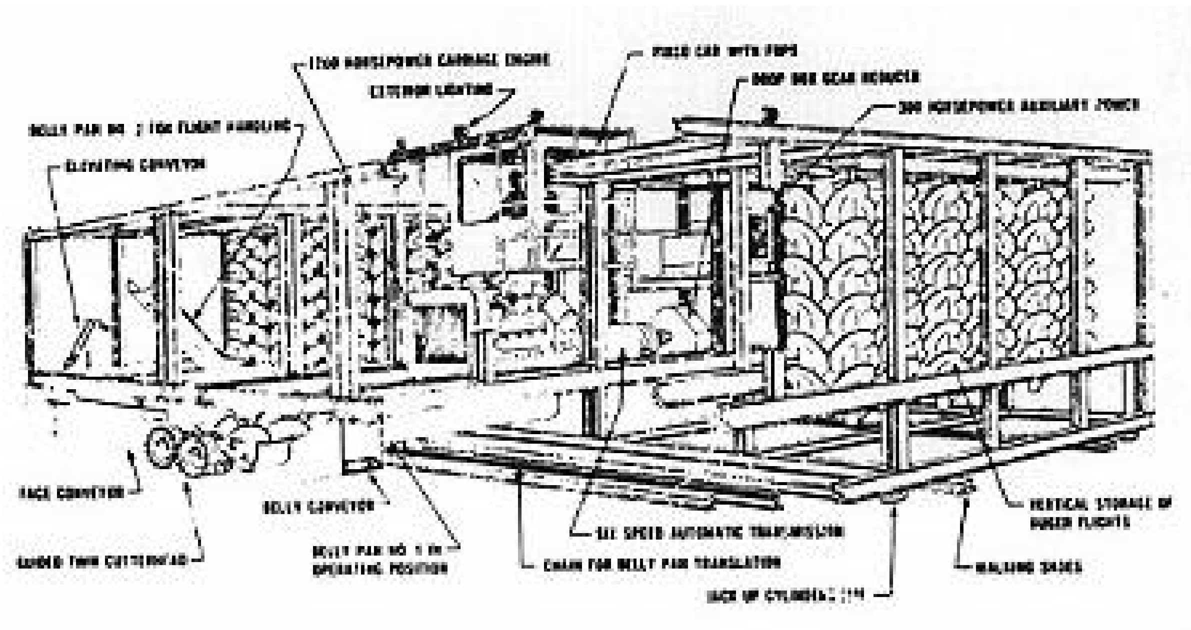


Figure 2.2 A typical Highwall Miner (Treuhaft, 1981)

Unless the operator is able to maintain positive control over the cutterhead throughout the boring operation, it is doubtful that significant amount of coal can be recovered from extended depths even if sufficient power is available for boring operation. The primary objective was to develop a boring machine which would provide at least 35% recovery from the coal seams under the following production criteria, while penetrating at least 500 ft into the coal seam.

Auger Diameter	Production requirement
< 30 inches	> 600 tons/shift
30 - 40 inches	600 - 1000 tons/shift
> 44 inches	> 1000 tons/shift

Moreover, the machine should be able to maneuver on a 70 feet wide bench so as to be completely compatible with the standard mining equipment typically used in the U.S.

2.2.1 Applicability of the system

Extended depth highwall auger mining system provides an excellent approach for extracting coal from thin seams. This method is amenable to relatively level lands and coal seam. In practice, a series of parallel trenches would be progressively excavated across the mining property. The coal between the trenches will be augered as the excavation proceeds.

This method of extracting coal is especially economical as less than 10% [Trauhaft, 1981] of the mined land would require overburden removal and reclamation. This situation is both environmentally and economically suitable and hence has potential in its applicability. Mining operation using Highwall Mining System is shown in Figure 2.3.

Coal handling requirements for Highwall Mining System is not critical, as the trenches are wide enough to allow free movement of vehicles and if necessary, for coal to be easily stockpiled in the pit. Alternatively, coal could be easily elevated from the pit via an elevating conveyors so as to reduce pit congestion.

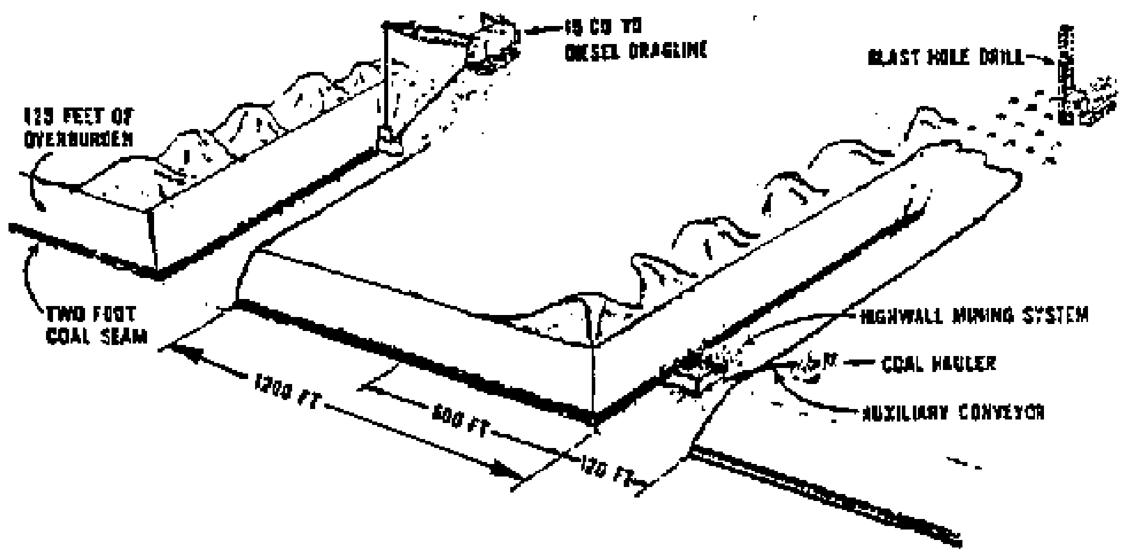


Figure 2.3 Highwall mining in progress (Treuhaft, 1981)

2.2.2 General Description

The Highwall Mining System is based on the dual auger configuration to maximize recovery and control without compromising flight handling and storage capabilities. The whole system is considerably larger than existing dual head auger system due to the flight storage facility that is provided with the machine. The Highwall Mining System differs from the conventional machines due to the presence of 1200 hp multi-speed auger drive train and vertical storage facility of augers. Such a high power multi-speed engine is required to achieve the desired production rate when boring at extended depths. To accommodate the system into narrow benches and increase the mobility, the whole unit is composed of three trailer-able units, two auger bays and one main carriage. The augers are made of alloy tubes, thus making it considerably lighter than the conventional auger flights. This reduces the dead weight to be carried and saves on power due to reduction in frictional losses. More power can now be utilized for productive work like cutting and conveying of coal. Conveyor belt system is used for transporting coal which is discharged from the auger bore hole. Coal is first loaded into a small belly conveyor which discharges its load into a small face conveyor which moves the coal to the outer edge of the Highwall Mining System structure. After leaving the face conveyor, coal is discharged onto a loading conveyor which elevates it into haul trucks or stock piles. The elevating conveyor is pinned to the Highwall Mining System, but can be easily removed and driven to the opposite side to maintain compatibility with the direction of advance.

The production capabilities of the advanced Highwall Mining System is dependent on time required for boring, flight handling, flight retrieval to and from the bays, and tramming of the machine from one hole to the next. However, it was found that tramming required a few minutes. As a result its effect on the production cycle is negligible compared to other elements. Major bottleneck lay in the time required for flight removal from the bays, and connecting them to existing auger flights. Since productivity is directly proportional to total time required per hole, careful attention must be given to those machine functions and operating parameters that specifically influence boring time and downtime. After careful observation of several auger sites, it was found that about 40% of total time is taken up by flight handling and making connections [Trauhaft, 1981]. To improve upon downtime, several changes were proposed. They are listed below.

- Increase carriage reaction speed for both flight installation and retrieval.
- Increase carriage advance speed during flight retrieval.
- Alter flight transfer to and from carriage.

The third point incorporates an improved flight handling system that required only lateral transport of the flights from the carriage. The lifting and lowering of the flights from the carriage is done in parallel with augering operation thus reducing downtime.

2.2.3 Steering the Cutterhead

The success of the Highwall Mining System lies in the operator's ability to maintain control over the cutterhead throughout the boring operation. Without control, it is doubtful that significant amount of coal can be won from extended depths. A guided cutterhead must meet one primary objective, that it should respond to operator directed commands to change the direction of the boring operation. The steering ability of conventional augers are restricted to the ability of the operator to control thrust. This is the only control available to the operator for steering. The problem is that the thrust available at the cutter head decreases steadily with increasing bore depth. As a result the operator slowly loses control over the cutterhead. This is a major reason why conventional auger holes are restricted in depth of around 150 feet. Single augers have been observed to go up and to the right under high thrust levels, while at low thrust levels the head tend to cut down. At higher penetration rate the couple action generated by the turning auger head exceeds the weight of the cutterhead. This causes the head to climb up at high penetration rate. At low cutting speed the weight of the head is dominant and hence the cutter moves downward. Two steering techniques that seemed feasible were investigated; namely *pivot steering* and *pressure steering*. Both these techniques require some penetration velocity in order to incorporate corrective forces at the cutterhead to initiate redirection. Each of the steering techniques is described below in detail.

2.2.3.1 Pivot Steering

In this method a turning moment is developed by applying forces to the rear end of the cutterhead assembly. Pressure pads are used to pivot the assembly in the direction of the desired steering correction. The pressure pads are extended until they push against the borehole wall which create the necessary moment to turn the cutterhead.

2.2.3.2 Pressure Steering

This method consists of hydraulic jack pads located immediately behind cutterhead barrels. These jacks are extended and forced against the walls to initiate a steering change. The interaction of the new forces from the jack pads with the cutting forces at the head change the dynamics of cutting process and redirect the head. The degree and length of time the pads are extended depends upon the responsiveness of the cutterhead assembly and can only be determined by experience.

Each of these steering methods require forward motion of the cutterhead to maximize its effectiveness and ideally corrections should be applied without sacrificing penetration rate. Since the steering system is designed to apply steering forces that are independent of the thrust, it is virtually independent of bore depth. However, these steering systems have one major drawback. Since implementation of these techniques require attaching additional structures on the head, they could be a hindrance to smooth flow of coal. A secondary requirement for guiding the auger cutterhead is a reliable sensing techniques that can resolve the position of the cutterhead with respect to the coal seam boundary. In conventional augering machines, operator usually determines the cutterhead's relative position by observing the products coming out of the borehole. Sometimes experienced operators can determine the position of the cutterhead by feel, though this ability diminishes fast with increasing borehole depth. Hence for extended depth cutting, availability of real time information on when the cutterhead intersects the non-coal boundary is of vital importance. This will allow the operator to make positional corrections in time to prevent the cutter going into roof, rather than wait until visual observation of material exiting the borehole. Numerous techniques are available today which can very accurately determine the cutterhead's position relative to the coal/non-coal boundary as well as adjacent drilled holes. These techniques are discussed in detail in Chapter 4.

2.2.4 Communication with the Cutter

A communication link between the cutterhead assembly and the mining machine is necessary for the operator to initiate steering command as well as receive sensor information.

Several techniques were considered namely 1) VLF Radio Propagation; 2) Current Induction; 3) Microwave Propagation. Out of these, VLF Radio Propagation was selected for the downhole communication link. This technique showed the highest likelihood of working at extended depths without requiring extensive mechanical modification to the auger flights. Testing also showed that signal attenuation by mounting the antenna in the worst configuration was only 30% of the transmitted strength. Thus under normal condition, a loss of not more than 20% can be expected. Furthermore, signal strength was not found to degrade by adding auger sections, since all significant losses occurred within first few feet of iron surrounding the antenna [Treuhaft, 1981].

2.2.5 Power Consumption

Power requirement for the whole process is a significant factor in assessing the boring depth capabilities of the Highwall Mining System. In general, total horsepower required to bore a single hole, includes power required for cutting and power required for conveying coal. Hence power consumption for the system can be examined in terms of cutting and conveying horsepower requirements. For a given set of seam conditions and fixed penetration rates, cutting power is independent of bore depth. It only depends upon penetration rate. Non-cutting power was found to be proportional to the product of bore depth and penetration rate.

Highwall Mining System is promising in its applicability. This system can be successfully applied in extracting very thin seams which are otherwise uneconomic to mine by open pit or underground techniques. Since mining by highwall entries require considerably less overburden removal, the system has high economic potential. This system also demands very low manpower compared to conventional mining techniques. However, to make the system widely available and acceptable in the market, several key issues have to answered. A very efficient guidance system have to be developed in order to provide control of the cutterhead in the hands of the operator. An efficient guidance system is key to the success of the whole mining system as it will result in high production, less mining losses in the form of support pillars, and waste extraction.

2.3 Cable Towed Plow

Apart from the mining methods discussed above, several other systems were referred. Two variations of this system were conceived, one with roof support and the other without roof support by Meyercheck et. al. in the early 90's. Though they represented two different systems, they worked on the basis of same principle. A thin seam longwall plow system was utilized for extracting coal. A typical thin seam cable towed plow is shown in Figure 2.4. The overall system consists of plow and an armored face conveyor drawn by steel cables attached to motors located in the head and tail gates. This system was proposed to excavate thin seams in the mountainous area of the Appalachian region of eastern United States. The benches cut on the mountain serves as the head and tail gates for the system. The face is advanced remotely by crowding the pan line with the cable tension system. The plow is dragged across the face by the cables which dumps coal into the Armored Face Conveyor. This process is repeated back and forth thus advancing the face. As the face advances, the goaf is remotely backfilled up to the edge of the conveyor. However, there are some uncertainties associated with the system. The cost of backfilling, and whether or not this can be accomplished within the framework of a low cost mining system, is questionable. Technically the feasibility of the system is also in question. Whether the cable drawn system will be able to produce enough tension for adequate face advance is not established as there is no mention of the system being used in any mine.

2.4 Roof Fall Tolerant Mining System

Another system that was conceptualized hand in hand with the previous one was the "roof-fall-tolerant" system [Meyercheck et. al., 1990]. In this system, no artificial roof support is required. This unique feature greatly simplifies the overall mining system and provide the opportunity to develop a very low cost system, which is a necessity in thin seam mining. In this method a small cutting device is made to advance in the seam to cut thin slots, with subsequent caving. The cutting machine consists of a long, loop chain saw cutting and conveying system that spans the entire mountain top. Coal is thus extracted in thin slots from the seam. This method is very similar to the plow AFC system discussed previously.

Only difference is that in the current system, coal is extracted in thin slots rather than in blocks.

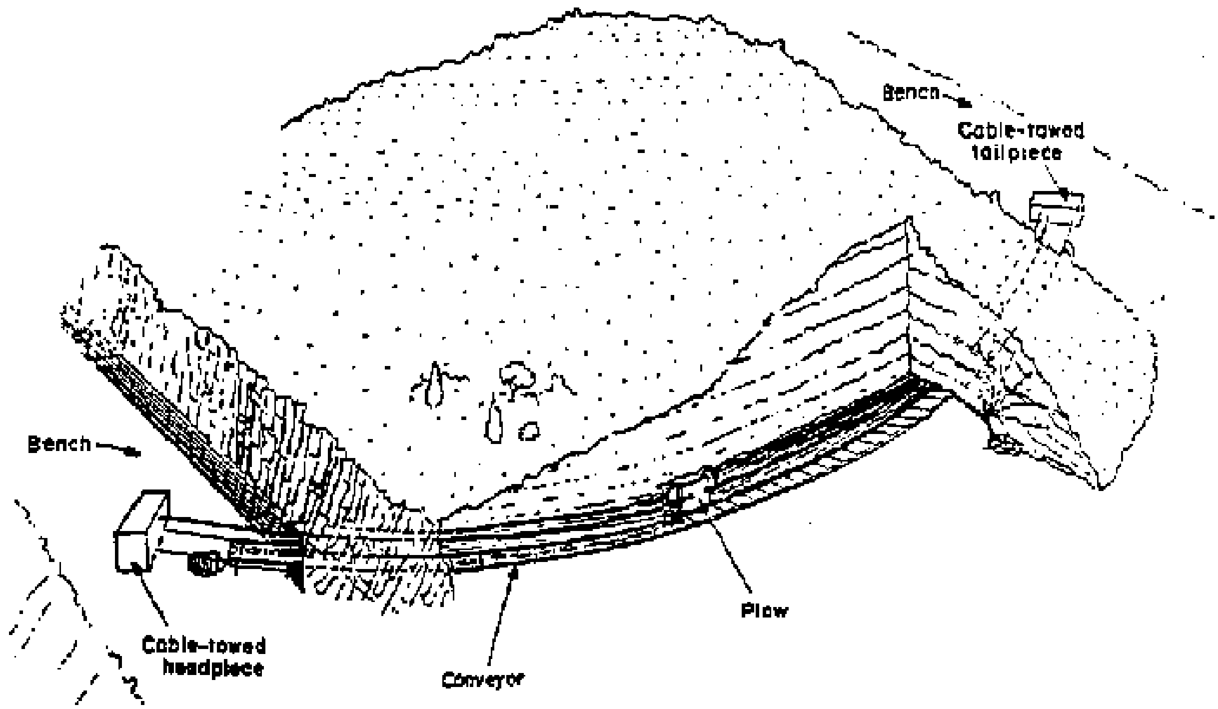


Figure 2.4 Cable towed plow for extracting mountain top deposits (Mayercheck et. al, 1990)

The systems discussed above have not been developed as a full scale machines and tested in any mine. Hence not much can be commented on their practical applicability. The systems have only been tested on laboratory models. Wood cutting chain saw was used as model cutting system and an 18 by 18 by 18 in block of layered cement like material was used to simulate the coal seam. Airbags were used to generate overburden pressure. A total of 15 blocks were tested upon with this cutting system. Blocks were subjected to an overburden pressure of 250 ft. Since the test blocks were only 18 in long, did not produce enough unsupported length for caving to occur. Hence the dimensions of the block were increased from 18 in to 30 in so as to develop a longer unsupported overburden. The model was found to cave gradually in the middle area as each subsequent pass of the saw cut out more coal.

The test revealed that the roof-fall-tolerant system is valid and has potential. Testing on small scale models supports the thin slot-cave hypothesis, which is the central concept behind this mining method. However only field trials can prove its efficiency and applicability.

3. System Description

Two different mining systems have been proposed in this study. These systems have been proposed after adopting ideas from previous works done by different organizations in the past. Both the systems does not depart significantly from the conventional approach of coal mining. Special features have been incorporated to accommodate the machine for conditions prevalent under thin seam mining. Although the systems have been conceptualized, much research and equipment development work is needed before implementation of these systems.

Auger mining, which was commonly used for extracting coal left by open pit mining, has been modified to be used underground in thin seams. Augering has been considered for mining thin seams because in the past multiple head augering has been successfully implemented on the surface by several mining companies. Balkan Auger has conducted successful multi-head augering of the Derby seam in Kentucky averaging 29 inches in thickness sandwiched between massive sandstone. With an 18 inch dual head auger, the company could achieve an average production of 500 tons per shift with production peaking at 650 tons per shift [Coal Age, 1974]. In another case, Fair-Quip augers were used in a 5-block seam in Kanawha County in West Virginia. These twin head augers could drill up to 250 feet in length, with typical auger production from 70 to 150 tons per hour [Coal Age, 1972].

A lot of research on highwall mining system was done during the early 80s by Skelly and Loy consulting firm and South West Research Institute. The investigations included study on extended depth highwall mining systems, high recovery cutterheads, and variable angle augers. Each system was reviewed with respect to 1) Physical design, 2) Recovery and production potential, 3) Safety and environmental considerations, and 4) Costs. The reports also present comparative studies on different systems illustrating their relative standing in various above mentioned categories. Apart from discussing the miner and the mining logistics, the SWRI report also discussed in detail various CID and web thickness sensing techniques. These techniques are discussed in detail in chapter 4. SWRI developed a guided auger head for greater penetration into coal seam without much deviation.

As a result, multiple head auger machines have been proposed for mining thin seams underground. Special direction control units need to be utilized to improve extraction, and hence make the operation feasible. Innovative logistic of mining and filling operations are used to increase extraction from panels by leaving smaller support pillars. Various sensitivity analyses have been done to evaluate the feasibility of the system. Details of these analyses are presented later in this chapter. Cost analysis has been done to estimate the approximate face cost of production.

The second system that has been proposed is based on the principle of a continuous miner. The whole mining machine is an integrated system comprising a cutting unit, a conveying unit and a ventilation unit. A detailed conceptual diagram of the system is presented in this chapter. This system is more flexible compared to auger mining and has a wider applicability and higher production potential. A comparative study between different proposed systems is presented. The whole approach of opening up mining entries has been modified to accommodate the machine in restricted space of underground openings without sacrificing much on the size and power. Various sensitivity analyses have been performed to highlight the attractiveness of the system. A new type of conveyor, which can turn corners, has been used to transport coal out from the face. The feasibility of using the same conveyor to fill the mined out area is being sought. A small computer program has been developed to perform the sensitivity analyses of the system.

3.1 Auger Mining

The layout of this system does not depart substantially from the conventional approach of developing an underground coal mine. Main mine headings and panel entries are developed in a pattern similar to a typical coal mine. Main headings have two parallel entries connected by breakthroughs. The headings and the panel entries will have a rectangular cross section area of about 20 ft by 8 ft (6 m by 2.5 m) to accommodate the mining machines. Continuous mining machines may be used for development of the headings and panel entries.

From the main heading, about 4,000 ft (1220 m) long panel entries will be driven perpendicularly about every 1,250 ft (380 m). Panel entries will be connected to ventilation return airways. The face layout required by the system is shown in Figure 3.1. In the panel entries, a coal augering machine will be used to mine the coal seam. Double headed augers with vertical and lateral steering control devices will advance about 600 ft (183 m) into the coal seam. This is shown in figure 3.2. Two augers 40 ft (12 m) apart will advance on the same side of the entry and will excavate a strip of coal 600 ft (183 m) long and 4 ft (1.2 m) wide every 16 ft (4.5 m) apart. The voids created by the excavation process will be backfilled and sealed immediately.

When one side of the panel is finished, the auger will mine the other side of the panel in the same fashion. The augers will then be employed again for mining the webs of the coal left between previously mined stalls by leaving a 2 ft (0.61 m) rib on each side. This operation will be repeated until the whole panel is mined out. Mining and immediate back filling of the stalls will maintain the integrity of the entries. Mined out coal will be transported to the main heading by belt conveyors. An inert gas, possibly nitrogen, will be injected into the stalls during the augering and backfilling operation to reduce the risk of methane explosion. Ventilation in panel entries will be provided by fresh air intake from the main heading to a return airway at the other side of the panel.

As mentioned earlier, guiding the auger heads is of utmost importance for the success of the system. With the present available technology, two different approaches of steering have been proposed: pivot steering, and pressure steering. A detailed discussion of these two methods of steering is provided.

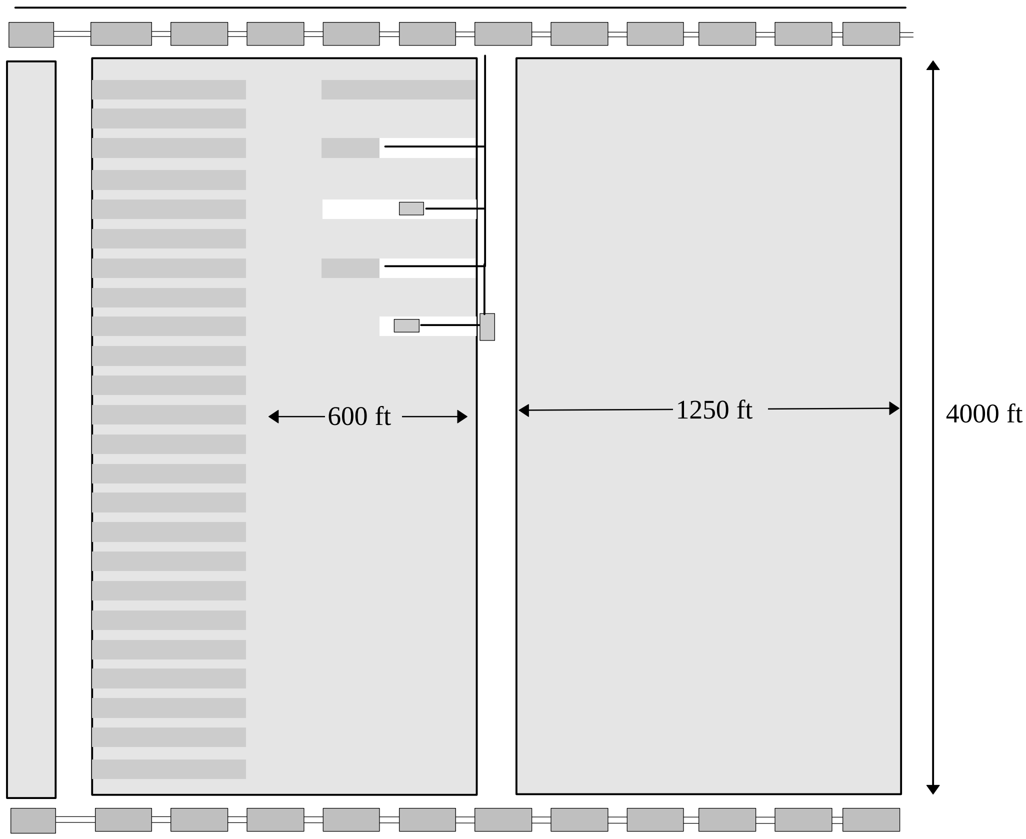


Figure 3.1 Panel layout for auger mining (Foreman & Shelton, 1974)

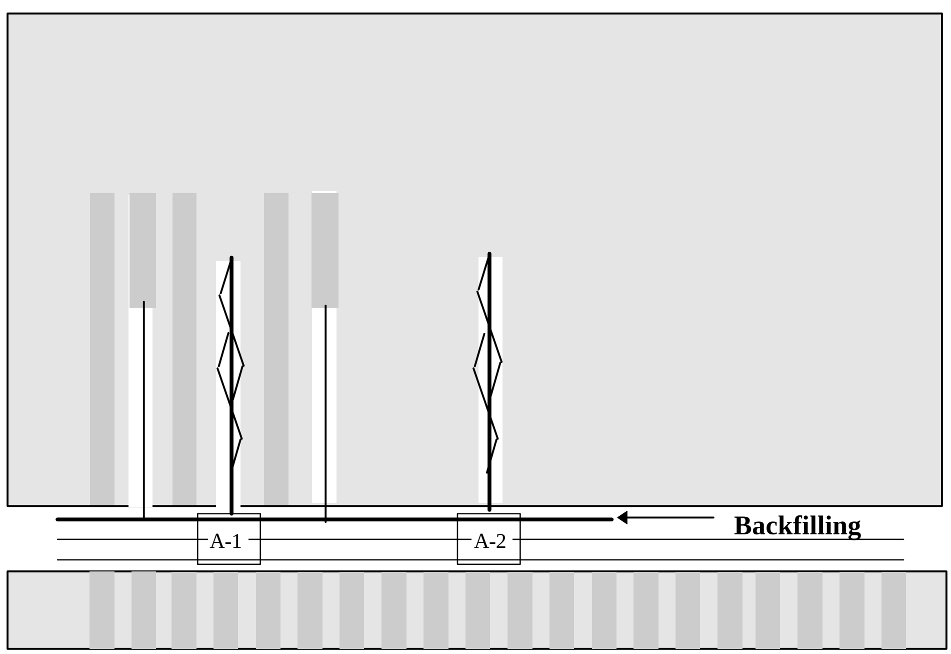


Figure 3.2 Mining and backfilling operation (Foreman & Shelton, 1974)

3.1.1 Pivot Steering

In this method, a turning moment is developed by applying forces to the rear of the cutterhead assembly. A turning moment is created by extending pressure pads in the direction of the desired steering correction. The pressure pads pushes against the borehole wall, thus creating a moment by applying direct force rather than by friction between the pads and the borehole walls. This moment causes the cutterhead to turn in the desired direction. Another method is also available for steering cutterheads. The ripper method works by extending the ripper teeth into the borehole's floor or roof to create a pivot point or moment. However, the effectiveness of the method depends upon the forces generated by the pivot teeth in relation to the directional cutting forces being exerted on the head. The method is limited by the ability of the ripper teeth to maintain sufficient force during the ripping process. The force depends solely upon the design of teeth, chipping depth and the mechanical properties of the material being chipped. From practical experience it was found that the ripper needed several teeth to be successful. This caused packaging and coal flow problems. As a result this method of steering is not recommended, especially in thin seams where there is much less clearance [Treuhaft, 1981].

3.1.2 Pressure Steering

In this method, hydraulic jacks located immediately behind the cutterhead barrels are extended to initiate steering. The jacks are forced against the borehole's wall. The interaction of the new forces from the jack pads with the cutting forces at the head change the dynamics of the cutting process and redirect the head. The responsiveness of the cutterhead to steering depends upon the force-time history of the pads against the borehole. It was found by theoretical calculations that this steering method required forward motion of the cutterhead to maximize its effectiveness, and ideally, correction should be accomplished without sacrificing penetration rate [Treuhaft, 1981].

Conventional auger steering corrections are controlled only by thrust, and, for the first 100 feet or so feet of bore depth, the longitudinal forces obtained by altering thrust tend to be sufficient to make the necessary steering corrections. However, with increasing depth, the control over the cutterhead slowly diminishes until a point is reached where the operator has no control over the cutterhead. Since the discussed methods of steering are independent of the level of thrust, they are virtually independent of bore depth [Treuhaft, 1981].

3.1.3 Production Potential

Production potential of a dual head auger is evaluated by determining the volume of 2coal extracted by the machine and multiplying the figure with the density of coal. Since the cutting machine has a dual head, it excavates an area which resembles a “figure eight”. The contour of the excavated region of a typical dual head auger is shown in Figure 3.3.
The area of the excavated region is evaluated from simple geometry and is approximately equal to $1.79d^2$, where d is the diameter of each auger head. The actual area is somewhat less than this because not all of the coal between the heads is removed [Trauhaft, 1981].

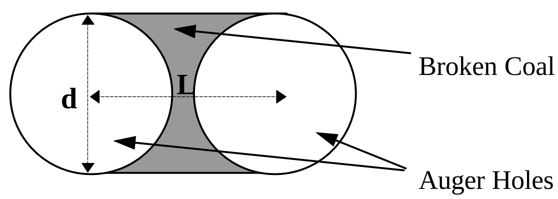


Figure 3.3 Contour of the region excavated by dual head auger

Some basic assumptions which were made for determining production potential, and subsequent sensitivity analyses are listed below. Sensitivity analysis on production potential was performed by varying parameters like penetration rate, flight length, and auger diameter. The outputs of the analysis are shown in Tables 3.1 and 3.2.

• Shift time	8:00 hr
• Auger production time	6:00 hr
• Depth of cut	600 ft (183 m)
• Auger diameter	24 in and 30 in
• Location change	1 min/flight
• Cross sectional area of cut	$1.79*d^2$

Table 3.1 Variation of production versus different penetration rate for a 24 inch dual head auger

Penetration Rate (ft/min)	Production (t/shift) *Flight Length 8 ft*	Production (t/shift) *Flight Length 10 ft*
6	319.25	348.75
9	378.69	420.97
12	426.25	479.09
15	459.71	523.12

Table 3.2 Variation of production versus different penetration rate for a 30 inch dual head auger

Penetration Rate (ft/min)	Production (t/shift) *Flight Length 8 ft*	Production (t/shift) *Flight Length 10 ft*
6	498.82	544.92
9	591.71	657.76
12	666.01	748.58
15	718.30	817.38

The corresponding graphical representation of the sensitivity analysis are shown in Figures 3.4. and 3.5 respectively.

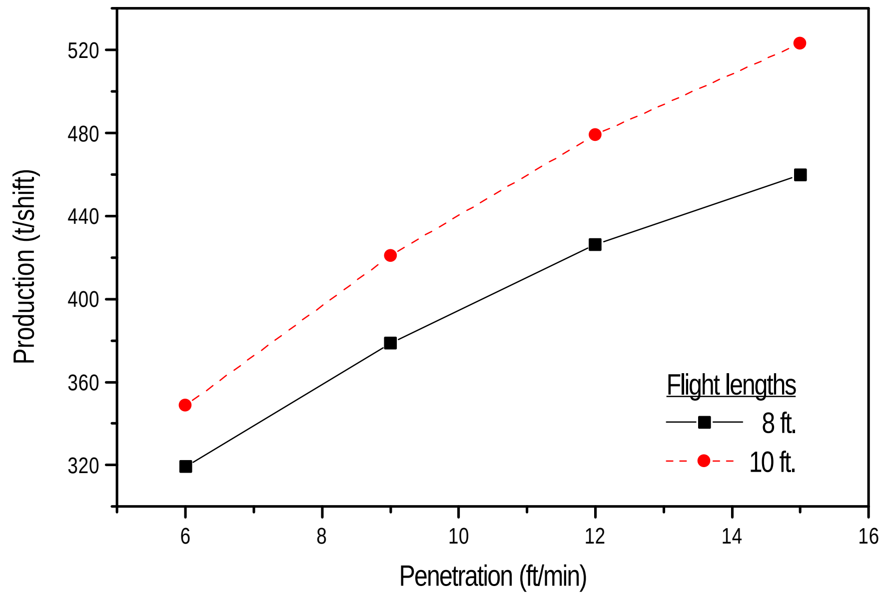


Figure 3.4 Relationship between production and rate of penetration (24 inch auger)

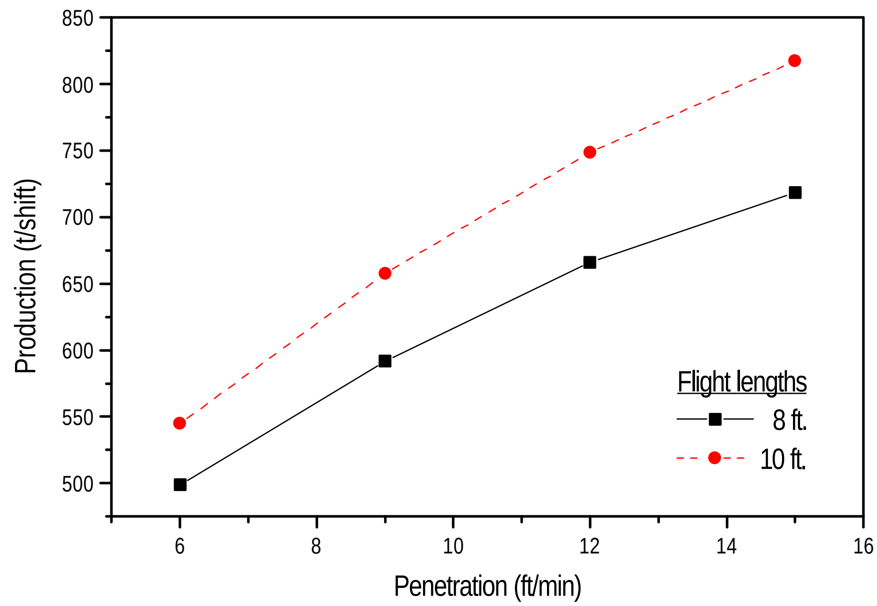


Figure 3.5 Relationship between production and rate of penetration (30 inch auger)

3.1.4 Power Requirement

The power required to remove coal during the boring process is a significant factor in assessing the boring depth capability of the auger mining system and in determining the system's design parameters with respect to its operational requirements. In general, the total horsepower required to bore a single hole can be divided into two components, namely cutting and non-cutting. These include the horsepower required for cutting, conveying, and frictional losses. In this case, the non-cutting horsepower term includes all contributions resulting from the mass of material being conveyed, the mass of the auger flights, and the frictional losses resulting from material and mechanical interaction.

For a given set of seam conditions and for a fixed penetration rate, cutting horsepower can be considered to be virtually independent of bore depth. Conveying horsepower, however, does vary with the penetration rate. The empirical relationship between required power, penetration rate, and bore depth is given below [Treuhaft, 1981].

Total Horse Power (THP) = $0.1 \cdot xv + 2 \cdot v^{1.7} + 14$ (1)

where v = instantaneous velocity (ft/min)

x = bore depth (ft)

Assuming 500 hp (373 kW) for each auger string and 1000 hp (756 kW) for dual head, penetration rate is given by

$v = 49.9 - 6.68 \, ln \, x$ (2)

Figure 3.6 shows the relationship between bore depth and penetration rate.

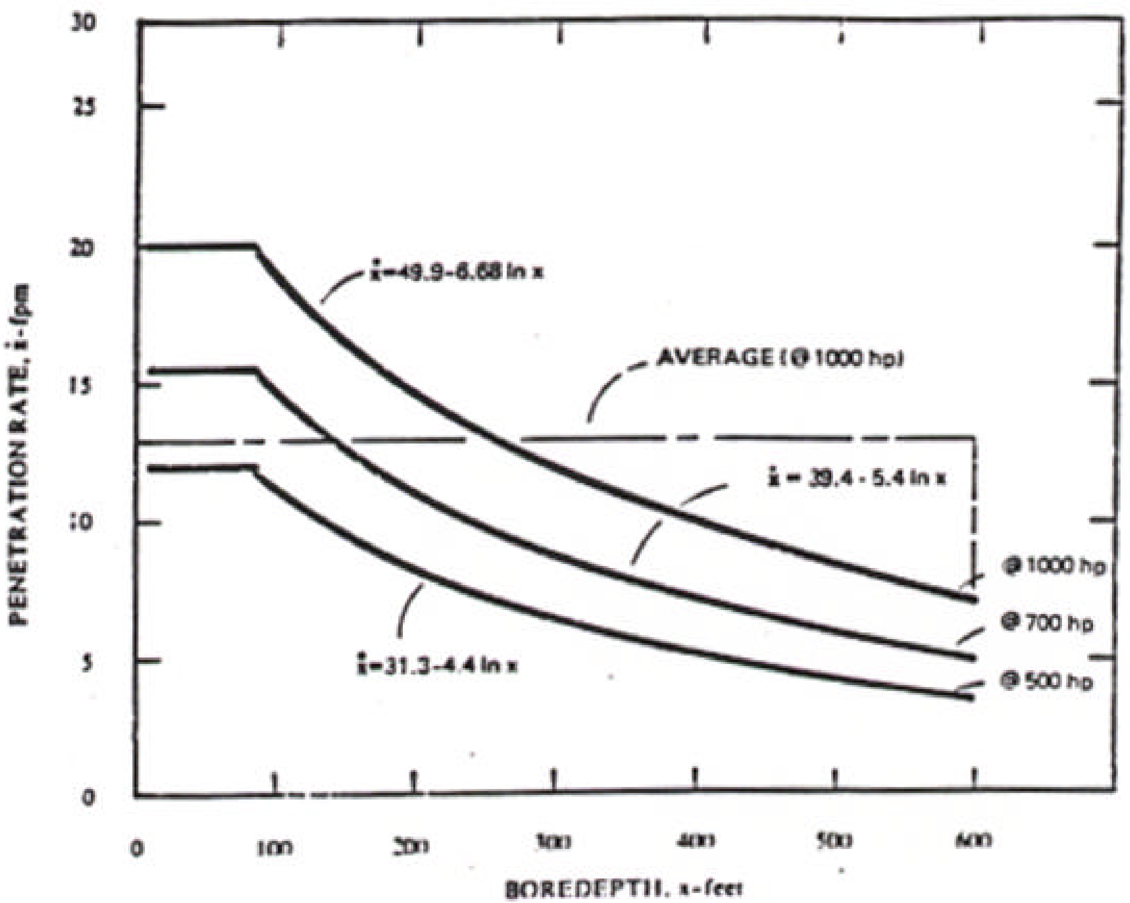


Figure 3.6 Penetration rate profile for 24" dual head auger (Treuhaft, 1981)

3.1.5 Ownership and Operating Cost

Ownership and operating costs for a double head 24 in. auger, using the straight line method of depreciation with no salvage value, is estimated below [Topuz et. al., 1997]

<u>Ownership Cost</u>	
Auger Machine	2,000,000
Auger Flights	420,000
Total Cost	2,420,000
<u>Average Investment</u>	
Auger (7 yr.) $ 2,000,000 @ 0.57	1,140,000
Auger Flights (3 yr.) $420,000 @ 0.67	280,000
Total Cost	1,420,000
<u>Annual Fixed Cost</u>	
Depreciation (7 and 3 yr.)	425,000
Interest etc. (16% @ 1,420,000)	227,200
Total Fixed Cost	653,000
<u>Annual Operating Cost (4000 hr./yr.)</u>	
Labor (2 shift @ 2 men @ $50,000)	200,000
Maint., repairs and supplies (25% dep.)	106,500
Power (750 kW @65% @ 4,000 @ 0.06)	117,000
Total operating cost	423,500

The annual fixed and operating costs amount to a sum of $1,076,500.

3.2 Self Advancing Miner

The Self Advancing Miner has been proposed for production from thin seams. Before conceptualizing this system, however, the shortcomings of the existing thin seam miners must be considered. The major problems with existing machinery are listed below.

- Lack of accurate coal/rock interface sensing device
- Lack of control over steering

- Inadequate coal transportation from the stalls to meet production potential

The self advancing miner was conceived to address these shortcomings

3.2.1 General Description

The self advancing miner is a completely integrated Mining System with two distinct components: a self-propelled cutter and a flexible and self propelled conveyor train. Though it uses the same panel entries as used in any conventional coal mine, it cuts coal at an angle of approximately thirty to forty degrees to the main entries. The panel layout of the system is shown in Figure 3.7. This enables the shearer and the conveyor train to negotiate the turns without requiring a large clearance. The self propelled cutter is a shearing drum with size commensurate with the seam thickness. It rides on a caterpillar which is powered by a flame proof electric motor. The shearing drum is powered separately by flame proof electric motors. The caterpillar provides the necessary forward thrust during the cutting cycle. The gathering arm at the bottom of the machine collects broken coal and loads it onto the chain conveyor. The chain conveyor discharges its load to the train of chain conveyors attached to the cutting machine. The chain conveyor is composed of 12 ft (3.7 m) units, attached back to back. Each conveyor unit is powered by a separate flame proof motor. The conceptual diagram of the Self Advancing Miner is shown in Figure 3.8. Stalls must be ventilated in order to prevent any chance of methane explosion occurring due to sparks generated during cutting of coal or electrical. To ventilate the stalls, rigid ducts are used. The ducts are attached to the side of the conveyor train and pulled in as the stall progresses. Centrifugal fans can be used to generate the necessary air pressure and quantity. Lateral steering will be achieved by providing differential rotation of the caterpillar. A built in coal/rock interface sensing device will continuously monitor the direction of cutting. Vertical steering will be provided by two hydraulic rams which will position the cutter above and below the horizon.

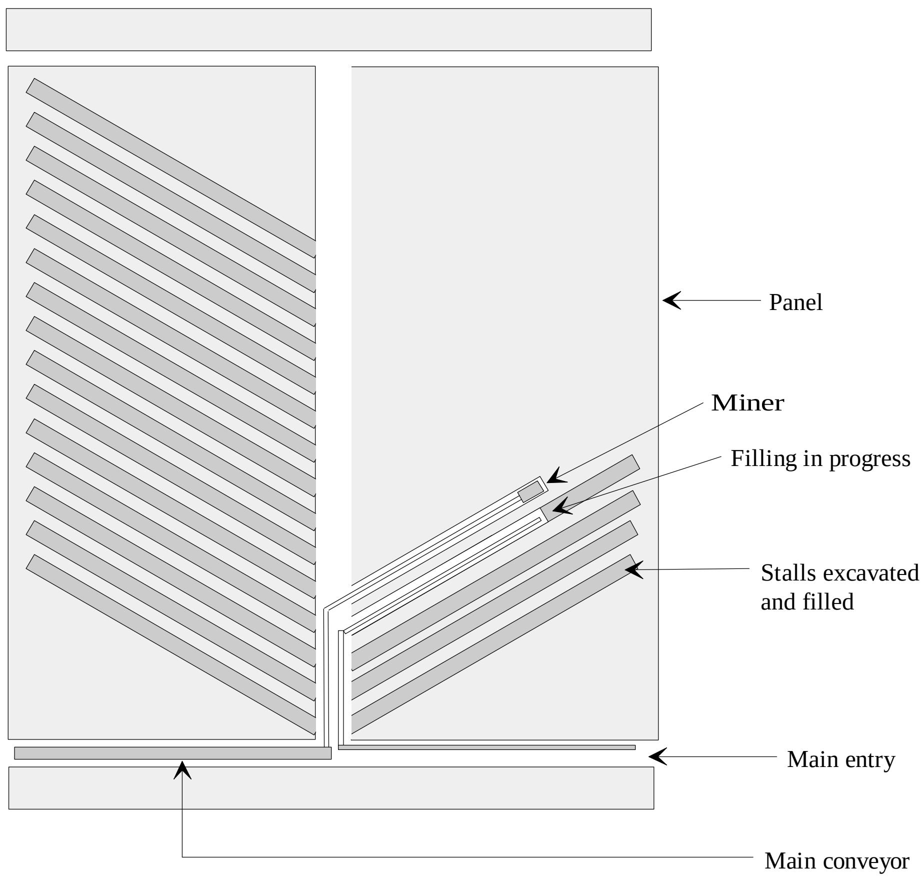


Figure 3.7 Face layout required by Self Advancing Miner

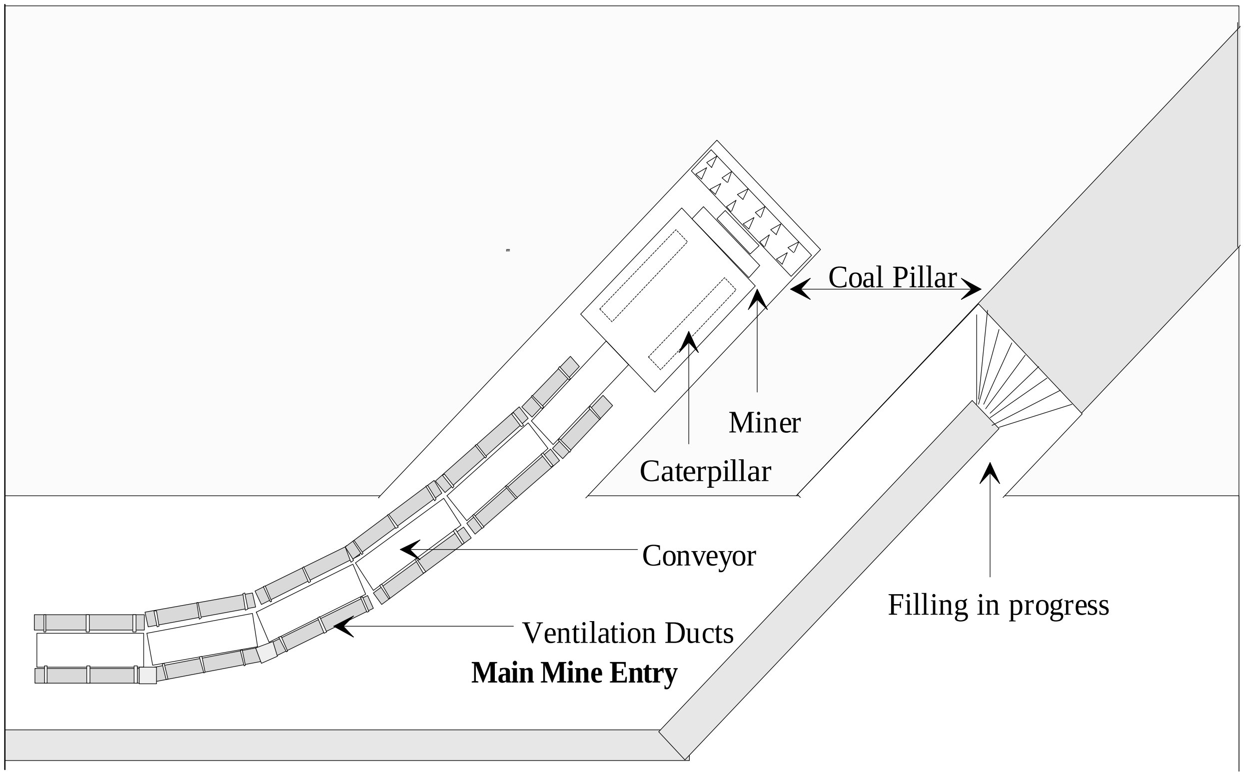


Figure 3.8 Schematic diagram of the Self Advancing Miner (not to scale)

3.2.2 Production Potential

Several parameters affecting the performance of Self Advancing Miner were varied to study their effect on its production potential. A program was developed in Visual Basic 3.0 to perform the sensitivity analysis. Several assumptions for the sensitivity analyses are listed below.

- Shift time 8:00 hr
- Production time 6:00 hr
- Width of cut 7.00 ft
- Withdrawal speed 24 ft/min

Parameters that were varied to estimate their sensitivity are:

1) Seam Thickness; 2) Cutting speed; 3) Change over time; 4) Depth of cut.

The output of the sensitivity analyses are listed in Tables 3.3 through 3.8, and their corresponding plots are shown in Figures 3.9 through 3.14 respectively.

Table 3.3 Production (t/shift) Vs. cutting speed across different seam thickness (depth of cut = 600 ft; location change time = 15 min)

Seam Thickness (ft)	Cutting Speed (ft/min)				
	3.0	6.0	9.0	12.0	15.0
2.0	516.90	886.56	1164.00	1379.88	1552.74
2.5	646.14	1108.20	1455.00	1724.88	1940.88
3.0	775.38	1329.84	1746.00	2069.82	2329.08
3.5	904.62	1551.48	2037.00	2414.82	2717.22
4.0	1033.86	1773.12	2328.00	2759.82	3105.42

Table 3.4 Production (t/shift) Vs. cutting speed across different depth of cut (seam thickness = 3.0 ft; location change time = 15 min)

Depth of cut (ft)	Cutting Speed (ft/min)				
	3.0	6.0	9.0	12.0	15.0
300	730.20	1202.16	1532.34	1776.30	1963.86
600	775.38	1329.84	1746.00	2069.82	2329.08
900	791.70	1378.62	1831.08	2190.54	2482.98
1200	800.16	1404.36	1876.80	2256.30	2567.82

Table 3.5 Production (t/hr) Vs. cutting speed across different location change time (seam thickness = 3.0 ft; depth of cut = 600 ft)

Location change (min)	Cutting Speed (ft/min)				
	3.0	6.0	9.0	12.0	15.0
5	808.74	1431.12	1924.86	2326.14	2658.72
10	791.70	1378.62	1831.08	2190.54	2482.98
15	775.38	1329.84	1746.00	2069.82	2329.08
20	759.72	1284.36	1668.42	1961.76	2193.12
25	744.66	1241.88	1601.88	1864.44	2072.16

Table 3.6 Production (t/shift) Vs. seam thickness across different cutting speed (depth of cut = 600 ft; location change time = 15 min)

Cutting Speed (ft/min)	Seam Thickness (ft)				
	2.0	2.5	3.0	3.5	4.0
3	516.90	646.14	775.38	904.62	1033.86
6	886.56	1108.20	1329.84	1551.48	1773.12
9	1164.00	1455.00	1746.00	2037.00	2328.00
12	1379.88	1724.88	2069.82	2414.82	2759.82
15	1552.74	1940.88	2329.08	2717.22	3105.42

Table 3.7 Production (t/shift) Vs. Seam Thickness across different depth of cut (cutting speed = 9 ft/min ft; location change time = 15 min)

Depth of cut (ft)	Seam Thickness (ft)				
	2.0	2.5	3.0	3.5	4.0
300	1021.56	1276.98	1532.34	1787.76	2043.12
600	1164.00	1455.00	1746.00	2037.00	2328.00
900	1220.70	1525.92	1831.08	2136.24	2441.46
1200	1251.18	1564.02	1876.80	2189.58	2502.42

Table 3.8 Production (t/shift) Vs. seam thickness across different location change time (cutting speed = 9 ft/min ft; depth of cut = 600 ft)

Location change (min)	Seam Thickness (ft)				
	2.0	2.5	3.0	3.5	4.0
5	1283.28	1604.10	1924.86	2245.68	2566.50
10	1220.70	1525.92	1831.08	2136.24	2441.46
15	1164.00	1455.00	1746.00	2037.00	2328.00
20	1112.28	1390.38	1668.42	1946.52	2224.62
25	1065.00	1331.28	1597.50	1863.78	2130.00

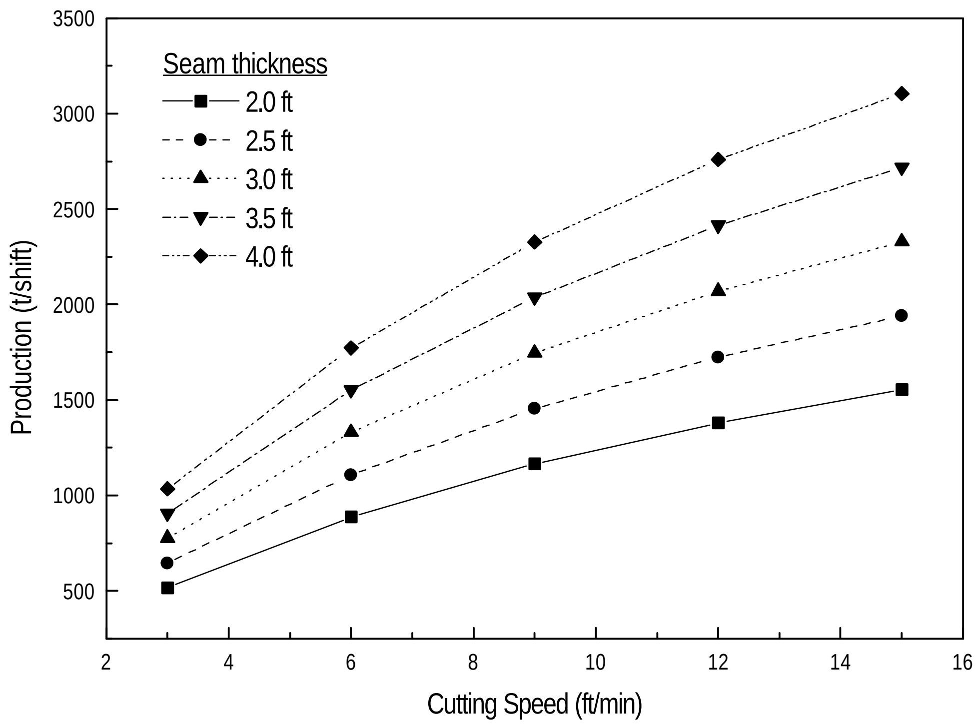


Figure 3.9 Relationship between production and cutting speed

(for different seam thickness)

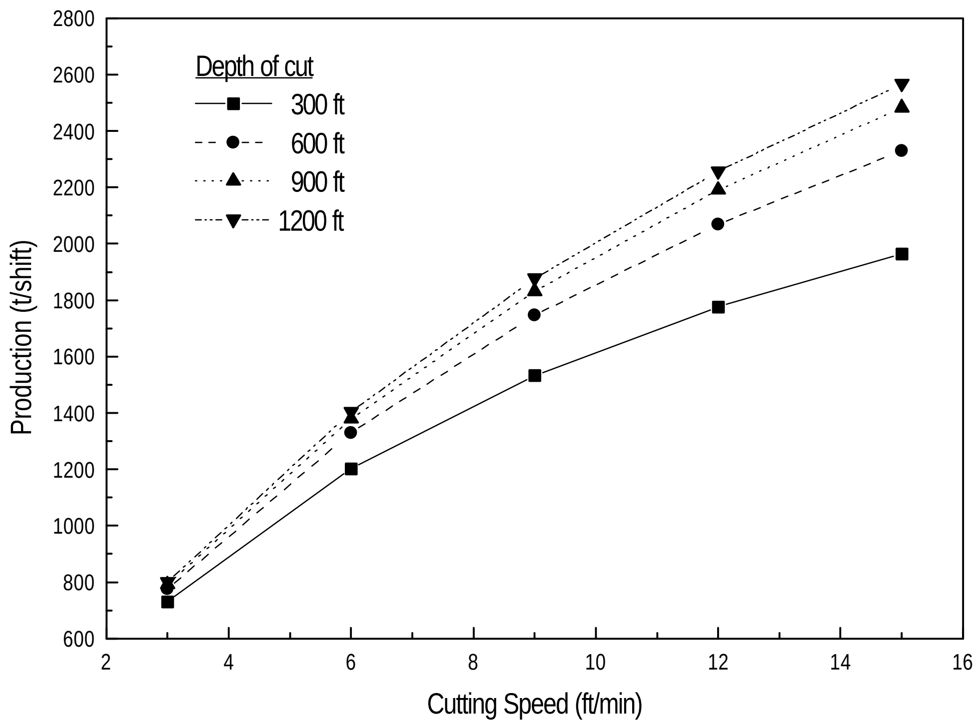


Figure 3.10 Relationship between production and cutting speed

(for different depth of cut)

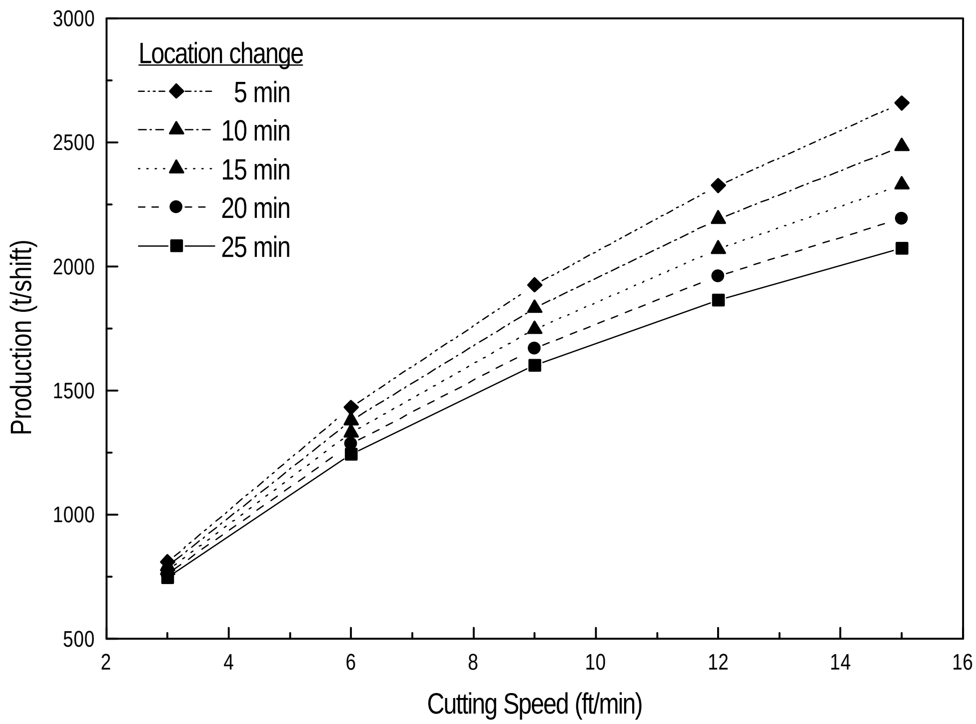


Figure 3.11 Relationship between production and cutting speed

(for different location change time)

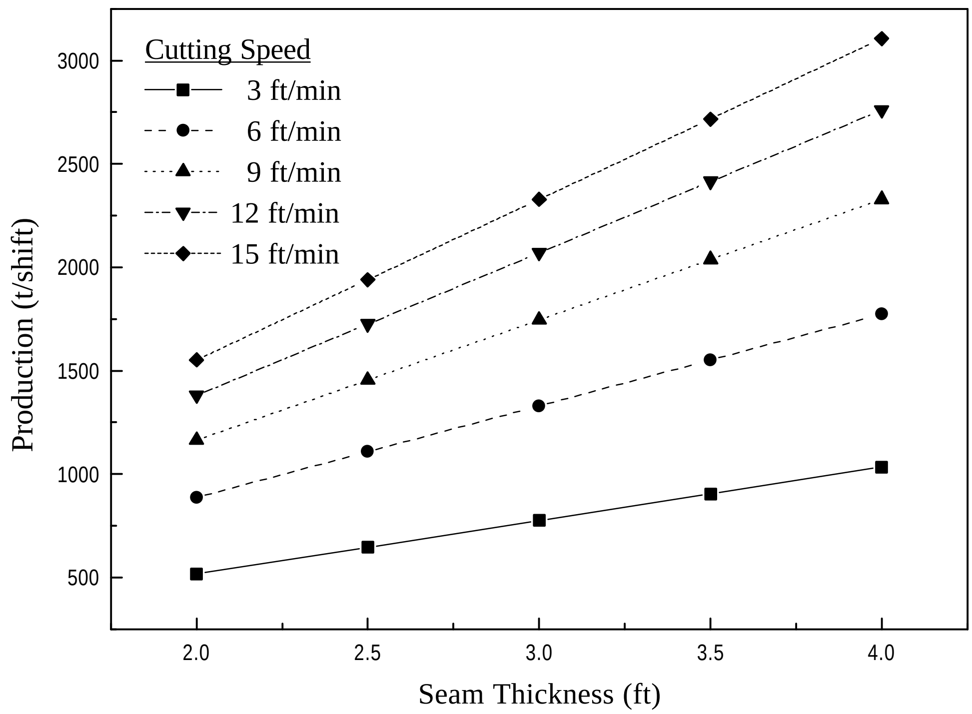


Figure 3.12 Relationship between production and seam thickness (for different cutting speed)

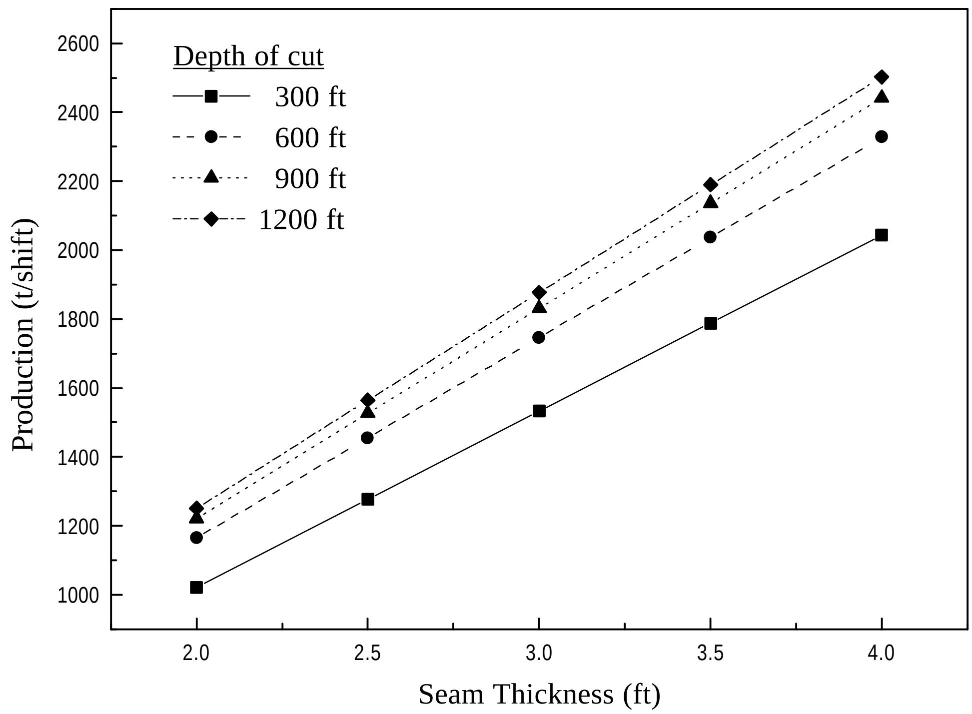


Figure 3.13 Relationship between production and seam thickness
(for different depth of cut)

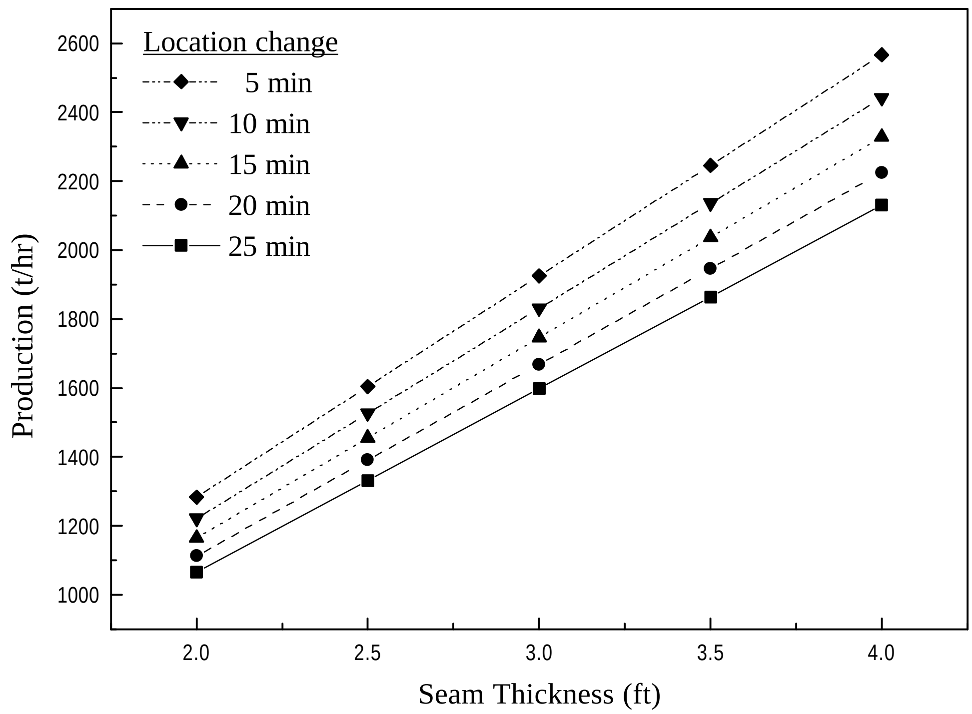


Figure 3.14 Relationship between production and seam thickness (for different location change time)

3.2.3 Ownership and Operating Cost

The ownership and operating cost of the self advancing miner, using the straight line depreciation method with no salvage value, is estimated below [Topuz et. al., 1997]:

<u>Ownership Cost</u>	
Miner (2 x 7 x 12 ft)	$2,000,000
Conveyor (12 ft @ 50 units)	$1,500,000
Total Cost	$3,500,000
<u>Average Investment</u>	
Miner & Conv. (7 yr.)$3,500,000 @ 0.57	$1,995,000
<u>Annual Fixed Cost</u>	
Depreciation, (7 yr.)	$500,000
Interest, etc. (16 % @ $1,995,000)	$319,200
Total Annual Fixed Cost	$819,200
<u>Annual Operating Cost</u>	
Labor (2 shift @ 3 men @ $50,000 yr.)	$300,000
Maint., repairs and supplies (50 % depre.)	$250,000
Power (1,750 kW @ 60 % @ 4000 @ $0.06)	$252,000
Total Annual Operating Cost	$802,000

The sum of Total Annual Fixed and Operating costs is $1,621,000.

3.3 Recovery and Cost Comparison

The maximum achievable recovery, under ideal conditions, for the self advancing miner and dual head auger is shown in Figure 3.15. Cost figures for the 24 in. dual head auger and self advancing miner are shown in Table 3.9. The following assumptions were made for calculations.

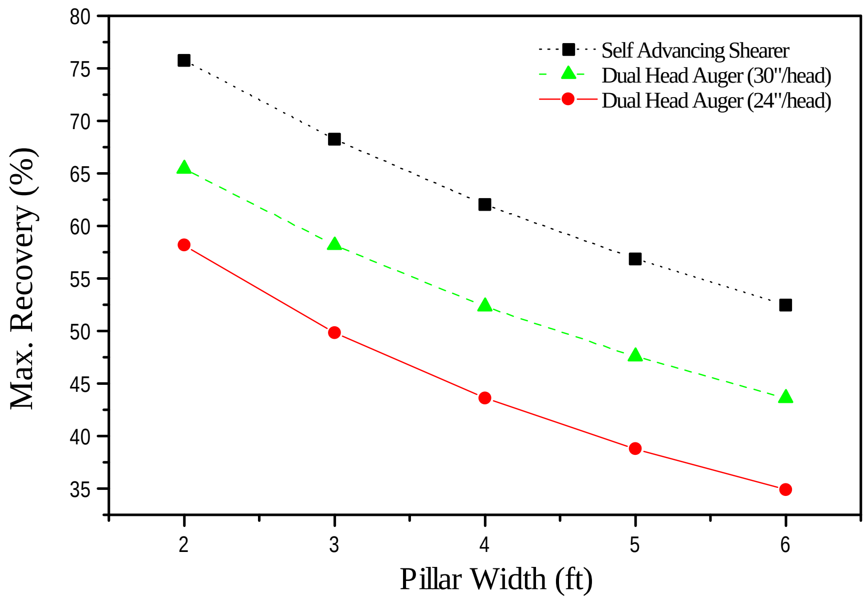


Figure 3.15 Relationship between % recovery and pillar width

- All production figures assume 2 shift / day and 250 days / year.
- All cost figures represent the cost of face operation only. They do not include the cost of development, ventilation, backfilling, etc.
- Auger figures are modified and updated from Skelly and Loy, 1981. Production is based on an average penetration rate of 10 ft/min (3.05 m/min).
- Self Advancing Miner figures assume 6 ft/min (1.8 m/min) penetration rate.
- Production of C.M. is based on CONSIM (7-men section) [Topuz et. al., 1989]

Table 3.9 Cost comparison between different system

Cost Item	24" Auger	Self Advancing Miner	C.M. (3 ft.)
Capital Cost	2,420,000	2,200,000	2,200,000
Avg. Investment	1,420,000	1,254,000	1,257,140
Annual Fixed Cost	653,000	516,000	520,000
Annual Op. Cost	423,500	709,500	1,180,000
Total Annual Cost	1,076,500	1,225,000	1,700,000
Production (t/shift)	390	880	600
Annual Production	195,000	440,000	300,000
Production Cost ($/t)	5.52	2.78	5.67

Apart from the above analysis, air requirement for the whole system was evaluated. To ventilate the stalls, rigid ducts will be used. The ducts will be attached on the side of the conveyor train and pulled in as the stall progresses (Figure 3.8). A centrifugal fan will be used to generate the necessary pressure and quantity. The maximum pressure drop in a 600 ft stall having a seam thickness of 2 ft is listed along with other pertinent assumptions.

- Width of cut : 7 ft (2.13 m)
- Duct diameter : 1.0 ft (0.33 m)
- Amount of air : 3000 cfm

Considering two situations; 1) one rigid duct, and 2) two rigid duct. The duct diameter used in the calculation is one foot. The logic behind using 3000 cfm is due to the fact that this quantity is most widely used in ventilating faces where continuous miners work.

Pressure drop = $H_v + H_f$

$H_v = rv^2/2$ $\qquad$ H_f = 2687.42 Pa (from Peabody ABC Corp)

$= 1.2 * 0.073/2$

$= 226.98$ Pa

Therefore total pressure drop = 2914.40 Pa or 2.9 kPa. If however two rigid duct is used pressure drop per duct is 2914.4/4 = 728.6 Pa. This is a significant improvement and must be weighed against the extra investment that is required for the extra ducts.

4. Coal/Rock Boundary and Web Thickness Sensing Techniques

The coal/rock interface detection is a key issue that needs to be addressed for an efficient application of remote mining in thin seams. Since returns from these seams are much less as compared to any thick seam mined by longwall or continuous mining methods, there is very limited room for any kind of quality variation in the output from mining these seams. Mining anything other than coal can lead to a reduction in productivity and consequently loss of profit due to contamination of coal with roof or floor material. Cutting of harder roof material may also cause slower advance and rapid bit failures which may further burden the profitability of the mine. As a result a technique has to be developed, be it crude or sophisticated, for detecting whether the remote miner is within the coal seam or cutting adjacent strata. These are discussed under the coal/rock interface detection techniques. Apart from detecting the interface, there is a need for maintaining proper direction of cutting. This will help maintain the integrity of the openings by keeping the pillar width between them constant. There are several techniques now available to detect the pillar width and are discussed under the web thickness sensing techniques.

4.1 Coal/Rock boundary sensing techniques

From the very beginning, the mining industry has been plagued by this problem. The developers of Collins Miner [Lansdown et. al., 1963] faced the problem of vertical position indication. They had to resort to the crude method of visual indication of vertical position. The state of the art at that time was not developed enough to provide them with a better and more efficient method of interface detection. However, currently various new techniques have either been developed or are in a prototype stage for this purpose. Many experimental applications of these techniques have shown that they work well, with certain limitations associated with each of them. More than twenty basic type of Coal-Rock Interface Detection (CID) concepts have been investigated in the US and abroad beginning in the early 1960's. Each concept showed limited success under site specific geological conditions. This chapter lists some of the most common and efficient methods of coal rock interface detection techniques available today along with their application. These techniques have been shown to have great potential on experimental basis, but their reliability should be tested on field and under continuous performance over a period of time.

4.1.1 Nucleonic Sensing Techniques

Nucleonic sensing method was used for guidance control in the Collins Miner [Thomas, 1963]. This method of sensing involves the use of gamma ray to be reflected from adjacent surface and determine the type of surface. For the purpose of generating gamma rays, a radio active source, (thulium 170) was used. A portion of this is scattered back into a radiation detector consisting of geiger tubes, whose magnitude depends upon absorption characteristics of the strata. Much less radiation is received for rock strata than for coal and if varying thickness of coal are interposed in between the detectors and adjacent strata, the output can be calibrated to read inches of coal. Pulses of current are generated when radiation enters the detectors. This current is locally amplified and then transmitted back to the operator's cabin, where it is read as inches of coal. Some initial difficulties were met by the design team during field trial at New Lount Colliery [Lansdown et. al., 1963]. The main problem was to maintain contact of the probe with the floor. Due to accumulation of coal dust on floor the sensor often produced erroneous readings. The alternative of sensing the roof interface instead of floor was also considered. From a mining aspect, this offers a better prospect of leaving the roof undamaged, since in thin seams, with little margin for maneuver, cutting floor may be much more acceptable than cutting roof.

Though this technique has potential for detecting the interface, its use however may pose a health hazard for the personnel from harmful levels of radiation. In case of Collins Miner, the source was armed and disarmed automatically as the first section of the push rods were inserted at the end of the machine. As the machine had to advance some 5 ft before this can happen, the probe was well inside the stall when the source was armed. An alarm circuit was also installed to give warning of any failure in the automatic disarming mechanism.

4.1.2 Natural Gamma Radiation (NGR)

Natural gamma radiation works on the fact that strata adjacent to coal is considered to have a higher levels of naturally occurring radioactivity. They contain minute quantities of radioactive potassium, uranium and thorium which radiate gamma rays. These gamma rays are attenuated by coal interposed between the detectors and their natural occurring sources in adjoining strata. As a result, thickness of coal can be determined by the amount of NGR attenuation. It has been found experimentally that NGR count decreases exponentially as a function of coal thickness. To date, only two Coal Interface Detection (CID) sensors, both based on natural gamma radiation principle have been commercially available, namely Salford Electrical Instruments (SEI) 801 Natural Gamma Coal Thickness Indication System and the American Mining Electronics (AME) Model 1008 Coal Thickness Measuring System [Mowrey, 1991]. Both these devices are capable of providing coal thickness measurement from 2 to 50 cm. This is adequate for most mining situations in which a layer of coal must be left below the roof or above the floor.

The first NGR based coal interface detection technique was developed by National Coal Board (England). It was found that British shale, which formed the strata adjoining coal in most cases have similar gamma radiation, and that coal acts as an attenuator for this radiation. The first commercial version of NGR detector was developed in 1980 [Mowrey, 1991] consisting of thallium-doped cesium iodide crystals and weighed about 114 lb. The output pulses were counted which determined the coal thickness. The equipment had to be calibrated on site and upon calibration could read coal thickness in 0.8 inch increments.

In order to verify whether NGR technique is a viable option for measuring coal rock interface in the US, the Bureau of Mines has been collecting geological and mining data and representative coal and rock samples, (roof and floor) from number of major coal mines [Mowrey, 1991]. Data from more than 300 mines have been collected and analyzed. The result indicate that 88% of mines have either shale, draw slate, or soapstone as immediate roof. In 90% of the mines, the immediate floor strata consists of either fireclay, shale, or draw slate. These findings indicate that most of the immediate roof and floor strata are shaly formations and therefore, should have higher levels of naturally occurring gamma radiation than coal. This in turn indicates that the NGR technique should be suitable in many U.S. coal seams [Mowrey, 1991].

4.1.3 Optical Reflectance Method

Optical reflectance method was tested during the development of the highwall miner in the early 80's [Treuhaft, 1981] for guiding auger heads. The principle behind this sensing technique is to measure the relative reflectance of coal against other non coal surfaces like shale, sandstone, etc. and accordingly determine whether the miner is cutting coal or roof material. The degree of contrast is even greater in the infrared region of the spectrum. Extensive experimentation was done on several coal samples from eastern, central and western coal in the US and in each case good contrast was seen at the interface [Treuhaft, 1981]. The major down hole component of the optical relative reflectance sensing sub system consist of a circular kerf saw and the optical scanning sensor. The major surface component is composed of a visual display unit. The display unit is designed to imprint varying levels of gray to black on chart paper, thereby denoting the transition zone across the boundary. As the cutterhead advances into the coal seam, the circular saw, operating along the cutterhead centerline cuts a kerf in the roof. The optical scanner then continuously monitors the relative reflectance of the freshly cut wall reproducing several consecutive reflectivity scans on the operators console.

The major problem in employing the scanner system is keeping the sensors clean. Certainly if the sensors are to operate properly, their signals must have an unobstructed path to and from the kerf path. Partial blockage of the path will greatly decrease the system's sensitivity. In order to keep the sensors clean compressed air was continuously directed onto the surface by a small electrically driven compressor. Another potential problem that could arise when employing this technique concerns the requirement for the kerf saw to cut a sufficiently smooth surface so that aberration in surface geometry will not produce erroneous indication for the sensors. In addition the saw must be able to cut material that are much harder and more abrasive than coal and shale. To avoid any breakage of the saw, carbide tipped teeth were used and tested on variety of coal roof interfaces which performed satisfactorily [Treuhaft, 1981].

4.1.4 Vibration

Vibration based interface detectors work on the principle of identifying the various mechanical, seismic and acoustic vibrations produced by the mining machine. Each strata has a different vibration signature that is directly related to the geologic material being cut, the type of mining machine being used, and the condition of the bits. The signature of vibrations can provide the operator with the information of whether the machine is cutting soft coal or hard roof material. For a given set of conditions, a mining machine vibrates differently and generates different seismic waves while cutting different strata and then use adaptive signal discrimination to classify these vibrations. There are three basic type of vibration based Coal Interface Detectors (CID) concepts that Bureau of Mines investigated: machine vibration, in seam seismic, and acoustic [Mowrey, 1991]. Figure 4.1 shows a generalized overview of these three methods.

The machine vibration technique involves in mounting one or more vibration sensors on the mining machine, preferably as close to the cutter bits as possible. The vibrations due to cutting are picked up by these sensors and monitored to distinguish any change in the signature. Any change in the vibration signature would indicate the machine having left the seam. To improve detection, the sensors should be mounted away from extraneous machine noises. The advantage of using machine vibration technique as CID is that the sensors are mounted at a fixed distance from the cutter bits, and does not have to be mounted and remounted after mining certain distance. However, they are more likely to pick up machine noise and hence have low sensitivity.

The in-seam seismic technique involves mounting sensors on the roof , coal seam, or floor at one or more locations near mining operation. This is shown in Figure 4.1. These sensors are generally mounted on roof bolts and close to mining operation. The advantages associated with in-seam monitors is that the cables and personnel are out of the way of the machine. This in turn help in reducing machine noise and make the system more sensitive. However these technique has a very limited applicability in thin seam mining, as it will be impossible to send personnel in stalls to install these sensors [Mowrey, 1991].

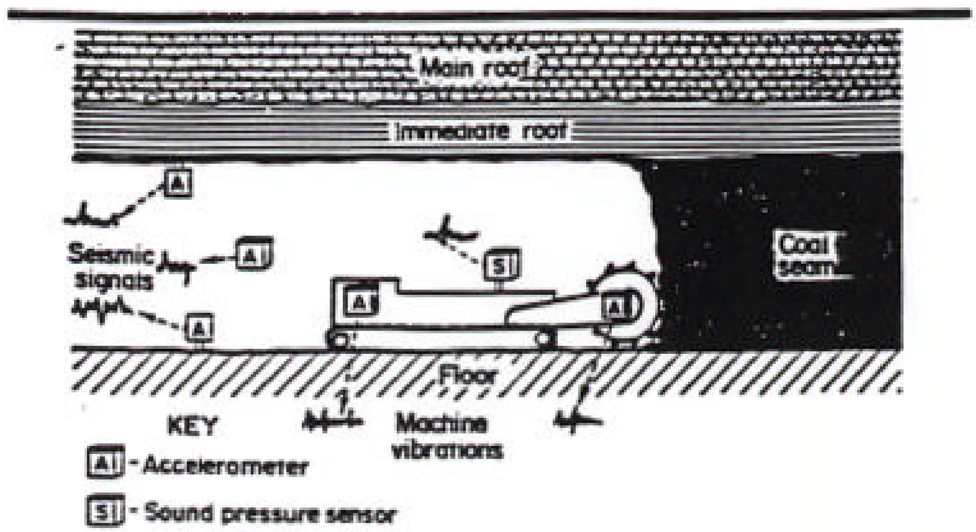


Figure 4.1 Schematic diagram of vibration based CIDs (Mowrey, 1991)

4.1.5 Infrared based Coal Interface Detector

Infrared (IR) based detectors work on the basis of sensing the change in cutter bit temperature when cutting material that is harder than coal like roof or floor material. IR camera is used to measure the surface temperature of coal and rock in the vicinity of the cutter bits. For this system to be successful, a definite contrast in mechanical properties must exist between coal and the adjacent strata notably in hardness. A particular advantage of IR over ordinary visible light system is that IR radiation can penetrate through dust and water sprays without much attenuation. Due to this reason, the sensitivity of this system is much higher than an ordinary video camera, which has to be constantly cleaned in order to function properly.

Development of technology over the last 10 years has led to the development of IR cameras which can detect temperature difference of 0.1^0 C. An IR camera works on the principle that an area of interest is scanned for thermal radiation in very much the same way as an optical camera does. The monitor displays the thermal picture in which the intensity of the objects in the scene represent their approximate temperature.

To investigate the feasibility of low cost IR imaging for detecting coal rock interface, USBM conducted a test using IR camera named Pyroviewer Model 5400 [Mowrey et. al., 1995]. This camera is very small, light weight, and operates on 12 V dc supply. This camera is capable of measuring temperatures from -30^{0}C to 1,100^{0}C with a resolution of 0.2^{0}C at 25^{0}C. It should be noted that this camera is sensitive to mechanical vibration and any impact may produce unwanted noise on the data. Thus, if the camera is held stationary, it will only detect objects in relative motion such as rotating bits on the drum. The Pyroviewer video images show up on the display unit as circular in shape with hotter objects as bright or white and cooler objects as dark or black. Moreover, this camera measures thermal radiation in the 8 to 14 micron range. At this wavelength, the atmospheric attenuation of IR is least. This wavelength is also ideal for measuring temperatures which are around the ambient.

Evaluation of Pyroviewer was done on a highwall mining site. It was mounted on the rear end of the miner and was approximately 20 ft away from the coal face. The seam was approximately 14 ft thick of which 12.5 ft of coal was normally extracted. The remaining 1.5 ft of coal was left as a support on the roof. For testing purpose, the miner was deployed in a previously mined entry. The cutter drum was allowed to cut the roof coal and subsequently the overburden at some spots. The Pyroviewer was able to detect the change in bit temperature when the bits started cutting the roof material. The data collected was of marginal quality due to the inability of the camera to focus properly. However this test showed that the technique can serve the purpose of interface detection. To improve upon the quality of results, a better and cheaper IR sensitive cameras has to be developed [Mowrey, 1995].

4.2 Web Thickness Sensing Techniques

Various web thickness measuring techniques are listed in Table 4.1 along with the necessary references.

Table 4.1. Web thickness sensing techniques and their references

Web thickness sensing techniques	References
Probe drill	Treuhaft, M. B., ’81, DEAC 01-76ET12213
Radar based system	Mowrey, G. L., et. al., ’95, Society of Mining Metallurgy and Exploration.
Acoustic sounding technique	Treuhaft, M. B., ’81, DEAC 01-76ET12213
Inertial navigation system	Schiffbauer, W.H., July ’97, Fourth International Symposium on Mine mechanization and Automation

All these are techniques are discussed in sections below.

4.2.1 Web Thickness measurement using Probe Drill

In order to measure the web thickness between the stalls, a probe drill can be used. Conceptually this method is very simple, as the web thickness is measured by how much the drill had to be extended before it punctured into the neighboring stall. By knowing the amount of extension, the web thickness can be measured. Implementation of this method would require the miner to be stopped from time to time for measurements. If deviations are not expected to be very high then occasional measurements would be adequate to maintain or confirm alignment. During the development of Highwall Miner [Treuhaft, 1981], this concept was implemented to measure web thickness. The unit consisted of hydraulically powered drill mounted horizontally perpendicular to the longitudinal axis of the augers. When required the operator would stop the miner and initiate the probe drills. The drill bit is made to advance through a lead screw system until it encounters the borehole walls. The change in pressure activates the revolution counter which counts the number of revolutions required to drill through the web. This count directly translates into web thickness. The revolution counter is automatically deactivated when the drill punctures the web into the adjacent borehole. The whole operation took only 40 seconds [Treuhaft, 1981] which will not significantly affect the production time. Hydraulic power to the probe drill was supplied from the pump attached to the left auger. Hence the left auger has to rotate during this operation.

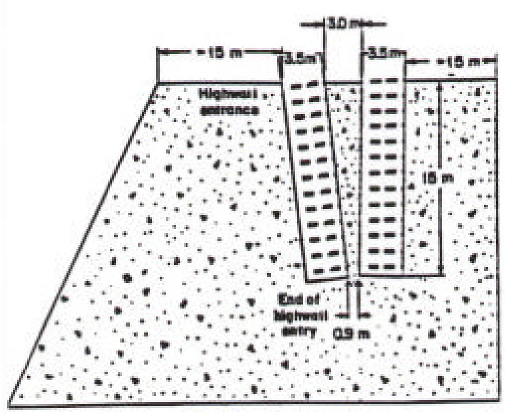
4.2.2 Radar Based Rib Thickness Measurement Technique

The idea that Radar can be used to monitor coal rib thickness was introduced by Cook (1975). He found that coal is relatively transparent to radar waves. The success of a radar system at a given location primarily depends upon two electrical parameters of the material being tested, namely electric conductivity and dielectric constant. Electrical conductivity is the ability of the material to conduct electrical current. It has been observed that higher the electrical conductivity the lesser is the radar penetration depth. Electrical conductivity vary greatly for different geological materials and is primarily governed by the water content, amount of dissolved salt and on the density and temperature of the material. Dielectric constant measures the capacity of a material to store electrical charge when an electrical field is applied on them. Dielectric constant for coal varies between 4 to 5 and that of air is 81. It is known that radar waves are reflected when there is a large change in dielectric constant in the medium. Reflection is even greater if the surface is planar and perpendicular to the direction of radar. Hence coal air boundary serves as an ideal interface for such change along which radar waves can be reflected back. It may be noted here that dirt bands and partings in coal seams may greatly reduce the efficiency of the system. Another consideration is the existence of air gap between the antenna and the formation. Ideally the antenna should be placed as closely as possible to the formation to minimize energy loss. It has been found that the spacing should be no more than 4 in. (10 cm). By measuring the time taken for the radar wave to reflect back, one can determine the web thickness.

Geophysical Survey Systems Inc. produces a commercial system using ground probing radar (GPR) to measure web thickness. This system can produce instant continuous-profile radar records that indicate the location and depth of strata and objects buried upto 20 ft depending upon the frequency used [Mowrey et. al., 1995]. Data is displayed in wiggle plot format. The commercial system named SIR-10 was used by USBM to study the feasibility of using such systems in determining rib thickness and to obtain information regarding the depth of penetration and effect of air gap on GPR response [Mowrey et. al., 1995]. For this purpose two highwall entries were driven the first one being straight and 50 ft deep and the second one slightly angled and also 50 ft deep. The angle was made such that rib thickness varied from 10 ft at the face to 3 ft at the end of the borehole as shown in Figure 4.2. The GPR unit was moved up and down the hole several times using 500 MHz signal. In the first experiment, the unit was placed directly against the borehole wall and in the subsequent experiments 4 in. and 2 in. away from the wall. Styrofoam pieces were placed between the wall and the antenna. In all the three cases the radar was able to see through the coal rib at the thickness ranging from 3 ft to 10 ft. Figure 4.3 shows the output obtained from the GPR unit as wiggle plots. This system however has one severe deficiency. The first hole has to be drilled without the help of any guidance. Once the first hole has been drilled, the rest of the holes can be drilled almost parallel with the help of this system. As a result if the first hole is not straight, the whole set of boreholes would deviate and may result in reduced extraction. For the GPR system to work properly, the following requirements are to be met:

- A continuous portion of the coal seam should be relatively clean with little or no partings or dirt or clay bands
- Rib thickness should be between 2 ft. to 10 ft.
- GPR antenna should be within 2 in. of the rib

The major advantages of the system is that drilling is not required to find out the rib thickness as is the case with probe drill. Measurement is obtained without any stoppage of production as the system is capable of continuously monitoring rib thickness [Mowrey et. al., 1995].



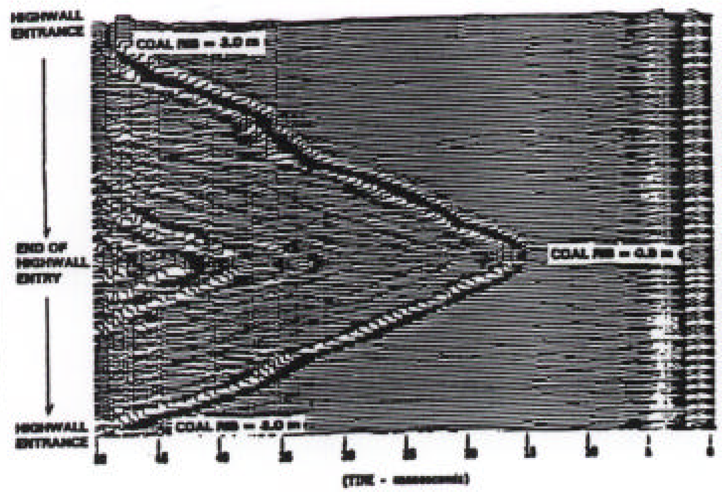


Figure 4.2 Testing site for GPR (Mowrey, 1995)

Figure 4.3 Wiggle line plot (Mowrey, 1995)

4.2.3 Acoustic Sensing Technique

Acoustic sounding technique for measuring the web thickness works on the principle of attenuation of sound wave by coal. Even though acoustic attenuation in coal is known to be very high, the application of this technique in the web thickness sensing should have some probability of success since the required direction of sound propagation is along the most favorable lamination. The coal-air interface at the adjacent bore hole should also provide stronger reflection than the coal-roof interface. Laboratory testing showed that acoustic signals in the frequency range below 1MHz can be made to propagate several inches through coal samples and that the reflected waves from the coal-air interface can be detected [Treuhaft, 1981]. For the system to work efficiently, contact must be between the ranging device with the coal surface must be maintained at all the time. This may cause some problem with smooth flow of coal. Special transducers would have to be developed to assure proper surface contact so as to couple the input and output energies. One possible solution is a retractable unit that can be periodically extended to the borehole wall in much the same way as a probe drill. However this system does not require that augers be kept moving during measurement like probe drill and hence can be used during flight change. However not much work has been done to develop the product and hence its exact potential cannot be established [Treuhaft, 1981].

4.2.4 Inertial Navigation System

The inertial navigation system has been tested out by National Health Institute for Occupational Safety and Health (NIOSH) and Pittsburgh Research Center (PRC). After investigating several different types of navigation system, the Honeywell Ring Laser Gyro Inertial Navigation System showed the most promise and hence selected for further study [Schiffbauer, 1995]. The system was tested out in a highwall mine in Wyoming. The system required three control points which were placed close to the southern edge of the pit approximately 600 ft. (180 m) apart in a straight line. The coordinates of the control points were determined from Wyoming state plane coordinate system. An automated transits were fixed on one of the reference point and a target was fixed to the top of the second. The transits were initialized and raw data collected to calculate the coordinate of the miner. After the evaluation of the initial position, the transits were put into remote mode, and it was set to search for the target placed on the miner. The horizontal angle, vertical angle and the slope distance obtained from the target on the miner generated accurate position of the miner.

All these techniques have shown promise in detecting the coal rock interface. They have been able to recognize the interface with success. However their applicability may not be universal and may not work under all circumstances. Therefore it is suggested that a combination of these systems be used and the data gathered from them be integrated to process the information. However, one cannot overlook the cost associated in such combining effort. It is concluded here that the necessary technology is available to perform the task of CID, but more effort is needed in order to make it ready for mining industry.

5. Backfilling Techniques

Just as an efficient interface detection would help increase the productivity of the thin seam mine, another important operation which would certainly improve extraction and productivity is backfilling operation. Although backfilling operation requires substantial capital investment, it can improve productivity by providing better strata control thereby increasing coal recovery. In thin seams this is of utmost importance as the survival and profitability of the mine depends upon high coal production and high extraction thereby increasing the mineable reserve substantially. However before discussing backfilling in detail, it must be added here that this operation may not always be needed. The strata may be such that it is self supportive or there is very low overburden pressure and hence does not require additional support to maintain the integrity of the openings. But one is unlikely to encounter such ideal conditions in future since the better and shallower seams in the southern Appalachia have all long been exhausted, leaving out those which were less favorable to mine.

Apart from serving the purpose of strata control, backfilling also help in improving environmental conditions related to mining and mineral processing. The whole operation, as mentioned earlier is an additional cost item in mining. However ground control, land use, and environmental advantage may make this operation an economical choice. In addition, urban and industrial waste which have very high disposal cost, may be selectively mixed with mining waste material which may further improve the economics of the underground disposal. This chapter discusses different filling techniques as well as fill material which can be used economically for filling purpose.

Along with the various advantages mentioned above, the major disadvantage of backfilling operation includes 1) addition of a new operation to the mining cycle, 2) possibility of reduced production rate caused by problems associated with backfilling. These disadvantages are more of traditional nature and must be dealt with if backfilling is to improve recovery and production in thin seam mining. Backfilling systems are characterized and identified by the mechanism used for transport and emplacement of fill material. The overall characteristics, adaptability and limitations of each system depend on the individual transportation and stowing elements of the system.

Backfilling systems can be classified as gravity stowing, hand stowing , mechanical stowing, pneumatic stowing and hydraulic stowing. Only the last two methods have been discussed here as the other methods are either technically and economically infeasible or lack of effectiveness under the given set of mining conditions. Hand stowing require the presence of personnel near the face for packing dirt, which is almost impossible due to very low seam height. Hence technically this system cannot be applied for filling the stalls with waste material. Applicability of gravity filling is also very limited especially in the southern Appalachia as most of the coal seams are relatively flat. Gravity filling requires high dipping coal seams so that fill material can roll into the void created by mining. Since most of the coal beds in US has dips less than 5 degrees, which make this method technically unsuitable. Mechanical filling has been left out mainly due to its lack of effectiveness. In this method waste material is introduced using some kind of mechanical device like conveyor belts, into voids created by mining. While mining thin seams, efficiency of support by filling must be very high in order to increase extraction. Mechanical filling causes waste material to be packed very loosely which results in weak filling thus defeating the purpose. As a result, mechanical filling has not been included in the study.

5.1 Backfilling Material

Apart from selecting the method of backfilling, another important aspect is using right the kind of fill material. The material should have the following characteristics:

- widely available
- affordable
- easy to handle
- does not pose any form of health or environmental hazard
- provides the desired strength
- reach the desired strength rapidly

Traditionally most widely used back fill material has been waste rock generated during development work and mill tailings from preparation plant. However, during thin seam mining, amount of development is kept at minimum for economic reasons. Hence, very little waste rock is available from the development phase to be used for backfilling. One has to look for other fill material to serve the purpose.

Waste materials to be disposed of underground may be classified according to physical properties such as dry or wet and particle size such as coarse or fine. Chemically they can be further be classified as reactive or non-reactive material. Reactive materials refer to those kind which may produce contaminants upon oxidation, weathering or other chemical process. So fill material should be carefully chosen, otherwise they can cause environmental damage and loss of profit due to law suits and damage claims. Results from recent laboratory tests show that a mixture of cleaning-plant tailings with suitable percentage of cementicious or cheap pozzolanic material can form a monolithic pack having adequate strength to meet the design strength [Zadeh et. al., 1987].

After considering several mixes [Hii et. al., 1990], the mix with following composition was found to have the desired properties in terms of curing time and strength.

Coal washery refuse	75.00 %
Portland cement	10.80 %
Calcium chloride	0.30 %
Water	13.90 %

The strength versus curing time for first 24 hr and for next 10 days are shown in Figure 5.1 and 5.2 respectively. Although the waste generated by coal preparation plants and coal burning power plants are the natural choice as fill material, they may not be presently available at mine site. An ongoing research investigates underground de-shaling of coal [Luttrell et. al., 1996]. This research may make fill material readily available underground. It is interesting to note that thin seam mining systems may not produce sufficient waste material to fill the voids due to the fact that thinner coal seams are generally much cleaner

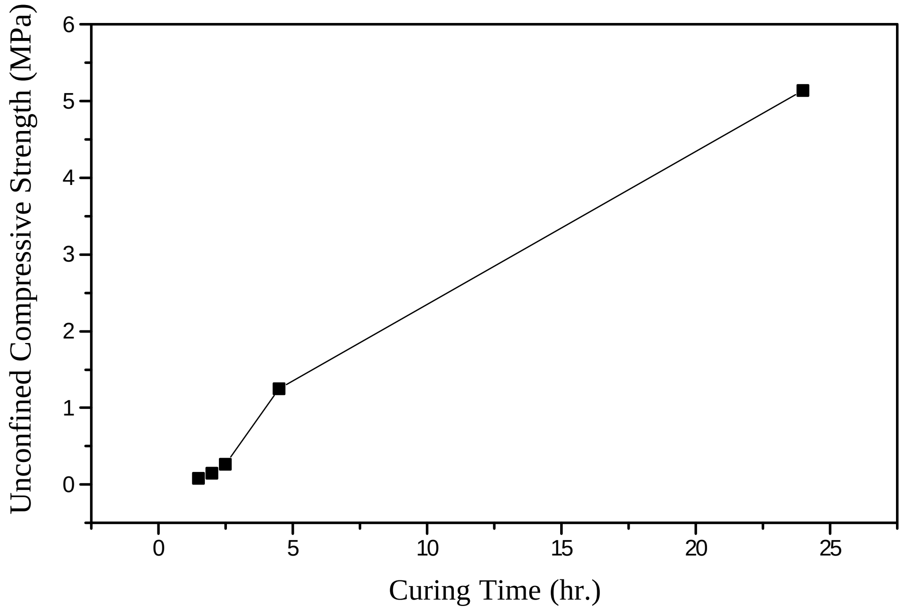


Figure 5.1 Compressive strength and curing time (Hii et. al., 1990)

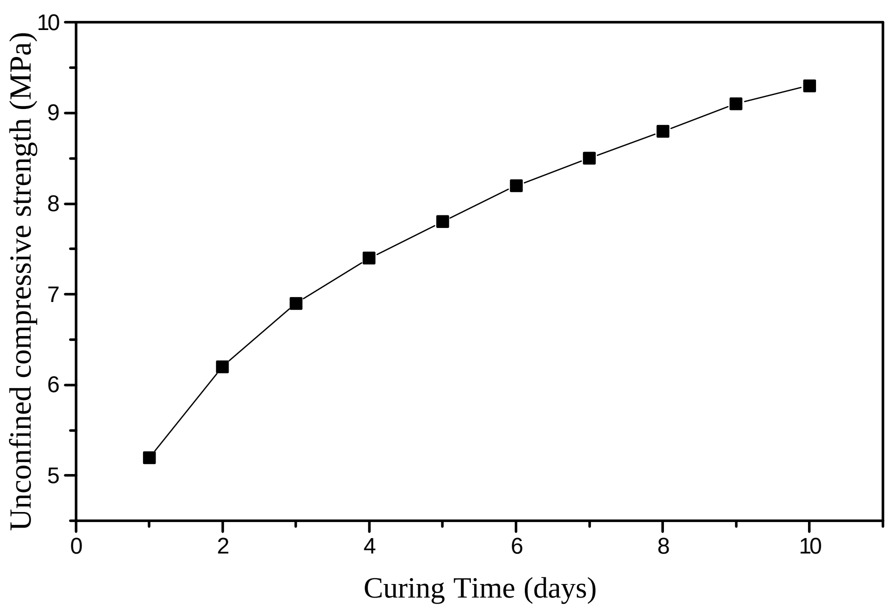


Figure 5.2 Compressive strength and curing time (Hii et. al., 1990)

and contains less partings. The situation may necessitate importation of waste material from external source [Topuz et. al., 1997]. Some possible candidate materials are shown in Table 5.1 which can serve as backfilling material.

Table 5.1 Material for underground backfilling of coal mines

Suitable - Zero level emission

- Refuse of coal cleaning plants
- Fly ash and bottom ash of power plants
- Glass and ceramic waste
- Demolition waste
- Asbestos containing waste

Limited suitability

- Fly ash, bottom ash and dust from waste incineration plants
- Waste from metallurgical plants and foundries
- Polluted slag

Waste excluded

- Waste deriving from plants and animals as well as processing plants
- Waste of chemical and synthetic plants
- Residential waste
- Radioactive waste

Source: Dartsh et. al., 1994

5.2 Backfilling Methods

Backfilling method involves the task of injecting waste material into the voids created by mining operation. Ideally, this operation should take place immediately after coal extraction. Since the mining systems discussed are all continuous in nature and has high production potential, the filling systems should also be of continuous nature. As a result, only pneumatic and hydraulic filling techniques are discussed. Characteristics of each backfilling system has been classified into three broad categories: technical, economical, and environmental. Table 5.2 provides the detail of the specific attributes considered within each category. Ratings have been assigned to each of these attributes based upon how they effect mining operation in general. The higher the rating, the more favorable the system is to the whole mining operation depending upon the given set of conditions prevailing in the mine. Table 5.3 lists the rating of pneumatic and hydraulic backfilling systems [Topuz et. al., 1997].

It may be noted here that volume of the void needed to be filled in case of self advancing miner working in a 2 ft thick seam is 2 ft x 7 ft x 600 ft = 8400 cu. ft. Fly ash has bulk density between 35 - 45 lb./cu. ft. If waste rock is mixed with fly ash the whole material will have bulk density higher than 45 lb./cu. ft. Assuming the bulk density of the fill material is 50 lb./cu. ft., total amount of material required to fill one stall is approximately 190 tons. Currently pneumatic stowers with capacity as high as 70t/hr are available. Such stowers have been successfully used by MICON for blind filling of auger holes. MICON completed the stowing of 63 auger holes about 80 ft long in only 11 working days achieving 80% compaction [Mason, 1986]. However, the main bottleneck of using pneumatic as well as hydraulic backfilling system in this case is the associated idle time. The individual pipes members have to be disconnected from the pneumatic or hydraulic pump to reduce the length as filling progresses. Pipes are available in 12 ft - 16 ft units. If 12 ft pipes are used, then total number of pipes required is 50. If 2 min is required to disconnect one pipe from the pump and reconnect again, then total time the system is idle is 49 x 2 = 98 min which is approximately 1 hr. 40 min. This can be done by using sleeves to join the pipes so that they can be attached and dismantled by simple twist. The pipes can be retracted using a winch. To work around this problem, rubber hoses may be used to transport fill material to the face. The cross cuts which are 20 feet wide provides ample room for the hose to turn 45^0 into the stall.

Table 5.2 Attribute ratings table for backfilling system

Attributes	Attribute Rating	
Technical	5	1
Mining method	Has no effect on the operation	Has negative effect on the operation
Availability of equipment	Available	Not Available
Convergence	Low (20%)	High (50%)
Backfilling rate	High	Low
Mine floor	Does not influence operation	Detrimental on operation
Mine roof	Does not influence operation	Detrimental on operation
Flexibility	Very flexible	Inflexible
Seam thickness	Does not effect operation	Detrimental on operation
Dip of seam	Does not influence operation	Detrimental on operation
Pre-treatment	Not needed	Screen, crush, blend, dewater
Economical		
Effect on productivity	No effect	Decreases heavily
Equipment cost	Minimal	Expensive
Cost of filling material	Minimal	Expensive
Availability of fill material	No effect on operation	Restricts operation
Availability of water	Available or not necessary	Necessary, not available
Labor requirement	<= 3 men	>= 10 men
Environmental		
Health hazards	None	Present
Effect on ground water	Less than surface disposal	More than surface disposal
Subsidence	None	Noticeable

Table 5.3 Ratings of selected stowing system

Attributes	Backfilling Techniques	
	Hydraulic	Pneumatic
Technical		
Mining Method	3	3
Availability of equipment	4	4
Backfilling rate	5	5
Convergence	5	4
Mine floor	3	5
Mine roof	4	4
Flexibility	3	4
Seam thickness	5	3
Dip of seam up/down	1/5	3/5
Pre-treatment	3	1
Economical		
Effect on productivity	4	3
Equipment cost	3	4
Cost of fill material	5	5
Availibility of fill mat.	1	1
Availibility of water	1	5
Labor reqd.	4	4
Environmental		
Health hazard		
· Stowing persnl.	4	2
· Prod. Persnl.	4	3
Effect on ground water	3	5
Surface subsidence	5	4

5.2.1 Pneumatic Filling

Pneumatic filling is in a way transporting bulk material through a pipeline by applying either negative or positive pressure air stream and subsequently dumping the material in voids. It can also be described as harnessing of air movement to accomplish work. Material is feed into the system using a pipe feeder system shown in Figure 5.3 [Burnett, 1995]. Air is supplied to the pneumatic pipe feeder at 100 psig and expands to pipeline pressure of 4 psig [Burnett, 1995]. The material should be dry and free flowing. It must be noted that pump connected to the compressor used for conveying may get unbalanced due to wear and tear which will reduce the efficiency of the system due to air leakage. The pump should not be allowed to operate for a sustained period of time without material or substantially under loaded. Hence material should always be available for conveying.

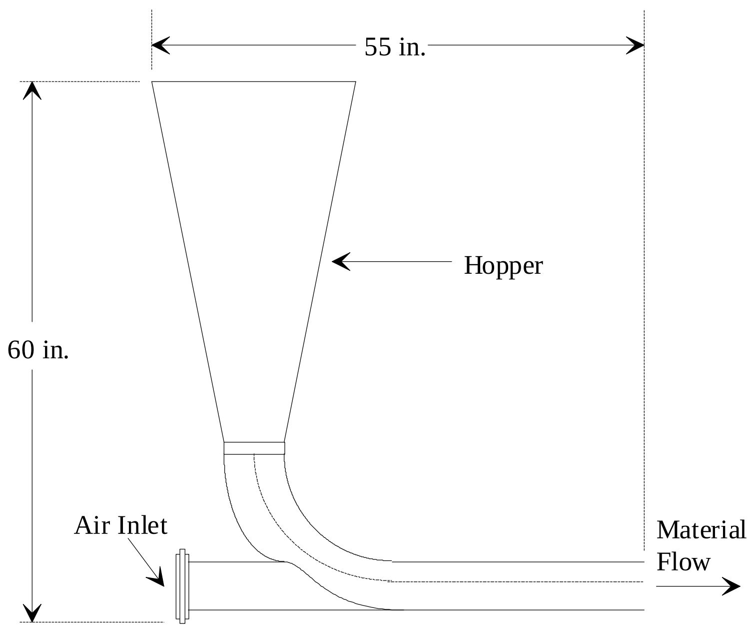


Figure 5.3 Pneumatic pipe feeder system (Burnett, 1995)

5.2.2 Hydraulic Backfilling

Hydraulic backfilling of voids involves the utilization of water as a transporting medium of solids. The typical components of this system includes a mixing installation for making the slurry, a slurry pump or pumps, and pipes for distributing the fill material underground. The mixing installation includes a mixing tank into which prepared fill is fed and mixed with water to form an evenly mixed slurry [Young-On , 1990]. Apart from mixer, slurry pumps are used for transporting the mixture.

The material chosen to be used for hydraulic backfilling requires large quantities of cement (about 10%). Hence before using hydraulic filling, the economic benefits derived from it should be stringently measured. Moreover, the fill material has very low water content. Hence it must be flushed very quickly into the mined out area as soon as possible, otherwise there is a danger of the whole mixture setting in the pump and the pipes. Fly ash and very fine washery refuse show thixotrophic properties which prevent the slurry from settling down. They also help is reducing friction on pipeline wall. The chances of setting can be further reduced if cement is mixed at the end of the transportation phase just before introduction into the mined out stall. The use of high density fill has been successfully implemented in Grund mine in West Germany. Highly consistent paste like fill material has been successfully transported over a distance of 6500 ft with discharge rate of 137 fpm [Crandell, 1992].

Major bottle neck of hydraulic filling over pneumatic filling is the availability of the transporting medium that is water. It may or may not be widely available in the mine and hence hydraulic filling cannot be applied universally. On the other hand pneumatic system does not have any such bottleneck. But hydraulic fill provides a much better fill strength compared to pneumatic fill. As a result where high strength filling is required due to high strata pressure and/or weak pillar strength, hydraulic fill will perform better compared to pneumatic fill. Hence the choice of backfilling system is site specific and no general rule of thumb is available for selection.

There are two types of hydraulic transportation system namely 1) open hydraulic system, and 2) circulating hydraulic system. The open system requires only one length of pipe from the slurry preparation plant to the face. For this system, mine water is supplied from existing dewatering system within the mine. The pumped out water is used to transport the slurry down to the face for final injection into the stall. Circulating hydraulic system on the other hand works on a closed loop where water used for transporting the fill material is recycled and returned to the surface to be used again. Hence depending upon site condition, any one of the system can be used for transporting fill material. Various different types of pumps are available in the industry for pumping slurry. They are listed below [Zhou, 1993].

- Plunger pump
- Piston pump
- Double piston pumps
- Rotary ram pump
- Centrifugal pumps
- Hydraulic exchange pumps

All these pumps are capable of producing high flow rate and high heads, but only hydraulic exchange pump is capable of handling high concentration slurries of the order of 60% solids [Zhou, 1993]. The total flow rate required to fill alternate stalls is shown below.

Amount of fill material required = 570,000 x 0.67 = 381,900 tons/yr.

= 180,000,000 lb. (approx.)

Volume of material required per hour assuming filling is done 6 hr. per shift, 2 shift per day, 250 days per year = 180,000,000 /(250 x 2 x 6 x 50) = 1200 cu. ft/hr

If concentration of material to be used is 85% then amount of water required is given by the following relation [Zhou, 1993].

$Q_w = Q_c(1-C_v)/C_v$ (5.1)

where

Q_w = volume flow rate of water

Q_c = volume flow rate of solid fill material

C_v = concentration of solid

= 1200 x15/85 = 211 cu ft/hr.

Amount of slurry that the system should be able to handle = 1200 + 211 = 1411 cu ft/hr.

= 180 gpm (approx.)

Pumps of such capacity are widely available. However, another major cost center associated with hydraulic filling are the pipes. For the purpose of filling, abrasion resistant steel pipes have to be used for transporting the slurries to the panels and epoxy coated fiberglass slurry lines for injecting the material into the stalls [Young-On, 1990]. These are substantially more expensive than PVC pipes which can be used in case of pneumatic filling.

6. Cost Analysis

This chapter combines mining, backfilling and development costs. For mining it is assumed that the self advancing miner will be used. Furthermore the following assumptions are made:

Seam thickness	: 2ft.
Width of cut	: 7 ft.
Depth of cut	: 600 ft.
Density of coal	: 0.041 st./cu. ft.
Density of fill material	: 0.055 lb./cu. ft.
Pillar width	: 4 ft

Two situation exists while considering back filling the stalls 1) filling every stall; 2) filling every alternate stall. Amount of fill material required per ton of coal mined is determined by the following calculations.

Production from each stall is 2 x 7 x 600 x 0.041 = 345 st
Amount of material required to fill the stall completely is 2 x 7 x 600 x 0.055 = 462 tons
If every stall is filled, then amount of fill material required is:
462/345 = 1.34 tons per short ton of coal.
If every other stall is filled amount of fill material required is:
465/690 = 0.67 tons per short ton of coal.
Hence to fill up one panel, amount of fill material required for both the cases are:
1.34 x (2 x 3900 x 1200 x 0.041 x 0.63) = 320,000 tons and
0.67 x (2 x 3900 x1200 x 0.41 x 0.63) = 165,000 tons.

Sensitivity analysis is performed to determine how the total cost varies with filling cost per ton. Total cost includes panel development, mining, and filling costs. Fixed and operating cost of the operation is $1,621,200/yr (from section 3.2.3). In one year, number of panels that can be mined is calculated under the following assumptions:

Cutting speed	: 9 ft/min
Withdrawal speed	: 24 ft/min
Location change	: 15 min/stall

Production per shift obtained from Figure 3.12 is approximately 1140 tons. Assuming 2 shifts per working day, production per day = 2280 tpd. If there are 250 working days in a year, production per year = 570000 st.

This amount of production will come from 2.5 panels of dimension 4000 ft by 1320 ft. If every alternate stall is filled, amount of fill material required is 0.67 x 570,000 = 381,900 tons. Amount of development needed to block two production panels can be evaluated be considering the panel layout shown below in Figure 6.1.

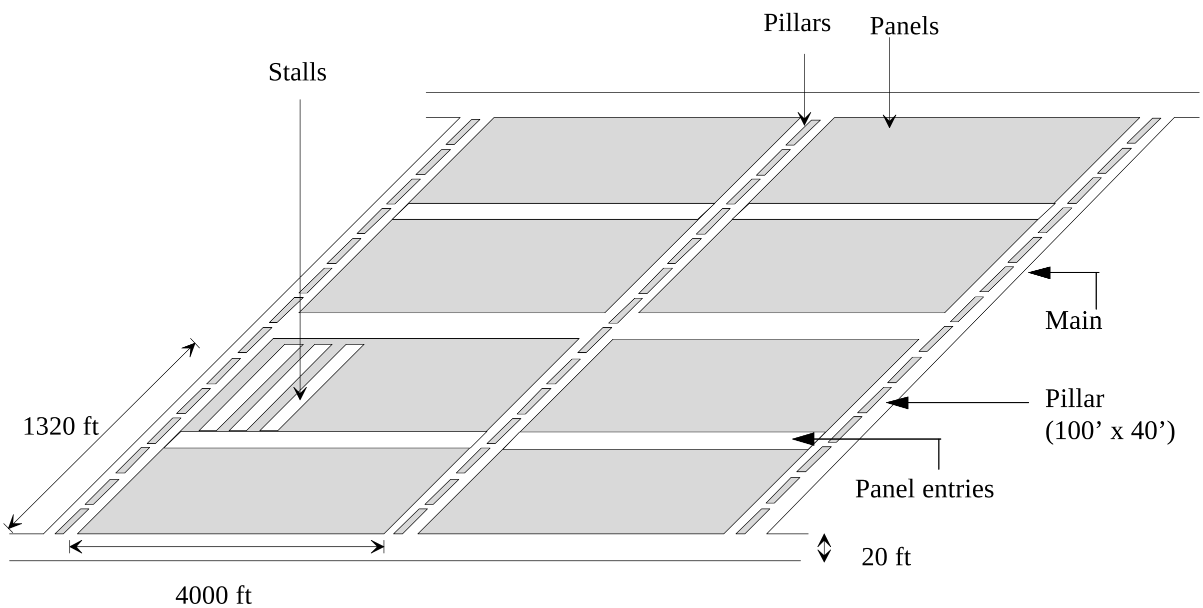


Figure 6.1 Panel layout for self advancing miner

Development of main headings	= (1320 + 20) x 2	= 2680 feet
Development of breakthroughs	= (4000/120) x 40	= 1400 feet
Development of panel entries	= 2 x 4000	= 8000 feet
Total development to block two panels:		= 12080 feet

Table 6.1 shows the result of sensitivity analysis performed by varying filling cost across different panel development cost assuming alternate stalls are filled.

Table 6.1 Cost/s.ton versus filling cost (for different development cost/ft)

Filling Cost ($/s. ton of coal)	Development Cost ($/ft)			
	$ 150/ft	$ 200/ft	$ 250/ft	$ 400/ft
5	$ 11.02	$ 12.08	$ 13.14	$ 16.32
10	$ 16.02	$ 17.08	$ 18.14	$ 21.32
15	$ 21.02	$ 22.08	$ 23.14	$ 26.32
20	$ 26.02	$ 27.08	$ 28.14	$ 31.32

The corresponding plot is shown in Figure 6.2.

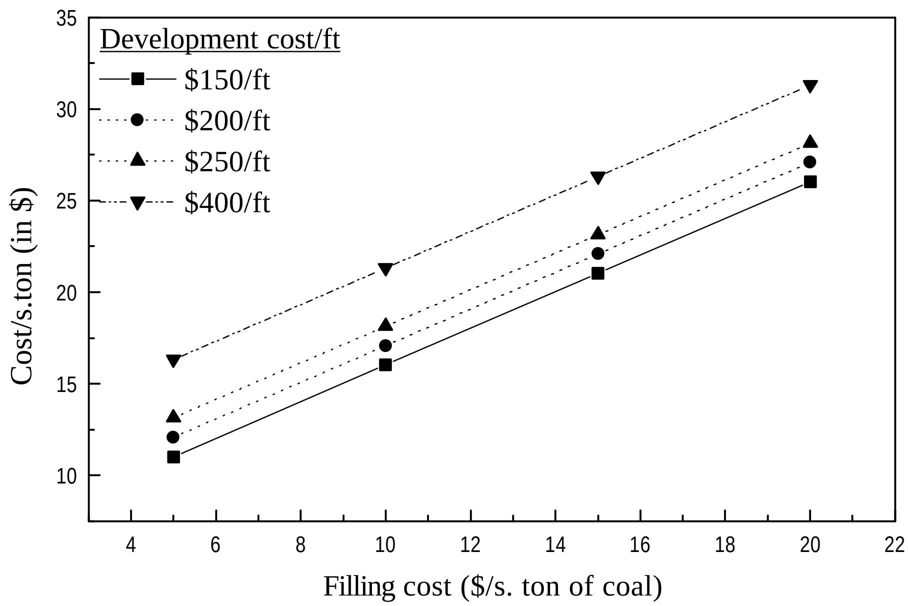


Figure 6.2 Relationship between cost/s.ton and filling cost/ton

(for different development cost/ft)

It can be observed from the above figure that filling cost is very crucial to the overall face cost of the project. It must be noted that the cost of mine entries, surface, and other necessary costs need to be added in order to determine the overall cost of mining.

7. Conclusion and Future Work

In this thesis, two mining systems have been proposed along with their production potential and estimated costs. It is observed that predicted production as well as extraction percentage for both the systems are large enough to make thin seam mining profitable. From the preliminary studies conducted it appears that self advancing miner (SAM) has a better potential than the auger mining system. It has much higher production rates compared to auger mining with not much difference in yearly operating cost (see sections 3.1.5 and 3.2.3). However, substantial work is left to be done be equipment manufacturers, coal producing companies, and research and development organization to make these systems viable in the future.

Some progress has already been made in the area of equipment development recently. Several companies including Fairchild International, Joy Mining Machinery, and Tamrock have all developed different types of thin seam miners. Fairchild and Joy have also developed conveyor units which can be articulated to form a long conveyor chain which can negotiate bends, very similar to the one required by self advancing miner (SAM). Now it lies in the hands of coal mining companies to use these equipment and follow the guidelines proposed in this thesis to determine whether thin seam mining will at all be profitable or not.

Several coal interface detection and web thickness sensing techniques have been discussed in this thesis. There are variety of such systems to select from depending upon the desired accuracy, safety and costs associated with each of them. Technology is well developed to provide an accurate detection of coal/rock boundary and pillar thickness to help enhance production compared to what was available a decade back. Out of all the systems discussed, natural gamma detection technique seemed to be more feasible than the other systems for detecting coal/rock interface. It is the only commercially available system. For the purpose of measuring rib thickness, radar detection technique seems to be feasible. Though no commercial system exists, there is a high probability that the development of such a system is not very far of in time.

Apart from discussing the mining systems, filling of the stalls or production cuts have been discussed. Though filling is an extra cost item for mining, it cannot be done away with mainly due to seam depths and strata pressure that one is likely to encounter while mining thin seams. Moreover, extraction should be higher in order to make the whole venture profitable. Some preliminary calculations have been included in the thesis to establish quantity of fill material and, capacity requirements to match up with production, as well as costs associated with filling. Filling seems to be a feasible option but availability of fill material is likely to pose some problem since small amount of waste material is likely to be generated during thin seam mining. Hence fill material may need to be imported from neighboring power plants, urban and industrial waste disposal sources. However, if high disposal cost material is available, then this may even improve the economics of the proposed system. From preliminary studies assuming that alternate stalls are filled total panel cost varied between $11/short ton to $31/short ton for different filling and development costs. If all the stalls are filled, total panel cost varies between $16/short ton to $51/short ton. It is found that the panel cost is more sensitive to filling cost than development cost.

Further studies must be done before safe and economic application of the thin seam miners. More research is needed in the area of ground control for the better understanding of roof and floor behavior under the proposed new mining layouts. Studies must be conducted to establish and maintain correct location of stalls, width of pillars, width of entries and mining sequence under given mining conditions which would enhance recovery and safety. Furthermore, their is a need for the development of inexpensive yet effective fill material for backfilling the stalls otherwise filling might become impossible under stringent economic returns.

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Appendix 1

This appendix displays some outputs from the program. Screen shots have been directly taken from the screen and displayed here.

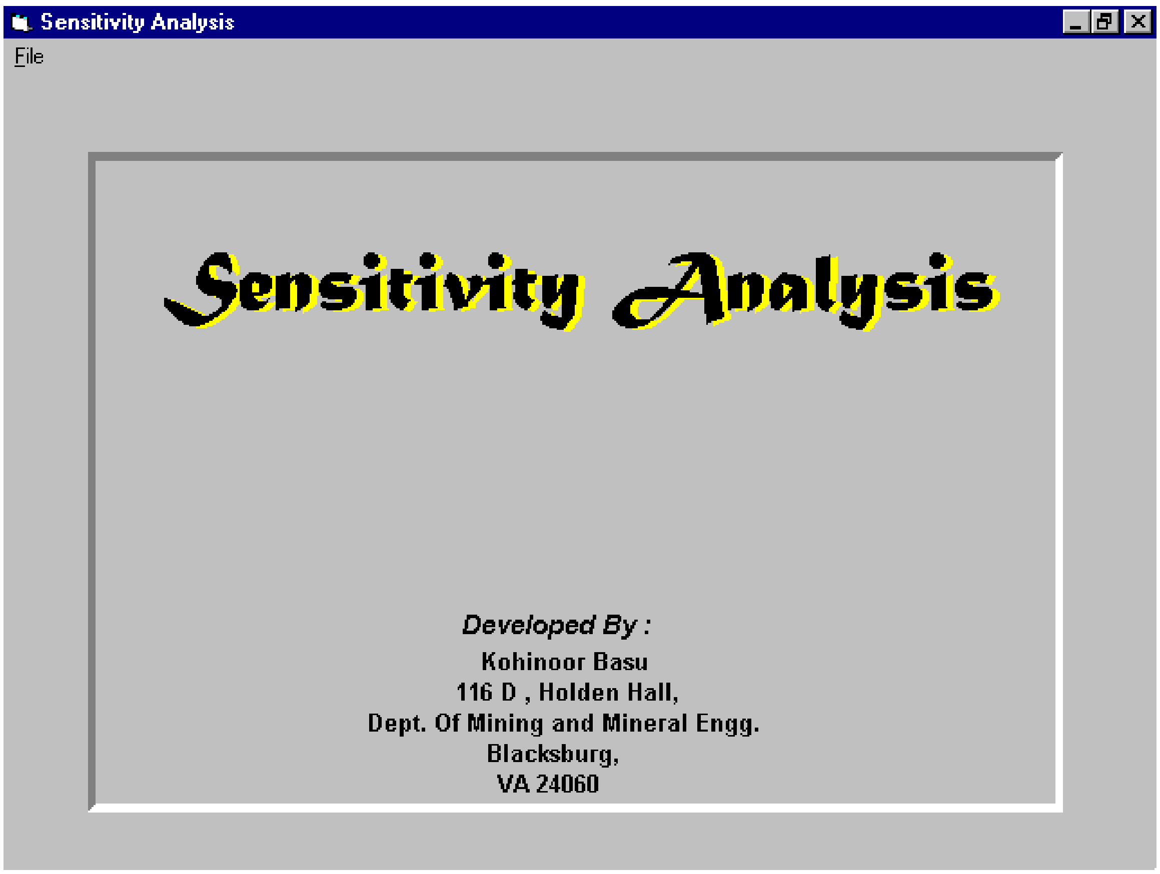


Figure 1.A Opening Screen

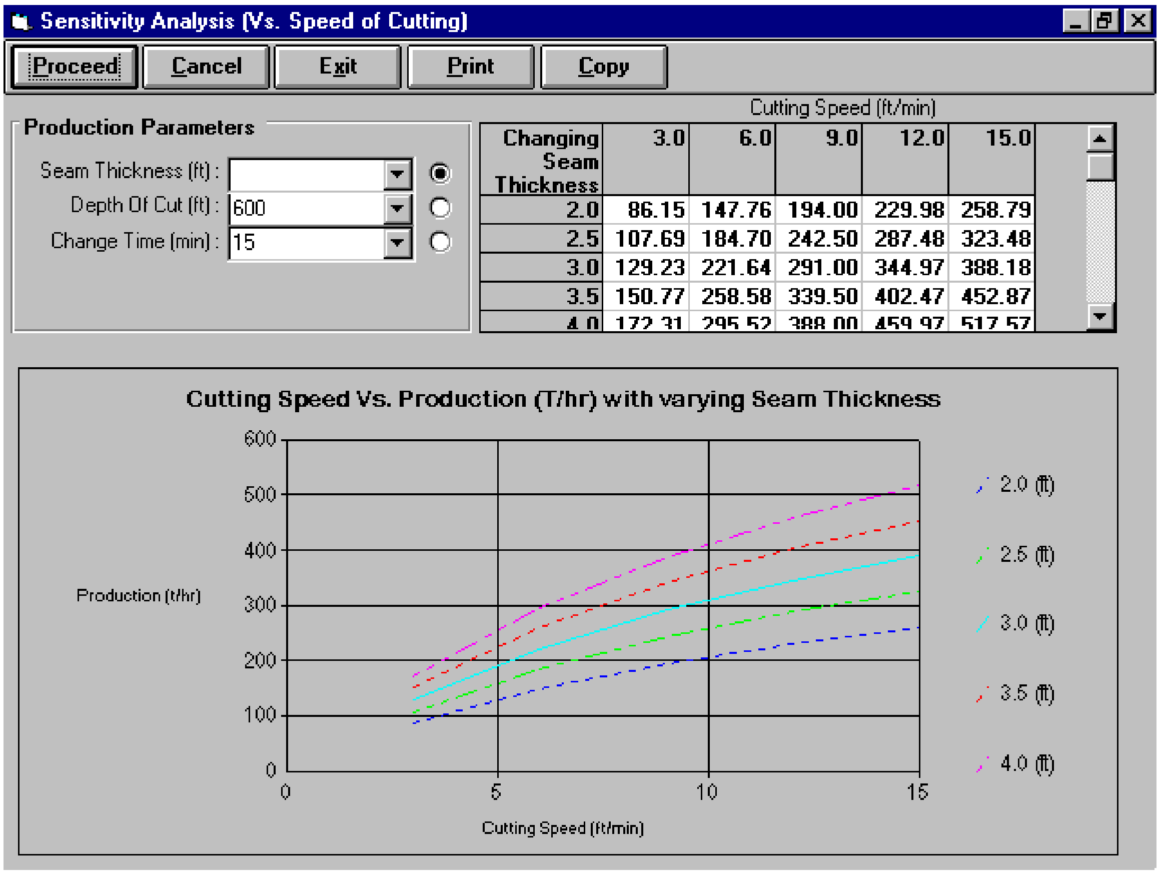


Figure 1.B Screen for performing sensitivity analysis with respect to cutting speed

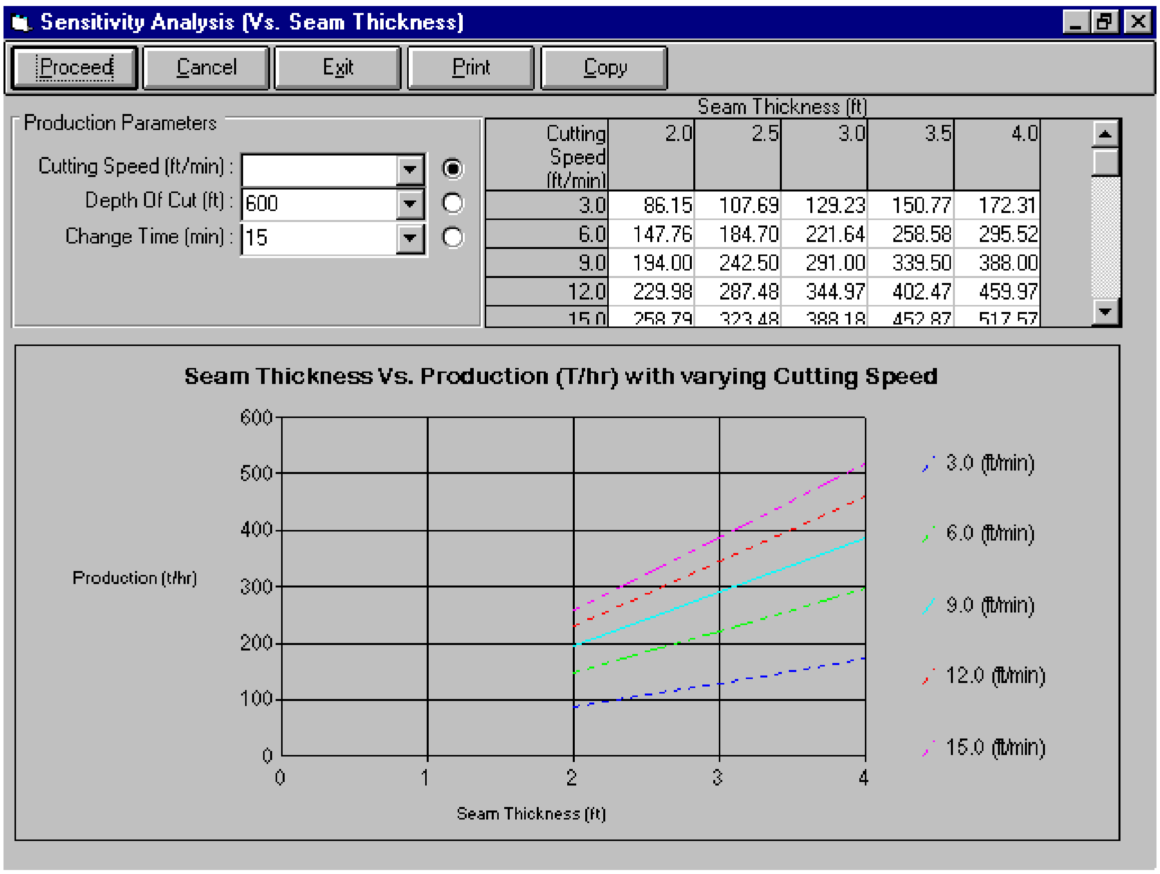


Figure 1.C Screen for performing sensitivity analysis with respect to seam thickness:

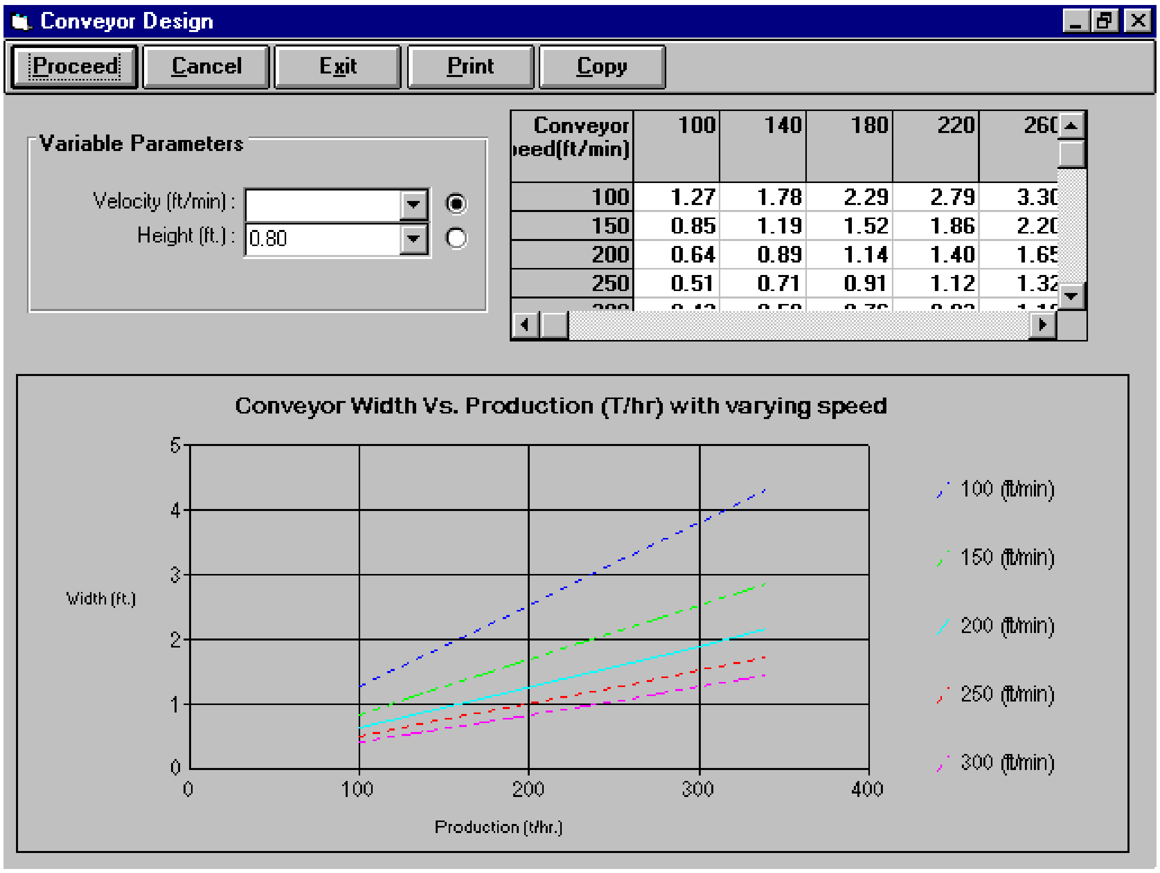


Figure 1.D Screen for performing sensitivity analysis to establish appropriate conveyor width

Appendix 2

This appendix lists some thin seam mining equipment which have been developed by various manufacturing companies in Virginia and West Virginia. They include companies like Joy Mining Company, FairChild International, and EIMCO. They are shown below along with their specifications.

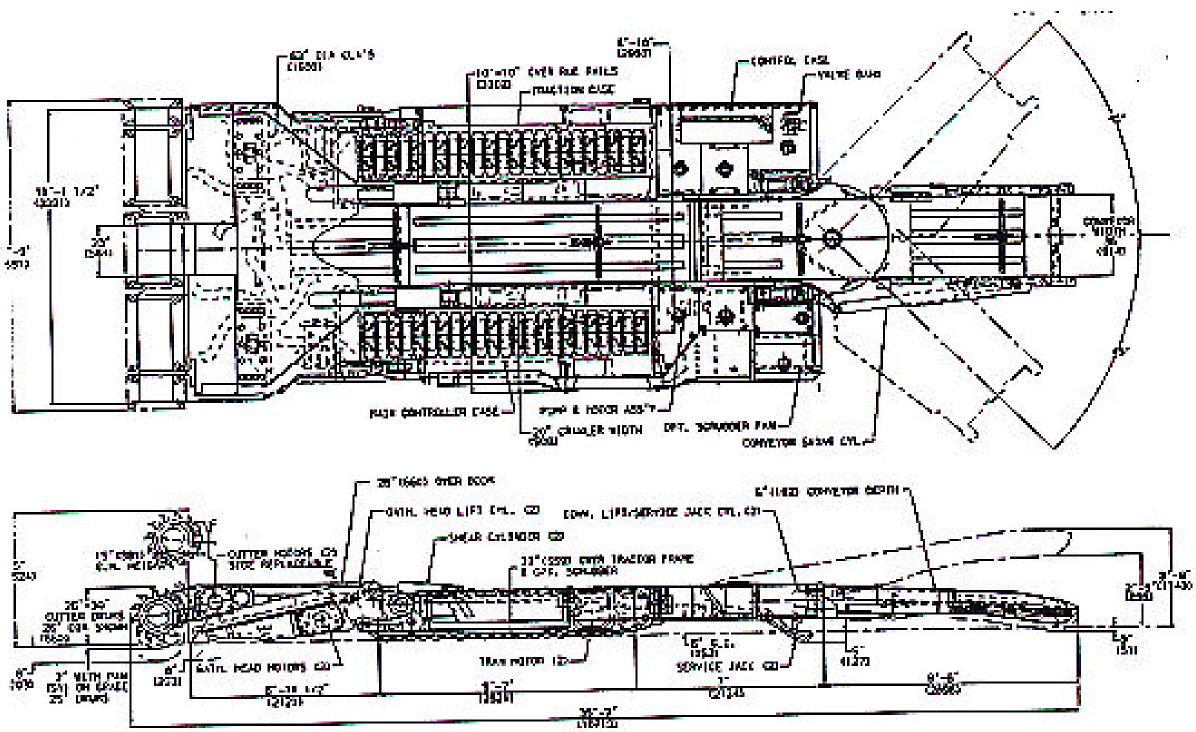


Figure 2.A EIMCO Dash Zero continuous miner

EIMCO
DASH ZERO

DASH ZERO CONTINUOUS MINER

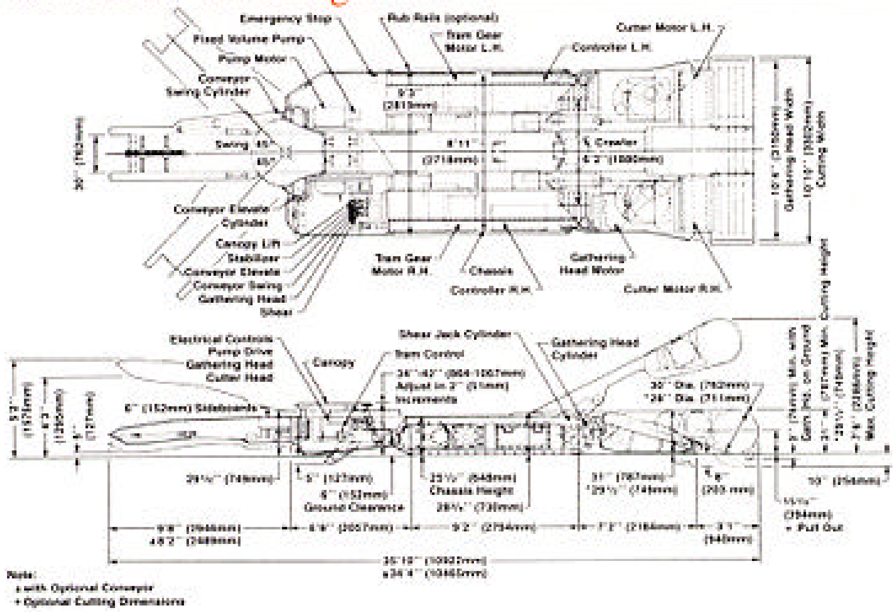
- 5’ 00” Reach, 175 HP Cutter Motors, 26” - 28” - 30” Drum Diameter
- 22” Frame, 26” over Cutter Boom, 6” G.C.
- Machine WT. 100,000LBS, 20” Crawlers, 21.6 PSI Ground Pressure
- Patented Core Breaker, Individualized Gear Cases, 11” - 9” Chainless Cutterhead
- 4300 CFM Scrubber, Single Intake
- Dual 50HP, 6-Finger 63” Dia. CLA Gathering Head
- 36” Wide, 475 FPM, Sound Deadened Conveyor with Replaceable Strips
- All Gear Drive, Modular, Infinitely Variable 0-65 FPM with Cutterhead Feedback Traction System
- Micro Processor, Two-Way Data Link Radio Remote Control
- Maximum Material Flow/Load Rate, Component Design and Component Interchangeability with other Dash Series Models
- Motors

    Cutter . . . .(2) @ 175 HP AC
    Gathering/Conveyor . . . . (2) @ 50 HP AC
    Pump . . . . (1) @ 45 HP AC
    Tram . . . .(1) @ 50 HP DC
    Scrubber . . . .(1) @ 5 HP AC

- Machine Power 950 VAC

Figure 2.B Specifications of EIMCO Dash Zero continuous miner

14CM10-10AA General Arrangement



All JOY products and services are sold subject to JOY's standard terms and conditions of sale, including its limited warranty. These will be furnished upon request. The company reserves the right to alter or improve the design or construction of its machinery as described herein and to furnish it, when so altered, without reference to the illustration or descriptions in this bulletin.

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Mining Machinery Division

Figure 2.C Joy 14 CM 10 - 10AA general arrangement

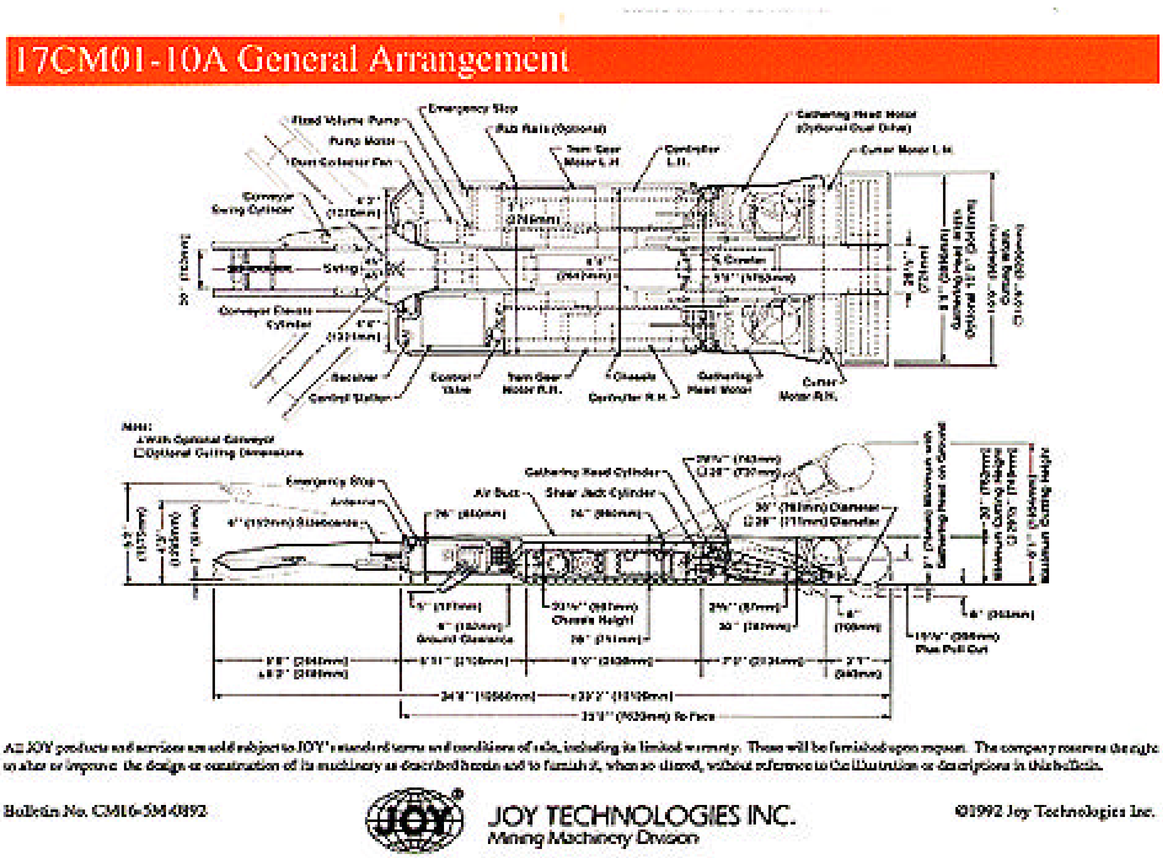

Figure 2.D Joy 17 CM 01 - 10A general arrangement

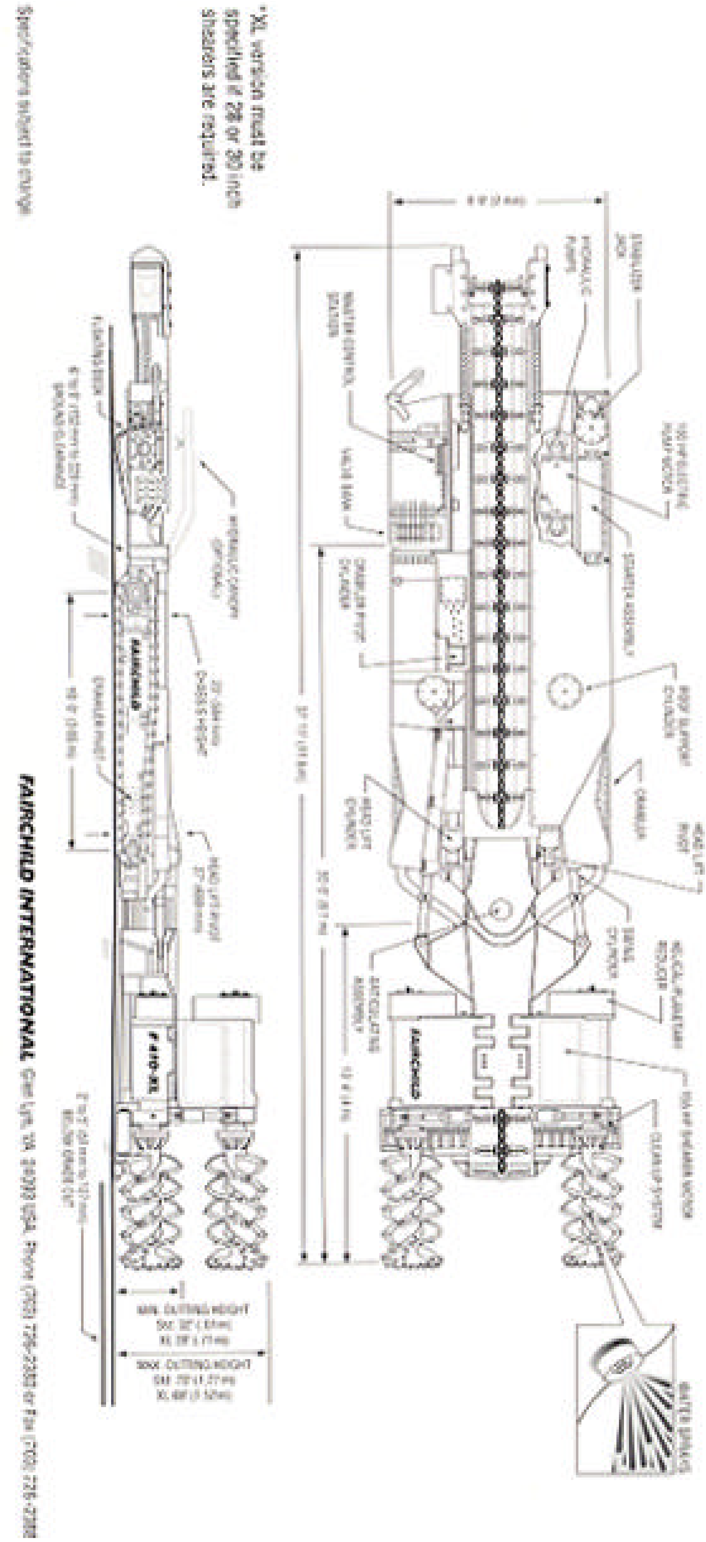

Figure 2.E FairChild F410 - XL general arrangement

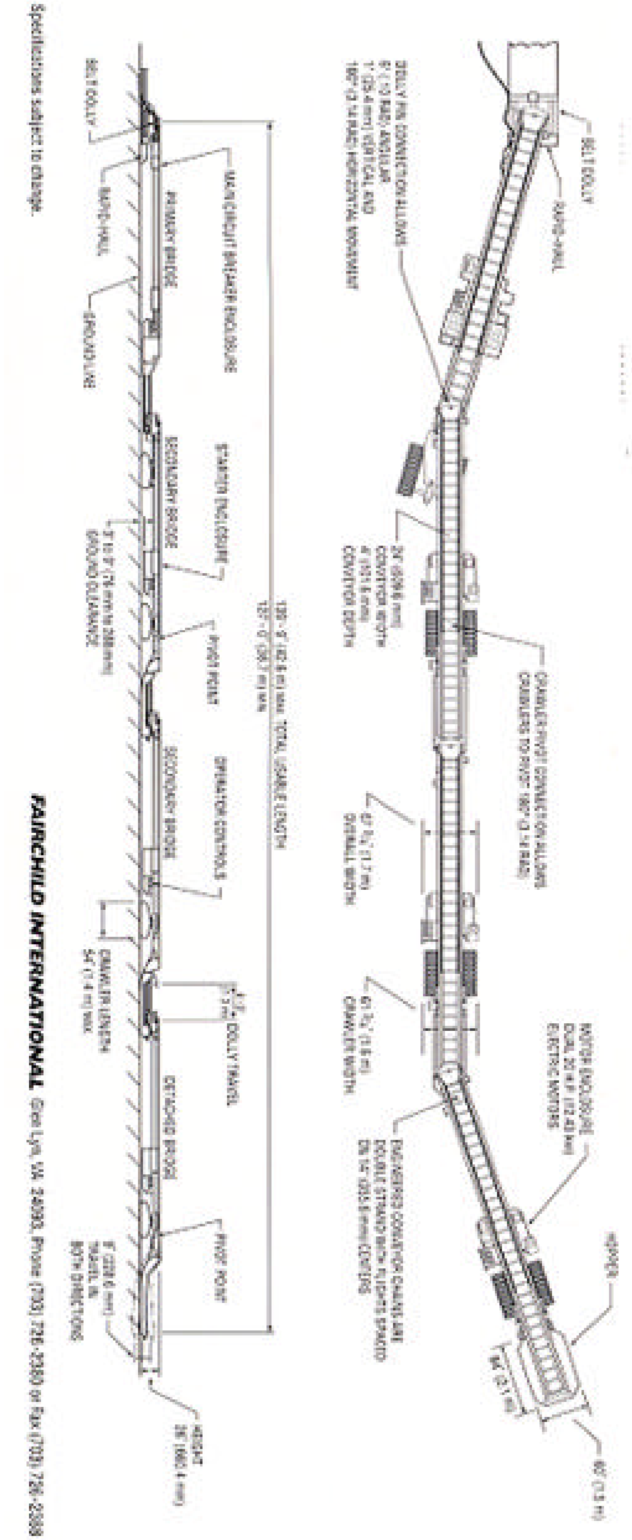

Figure 2.F FairChild Haul-Mark general arrangement

VITA

Kohinoor was born on September, 1971 in Calcutta, India. In August, 1994, he received his Bachelors of Technology in Mining Engineering from Indian Institute of Technology, Kharagpur. After completion, he worked for two years as an Information Technology consultant in Computer Exchange Pvt. Ltd. in Calcutta for two years. In August, 1996 he was granted a research assistantship by the department of Mining and Minerals Engineering, Virginia Polytechnic Institute and State University, where he started his graduate studies leading to a degree of Masters of Science in Mining Engineering.