

THE EFFECTS OF BACKFILLING ON GROUND CONTROL AND RECOVERY IN THIN-SEAM COAL MINING

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(ABSTRACT)

A large percentage of Southern Appalachian coal reserves are located in seams less than 36" thick. As thicker and currently more mineable, deposits are exhausted, methods of underground thin-seam extraction will have to be developed. These methods must be capable of removing coal efficiently and economically. Past experience with highwall mining of thin-seam coal has indicated that recovery rates tend to be lower than in conventional operations. It is suspected that this will also apply to underground thin-seam mining, regardless of proposed technology or mining method. A method of increasing recoveries from thin-seam mining operations is necessary in order to exploit thin-seam reserves.

Backfilling is one alternative that may find applicability in thin-seam coal mining. The ability of backfill to provide additional ground support may enhance coal recovery by allowing for the design of undersized pillars. Backfill has been used extensively in hard rock mining but has found limited use in coal mining. Its adaptability to thin-seam coal mining has been examined and is presented in this thesis. Backfill is capable of providing additional ground support by restricting lateral deformation of surrounding coal pillars and roof. This additional support can result in significant increases in recovery from thin-seam coal deposits. However, the overall feasibility of backfill is dependent on the in situ behavior of the fill material, the properties of the fill, the effects of the filling method on the total mining operation, and the cost of filling per extra ton of coal recovered. The influence of these parameters has been studied and indicate that, in certain situations, backfilling for the purpose of increasing recovery rates from thin-seam coal mines is feasible.

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CHAPTER 1. INTRODUCTION

Over the last century the Southern Appalachian coal fields have been mined extensively. The amount of total reserves has been reduced substantially due to this activity. Most of the thicker deposits have been depleted and those remaining will probably be exhausted within 20 years. In order to maintain a substantial amount of coal production beyond that time, coal found in thin-seams ($\leq 36''$) must be exploited. In order to shift the focus of coal mining to thinner seams, technology capable of extracting thin-seam coal efficiently and economically must be developed and refined. Based on existing technology used in highwall mining and underground mining, one such thin-seam mining method has already been proposed.

The Self-Advancing Miner has been designed to extract thin-seam coal from underground mines (Basu, 1997). After development of a 4000' long and 850' wide panel, the miner cuts into the coal seam at a height equal to the seam thickness, at an angle of 45° , and to an extent of 600'. The miner continues along the length of the panel leaving pillars between each cut. After one side of the panel has been mined, the miner proceeds to the opposite side and extracts the coal in the same manner.

Since the coal seams are so thin (24"-36") it is apparent that the theoretical recovery rates of this mining method will be fairly low and will decrease rapidly with the depth of mining. An additional source of ground support that allows for the design of smaller pillars is necessary in order to increase the recovery from thin-seam mining. Backfill is one alternative that must be considered. Although not common in coal mining, backfill has been employed successfully in other types of mining. The purpose of this report is to assess the feasibility of backfill as applied to thin-seam coal mining.

The first step in examining backfill is to study all relevant information regarding the history of backfill, backfill materials, material properties, and the support mechanism of backfill. Experience and research with different types of backfill materials, their engineering properties, and their underground behavior has lead to the formation of a vast knowledge base on backfill. The most important points and the ones that influence the backfilling of thin-seam coal mines are summarized in this thesis.

The extracted panel left by the miner forms the basis upon which recovery with backfilling can be compared to recovery without backfilling. The pillars left within the panel are defined, analyzed, and designed according to the functions that they provide. A pillar design procedure and design program is outlined and developed for this purpose. The pillar program calculates widths based on seam height, cut length, cut width, and overburden depth. From these calculated widths the theoretical rates of recovery from a single panel can be determined and used later for comparison with a backfilled panel.

Although the support capability of backfill is well known, it remains fairly difficult to quantify. The anticipated behavior and response of fill to deformations of the surrounding pillars and roof are analyzed here. A method for determining the amount of fill support and the resulting increase in pillar strength based on fill properties, coal pillar properties, and surcharge loading of the fill is developed. Design charts are presented which indicate the influence that these parameters have on the magnitude of fill support and ultimately on the increase in recovery due to backfilling. The supporting effect of backfill is incorporated into the original pillar design program so that new pillar widths can be calculated and the increase in recovery can be determined.

The final analysis of backfill is in terms of cost. Although backfill increases recovery, it is not necessarily economic. The cost of backfilling incurred in order to increase recovery should not exceed the current profit made per ton of coal (assuming all other things equal between the non-backfill mining operation and the operation with backfill). Therefore the cost of fill is determined per ton of extra coal recovered. Once again, the costs are presented in the form of charts indicating the dependence of fill cost on the magnitude of fill support and depth of mining.

Other considerations concerning backfill and a backfilling operation are discussed. Material availability, material properties, placement method, mining sequence, and other parameters affecting the overall feasibility of backfilling are reviewed. General guidelines and suggestions regarding these parameters are also included.

Based upon available information and past research, the analysis of theoretical recovery rates with and without backfill, and other important considerations, conclusions

and recommendations are drawn about the overall feasibility, and application, of backfill in an underground thin-seam coal mine.

CHAPTER 2. LITERATURE REVIEW

2.1 Background

The placing of backfill underground has predominantly been a practice employed in cut-and-fill mines. Backfill material is introduced underground into previously mined stopes to provide a working platform and localized support, reducing the volume of open space which could potentially be filled by a collapse of the surrounding pillars (Barret et al., 1978). Therefore, the placement of fill in open spaces underground tends to prevent the deformation of the surrounding rock mass into the mined out space, effectively increasing the strength, or load bearing capacity, of the surrounding rock mass. This type of support mechanism not only helps provide support to pillars and walls, but also helps to prevent caving and roof falls, minimize surface subsidence, and enhance pillar recovery (Coates, 1981).

The most common types of fill material are waste rock, mill tailings, quarried rock, sand, and gravel. Of these materials, waste rock and mill tailings, generated during the mining process, are the most accessible and usually the most economic. Waste rock and tailings come from three major sources: (1) development work of the mine, (2) washer-concentrator reject, and (3) old refuse piles. Development work can generate up to 25% of the needed backfill in the form of crushed stone, while mineral processing acts as the main source of backfill due to the quantity and quality of reject material (Arioglu and Biron, 1983). Additives such as portland cement, lime, pulverized fly ash, and pastes are also added to the fill in order to alter its characteristics and improve its effectiveness.

The method of placement of the fill is dependent upon the source of energy required to install the backfill material underground (Arioglu and Biron, 1983). Backfilling applications include hand stowing, gravity stowing, mechanical stowing, pneumatic stowing, and hydraulic stowing. Hydraulic stowing is the most advanced method of backfilling and is best suited for use with mine tailings. Wet refuse from processing plants can be easily transported over long distances, and eventually underground, via pipelines. Hydraulic placement provides an extremely flexible system

in terms of changes in orebody geometry, such as stope width, dip, and length and the stowed tailings constitute an excellent support system (Wayment, 1978).

Backfilling of coal mines in the past has been primarily restricted to abandoned mines (Bloomfield, 1984). Coal refuse piles, which become physically unsafe in terms of slope stability and acid-mine drainage, have been the source of fill for these abandoned mines. Aside from avoiding surface disposal of coal waste and relieving the above-ground surface of an environmental hazard, coal waste has been placed to provide support to the roof and remnant pillars left after mining operations have ceased (Bloomfield, 1984). The practices employed in backfilling abandoned coal mines can be, and have been, adapted for active coal mines, resulting in favorable stability, safety, environmental, and economic conditions.

Advantages of stowing coal waste underground include the elimination of the environmental, health, safety, and social problems associated with surface disposal (National Academy of Sciences, 1975). Coal waste, rejected from processing plants, is currently placed on the surface in waste embankments and ponds. By recycling this waste and using it in some form as backfill, environmental problems such as slope failures, dam failures, and air and water pollution may be avoided. The accessibility of coal refuse and plant tailings as fill material may also provide an economic advantage over the costs incurred during the permitting, construction, and reclamation of surface disposal sites.

Although the environmental advantages of using coal refuse underground as fill material are significant, the major concern with backfilling is its effectiveness towards improving coal recovery and underground stability. Aside from its environmental benefits, the main purposes of fill can be summarized as follows (Arioglu et al., 1986):

1. To provide support to openings and pillars
2. To make pillar recovery safer, thus enhancing productivity
3. To minimize surface subsidence

The ground support function is the most important role for fill as a structural component. The presence of fill in an opening can prevent large scale movements and collapse of openings merely by occupying voids left by mining (Aitchison et al., 1973). As time goes by and the pillars start to deform, the fill material acts to restrict this movement,

effectively increasing the strength of the pillars and improving the stability of the mine workings. The additional roof and pillar support provided by the fill allows for greater coal recovery by increasing extraction ratios (Bloomfield, 1984). The strengthening effect of the fill permits the design of smaller pillars, enhancing coal recovery and sometimes eliminating the need for a secondary operation dedicated to pillar recovery. Backfilling is also a means of subsidence control in mines where pillar support of the overburden is inadequate. The placing of fill significantly reduces surface damage from subsidence by lending lateral support to the pillars and by limiting the volume of voids. While backfilling a mine does not completely eliminate subsidence, it has been shown to reduce it by 33% as compared to a non-backfilled mine (National Academy of Sciences, 1975). Hydraulic backfilling immediately following coal mining operations has created even greater reductions in surface subsidence. Subsidence as small as 5 to 15% of extracted seam thickness has been reported, much less than the 75% usually created by room and pillar operations (Vorobjev and Deshmukh, 1966).

Improved ventilation is a by-product of the use of fill for increased ground control. Movements of the coal strata tend to damage stoppings, barriers, over/undercasts, and other ventilation apparatus, increasing leakage throughout the mine (Hartman et al., 1982). The average leakage in a coal mine is 50%, from the fan to last open crosscut, and much of this can be attributed to damage caused by ground movements. Backfilling is a very effective way of reducing this leakage and insuring that an adequate supply of fresh air reaches the coal face. By preventing leakage through mined out areas, fill also improves the utilization efficiency of ventilation air, which has been shown to lower costs for the purchase and running of fans.

There are several disadvantages of using coal refuse as backfill; these have lead to the limited use of backfilling in U.S. coal mines. The main disadvantage of stowing coal waste underground is the resulting increase in cost. In order to employ a hydraulic stowing system a large capital investment is required to build and maintain the processing plant, pumps, mixers, pipelines, and bulkheads that make up the system. Obtaining enough fill material to occupy underground voids is also a problem. The coal refuse produced during the cleaning process is only about 25% of the volume of the raw coal removed; therefore totally filling the mine voids with waste from the mining operations is

not possible (National Academy of Sciences, 1975). In order to completely backfill the mine, materials from alternate sources must be acquired. Options include crushed stone, sand, and gravel, but these all incur more costs. Other waste materials, such as pulverized fly ash and sulfate sludge from coal burning power plant operations must be considered. These materials may be able to provide the additional volume of fill at a lower cost than aggregate materials.

Although placement of coal refuse underground alleviates the problems associated with surface disposal, it presents new environmental considerations. Fill material consisting of high sulfur coal and spoil has the potential to react chemically with air and water to produce sulfuric acid (Sleeman, 1990). If oxidation of the coal waste occurs and sulfuric acid is produced, the acid will drain through the fill, possibly entering and polluting the groundwater.

The biggest single problem associated with coal tailings backfill is water. Although the easiest and cheapest way to transport fill underground is as a tailings-water slurry (hydraulically), the introduction of water creates a significant problem underground (Wayment, 1978). Highly saturated fills have lower strengths than unsaturated fills as additional water reduces the cohesion between fill particles. Although adding cementing agents to the fill can mitigate this problem, water also tends to influence the effectiveness of cemented fill. Segregation may occur during the filling process as coarse particles of cement and fill are deposited faster than the fines, which tend to be held in suspension for a longer period of time (Coates, 1981). A large fraction of the coarse particles may end up being placed close to the discharge pipe while the fines are carried further away from the pipe, resulting in non-uniform distribution of the fill and possible changes in its predicted properties and behavior.

In order to predict the ground support capabilities of a backfilling system it is important to define, and understand, important properties of the fill material. Since most fill materials can be defined as either granular or fine-grained soils, the evaluation of any fill property involves applying the principles of soil mechanics. Addition of cementing agents to a fill transforms the fill from a natural material (i.e., soil) to an engineered material, and thus requires further evaluation of the effects of cement addition.

2.2 Uncemented Backfill

Any fill material placed underground without being mixed or treated with cementing agents or binding chemicals is considered uncemented. Whether it be waste rock, tailings, or sand, the fill material itself is nothing more than a soil. Therefore, the mechanics of uncemented fill is essentially soil mechanics and important fill properties can be expressed using soil mechanics principles (Coates, 1981).

2.2.1 Properties of Uncemented Fill

Adequate support of underground mine voids, and the surrounding pillars, is the fundamental design criteria that any fill material must meet. Predicting a fill's potential to meet the design criteria is based on existing or modified soil properties of the fill. There are three main properties that govern the behavior of a fill material. These are shear strength, compressibility, and permeability (Schiffman et al., 1988). The stability of the backfill is dependent on the shear strength of the fill, while the compressibility and permeability are directly related to the settlement, drainage, and consolidation behavior of the backfill. Analysis of these three properties not only indicates a fill's potential behavior, but it also reveals other fill properties upon which shear strength, compressibility, and permeability are correspondingly dependent.

Success of a backfill system is based upon the fill material's ability to provide underground support. The most important ingredient for achieving success is proper evaluation of the shear strength of the fill. Shearing resistance in any fill material is the result of resistance to movement by bonds formed across interparticle contacts (Terzaghi et al., 1996). The nature of these bonds are dependent on fill mineralogy, composition, and density, so any evaluation of the shear strength of a fill material is dependent on the examination of these properties.

The shear strength of any fill may be attributed to three basic components (Cernica, 1995):

1. Frictional resistance to sliding between solid particles
2. Interlocking and bridging of solid particles to resist deformation
3. Cohesion and adhesion between soil particles

The preceding components indicate the importance of determining and describing the grain-size distribution of a fill. Most hydraulic backfills are prepared from concentrator tailings by cyclone treatment in order to remove the clay-size fraction (any material smaller than 0.074mm). Thomas (et al., 1979) collected fills prepared from 6 Canadian mines and analyzed their size distributions. The distributions are given in Table 2.1. The analyses show that, for the most part, the greatest percentage of material falls between 30 and 300µm, which corresponds to medium sand and silty-sand type soils (Brady and Brown, 1992).

Table 2.1 - Size Analysis of Hydraulic Fills (from Thomas et al., 1979)

Size Fraction (µm)	Weight %					
	Mine 1	Mine 2	Mine 3	Mine 7	Mine 8	Mine 9
+3340	0.00	0.00	0.00	0.00	0.07	0.00
2630-3340	0.00	0.00	0.00	0.00	1.09	0.00
1180-2360	0.00	0.00	0.00	0.00	9.01	0.00
850-1180	0.00	0.00	0.00	0.00	8.16	0.13
600-850	0.00	0.00	0.00	0.00	11.00	1.39
425-600	0.70	0.05	0.00	0.20	8.28	6.02
300-425	2.50	0.21	0.00	0.59	6.44	12.86
212-300	8.30	1.62	0.80	3.16	6.37	19.73
150-212	15.70	2.33	3.20	8.69	7.08	18.88
106-150	17.10	11.08	17.20	22.50	5.71	18.33
75-106	20.90	11.67	22.80	17.30	4.79	10.04
53-75	11.00	20.52	13.20	17.53	3.57	4.90
40-53	7.50	9.32	14.90	6.78	1.17	4.04
30-40	7.10	14.25	14.90	15.93	3.36	1.09
20-30	2.40	15.72	8.00	5.14	4.04	0.69
15-20	1.40	7.91	2.20	1.11	4.39	0.43
10-15	0.80	1.20	0.60	0.30	2.88	0.23
-10	4.80	4.01	2.20	0.79	12.60	1.25

From soil mechanics, sandy soils are designated as cohesionless; thus any resistance to deformation by a fill is created by friction between grains. The shear strength of a cohesionless fill can be represented by the following equation (Bell, 1992):

$$S = s \tan f \quad [2.1]$$

where,

S = The shear strength of the fill (lb/ft²)

s = The normal stress on the failure plane (lb/ft²)

f = The angle of friction of the fill

Equation 2.1 suggests that the shear strength of a fill will increase as its friction angle increases. This is an important point when analyzing the grain size distribution of a fill. Well-graded fills have higher friction angles than poorly-graded fills. Well-graded fills also exhibit a large range of fine and coarse particle sizes; this tends to decrease void space between grains, in turn increasing the frictional resistance of the fill particles.

The mineral composition of the fill indirectly affects the frictional resistance of the fill grains. Mineralogy controls important grain characteristics such as size, shape, surface attributes, angularity, and strength of particles. Angular grains interlock more effectively than rounded ones, creating a larger friction angle and increasing fill strength. Fills consisting of hard particles with rough surfaces that oppose grain movements offer greater resistance to deformation and displacement.

In addition to grain size and type, the shear strength is dependent on fill density. The density of a fill is dependent on the amount of solids and the amount of voids that make up the fill. As the void ratio decreases the amount of space between grains is reduced resulting in a denser fill. The increase in density of the fill implies an increase in interparticle contact area, and thus, in shearing resistance of the fill (Terzaghi et al., 1996). The tightly packed particles of a dense fill display greater resistance to shear forces as grains must be forced up and around adjoining grains.

When quantifying the shear strength of a fill it is important to consider the effect of water. Since most backfilling is done hydraulically, fill materials are saturated to some degree upon placement and tend to remain that way afterwards. Moist (partially saturated) sandy fills can have some cohesive strength, or apparent cohesion. Meniscus forces arising from surface tension in pore water produce intergranular forces between the individual fill particles. Therefore the soil exhibits cohesion as the intergranular forces produce frictional resistance between particles (Schroeder and Dickenson, 1996). The shear strength of the fill can then be expressed by the following equation (Cernica, 1995):

$$S = (s + u_c) \tan f \quad [2.2]$$

where,

S = The shear strength of the fill (lb/ft²)

s = The normal stress on the failure plane (lb/ft²)

u_c = The capillary force in the fill (lb/ft²)

f = The angle of friction of the fill

The water pressure can also be positive, reducing the shear strength of the fill. Frictional resistance between fill particles is reduced by the build-up of pore water pressure. Pore water pressure tends to compress fill particles but does not create any shear effects between fill particles. Therefore, pore water pressure acts against the normal force within the fill that induces interparticle friction, and as a result the shear strength of the fill is reduced (Schroeder and Dickenson, 1996). This phenomenon is represented in the following equation (Bell, 1992):

$$S = (s - u) \tan f \quad [2.3]$$

where,

S = The shear strength of the fill (lb/ft²)

s = The normal stress on the failure plane (lb/ft²)

u = The pore water pressure in the fill (lb/ft²)

f = The angle of friction of the fill

It can be seen that as the pore pressure nears σ , the shear strength of the fill nears zero. Not only is this important in terms of the fill not being able to provide the proper ground support, it is also important because it may result in liquefaction of the fill. If the fill is not confined, and reaches a state of liquefaction, it may fail and begin to flow as a mass, creating an underground hazard. Proper steps must be taken to avoid liquefaction after fill placement, including monitoring of underground blasts and fluctuations in the water table. Loose, fine, silty-sand fills are vulnerable to effects from dynamic loads, such as blasts, or sudden fluctuations in the water table (Cernica, 1995).

As surrounding rock masses deform into a backfilled area, pressures are exerted on the fill which cause vertical and lateral compression of the fill. The extent of compression is dependent on the density of the fill and the magnitude of the pressures. Therefore, the compressibility of a fill material can be measured in terms of its void ratio and the effective stresses to which it is subjected (Schiffman et al., 1988).

Although vertical and horizontal pressures are exerted on backfill, settlements due to lateral deformations are generally small when compared to those from vertical compression (Terzaghi et al., 1996). Therefore, the compression of a fill is one-dimensional and only vertical stresses are taken into account when calculating the total settlement. Underground, when the roof begins to deform into the fill, the vertical stresses acting on the fill can be attributed to the overburden pressure. In this scenario the overburden pressure is the effective stress which compresses the fill, resulting in a direct decrease in the volume of voids, or void ratio, of the fill.

The reduction in the volume of the fill can be attributed to the following factors (Cernica, 1995):

1. The escape of water and air from the voids
2. Compression of the fill particles
3. Compression of water and air within the voids

The total volume reduction caused by these factors is known as the settlement of the fill. Mathematically, the settlement of any fill can be calculated given the original height of the fill and the change in void ratio that results from loading caused by the overburden pressure (Bell, 1992).

$$\Delta h = \frac{h_1 \Delta e}{1 + e_1} \quad [2.4]$$

where,

Δh = The settlement of the fill (in)

h_1 = The initial height of the fill (in)

e_1 = The initial void ratio of the fill

Δe = The change in void ratio

Figure 2.1 is a visual illustration of the relationship given in Equation 4.

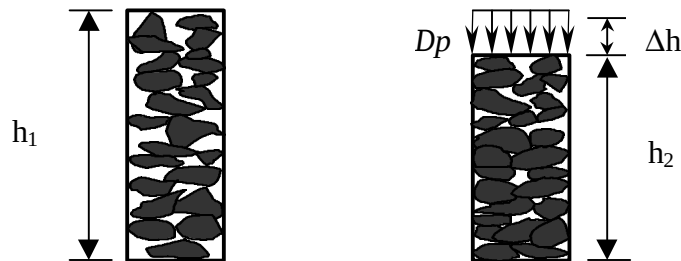


Figure 2.1 – Vertical Compression of Fill

It is evident that as Dp is applied, the volume of the fill decreases as particles are forced closer together. The void ratio decreases as air and water are dissipated from the voids and the density of the fill increases. This process of settlement due to pore water dissipation is time dependent and is referred to as primary consolidation. Most surface settlement, or subsidence, will occur during primary consolidation of the fill and can be predicted using one-dimensional consolidation theory.

It is useful to apply one-dimensional consolidation theory to fill materials in order to determine the magnitude of surface settlements above backfilled areas. However, it is a very simplistic approach to surface subsidence predictions. Since pillars and walls around backfilled areas will deform, fill material will be subjected to horizontal compression. Horizontal compression of the fill will, to some extent, cause the fill to expand vertically, altering the fill height and void ratio. Although one-dimensional consolidation theory may not be sufficient in determining surface settlements above filled areas, it is helpful in describing a fill's load-deformation characteristics.

The permeability of a fill is the measure of the ease with which water flows through its voids (Schroeder and Dickenson, 1996). Upon placement it is desirable to have a fill drain as quickly as possible so that the fill material can settle and reach its operating density. This process of drainage is dependent upon the permeability of the fill. The importance of permeability can be seen in the fact that hydraulic mine backfill must be dewatered from a moisture content of 30-40% (by weight) to around 20% before being able to perform its ground support role (Thomas, 1978). The processes by which dewatering occurs are decanting, exudation, and percolation, all of which are dependent upon the permeability of the fill.

It is common practice to specify fill permeability in terms of percolation rate. From Darcy's Law the coefficient of permeability can be defined as (Smith, 1990):

$$k = \frac{Q \times L}{A \times \Delta h} \quad [2.5]$$

where,

k = The coefficient of permeability (ft/s)

Q = The volume rate of flow through the fill (ft³/s)

A = The cross-sectional area of the fill (ft²)

Dh = The head differential across the fill (ft)

The percolation rate is the same as the coefficient of permeability except that the hydraulic gradient, $\Delta h/L$, is equal to unity. Therefore, the percolation rate of any fill material can be written as:

$$k_p = \frac{Q}{A} \quad [2.6]$$

where,

k_p = The percolation rate of the fill (ft/s)

Q = The volume rate of flow through the fill (ft³/s)

A = The cross-sectional area of the fill (ft²)

The percolation rate of any fill applies to the flow of a unique fluid at a unique temperature and is a combined property of the fill material and the fluid flowing (Thomas, 1978). Since the percolation rate is dependent on the fill material, the grain-size distribution and the porosity affect the drainage behavior of the fill. As for the fluid flowing through the fill, in this case water, the viscosity and degree of saturation of the fill also influence its dewatering capabilities.

The size of fill particles determines the size of individual pores (Terzaghi, 1996). The grain-size distribution of a fill gives an indication of the size and “connectability” of the pores available for transporting water. Basic predictions of percolation rates through fill can be made using the Hazen relationship for filter sands. The relationship is based on data obtained from the grain-size distribution of the fill and is as follows (Hausman, 1990):

$$k_p = C(D_{10})^2 \quad [2.7]$$

where,

k_p = The percolation rate of the fill (in/s)

D_{10} = The effective grain size (in)

The effective grain-size is taken from the size distribution data at the “10% finer” point. The constant, C , can be related to the uniformity coefficient of the fill, also calculated from the size distribution data (C typically around 100). Equation 2.7 indicates that a fill with a large percentage of fine material will have slow drainage capabilities. Extreme

fines, called slimes, are particles that are –38 microns in size; their presence causes a considerable reduction in percolation rates (Herget and De Korompay, 1978). In order to prevent blockages among the pores, and reduced percolation rates, slimes must be classified out of fill material before placement.

Reductions in the void ratio of the fill, due to compression and consolidation, also act to lessen the percolation rate of the fill. As the volume of the connected voids decreases due to particle consolidation, both the porosity of the fill decreases and its capacity to drain decreases. The influence that porosity has on percolation rates is represented in the following formula (Pirson, 1950):

$$k_p = \frac{n^3}{5(S_v^2)(1-n)^2} \quad [2.8]$$

where,

k_p = The percolation rate of the fill (in/s)

n = The porosity of the fill

S_v = The total grain surface per unit volume of reservoir

Equation 2.8 is not meant to be used to calculate percolation rates; rather, it only illustrates how a change in porosity can effect a fill's percolation rate. Since the total grain surface is inversely proportional to the grain size, it can be seen that for a given grain size, the change in percolation rate due to compression will depend primarily on porosity (Herget and De Korompay, 1978). If the porosity of a fill were reduced during compression by 10%, the percolation rate of that fill would decrease by 40%, perhaps leaving the fill incapable of dewatering properly.

The viscosity of the water and the degree of saturation of the fill also tend to have an affect on the percolation rate. The viscosity of water decreases as the temperature decreases; thus the underground environment of the backfilled area must be considered when calculating percolation rates. Fills with higher levels of saturation tend to have higher percolation rates (Lambe and Whitman, 1969). A reduction in saturation of about 10% can result in a percolation rate decrease of 50% (Herget and De Korompay, 1978).

It is important that a fill dewater quickly so that it can begin to support the underground workings. Effective drainage results in the dissipation of pore water

pressures and speeds up the process of primary consolidation allowing the fill to settle and densify, enhancing its stability.

2.3 Cemented Backfill

Cementing agents, or binders, are commonly added to backfill materials in order to manipulate their behavioral characteristics. The addition of chemical binders to tailings backfill produces materials of relatively high initial stiffness. This stiffness allows fill materials to stand freely upon pillar extraction and yields cemented fills that can have improved strength characteristics (Lamos and Clark, 1989).

2.3.1 Components of Cemented Fill

The main component of cemented fill is the fill material itself. It has been pointed out that most fill materials are comprised of waste materials generated during the processes of mining and mineral processing. The largest component of this waste is mill tailings. Cementation of the fill occurs as bonds are formed between fill particles at grain contact points.

Many different types of binding agents are used, the most common being portland cement. Portland cement consists primarily of calcium, silicon, aluminum, and oxygen which together hydrate with the water present in a fill to form a cemented material (Coates, 1981). As hydration of the cement occurs a precipitate is formed which adheres to the surfaces of the fill particles. The hardening of the precipitate transforms the fill from a soil to a soil-cement. The two most important strength producers in portland cement are tricalcium silicate and dicalcium silicate. Tricalcium silicate hydrates first and is responsible for the early strength of the cemented fill (Alymer, 1978). The hydration of the tricalcium silicate also yields hydrated lime which, when combined with pozzolanic admixtures, produces additional cemented materials.

Admixtures containing pozzolanic materials not only take advantage of the excess lime; they are also used to curb costs by reducing the amount of Portland cement needed for stabilization. Fly ash and smelter slags are the most popular pozzolans used as admixtures. Fly ash is a product of coal combustion, while smelter slags are the by-

products of metallurgical processes. When mullite, a glassy component of fly ash, reacts with water and lime, cementitious calcium silicate and calcium aluminosilicate are formed (Hausman, 1990). The process is much the same for smelter slags. Pozzolanic properties of ferrous slags are created when the slags are cooled to a “glassy” and highly vitreous state by rapid quenching with water (Atkinson et al., 1989). In this state, smelter slags will react with water and lime, further stabilizing the fill.

In addition to portland cement (or any admixture) and the fill material itself, the final ingredient in cemented fill is water. Water is necessary to ensure that proper hydration of the cement occurs. If proper hydration of the cement does not occur, the fill will not meet its required strength and stiffness. Since tailings backfill is fairly saturated to begin with and more water is usually required to pump it underground, the water content of tailings backfill is always in far excess of that required for hydration of the portland cement (Brady and Brown, 1992). The main concern then is the pH of the water and the amount of sulfate salts present in the water. Acidic water and sulfate salts can attack the cement bonds within the fill, leading to a loss of strength, durability, and stability.

2.3.2 Properties of Cemented Fill

As with uncemented fill, certain properties of cemented fill are used to characterize its behavior. However, soil mechanics alone cannot describe the properties of cemented fill. The addition of cement to fill transforms the fill from a soil to an engineered material. The addition of cement and other pozzolans, and their compositions, make the process of fill evaluation more complex. Detailed studies and laboratory works have been carried out in order to examine cemented fills and the properties that will be important in their design. Past work has shown that the important properties of cemented fill are (Knissel and Helms, 1983):

1. Strength (i.e., uniaxial compressive strength)
2. Deformation behavior
3. Cohesion and angle of internal friction
4. Density and porosity
5. Consistency of the mixture

As with uncemented fill, strength is the main concern for cemented fills. The properties of the fill material such as friction angle, density, and porosity have the same effect on a cemented fill's strength as they do for uncemented fills. However, for an uncemented fill, importance was placed on understanding the drainage behavior of the fill material and its percolation rate, which is not the case for cemented fills. With the addition of cementing agents to a fill material the permeability, of cemented hydraulic fills is rarely a concern (Thomas, 1978). Furthermore, fill dewatering is assisted by cement hydration and the possibility of fill liquefaction is eliminated. Compressibility is also not found among the important properties of cemented fill. Although cemented fills do compress, they tend to provide relatively high strengths at low strain values, such that consolidation is dependent mainly on plastic deformations, or creep, of the cemented fill. This is in opposition to the immediate, elastic settlements that occur in uncemented fills. Disregarding compressibility and percolation, examination of cemented fill properties focuses mainly on the effects of cement addition.

The purpose of cement addition is to improve the strength properties of the backfill material. The most noticeable increase in strength between an uncemented fill and one treated with cement is in the shear strength. Cement bonds that form between fill particles provide a cohesive component to the fill's shear strength which is absent in an uncemented fill. The shear strength of a cemented fill can be expressed using the following equation (Terzaghi et al., 1996):

$$S = s \tan f + c \quad [2.9]$$

where,

c = The cohesion of the cemented fill (lb/ft²)

Table 2.2 indicates the improvement that cement addition can have on a fill. Specimens consisting of a mixture of portland cement and sandfill were tested for their cohesion after curing times of 7 and 28 days.

Table 2.2 - Cohesion of Cemented Fills (from Thomas et al., 1979)

Cement Content (wt %)	Curing Time (days)	Specimens Tested	Cohesion (MPa)
0	205	11	0.03
4	7	22	0.13
	28	23	0.15
8	7	24	0.24
	28	24	0.31
16	7	24	1.02
	28	24	1.46

Table 2.2 not only indicates an increase in cohesion with increasing cement content, it also indicates an increase in the cohesion with curing time. In addition to cement content and curing time, other factors such as fill density, water content, temperature, and humidity have been shown to influence the strength of cemented fills.

The cement content greatly influences the properties of cemented fill. Table 2.2 was already noted as showing increasing cohesion with increasing cement content. It has also been determined that the compressive strength, regarded as the single most useful measure for comparing the relative strengths of cemented backfills, increases with cement content (Lamos and Clark, 1989). Mitchell (1976) described the unconfined compressive strength of cement-stabilized soils as increasing linearly with cement content (Cernica, 1990). A similar relationship has been shown to exist for cement-sandfill mixtures used underground, as seen in Figure 2.2.

Although the cement content contributes directly to the strength of the fill, the final amount of cement is controlled more by economics than by design strength parameters. Portland cement addition represents a significant cost component, particularly in richer mixes that contain 10% cement by weight (Thomas, 1973). Less important than economics is the problem of shrinkage cracking. Hydration of the cement and drying of the fill may cause cracking to occur within the fill, thus weakening it. Shrinkage cracking is common in cement-stabilized soils where an upper limit of 10% cement is typical (Cernica, 1990).

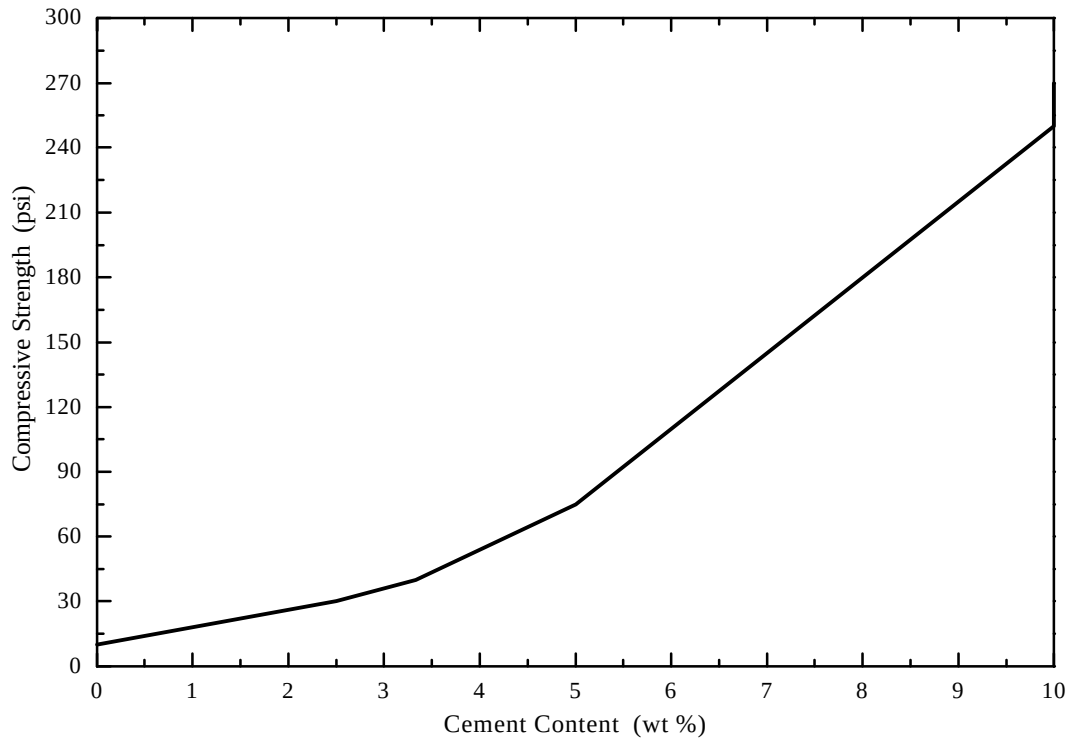


Figure 2.2 - Unconfined Compressive Strength vs. Cement Content

(After Crandell, 1992)

The curing time, temperature, and humidity have also been shown to have an effect on cemented fills. As with normal concrete, the strength of a cemented fill increases with curing time. Knissel and Helms (1983) tested specimens of differing water/cement ratios and found that ultimate fill strength is achieved after long periods of up to 90 days. Elevated curing temperatures also accelerate the gain in strength of cemented backfills provided that full saturation is maintained and the temperature does not exceed a critical level. Thomas (1969) stated that for common backfills, a curing temperature of 50°C is a safe limit (Lamos and Clark, 1989). As opposed to time and temperature, the strength of cemented fill decreases with humidity. Compressive strength increases more rapidly when dry cured rather than wet cured (Corson, 1970). Although these factors affect the strength of cemented fills, they may be impossible to control due to the underground environment. It is important to note that, excluding curing time, few mining environments will be able to provide the ideal hot and dry environment that can help improve a cemented fill's strength.

As with uncemented fill, some soil properties of cemented fill are related to the strength of the fill. These properties include density, porosity, and water (moisture) content. As with uncemented fill, it is desirable to use a dense fill, one with as low a porosity as possible. McCreedy and Hall reported that any increase in pulp density of the fill yields a significant increase in strength. Dense fills tend to be more tightly “packed” with less pore space between particles. Therefore the available amount of cement needs to fill less pore space and the cement crystals are of a stronger type. However, the relationship between water content and cemented fill strength is much more ambiguous than density and strength. As explained previously, it is vital that enough water is present so that complete hydration of the portland cement can occur. Yet numerous lab studies have shown that compressive strength decreases with increasing water content (Lamos and Clark, 1989; Knissel and Helms, 1983). Knissel and Helms (1983) concluded that moisture content is the single most important factor governing the strength of any cemented fill.

Laboratory work has also indicated that the grain-size distribution of a fill material is important in optimizing its strength. Modification of a fill’s grain-size distribution is a useful tool for improving a fill’s strength characteristics. In theory, a certain gradation of particle sizes is desirable to produce a more dense, minimum void mix (Alymer, 1973). The “desirable” size distribution can be described as well-graded, containing coarse to fine particles. Thomas (1971 and 1983) indicated that as the fines content of a fill material was increased, the strength increased. Figure 2.3 illustrates the effect that fines addition can have on compressive strength. Three cemented fills, prepared from deslimed tailings and distinguished according to their grain-size distributions, were tested for strength. Although comparable, the strengths of the medium and coarsely graded fills are less than that of the well-graded fill that contains more fine particles. Despite the strength improvement, the amount of fines may also have detrimental effects. Material finer than a No. 200 sieve may form a coating on the coarser particles, weakening the bond between the cement and those particles. The surface area of fine grained material is large in comparison to coarse grained material; therefore more water is required to “wet” the surface grains, insuring that proper cement

hydration occurs. The need for more water may limit the improvement created by fines addition.

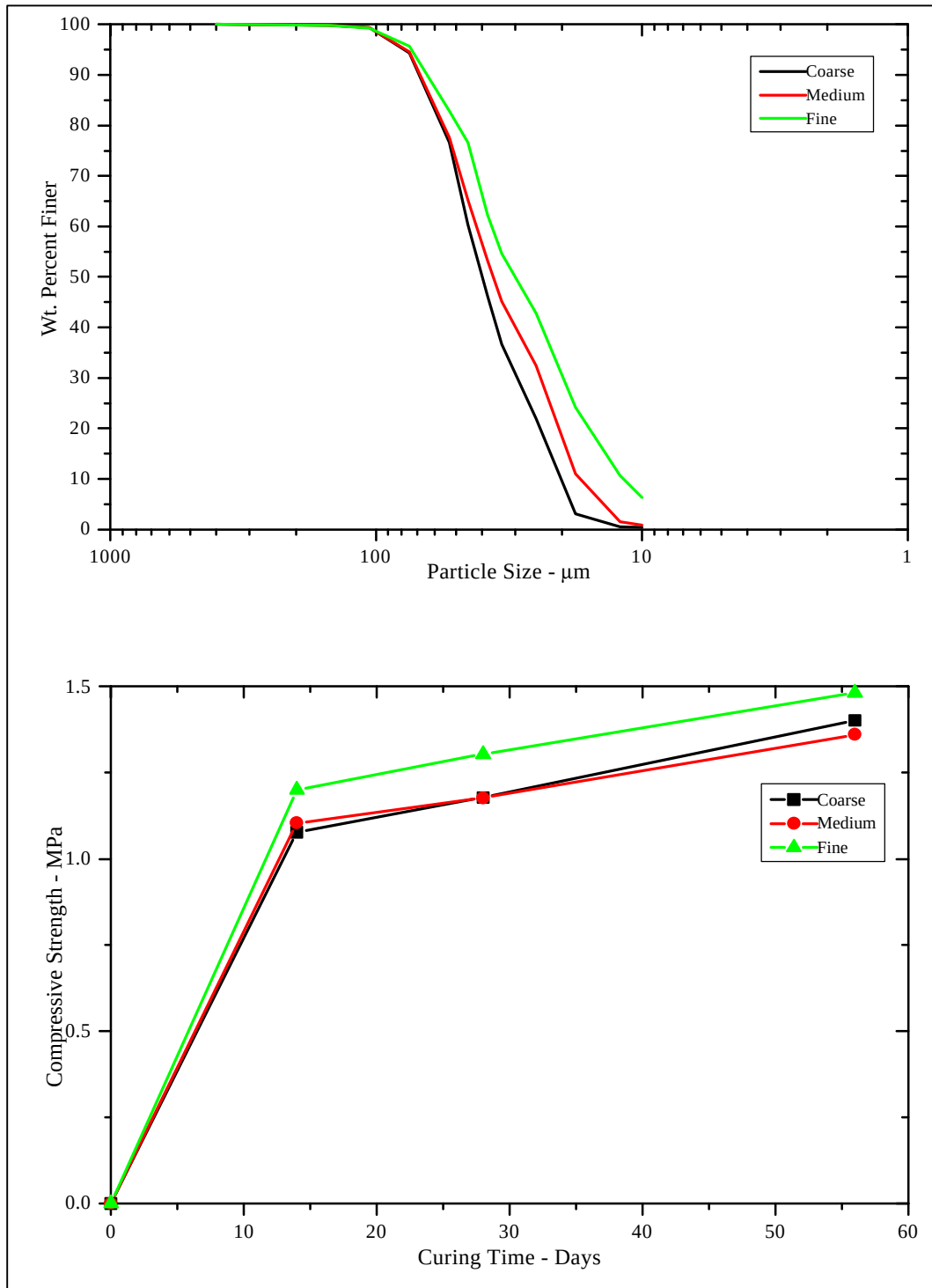


Figure 2.3 – Effect of Grain Size on Strength
(After Thomas, 1973)

Also detrimental to the cement bonds is the presence of certain harmful substances. These impurities include organics, silt, clay, shale, coal, and lignite. Organic materials may delay the setting and hardening of the cemented fill, reduce strength gain, and cause deterioration. Coal and lignite will affect the cemented fill's durability if they are present in large amounts (Kosmatka and Panarese, 1994). Reducing the amounts of impurities is desirable but limited by the type of fill material that is accessible and its composition.

As mentioned earlier, some of the portland cement can be replaced by pozzolans. The main purpose of using pozzolanic materials is to reduce portland cement costs. The two most popular pozzolanic materials used in backfill operations are fly ash and granulated blast-furnace slags. Both react with calcium hydroxide released during the hydration of portland cement to form cementitious compounds, contributing to the strength development of the fill by removing the weakening effect of the calcium hydroxide (Thomas and Cowling, 1978). Extensive testing has indicated that there are differences in strength gain between cement-only and cement plus pozzolan fills. The strength gain is dependent on the type of pozzolanic material used, the amount of the pozzolanic material (by weight) added, and the curing conditions. Generally, the strength development of a fill with fly ash or slag is similar to a fill with cement (Kosmatka and Panarese, 1994). However, low to moderate cement replacement with reactive (high calcium content, Class C) fly ash in a fill can lead to increased 28-day strength. Test work with fly ash at O'okiep Copper's Carolusberg Mine indicated that test specimens containing 6.5% fly ash and very little cement reached a "target" strength 60 days earlier than specimens containing 6.5% cement (Atkinson et al., 1989). Atkinson's work also showed that the use of modified copper slag as a cement substitute produced a fill that reached target strength about 55 days sooner than a cement-only fill. Also, when the curing temperature of the slag fill was increased from 22°C to 40°C, another 15 days was cut from the curing time. Correspondingly, increasing the weight percentage of pozzolanic material increases the compressive strength of the fill. Thomas and Cowling (1978) concluded that increasing copper slag addition is consistently beneficial at cement contents of 3% and greater.

Further testing on a copper-nickel slag supported the trend of increasing the slag content in order to increase the strength (McGuire, 1978). Those tests also recommended that the slag-cement ratio remain at 1 although an increase would mean greater fill strength. The recommendation was based on the fact that some furnace slags will have to be modified in order to provide advantageous characteristics. The copper slag from Carolusberg Mine was modified by increasing the amount of limestone added to the reverberatory furnace charge. This process increases the calcium oxide content of the slag, and its lime to silica ratio, making it more reactive and “more” pozzolanic. This type of process or others such as grinding and quick quenching may be necessary in order to enhance the admixture’s pozzolanic properties. Performing these procedures incurs costs that in turn limit the extent of pozzolanic addition. Thus, the use of pozzolans, which certainly enhances fill strength, is mainly a decision based on economics rather than one concerned with maximizing fill strength.

2.4 Coal Waste Backfill

Backfilling active coal mines is technically feasible but the process has had limited applicability due to economic constraints. In order to have any applicability, backfilling of coal mines will have to be done using the most freely available fill material: coal waste generated during the mining and coal preparation processes. Additionally, information about the properties of coal waste and the factors of influence are necessary if backfilling of coal mines is going to be made economic (Knissel and Helms, 1983). The previous sections have pointed out the important fill properties and the factors that influence them. Furthermore, for cemented fills, the influence of cement and other admixtures on those properties was discussed. Evaluation of coal waste also focuses on these properties and what influences them, but property ranges and specific physical interactions can be defined more clearly since a specific fill material has been chosen.

2.4.1 Properties of Coal Waste

Particle size and grain size distribution help to define the types of coal waste generated during the mining and mineral processing stages. Although coal waste can have a broad distribution of particle sizes, most waste can be divided into either coarse coal refuse or fine coal refuse. Fine coal refuse is generally classified as material smaller than 0.7 mm and is usually collected from the slurry or tailings stream of the coal preparation process (Stewart and Atkins, 1981). Coarse coal refuse is then the material greater than 0.7 mm and up to about 74mm (3”) in size, and is separated from the fine material during coal preparation. Figure 2.4 depicts a typical grain size distribution curve for coal waste taken from a coal preparation plant in Australia.

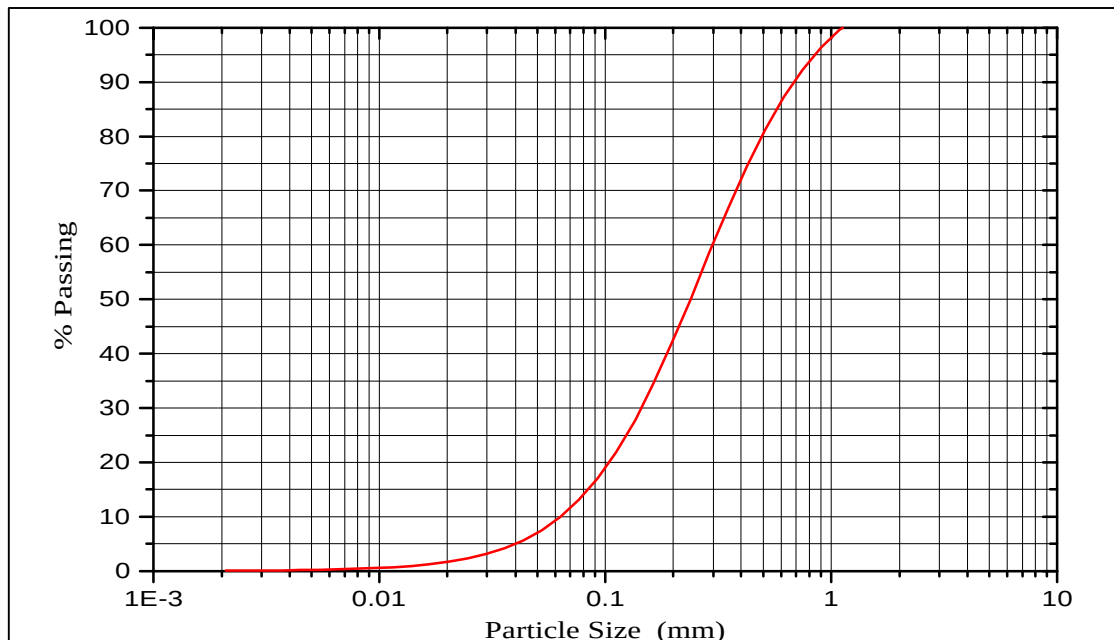


Figure 2.4 – Typical Size Distribution of Coal Waste
(After Indraratna et al., 1994)

Although the distribution curve in Figure 2.4 is unique to the mine from which the waste was taken, it falls within a distribution range proposed by Taylor (1987) and shown in Figure 2.5.

The upper and lower boundaries set forth by Taylor are distributions for cyclone overflows/flotation rejects and cyclone underflows, respectively. Figure 2.5 indicates that cyclone underflows typically contain fine gravel to fine sand particles, whereas the overflows usually contain coarse silt to fine silt sized particles. The distribution of a coal

waste falling within those boundaries, like the one in Figure 2.4, will typically consist mainly of fine sand and coarse to medium silt particles. Therefore, as a soil, coal refuse can be classified as silty sand or as a poorly graded sand-silt mixture (SM by Unified Classification System).

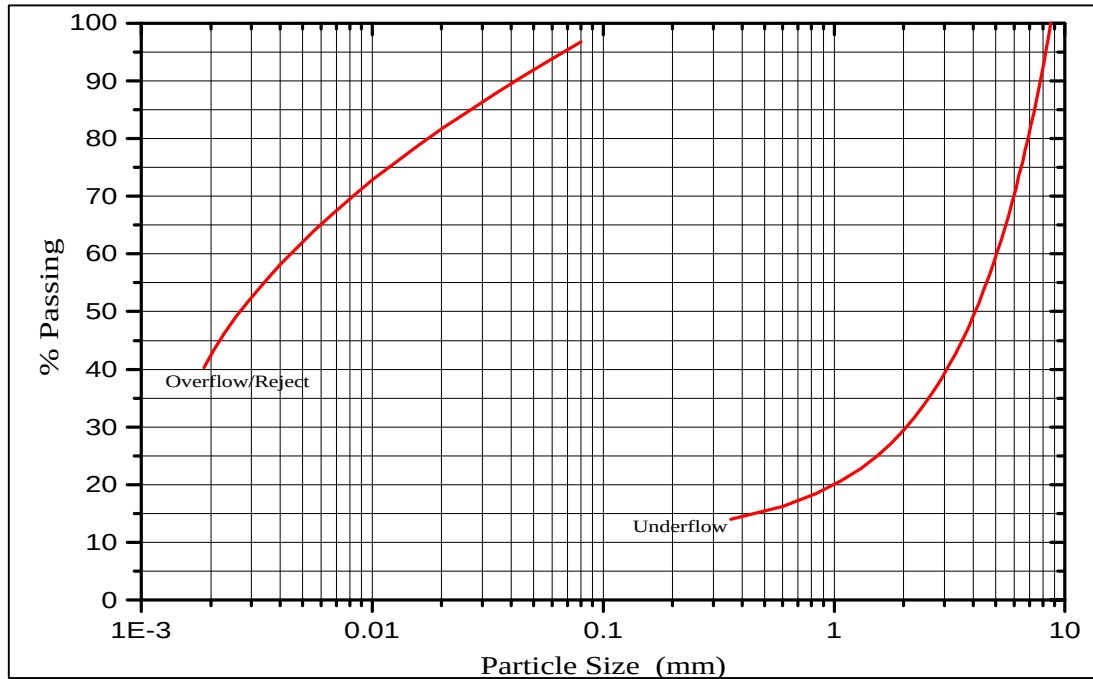


Figure 2.5 – Upper and Lower Bounds for Coal Waste Distributions
(After Taylor, 1987)

Since coal waste is classified as a sand-silt mixture, it is assumed to be cohesionless. Therefore, the shear strength of uncemented coal refuse is dependent on its internal angle of friction. As mentioned earlier, f is influenced by density, mineral content, particle size and shape, and grain size distribution.

Coal refuse exhibits a wide range of densities due to physical differences between the various coals mined around the world, and because of the various coal washing techniques practiced. Fell (et al., 1992) specified a range of specific gravity for Eastern U.S. coal tailings of 1.5 to 1.8. Other data has indicated a more “universal” range of 1.4 to 2.6, with a mean of about 2.2 (Indraratna et al. 1994, Nandy and Szwiliski, 1987). Since the wide range of specific gravities is partly due to differences in coal cleaning processes, it is inherent that the density of coal waste can be modified during the washery process. The greater the amount of coal particles removed during cleaning, the greater the

specific gravity of the remaining waste (Indraratna et al., 1994). Coal particles tend to have a lower specific gravity, around 1.3, than the other material present in tailings, therefore their removal increases the overall density of the waste. An increase in coal refuse density tends to increase the friction angle of the refuse and its shear strength.

Along with residual coal, coal refuse can consist of a wide variety of minerals. Although coal wastes vary differently from one coal basin to another, the predominant mineral components of most coal wastes are clay rocks, such as kaolinite, illite, chlorite, etc. (Canibano and Leininger, 1987). Quartz is also present, mainly in the coarse fraction, along with pyrites and carbonates. Bland (et al., 1976) sampled fine, coarse, and slurry refuse from Kentucky and found the following mineral compositions:

Table 2.3 - Average Mineralogy of Kentucky Coal Waste
(from Bland et al., 1976)

Mineral	Average %	Range %
Illite	37.86	16.42-50.39
Kaolinite	34.39	16.06-72.45
Quartz	16.40	7.56-31.88
Pyrite	3.89	0.47-11.81
Hematite	2.45	0.00-14.30
Calcite	1.72	0.00-16.38
Magnesite	0.75	0.00-3.27
Apatite	0.56	0.11-3.83

Table 2.3 indicates that the main constituents of the refuse are the clay minerals illite and kaolinite. Since clays have low friction angles, the presence of these minerals will tend to decrease the shear strength of the coal tailings. Clay minerals also attract water, which has a tendency to surround the clay minerals, decreasing frictional contact between particles and shear strength (Terzaghi et al., 1996). If quartz is present in fairly large percentages it may help to strengthen the refuse. The surfaces of quartz particles are very rough and contact between those rough surfaces provides increased frictional resistance to sliding and shear.

The shearing resistance of the tailings is also dependent on the size and shape of the clay and quartz minerals. Angular particles such as quartz interlock and provide more frictional resistance than thin, platy clay particles that tend to interact face-to-face and

have less frictional resistance. In general, most coal waste particles are long and flat with some degree of angularity, exhibiting both face-to-face and interlocking behavior (Knissel and Helms, 1983 and Indraratna et al., 1994).

The other two important soil properties of coal waste are compressibility and permeability. Along with shear strength they are helpful in determining the engineering behavior of fine and coarse coal waste.

Consolidation tests have been performed on coal wastes to assess their compressibility under large vertical loads. In general, when vertical loads are applied to coal waste, drainage of water from the waste occurs fairly rapidly, allowing for the dissipation of pore water pressures and immediate settlement of the waste. Williams and Kuganathan (1992) found that pore pressures usually dissipated within 2 hours of the application of vertical stress and that after full dissipation there was little on-going settlement of the coal waste. Therefore, after initial settlements have occurred, the long-term settlements of the waste are assumed to be negligible and are usually ignored (Indraratna, 1994). Since long-term settlements of coal waste are neglected, the overall settlement of the waste can be estimated using Equation 2.4 (given earlier).

The compressibility of coal waste is very much dependent on the waste's drainage capacity, or permeability. An approximate range of coal waste permeability, based on data collected from similar fine-silty-sands, is 0.1×10^{-4} to 20×10^{-4} cm/s (Cernica, 1992). This range of permeabilities is consistent with results taken from actual laboratory tests on coal waste that indicate a range of 0.02×10^{-4} to 16×10^{-4} cm/s (Bowman, 1992; Chen and Baker, 1978; Drnevich et al., 1976; Indraratna et al., 1994; Williams and Kuganathan, 1992). It was pointed out earlier that Hazen's Relationship could be used to estimate permeabilities for soils and fills based on data taken from the grain-size distribution of the fill. However, Drnevich (et al., 1976) found that Hazen's relationship grossly overestimated the permeability of the mine spoil materials. Chen and Baker (1978) also found that relationships between permeability, void ratio, and particle size commonly used in soil mechanics do not apply to coarse and fine coal waste. Using their measured results and regression analysis, Chen and Baker developed relationships between permeability, void ratio, and particle size for coarse and fine coal refuse. For coarse refuse the relationship is:

$$k = 393e^{3.631}(D_{50})^{1.311} \quad [2.10]$$

where,

k = permeability (in/s)

e = void ratio

D_{50} = average particle size (in)

For fine coal refuse the permeability can be expressed as:

$$k = 2033e^{3.486}(D_{50})^{1.235} \quad [2.11]$$

Equations 2.10 and 2.11 illustrate the dependence of k on void ratio. As the coal waste is compressed and its void ratio decreases, its permeability will also decrease. If the permeability becomes too low to permit adequate drainage, pore pressures may develop within the refuse and effect the strength and compressibility behavior of the coal refuse. However, it is also beneficial that the permeability decreases as compression increases because it is a process that helps to prevent the generation of acid mine run-off.

Decreasing void ratio and permeability help to isolate oxygen and water from acid generating ingredients, such as pyrite, and thus can reduce acid mine drainage (Chen and Baker, 1978).

2.4.2 Cemented Coal Backfill

As with other fill materials, it may be necessary to stabilize and strengthen coal waste with cementing agents. In general, cement addition will have the same effects on coal waste as it does on other fill materials (see previous section). The strength of the coal waste will be increased and its support capabilities enhanced. The relationships developed in the previous section between fill properties and fill strength also hold true for cemented coal waste. However, the gain in strength upon cement addition for coal waste is lower than for other fill materials due to the mineral and chemical composition of the coal.

Organics and other impurities are more prevalent in coal and tend to affect the strength and durability of cemented coal. The most notable impurity is clay. Fine clay particles can weaken the bond between the cement paste and the coal waste by forming a thin coating on coarser particles. The presence of clay particles, which tend to attract and

adsorb water, increases the water requirements for mixing, thus reducing the strength of the cemented coal.

The durability of cemented coal is also less than that of other cement mixtures. Low density materials, such as coal, can cause pop-outs on the surface of the mixture and can also break down and disintegrate. Poor durability upon exposure to weathering elements may limit the use of cemented coal to mines where normal temperature (55° F) and high humidity are nearly constant (Bland et al., 1976).

Resistance to, and the effects of, chemical attack is another concern. The production of sulfuric acid as a consequence of pyrite alteration is common to coal mines. Sulfate attack on low durability cemented coal waste will decrease its support capabilities. Provisions may have to be made for limiting the amount of sulfate exposure after placement or for a mix design capable of providing more chemical resistance.

Laboratory work has been performed in order to determine and evaluate the strength characteristics of cemented coal. As with typical concrete mixes, the unconfined compressive strength of cemented coal is dependent upon water-cement ratio, curing time, and cement content. Knissel and Helms (1983) used samples of coal waste mixed with portland cement to illustrate this relationship, depicted in Figure 2.6.

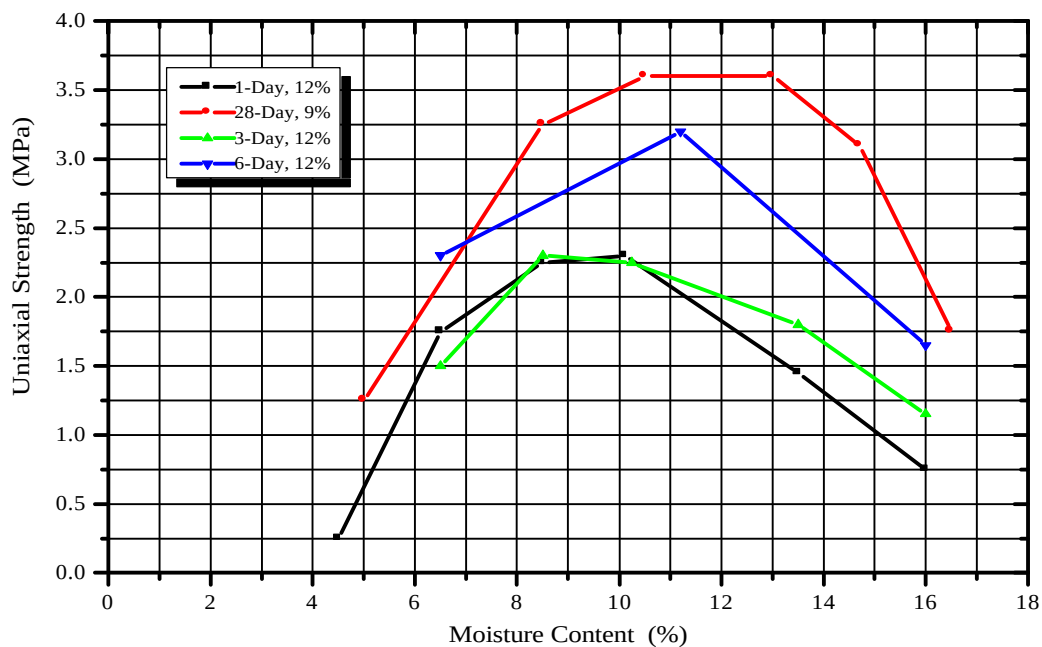


Figure 2.6 – Strength of Cemented Coal Refuse
(After Knissel and Helms, 1983)

Bland (*et al.*, 1976) found that when compared to typical concrete mixes, the compressive strength of cemented coal waste is about 40% to 80% lower. Bland was also able to consistently produce cemented coal samples with 28-day compressive strengths between 15-22 MPa (2000-3000 psi). However, these samples were “altered” in a few ways. The coal refuse was “washed” over sieves in order to remove a portion of the finer material that tends to coat the coarser material. The grain size distribution of the coal waste was changed by adding natural river sand (25% by total dry weight). The sand reduced the water requirement and made the mix more workable.

Thomas (*et al.*, 1987) tested 16 cemented coal samples, each containing 10% cement, and measured an average 14-day strength of 2.75 MPa (360 psi). The highest strength measured was 5.87 MPa (770 psi), which coincided with the highest dry density. Thomas’ results also showed that there was a tendency for coarser, less plastic waste to have higher compressive strengths. In addition, the study concluded that the short-term durability of cemented coal waste is controlled mostly by the properties of the raw refuse such as: size distribution, plasticity, slaking resistance, and sulfate content.

The preceding investigations indicate that cemented coal can develop a good amount of strength dependent on the refuse’s soil properties, the mix conditions, and the mix design. Without considering costs, it is feasible that cemented coal waste can be used in room and pillar coal mines in order to increase recovery. Pillar sizes can be decreased according to the amount of additional support provided by the cemented waste. Methods of quantifying the amount of additional support and relating it to pillar design are covered in the following section.

2.5 Support Mechanisms of Backfill

The mechanical effects of fill are not the same as those of primary coal pillars. Research and in situ testing have shown that fill cannot rigidly support the total weight of overburden and acts only as a secondary support system (Cai, 1983). Since the fill mass cannot support vertical loads and does not act as a “stand-alone” pillar, its mechanical effect must be defined in another way.

When fill is placed in an open void underground it does not come in immediate contact with the roof of the opening. As the roof begins to deform and displacements become larger than the “gap” between the fill and roof, the fill mass will begin to provide passive support. Likewise, when pillars begin to deform into the filled void the fill mass will provide lateral passive resistance. Passive resistance is defined as a state of maximum resistance mobilized when a force pushes against a soil mass and the mass exerts a resistance to the force (Hunt, 1986). This resistance is typically referred to as passive earth pressure and can be quantified according to the following equation (Thomas, 1979):

$$s_h = g \times h \times K_p \quad [2.12]$$

where,

s_h = earth pressure acting horizontally (lb/ft²)

g = unit weight of the fill (lb/ft³)

h = height of the fill (ft)

K_p = coefficient of passive earth pressure.

Equation 2.12 is valid for cohesionless fills and neglects any surcharge loading and frictional forces between the fill mass and pillar. The coefficient of passive earth pressure is equal to:

$$K_p = \tan^2 \left(45 + \frac{f}{2} \right) \quad [2.13]$$

where,

f = angle of internal friction of the fill.

For partially cohesive fills, or fills that exhibit some sort of effective cohesion, the passive earth pressure can be written as (Barker et al., 1991):

$$s_h = g \times h \times K_p + 2 \times c \times \sqrt{K_p} \quad [2.14]$$

where,

c = cohesion of the fill (lb/ft²).

It is important to note that these equations were developed using soil mechanics and may be limited in terms of adequately expressing the mechanics of fill loading due to rock deformations (Thomas, 1979).

Cai (1983) developed a formula for cemented fill that differs from Equations 2.12 and 2.14. He argued that since cemented fill has self-supporting characteristics, the lateral pressure it applies is evenly distributed along the full height of the pillar. The s_h calculated in Equations 2.12 and 2.14 is dependent on the weight of the fill and increases linearly from the top of the fill (where s_h is nearly zero) to the bottom of the fill (where s_h is maximum). Figure 2.7 is an illustration of the two pressure distributions along a pillar.

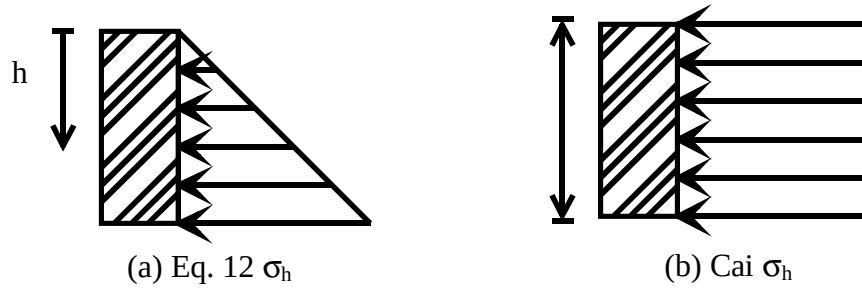


Figure 2.7 – Difference Between σ_h Distributions

Based on this distribution, and neglecting arching effects, Cai developed the following the following equation:

$$s_h = n \times \frac{g \times H \times a}{K_{p1} L} \quad [2.15]$$

where,

n = correction coefficient

g = unit weight of the overburden (lb/ft³)

H = overburden depth (ft)

a = width of open space (ft)

K_{p1} = coefficient of passive earth pressure of the pillar

L = width of the adjacent pillar (ft).

It was stated earlier that fill cannot rigidly support the total weight of the overburden. Although it may not seem that way, this is still the case for Equation 2.15. s_h is equal to the stress increment distributed to a pillar after mining and the stress increment is equal to the overburden load above the mined out space. The fill does not take any vertical load directly but rather relieves the pillar of the additional stress by providing lateral restraint.

The formula was evaluated by comparing calculated fill strengths, using Equation 2.15, with measured fill strengths from two mines in China. In one mine the calculated fill strengths correlated well with measured strengths; in the other, the calculated strengths were slightly smaller than the actual in situ strengths (Cai, 1983).

After passive resistance has been mobilized in the fill, the increase in strength of the surrounding pillars will be equal to the magnitude of the passive earth pressure. Thus the strength of the pillar increases according to the following formula (Guang-Xu and Mao-Yuan, 1983):

$$s_1' = s_1 + s_h \times K_{p1} \quad [2.16]$$

where,

s_1' = strength of pillar supported by fill (lb/ft²)

s_1 = strength of pillar (lb/ft²)

s_h = passive earth pressure provided by fill (lb/ft²).

The lateral pressure, s_h , can be calculated using any of the aforementioned equations.

A few conclusions and inferences regarding the support potential of fill can be drawn from Equations 2.12 through 2.16.

1. Passive, lateral restraint of the fill is mobilized upon pillar deformation.
2. The amount of lateral restraint is dependent on the physical properties of the fill and will increase as the density of the fill increases with progressive compression.
3. The development of moderate fill stresses may be adequate to provide a substantial increase in pillar strength. Confinement of a failed pillar can produce a residual strength that is a substantial proportion of peak pillar strength (Cleland and Singh, 1973; and Thomas, 1979).

Kejriwal and Ghose (1986) added to the preceding conclusions by stating that the main action of fill is to prevent the buckling of pillars. Since the maximum displacement due to buckling usually occurs at the center of a pillar, they recommended that the height of the fill should extend to more than one-half the pillar height and preferably to at least two-thirds of the total height. A series of lab tests conducted by Kejriwal and Ghose

indicated that the post-failure strengths of pillars increase substantially with the height of the fill.

The procedure used by Kejriwal and Ghose has been used by others to test the effectiveness of Equations 2.12-2.16. A pillar-backfill system is replicated by embedding “model” pillars, usually rock cores or cured cement mixtures, into fill material contained in a large diameter cylindrical steel mould (Fig 2.8). After the model pillar is embedded

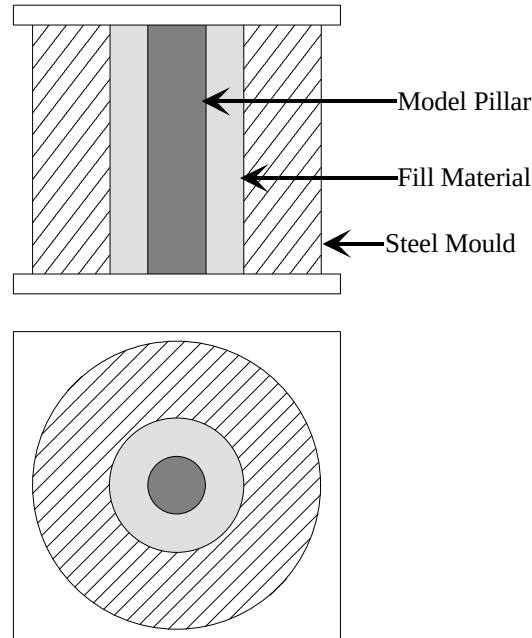


Figure 2.8 – Laboratory Model of Pillar-Backfill System
(After Yamaguchi and Yamatomi, 1989)

into the fill material a vertical load is applied to the composite system. The stress-compression relationship for the model pillar and the relationship between lateral stress in the fill and compression of the pillar are then recorded (Blight and Clarke, 1983). The results of studies employing this type of “mock-up” pillar-fill system verify that the formulas presented earlier correlate well with the actual support behavior of fill (Yamaguchi and Yamatomi, 1989; Blight and Clarke, 1983; Swan and Board, 1989; Aitchison et al., 1973).

In addition to increased pillar strengths and stability, backfilling will also help to reduce surface subsidence above coal mines. The approach to determining the amount of subsidence above a backfilled opening is different from the traditional methods of predicting coal mining induced subsidence. Long-term subsidence above a filled opening

can be estimated using a “beam on an elastic foundation” approach (where the intact roof rock above the fill is considered the beam and the fill is the elastic foundation). Assuming there is contact between the roof and fill, the magnitude of subsidence can be estimated using the following equation (Mitchell, 1983):

$$y_c = s_v / K \left[1 - 2(a+b)/c \right] \quad [2.17]$$

where,

y_c = amount of subsidence

s_v = vertical stress acting on the fill

K = backfill subgrade modulus

a , b , & c = coefficients dependent on roof characteristics.

Equation 2.17 is considered applicable when the depth of overburden is much larger than the span of the filled void. When the span is greater than the overburden depth or when the quality of the roof rock is poor, the amount of subsidence can be estimated by:

$$y_c = s_v / K \quad [2.18]$$

For an explanation on how to determine K , a , b , and c , refer to Mitchell (1983) and Wizniak and Mitchell (1989).

CHAPTER 3. THIN SEAM COAL MINING TECHNOLOGY

Economic extraction of thin seam coal reserves is not possible using conventional coal mining equipment or methods. A new method of extraction, applicable in seams less than 36" thick, is necessary in order to recover a significant portion of the thin seam reserves. Basu (1997) conducted a feasibility study in which two conceptual mining systems were proposed and evaluated for their applicability to thin seam mining. Each system was designed by integrating the separate processes of mining and backfilling and each was designed to remotely mine the coal. One system was deemed more practical and only this system is considered in this study.

3.1 The Self-Advancing Miner

Based upon past attempts, and failures, at mining thin-seams, the Self-Advancing Miner (SAM) was conceived. There were three major problems associated with past thin-seam methods that needed to be addressed by the Self-Advancing Miner:

- (1) Lack of an accurate coal-rock interface sensing device
- (2) Lack of steering control
- (3) Inadequate materials handling system.

These problems were alleviated by the design of the SAM. The Self Advancing Miner can be thought of as a remote-controlled continuous miner. A drum shearer, equal in size to the seam thickness, rides on a "caterpillar" and cuts its way through the coal. A conveyor train is attached to the caterpillar and carries broken coal away from the cutting face. A built in coal-rock interface detection device continuously monitors the direction of cutting. Control of lateral steering is provided by differential rotation of the caterpillar and vertical steering is provided by two hydraulic rams that position the shearer above and below the horizon. Figure 3.1 is a schematic of the SAM and its components.

It can be seen from Figure 3.1 that the stalls will be cut at an angle. The angle of the cut will be about 40° and is required so that the shearer and train conveyor can negotiate turns and corners. This requirement necessitates a different layout of the main

entries and panels (Figure 3.2). Initially, the main entries and panels were to be designed in a traditional manner with panels being developed using 90° cross-cuts.

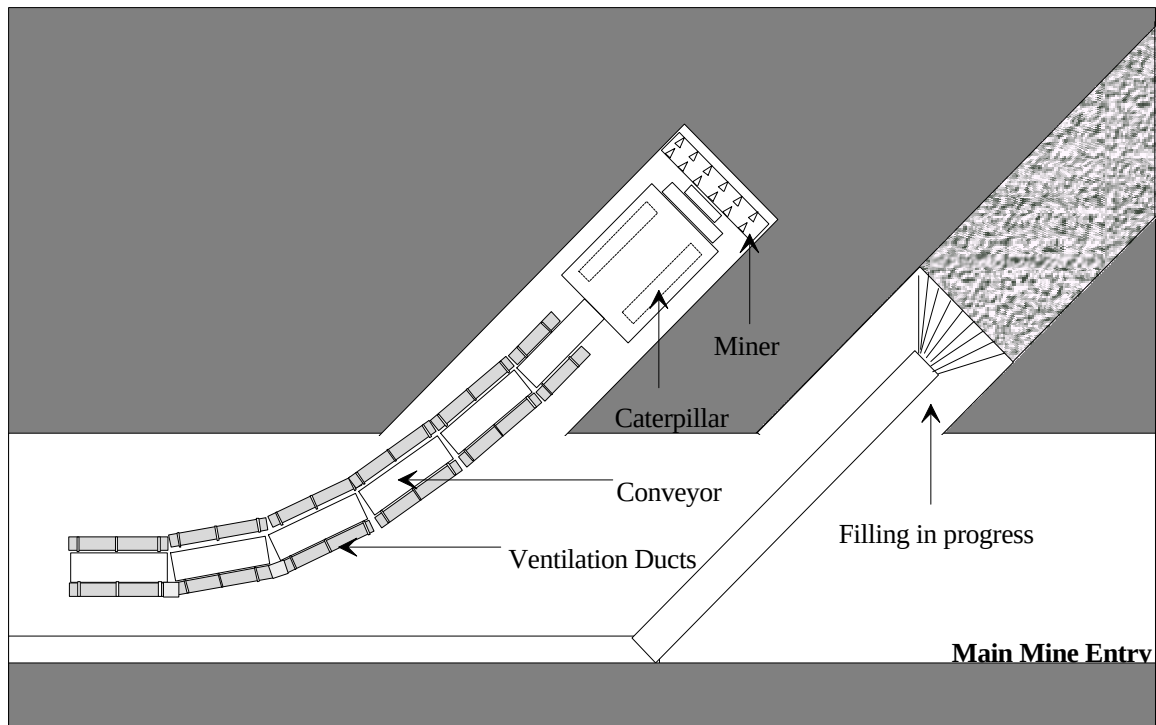


Figure 3.1 – Self-Advancing Miner
(After Basu, 1997)

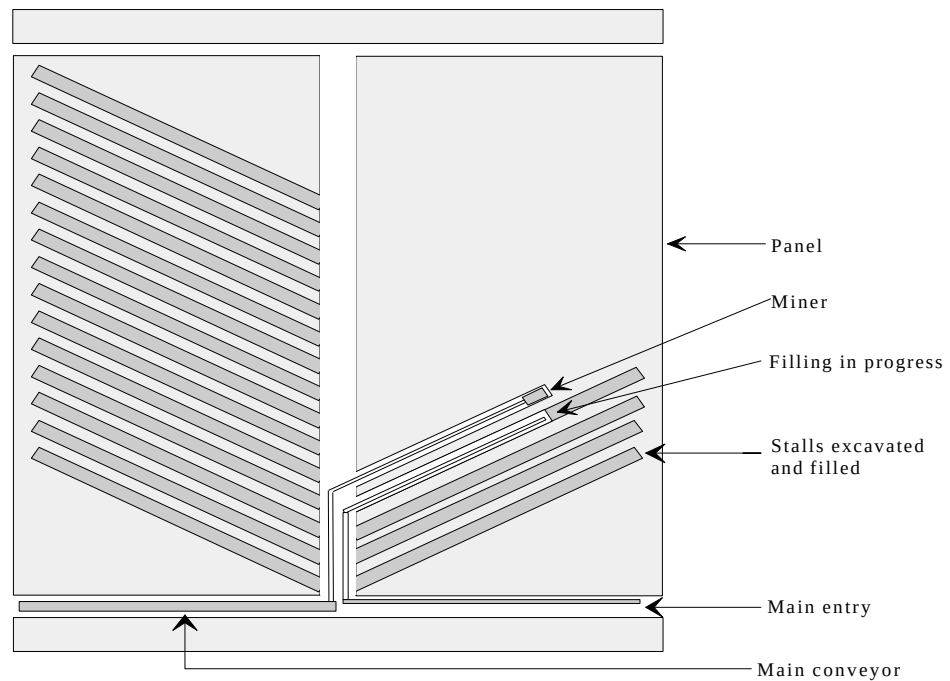


Figure 3.2 – Panel Layout for the SAM
(After Basu, 1997)

Each panel would be 4000 feet long and approximately 475 feet wide. The shearer would move through the main entry into the cross-cut where it would start cutting at an angle of 40° to 45° and to a length of 600 feet. However, it can be seen from Figure 3.2 that this layout requires a large portion of the coal seam to be left in place at both ends of the panel. An alternate layout has been designed which addresses this problem. In this case the cross-cuts are driven at an angle (40° - 45°) in order to develop the panels (Figure 3.3).

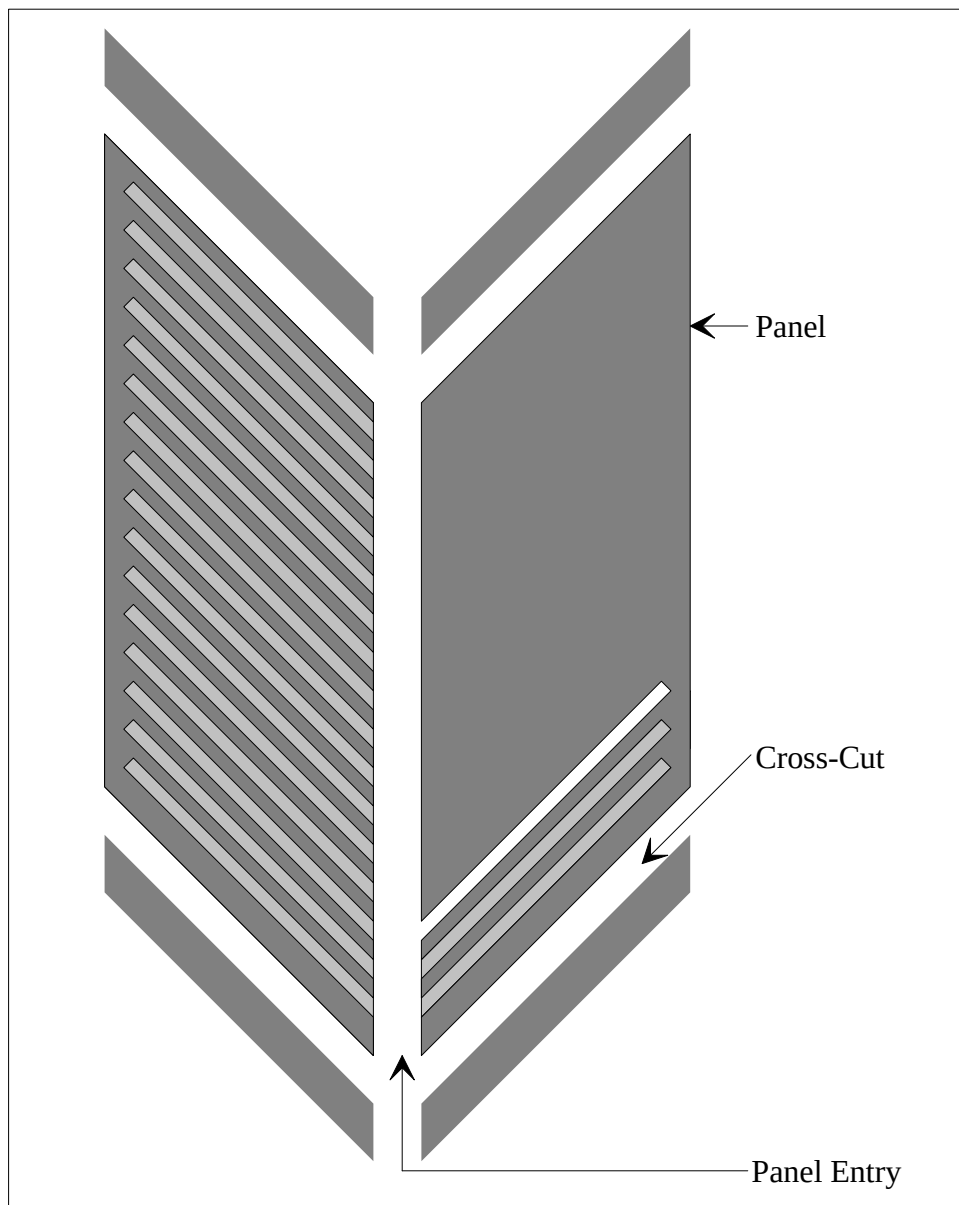


Figure 3.3 – Alternate Layout for SAM

It can be seen from Figure 3.3 that the alternate layout promotes maximum extraction of each panel. This layout is covered in more detail in the following section.

The entries and cross-cuts will be driven using a standard continuous miner. Pillars associated with the development of the main entries and cross-cuts will be designed to meet individual mine specifications and is outside the scope of this investigation. Currently the height of all entries and cross-cuts is assumed to be 6 to 8 feet and all widths to be 20 feet, the maximum by law. Obviously these dimensions will also be subject to individual specifications.

It is also important to take note of the unusual layout of the cutting face. Since the panels will be developed using 6 to 8 foot high entries and the coal seam thickness will only account for 2 or 3 feet, a significant portion of the panel “material” will be left above and below each cut (Figure 3.4). The face is analogous to that of a highwall mine,

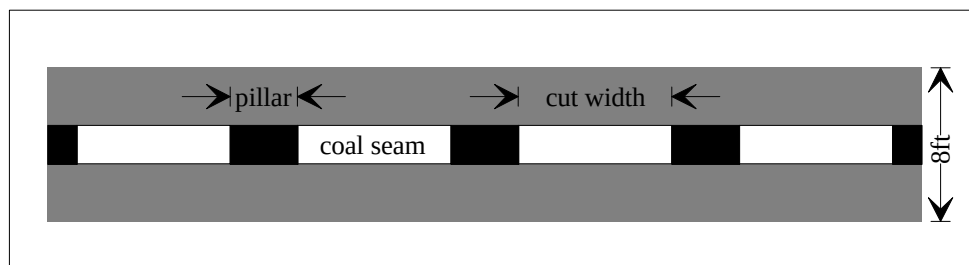


Figure 3.4 – Cross-Sectional View of the Cutting Face

a similarity that may aid in pillar design.

3.2 Detailed Panel Layout for the SAM

Figure 3.3 illustrated the alternate layout proposed for the Self-Advancing Miner. A more detailed look at the layout will indicate the approximate size and shape of the panels.

Figure 3.5 illustrates the panel geometry created by the development of entries and cross-cuts. Since the design of entries and chain pillars is unique to each mine, a three-entry system was chosen arbitrarily. As mentioned earlier, the cross-cuts will be driven at an angle, in this case 45°, in order to maximize panel recovery. The length of each panel will be 4000 feet. The width of each panel will vary with depth in order to accommodate a barrier pillar that will run through the center of each panel. However, the

width will be at least greater than twice the distance required for one cut, in this case 850 feet. Upon extraction of the panels, the barrier pillar and a series of pillars left between cuts will remain in every panel (Figure 3.6). Larger pillars will also be left at the ends of the panels to protect the cross-cuts. The sizing and analysis of these pillars is discussed in the following chapter.

In reality the actual layout of the mine will be dictated by the geometry of the coal seam and that will be distinct to each mine. However, assuming the SAM technology is employed, the geometry of each panel and each cut will prevailingly be the same. Therefore the stability of the pillars left between cuts can be analyzed regardless of differences between individual thin-seam coal mines.

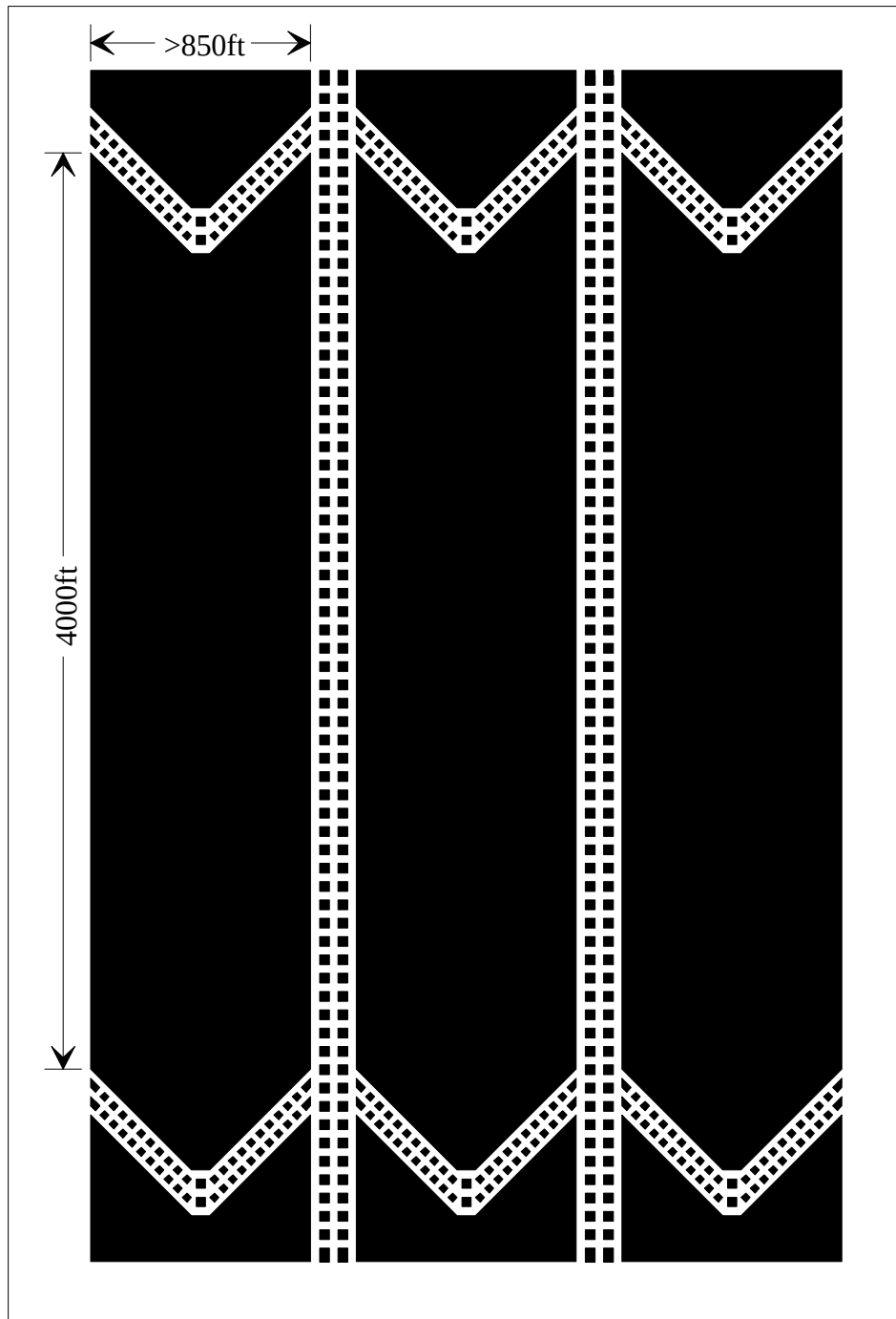


Figure 3.5 – Panel Development

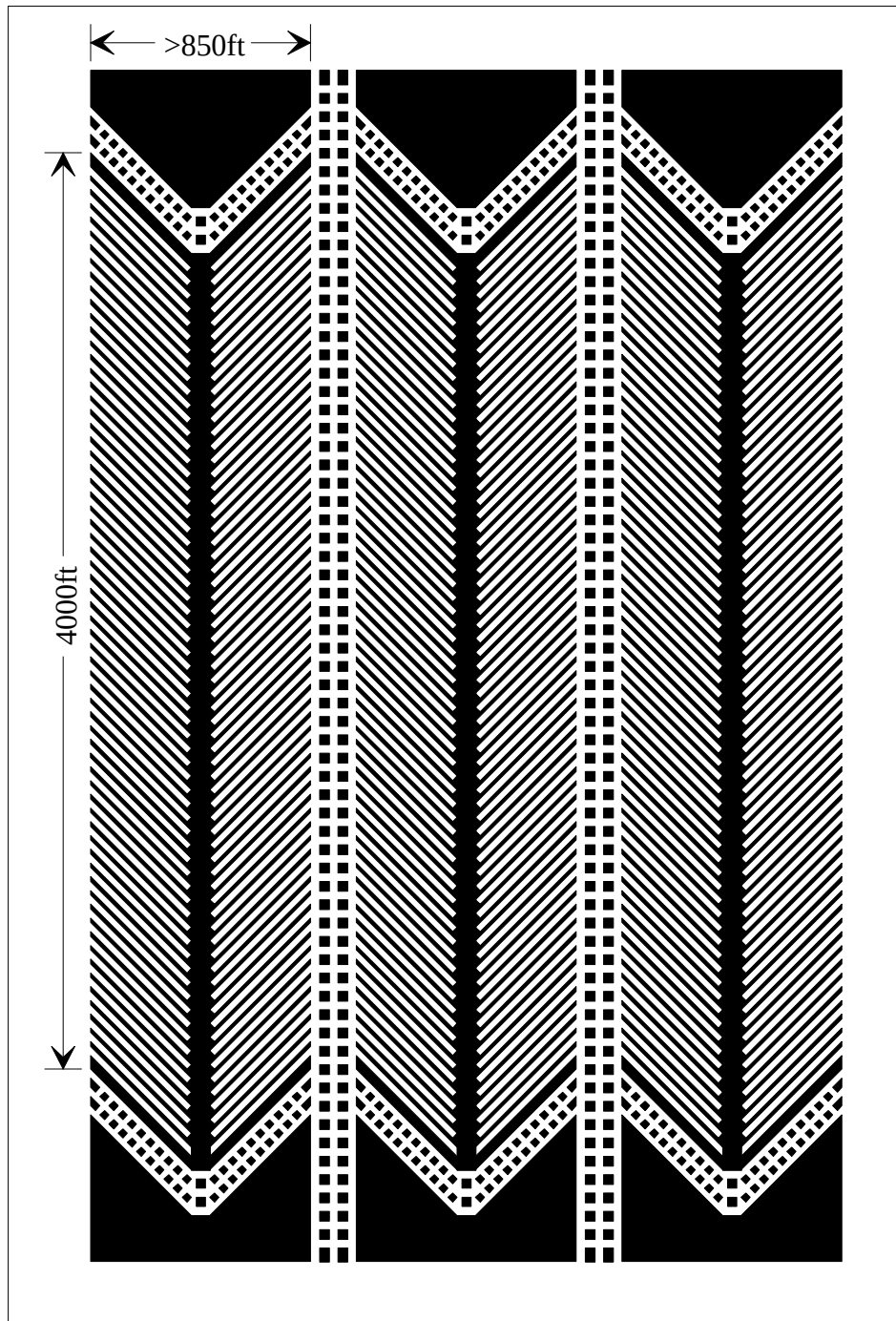


Figure 3.6 – Panels After Extraction

CHAPTER 4. PILLAR ANALYSIS AND DESIGN WITHOUT BACKFILL

The main purpose of pillars is to support the overburden and maintain the integrity of openings (Peng, 1986). Additionally, it is advantageous to design pillars as small as possible in order to maximize extraction rates. This trade-off between stability and maximum recovery is common to all room and pillar coal mines. However, for thin seam coal the compromise between stability and recovery is so great that mining of thin-seams has been deemed uneconomic. Lessening the impact of this compromise may make thin-seam coal mining feasible. Backfilling for the purposes of enhancing recovery could be the solution. In order to determine the feasibility of backfilling, the pillars left within the thin-seam coal mine were analyzed and designed without backfill so that comparisons could be made to the pillars left when backfill is applied. The no-backfill design served as a control so that increases in recovery due to backfilling could be measured.

4.1 Defining the Types of Pillars

Extraction of each panel by the Self-Advancing Miner will result in the development of three distinct types of pillars. The pillars can be distinguished by type according to size, shape, and function. The three types of pillars are (Fig 4.1):

1. Pillars left between SAM cuts
2. Barrier pillars left at ends of panels to protect cross-cuts
3. Barrier pillar left in the middle of each panel to separate SAM cuts.

There will always be four of the Type 2 pillars and one of the Type 3 pillars in each panel. The number of Type 1 pillars will vary with overburden depth, SAM cut width, and seam height but there will be at least a few hundred of them within each panel.

Although the pillars have been grouped into three main categories, each of which was analyzed and designed individually, it is important to remember that together the pillars act as one system and the influences they have on one another can impact their design.

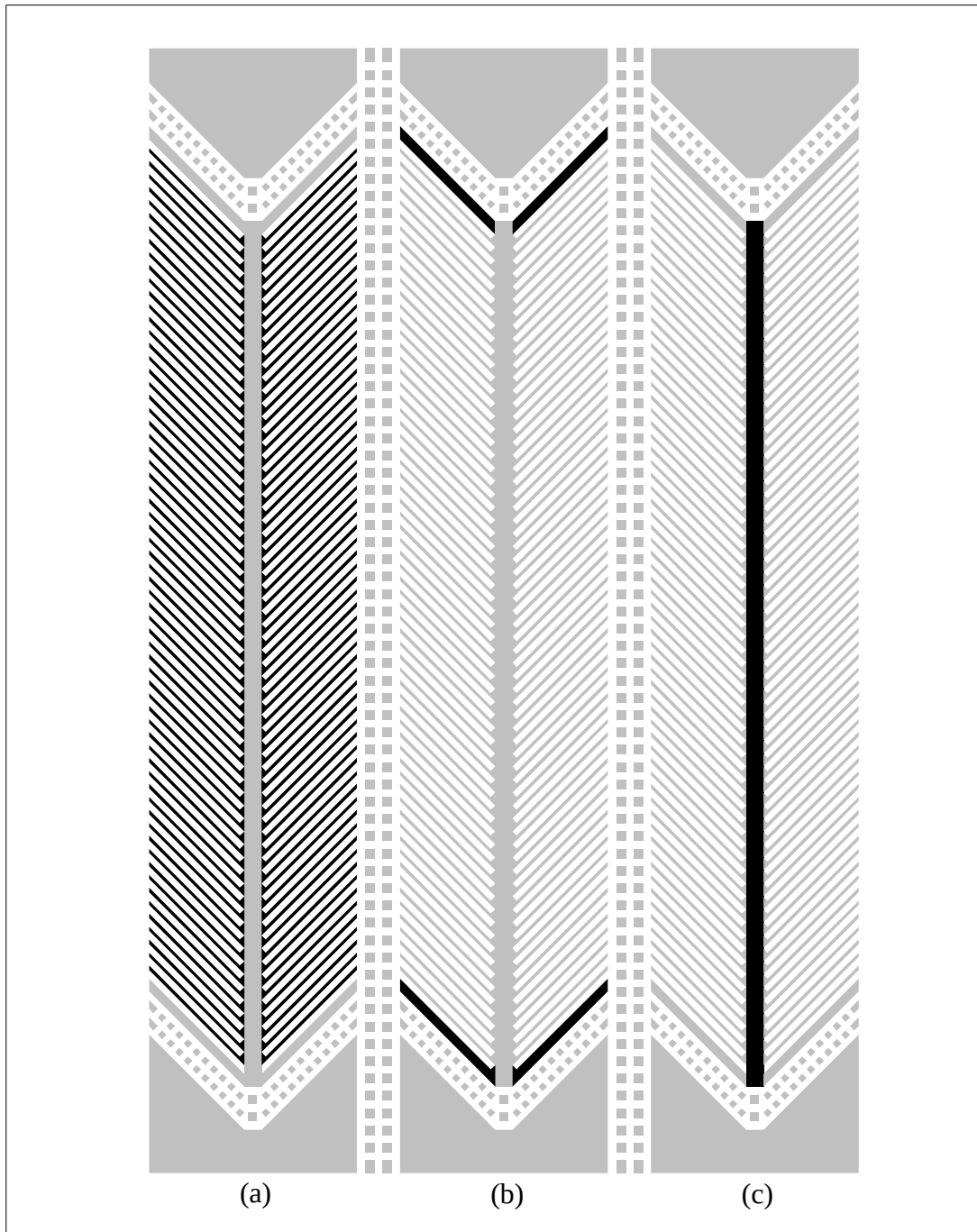


Figure 4.1 – Individual Pillar Types

- (a) Type 1
- (b) Type 2
- (c) Type 3

4.2 Analysis of Individual Pillar Types

4.2.1 Type 1 Pillars

Type 1 pillars will be left to support the openings made by the Self-Advancing Miner. Of the three pillar types, these are the most important in terms of stability and recovery. The majority of coal left for support will exist in the form of non-recoverable Type 1 pillars.

The basic shape of the pillars is that of a parallelogram (Figure 4.2). The rectangular shape of the pillars is transformed slightly due to the cut angle of the SAM.

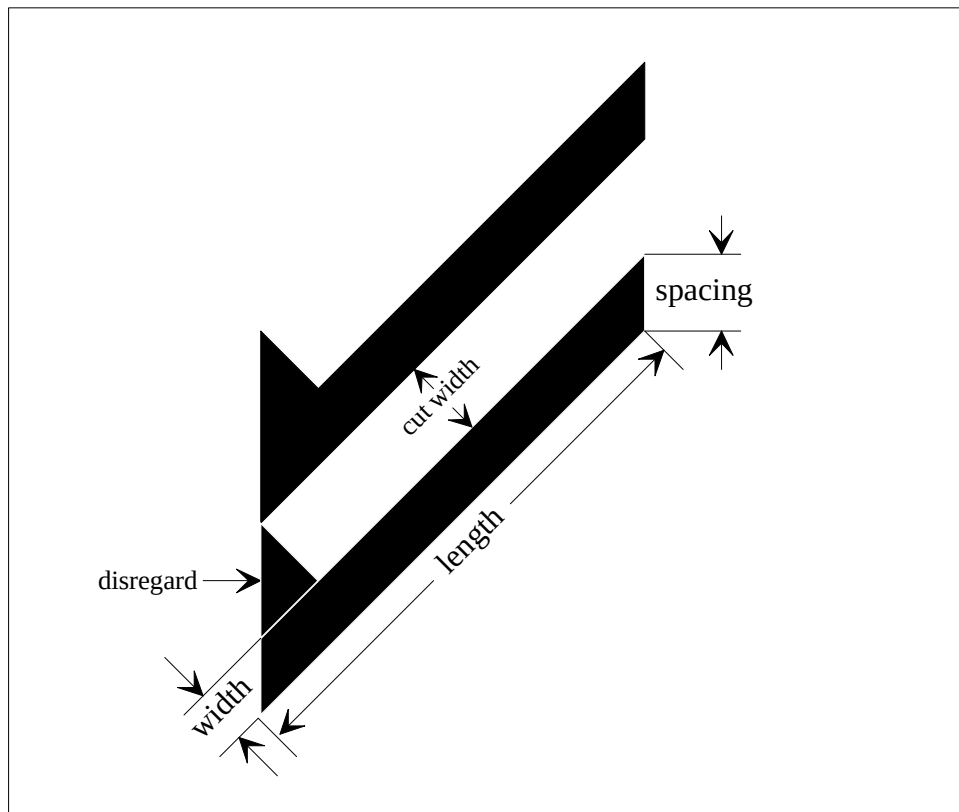


Figure 4.2– Geometry of Type 1 Pillar

In addition to the rectangular portion of the pillar, it can be seen in Figure 4.2 that a small triangular section of coal will be left at the end of each cut. This section of the pillar was disregarded in order to simplify the design of the pillar by considering only the rectangular geometry of the pillar. Figure 4.2 also indicates that the effective width of the pillar will be measured perpendicularly along the length of the pillar. The “spacing”

dimension is the actual distance left between cuts and can be determined by dividing the effective pillar width by the angle of the cut.

The shape of the Type 1 pillars will affect the pillar strength. Extensive testing has indicated that the width-to-height ratio of a pillar influences its strength (Peng, 1986). One of the better known pillar strength formulas, developed by Bieniawski, illustrates this point (Ross-Brown, 1978):

$$S_p = S_1 \left[0.64 + 0.36 \left(\frac{w}{h} \right) \right] \quad [4.1]$$

where,

S_p = pillar strength (lb/in²)

S_1 = in situ coal strength (lb/in²)

w = pillar width (ft)

h = pillar height (ft)

Since the Type 1 pillars will be located within such a thin-seam (24'-36"), the width-to-height ratio will almost definitely be greater than 1. Depending on overburden depth it may exceed 5. Bieniawski developed Equation 4.1 by testing square specimens and recognized that the formula underestimated the strength of rectangular pillars (Mark and Chase, 1997). More recently rectangular pillars and the effect of a pillar's length on its strength have been examined. A modified version of Equation 4.1, the Mark-Bieniawski Formula, has been developed. It indicates that the strength of rectangular pillars exhibits a dependence on the width-to-height and width-to-length ratios. The Mark-Bieniawski Formula is (Mark and Chase, 1997):

$$S_p = S_1 \left[0.64 + 0.54 \left(\frac{w}{h} \right) - 0.18 \left(\frac{w^2}{l \times h} \right) \right] \quad [4.2]$$

where,

l = pillar length (ft)

As with Equation 4.1, as w/h increases so does the strength of the pillar. Additionally, as w/l increases the strength also increases. Equation 4.2 has been used successfully within the *ARMPS* computer program to design pillars for room and pillar mining and it was used in the design of the Type 1 pillars (Mark and Chase, 1997).

Before moving on to the other pillar types, it is important to address the in situ coal strength term, S_1 , found in Equations 4.1 and 4.2. In the past much work was

dedicated to predicting the in situ pillar strength from tests done on laboratory specimens (Peng, 1986). However, Mark and Barton (1997) summarized the work of a group of researchers who have determined that the value of in situ coal strength falls between 780 psi and 1070 psi (5.4-7.4MPa). Mark and Barton also performed a statistical analysis that found no correlation between the stability of failed pillars and coal specimen strength, concluding that pillar design was much more reliable when a uniform coal strength was used. This uniform coal strength of 900 psi (6.2MPa) falls within the range determined by other research and was used in the design of Type 1 pillars.

4.2.2 Type 2 Pillars

The pillars left at each end of the panel are the Type 2 pillars. Their main purpose is to maintain the integrity of the cross-cuts. Based on this function, Type 2 pillars can be defined as barrier pillars. Barrier pillars are commonly designed to support entries or to separate mined or mined-out sections and their design is usually based on empirical formulas (Peng, 1986).

The geometry of the Type 2 pillar is basically the same as the Type 1 pillar. The only difference is that two of the four Type 2 pillars within each panel will not contain the “disregarded” triangular portion (Figure 4.3).

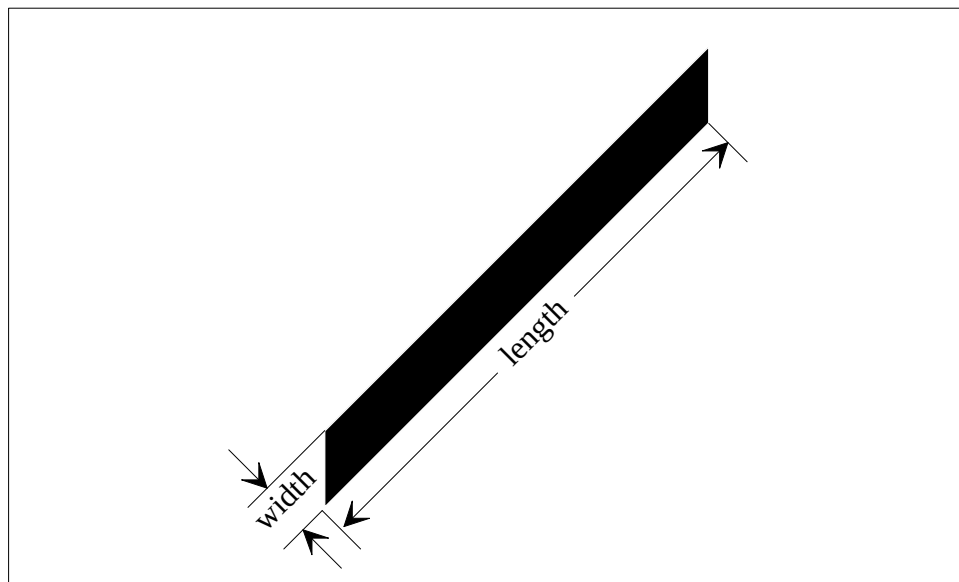


Figure 4.3 – Geometry of Type 2 Pillar

The measured dimensions of Type 2 pillars are the same as the Type 1 pillar. However, since the function of these pillars is to provide support to the cross-cut entries, analysis of their shape effect was not necessary to their design.

Barrier pillars are generally designed using rule-of-thumb type of formulas based on experience. Two of the more common formulas are the Mine Inspector's formula and the British Coal Operator's formula. The Mine Inspector's Formula is (Holland, 1973):

$$W_{bp} = 20 + 4h + 0.1H \quad [4.3]$$

where,

W_{bp} = width of barrier pillar (ft)

h = seam thickness (ft)

H = thickness of overburden (ft)

The British Coal Operator's formula is not much different from Equation 4.3. The guideline provided by that design equation is as follows (Peng, 1986):

$$W_{bp} = 45 + 0.1H \quad [4.4]$$

It can be seen that Equation 4.4 places no dependence on seam thickness. This is favorable to the design of the Type 2 pillars. Since the pillars will support a cross-cut entry on one side and a miner cut on the other, using Equation 4.4 simplifies their design. The dimensions and support needs for each span created by the miner cut and cross-cut will differ, but the stability of the cross-cut is more critical. Therefore the barrier pillar design provides a stable entry while being on the conservative side in regards to the cut made by the Self-Advancing Miner.

Additionally the barrier pillar design also ignores any effects of the coal-rock interface within the Type 2 pillars. Recalling that the coal seam is only 24" to 36" thick and the height of the entries about 6' to 8', there will be two coal-rock interfaces running through each Type 2 pillar that may be sources of weakness and concern. Assigning a pillar width independent of pillar strength helps alleviate this concern.

4.2.3 Type 3 Pillars

The pillar that will run through the middle of each panel, separating opposing SAM cuts, is the Type 3 pillar. This pillar was previously referred to as a "barrier" pillar but should not be confused with the Type 2 pillars. The Type 3 pillar will only act as a

barrier in terms of preventing opposing SAM cuts from punching through to each other. The pillar may also eventually act as a natural bulkhead if backfilling of the cuts is necessary.

Developing a design procedure for the Type 3 pillar is difficult. As with the Type 2 pillars, part of the Type 3 pillar will support the cross-cut entry and may need to be designed to do so. However, in comparison to the length of the pillar (4000') the portion of the pillar supporting the entry is minimal. Also, the width of the Type 2 pillar runs parallel with the length of the Type 3 pillar, so it can be assumed that the portion of the Type 3 that is bound by the width of the Type 2 pillar supports the entry. In Figure 4.4, "A" represents the section of Type 3 pillar that is supporting the cross-cut and its length

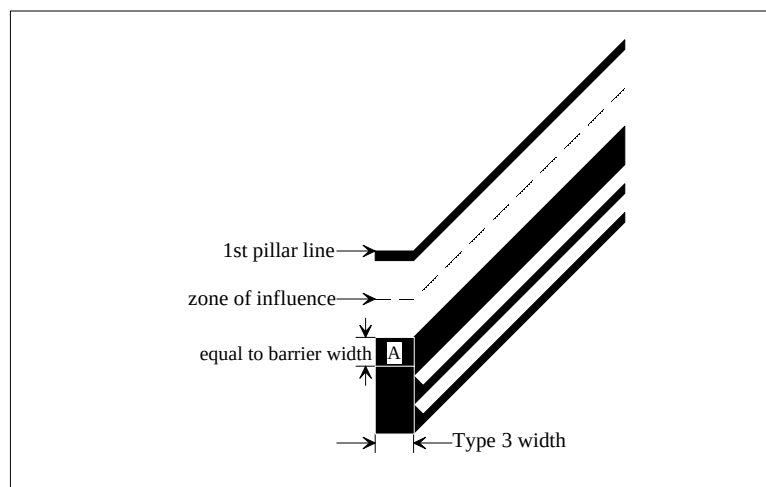


Figure 4.4 – Influence of Type 3 Pillar

is equal to the barrier pillar width. The width of the Type 3 pillar is not a consideration and thus does not require a design using a barrier pillar formula. Type 3 pillars do not have to be designed to support the cuts made by the Self-Advancing Miner either. Type 1 pillars will provide adequate support to those cuts without the need for any additional support from the Type 3 pillar.

The only functions that the Type 3 pillar will provide are as a “separation” pillar and as a natural bulkhead. Neither of these functions are structural or quantifiable making it difficult to design the pillar to meet its purpose. However, a conservative approach that requires designing the pillar as a structural component was taken. Therefore the design will meet the required functions of the pillar and the pillar will also provide additional support to the Type 1 pillars. The Type 3 pillar was designed using

the same pillar equations given for the Type 1 pillar (Equations 4.1,4.2) and using the same value of coal strength (6.2MPa, {900psi}). Additionally an area of influence, or support, was assigned arbitrarily to the pillar in order to determine its design width. Although this seems vague it is better than arbitrarily assigning a width to the pillar, the only other option.

4.3 Design of Individual Pillar Types

Customarily, pillar design is based on the assumption that the stress in a pillar is evenly distributed and equal to the original vertical geostatic stress divided by the pillar area/original area ratio (Farmer, 1985). Design of this nature also assumes that failure occurs when the pillar stress exceeds the strength of the pillar rock. This design can be referred to as the ultimate strength theory and ignores the fact that pillar failure can be progressive. Although progressive failure theories are credible, they require considerable development, and for this case the ultimate strength approach is sufficient.

Peng (1986) listed four items that must be considered in the design of pillars:

1. The expected load history, including premining pillar loading and mining induced abutment pressures.
2. Stress distribution within the pillar.
3. Pillar Strength.
4. Interaction between roof, pillar, and floor.

In this case the expected load history for each pillar type refers only to development loadings. These are the loads created by the mining of entries and by the mining of the coal seam by the Self-Advancing Miner. The stress distribution within each pillar was calculated using the simplest method, the tributary area method. This assumes that each pillar supports all the overlying strata that is “tributary” to its location (Farmer, 1992). The tributary method also assumes that loading is uniformly distributed within the pillar. In actuality the stresses within a pillar are highly non-uniform. However, Mark and Chase (1997) stated that although empirical formulas do not explicitly consider the effect of non-uniform distributions, they do imply a non-uniform stress distribution because of the shape effect. Considering this, use of the Mark-

Bieniawski Equation (Equation 4.2) may have helped to abate the simplicity of the tributary method. The pillar strength was already discussed in section 4.2.1 and an average coal strength of 900 psi (6.2MPa) was used in the design of the Type 1 and 3 pillars. The interaction between the roof, pillar, and floor is important but can really be only measured on an individual mine basis and was ignored.

Based on the previous assumptions and design considerations, along with the analysis provided in the previous sections, all of the pillars were then designed.

4.3.1 Type 1 Pillars

As mentioned earlier, Type 1 pillars will provide localized support to the openings created by the Self-Advancing Miner. There are no plans for pillar recovery or the formation of gob areas therefore abutment pressures will be minimal. There may be abutment pressures caused by the mining sequence but they were disregarded. The only loading that the pillars will be exposed to, and designed to withstand, is that of the weight of the overburden.

All pillars are designed with a safety factor against failure. Usually, the selection of a safety factor is based on mining experience or back-analysis of pillar failures. It is also dependent on specific knowledge of the underground environment such as roof and floor conditions. Since this type of information is unique to individual mines, a generic factor of safety was used for the Type 1 pillars. The *ARMPS* (Analysis of Retreat Mining Pillar Stability) computer program uses a stability factor of 1.5, which was determined to be adequate based on retrospective analysis of over 140 case histories (Mark and Chase, 1997). The back-analysis showed that when the stability factor was 1.5 or greater, 94% of the designs were satisfactory. A similar study performed on South African coal pillars indicated a range of 1.3 to 1.9 with an average factor of about 1.6 (Brady and Brown, 1992). Based on these results and the success of the *ARMPS* program, a stability factor of 1.5 was chosen for the Type 1 pillars.

The design of the Type 1 pillars was nothing more than solving for an unknown, the pillar width. The pillar length, cut width and angle, seam height, and overburden depth were all known variables upon which the pillar width is dependent. Additionally, the coal strength and stability factor had been set therefore the critical pillar width could

be back-calculated. The basic equation for pillar design, using the tributary method, is that the stability factor of the pillar is equal to the load-bearing capacity of the pillar divided by the actual load applied to the pillar;

$$SF = LBC / LT \quad [4.5]$$

where,

SF = stability factor

LBC = load-bearing capacity of pillar

LT = load on pillar.

The load bearing capacity of the pillar is the pillar's strength multiplied by its area and the actual load on the pillar is equal to the pressure exerted by the overburden multiplied by the pillar's area plus the area of one SAM cut. Assuming that the overburden pressure increases 1.1 psi/ft of depth and substituting Equation 4.2 into Equation 4.5:

$$SF = \frac{S_1 \left[0.64 + 0.54 \times \left(\frac{w}{h} \right) - 0.18 \times \left(\frac{w^2}{h \times l} \right) \right] \times A_p}{1.1 \times z \times (A_p + A_c)} \quad [4.6]$$

where,

S_1 = coal strength (psi)

w = pillar width (ft)

A_p = area of pillar (ft²)

h = seam/pillar height (ft)

A_c = area of cut (ft²)

l = cut/pillar length (ft)

z = overburden depth (ft)

The area of the cut and the area of the pillar can be written in terms of the pillar and cut dimensions and, due to the unique cut geometry, can get complicated. However, because the pillars are so long it was easier to use the rib pillar approach in expressing the areas. Treating the pillar as a rib pillar allowed the area of the pillar and the area of the cut to be expressed on a unit length basis. Thus, the angle of the cut was not required in the calculations, simplifying the equation. Using the rib pillar approach and substituting a stability factor of 1.5 and coal strength of 900 psi into the formula, Equation 4.6 can be written as:

$$1.5 = \frac{900 \left[0.64 + 0.54 \times \left(\frac{w}{h} \right) - 0.18 \times \left(\frac{w^2}{h \times l} \right) \right] \times w}{1.1 \times z \times (w + c)} \quad [4.7]$$

where,

c = cut width (ft)

Upon entering values for the seam height, cut length, cut width, and overburden depth, Equation 4.7 was used to calculate the pillar width that meets the design requirements. In order to verify the rib pillar approach a similar equation, based on the actual cut geometry and angle, was also developed and used to calculate widths that were compared to those calculated using Equation 4.7.

A *Mathcad 4.0* program was developed to solve Equation 4.7 for the critical pillar width (Appendix A). By inputting values for the cut length, seam height, cut width, and a range of overburden depths, the program is able to generate pillar widths that coincide with the different overburden depths. Figure 4.5 is a singular design chart for Type 1 pillar widths given a seam thickness of 24", assuming a cut width of 7' and cut length of 600', and ranging from a depth of 100' to a depth of 2000'. Figures 4.6 - 4.8 were

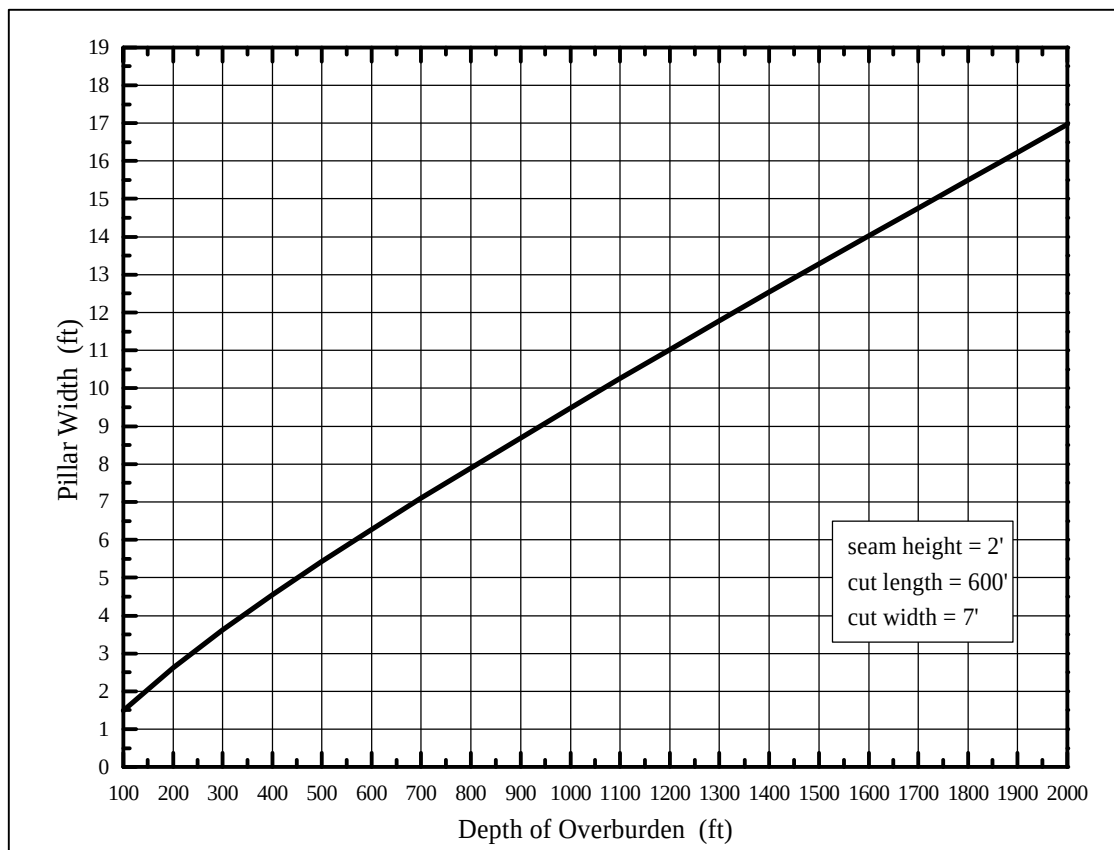


Figure 4.5 – Pillar Width vs. Depth for Type 1 Pillars

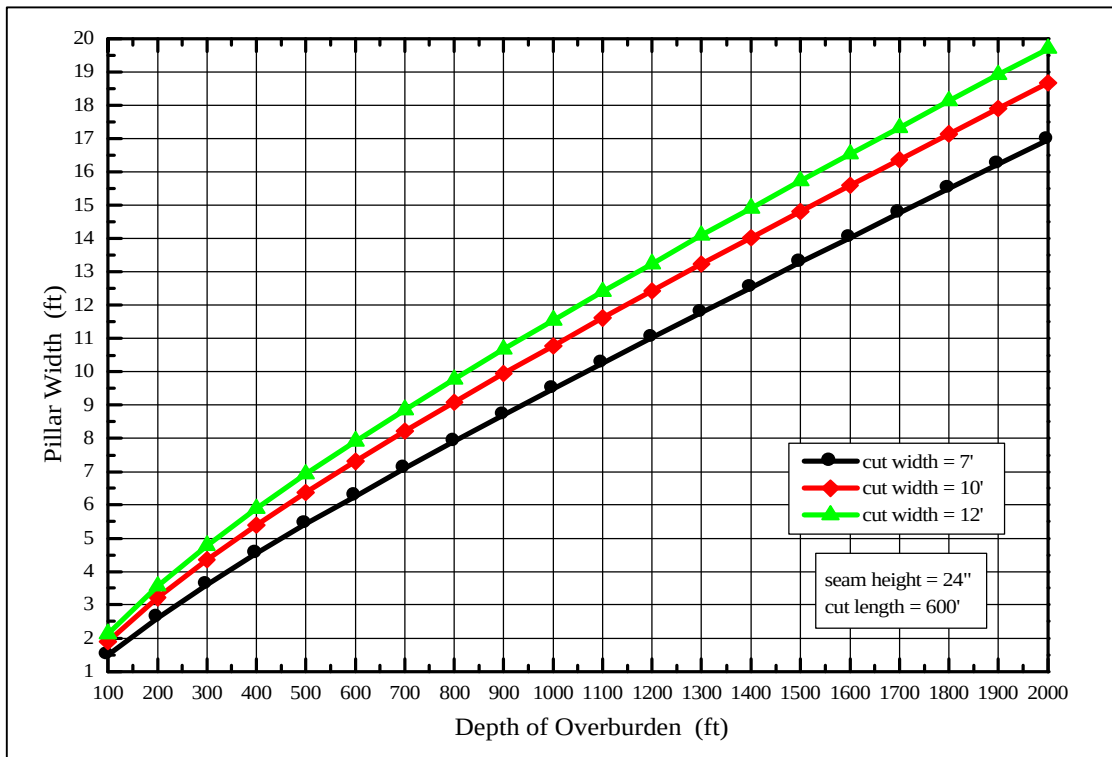


Figure 4.6 – Pillar Width vs. Depth for 24" Seam

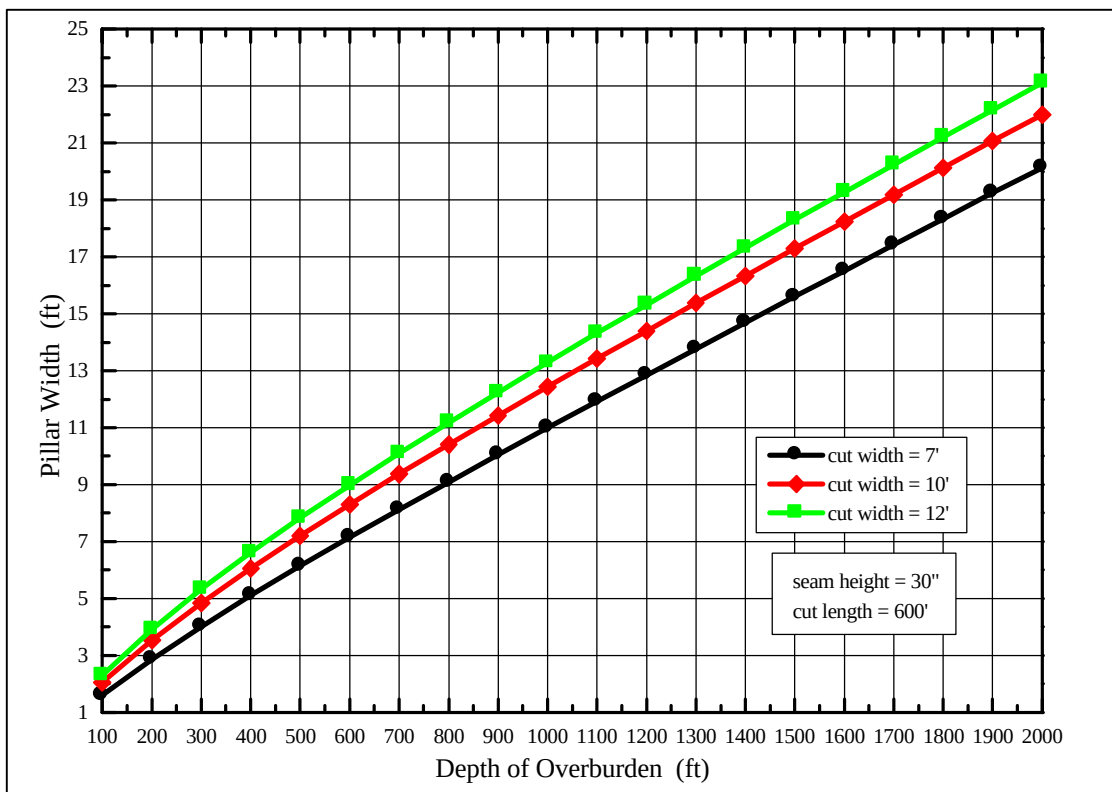


Figure 4.7 – Pillar Width vs. Depth for 30" Seam

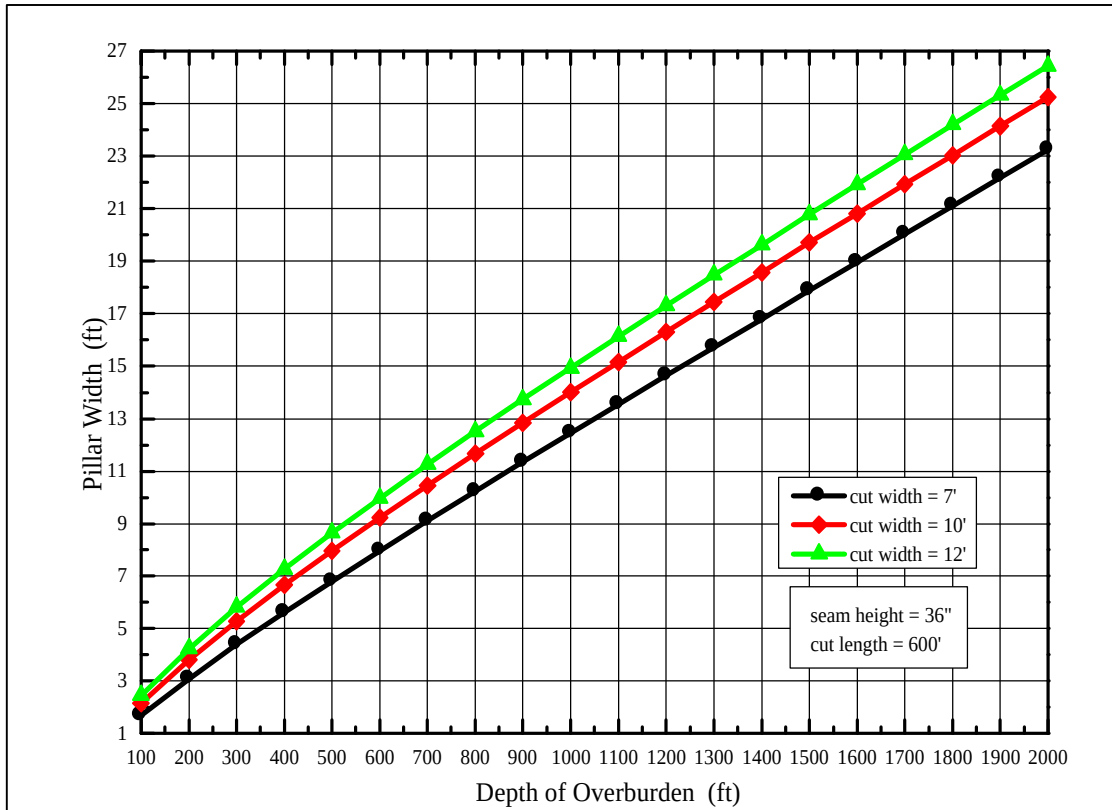


Figure 4.8 – Pillar Width vs. Depth for 36” Seam

developed in the same manner as Figure 4.5 and are for seam thicknesses of 24”, 30”, and 36”. They also incorporate variable cut widths of 7’, 10’, and 12’. The cut length was kept constant at 600’ for each design chart because the pillar widths remain fairly constant, due to the rib pillar approach, despite changes in length. In order to check the validity of the rib pillar approach and the results depicted in Figures 4.5-4.8, pillar widths were also calculated based on the actual geometry of the SAM cut. A second *Mathcad* 4.0 program, requiring additional input variables, was developed (Appendix A). The additional input included a value for the angle of cut of the SAM and a width for the entry from which the SAM was cutting from. The influence of this entry on the support capacity and design of the Type 1 pillars was ignored by the rib pillar approach. The width of the entry was assumed to be 20’ and the cut angle was entered as 45°. The results, generated using the same set of input parameters, compare favorably to those represented by Figures 4.5-4.8. Table 4.1 is a comparative example for a seam height of 24”, cut width of 7’, and cut length of 600’. Based on the results of the comparative

study, the rib pillar approach was used for all future designs including that of backfilled pillars.

Table 4.1 – Comparison of Geometric and Rib Pillar Designs

Depth	Pillar Width	
(ft)	(ft)	
	<i>Geometric Design</i>	<i>Rib Pillar Design</i>
100	1.6	1.5
200	2.7	2.6
300	3.8	3.6
400	4.7	4.5
500	5.6	5.4
600	6.5	6.3
700	7.4	7.1
800	8.2	7.9
900	9.0	8.7
1000	9.8	9.5
1100	10.7	10.3
1200	11.5	11.0
1300	12.2	11.8
1400	13.0	12.5
1500	13.8	13.3
1600	14.6	14.0
1700	15.4	14.8
1800	16.1	15.5
1900	16.9	16.2
2000	17.7	17.0

The calculated pillar widths are representative of an individual, Type 1 pillar designed to support openings along both of its sides created by the Self-Advancing Miner. Each individual pillar has a stability factor of 1.5. This is not to say that an entirely excavated panel consisting only of Type 1 pillars has a stability factor of 1.5. The overall stability of the panel will be influenced by the design widths calculated for the Type 2 and 3 pillars.

4.3.2 Type 2 Pillars

Since the function of Type 2 pillars is that of a barrier pillar, their design was quite simple. Using the British Coal Operator's Formula (Equation 4.4), the Type 2

pillars were sized in order to provide support to, and maintain the integrity of, the cross-cut entries. Additionally, the Type 2 pillar design helps to protect against an intersection between the cross-cut entry and the SAM cut on the opposite side of the pillar.

Only the overburden depth was needed to calculate the pillar width from Equation 4.4. The Type 2 pillar widths, as a function of depth, are shown in Figure 4.9.

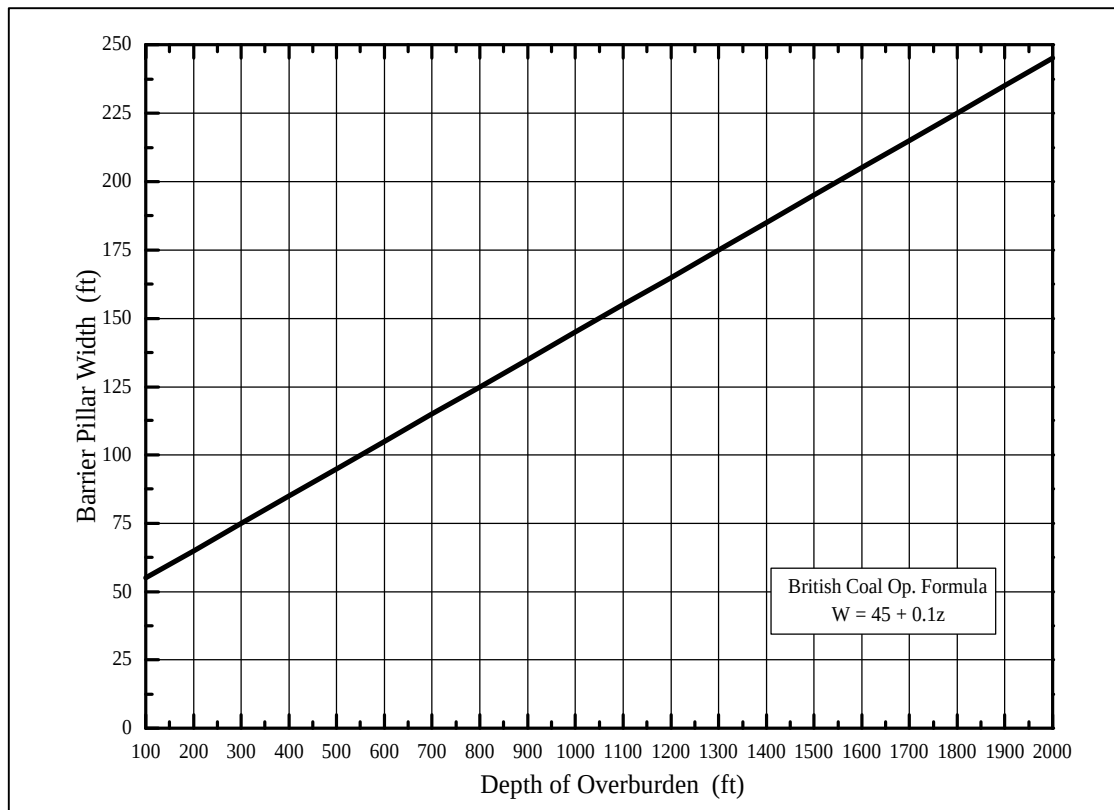


Figure 4.9 – Type 2 Pillar Width vs. Overburden Depth

The design widths in Figure 4.9 are based on experience and do not consider the strength of the coal or the thickness of the coal. Although the design is not based on actual coal mine conditions, it is easier to use in practice, and in this case, than other designs (Holland, 1973).

4.3.3 Type 3 Pillars

In section 4.2.3 the ambiguity of the Type 3 pillars was discussed. A suitable design procedure was difficult to develop because the functions of the Type 3 pillars cannot be categorized as structural. Without a need for the Type 3 pillars to provide

support it seemed that traditional pillar design, like that used for the Type 1 pillars, and barrier pillar design were both unnecessary. However, since there has been no previous experience with this type of pillar, rule-of-thumb type of designs used in similar situations do not exist. Therefore, the Type 3 pillars were designed in basically the same manner as the Type 1 pillars, using the rib pillar approach and calculating pillar widths based on pillar strengths and overburden pressures. This design was used for two reasons: (1) it calculated pillar widths based on actual conditions as opposed to arbitrarily assigning widths without previous experience, and (2) it provided the Type 3 pillars with the capacity to contribute additional support to the Type 1 pillars if necessary. The second reason is important because all of the SAM cuts may not be perfectly parallel to one another. If two cuts fan slightly towards one another, the width of the Type 1 pillar between the cuts will not meet its design specification, particularly in the middle of the panel where the cuts end (Figure 4.10). By assigning an area of influence to the Type 3 pillar, the pillar may compensate for the thinning, “mis-shaped” Type 1 pillar. For the design, an area that extends 50’ outwards from the Type 3 pillar was used. As with the

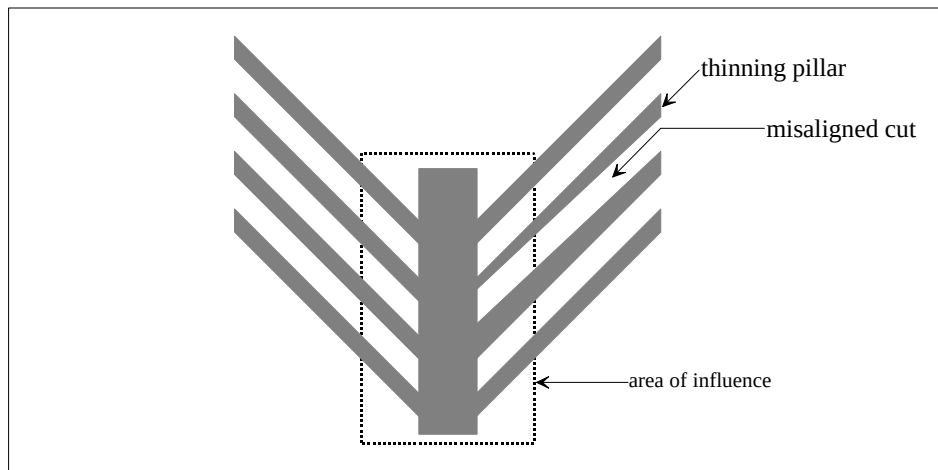


Figure 4.10 – Effect of the Type 3 Pillar on a Misaligned SAM Cut

Type 1 pillars, the calculated widths are given in a design chart (Figure 4.11) as functions of overburden depth and for a range of seam thicknesses and with the length of each Type 3 pillar was set at 4000’.

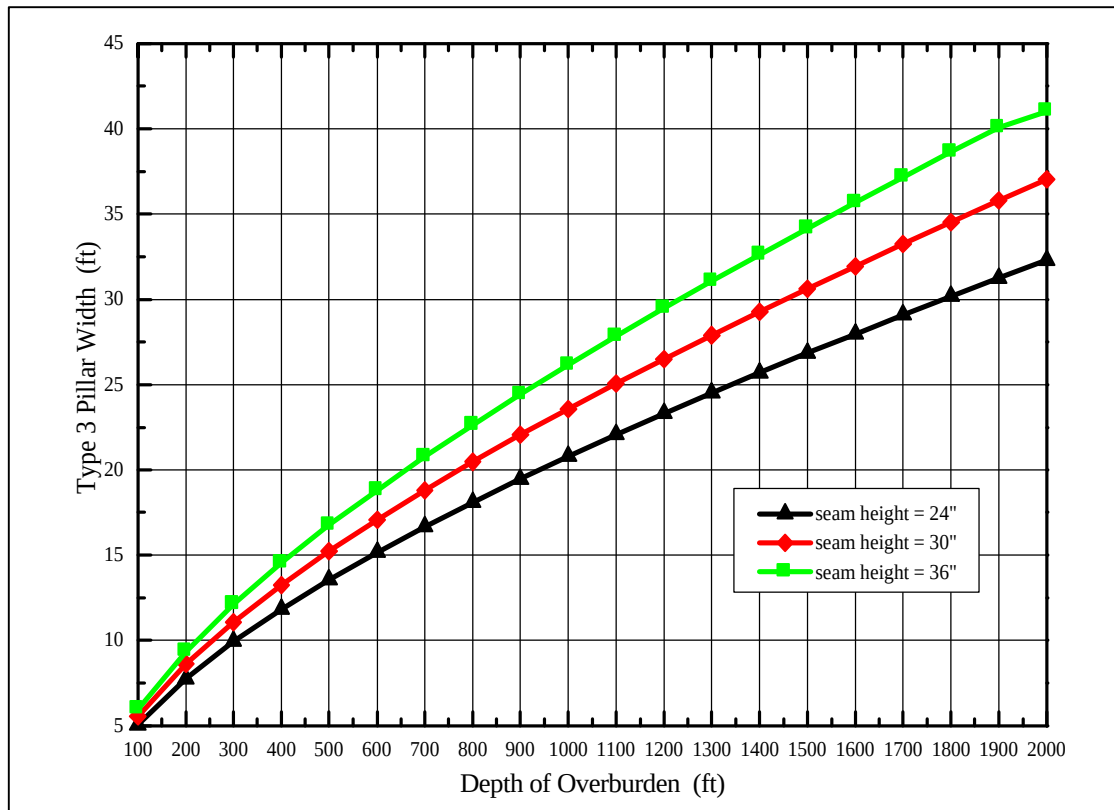


Figure 4.11 – Pillar Width vs. Overburden Depth for Type 3 Pillars

The effectiveness of the Type 3 pillar design cannot readily be measured without actual underground experience. From the preliminary analysis it doesn't seem that the Type 3 pillars will be under any unusual stresses; however, a more detailed study may be necessary if the pillars are required to withstand large loads. The design does provide pillar widths that seem suitable for the Type 3 pillar's function as a natural separation between SAM cuts and quite possibly as a natural bulkhead.

4.4 Panel Recovery Rates Based on Pillar Design

The design widths calculated in sections 4.3.1-3 were used to determine the theoretical extraction rates from a singular panel. Since there are no plans for pillar recovery, the extraction rate for a panel is directly dependent on the volume of coal left in the form of pillars. Spreadsheets were used to determine the volume of coal extracted and the overall recovery rate from a panel (see Appendix B). As with pillar widths, the rate of recovery decreases with increasing overburden depth (Figures 4.12-14).

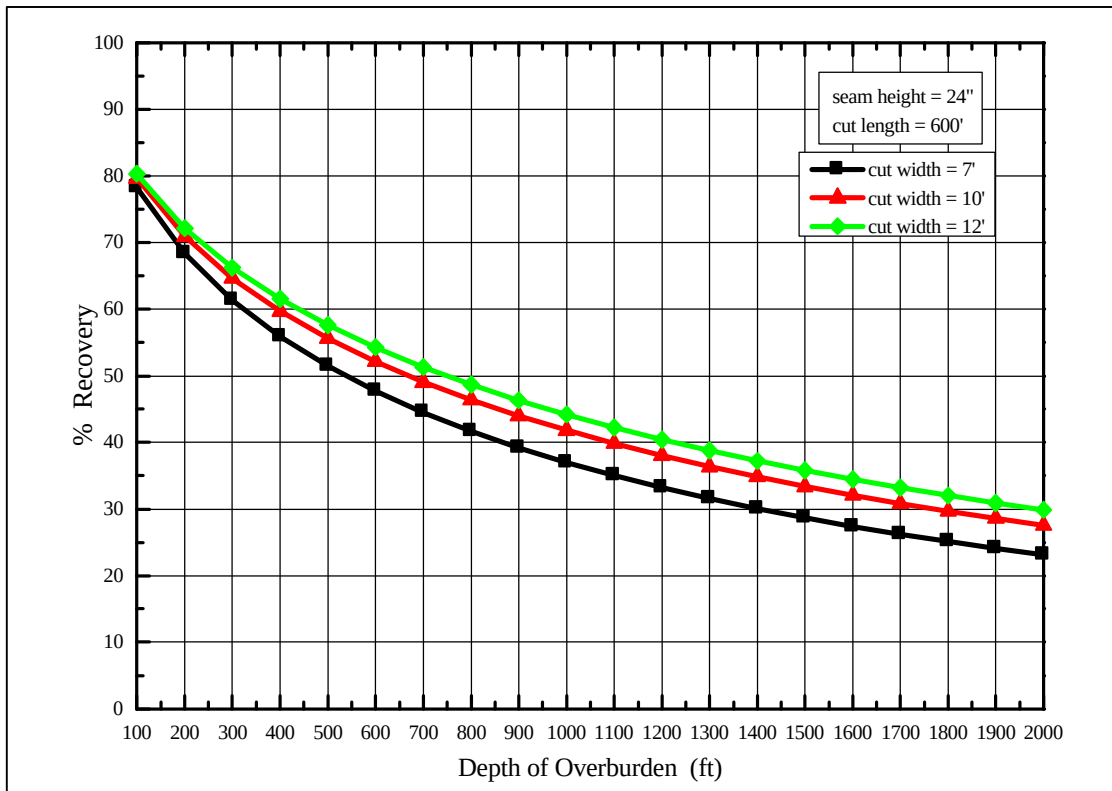


Figure 4.12 – Recovery vs. Depth for 24” Seam

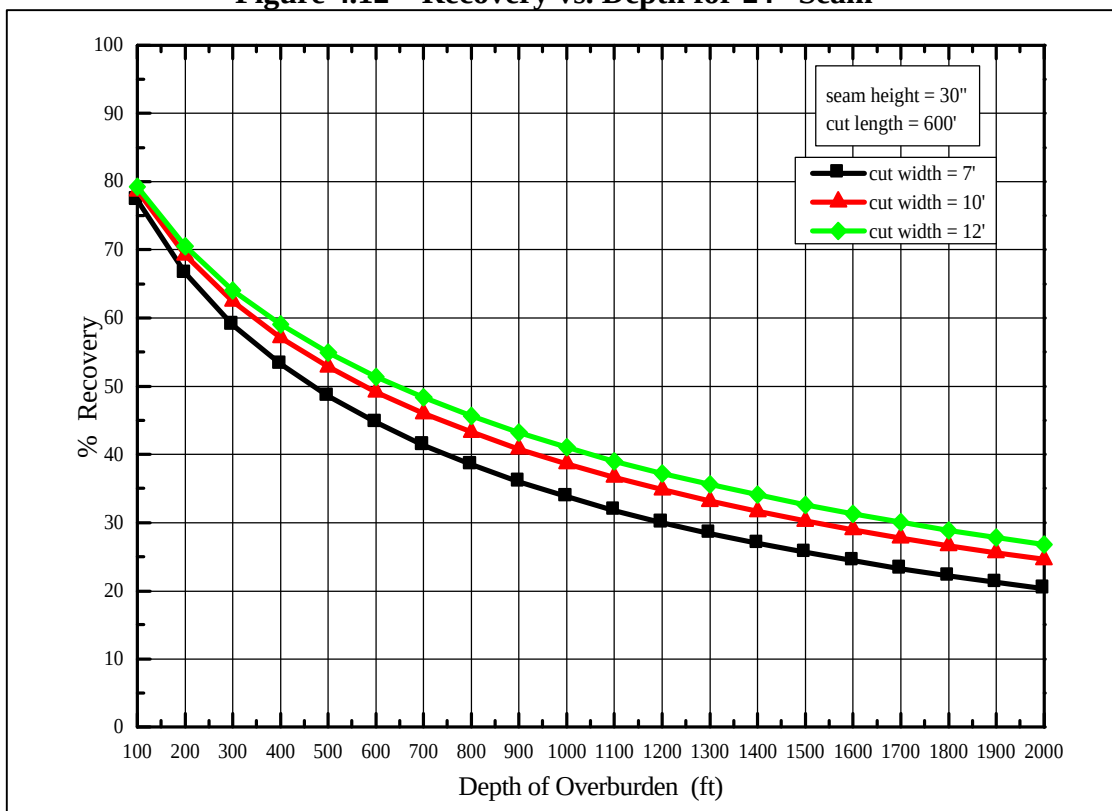


Figure 4.13 - Recovery vs. Depth for 30” Seam

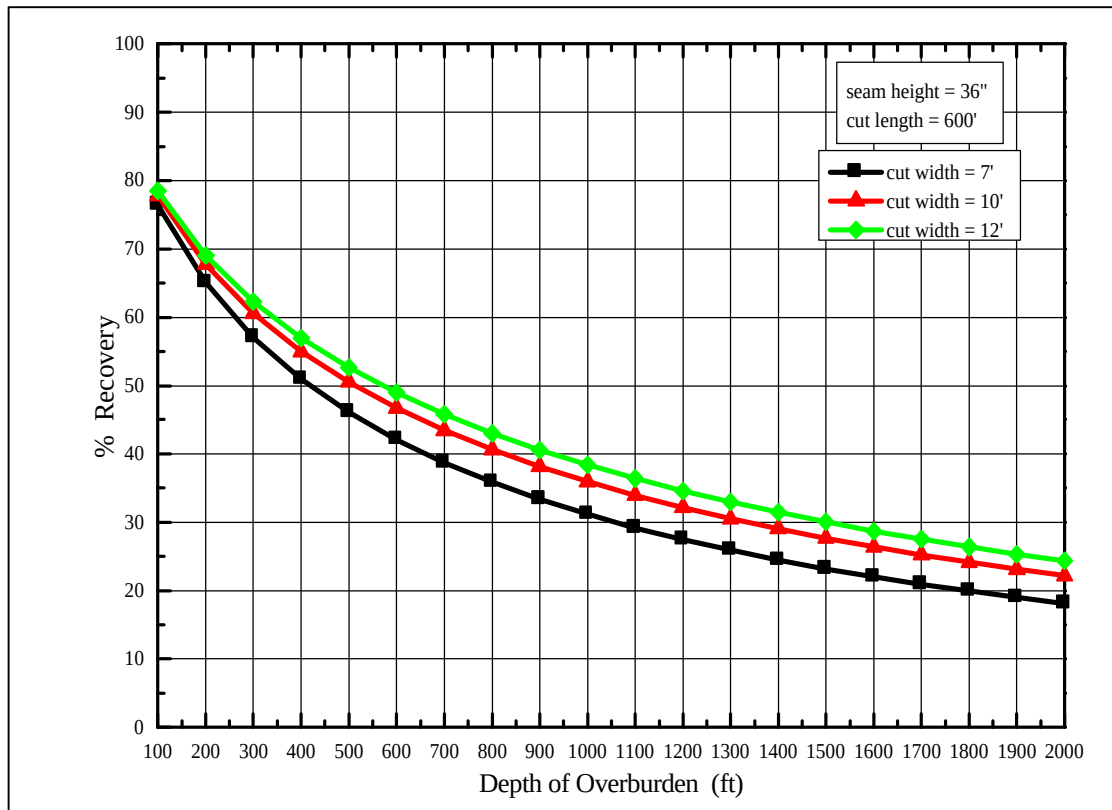


Figure 4.14 – Recovery vs. Depth for 36” Seam

Figures 4.12-14 also indicate that recovery rates are fairly independent of seam height and cut width. Therefore, the amount of coal extracted from a panel is most sensitive to changes in pillar dimensions and improvements in recovery are possible if the pillar widths are decreased. The widths of the Type 2 and 3 pillars are not as critical because the percentage of coal left in those pillars is small in comparison to the Type 1 pillars. Approximately 70% to 80% of the coal left within each panel will be in the form of Type 1 pillars (see Appendix B). A decrease in the widths of the Type 1 pillars will provide the most substantial benefit in terms of extraction and recovery. Because they have little influence on recovery, the Type 2 and 3 pillars will be left alone.

Backfilling each individual cut made by the Self-Advancing Miner may provide the kind of decrease in pillar widths that is needed for improved recovery. However, the increase in recovery must be substantial enough to justify the cost of backfilling in addition to improving the overall economics of the mining operation. The design procedure for pillars supported by backfill is investigated and outlined in the following

section. Increases in recovery due to backfilling, backfill mix design and components, backfill quality control, and the overall feasibility of backfilling are also covered in the following sections.

CHAPTER 5. ANALYSIS AND DESIGN OF BACKFILLED PILLARS

There is no current procedure for the design of pillars that can easily take into account the effect of backfill. Although the support mechanisms of backfill are understood (section 2.6), they have been developed qualitatively rather than quantitatively. The majority of research has focused on the properties, strength, and effects of backfill without clearly defining a design approach. In cut-and-fill mining the design of backfill has been developed mainly from experience. Currently there is little or no experience with backfill in underground coal mining. Therefore it was difficult to incorporate the effect of backfill on the Type 1 pillars, in a quantifiable manner, into the design procedure used in the previous section. A very simplistic approach is covered in the following section.

5.1 Analysis and Integration of Backfill Support Mechanism

The most important concept of backfill support is that the fill itself does not support the overburden. The additional strength that the fill transmits to the pillars is imparted as a horizontal pressure along the sides of the pillars. The resultant increase in strength is due to the confinement provided by the backfill. Recalling Equation 2.12, the weight of the fill itself provides some lateral resistance to the pillars. However, the magnitude of that resistance, even for strong, dense fills, would be too small to have any affect on the overall strength of the pillar. An increase in the horizontal pressure exerted by the backfill is necessary and can occur by using a cohesive fill (Equation 2.14) and/or by applying a surcharge load to the fill. The cohesion of the fill is dependent on its properties and mix design (sections 2.2-2.5) while any surcharging loading will be a result of roof and pillar deformations. As the overlying strata deforms into the fill, the lateral pressure exerted by the fill increases by an amount equal to the surcharge load multiplied by an earth pressure coefficient. Equation 5.1 considers the added affects of cohesion and surcharge loading:

$$s_h = (g_f \times h_f + p) \times K_p + 2 \times c \times \sqrt{K_p} \quad [5.1]$$

where,

s_h = earth pressure acting horizontally (lb/ft²)

g_f = unit weight of the fill (lb/ft³)

h_f = height of the fill (ft)

p = surcharge load (lb/ft²)

K_p = coefficient of passive earth pressure.

c = cohesion of the fill (lb/ft²).

The actual value of p is difficult to estimate and is dependent on many factors, but it is probably the most important factor in determining the strength increase induced by backfilling.

Figure 5.1 illustrates the process by which lateral pressure will develop within the fill. The resistance is contingent upon deformation of the roof and pillars; thus the fill acts as a passive support. It should be noted that in order to restrain the pillars, the

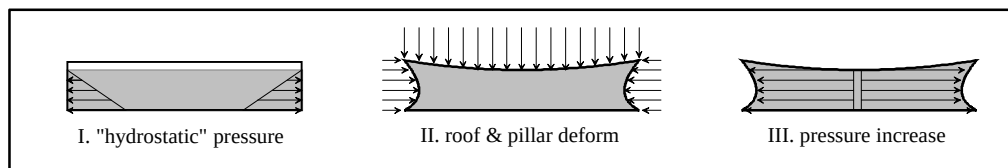


Figure 5.1 – Development of Maximum Fill Pressure

backfill should have similar or greater initial stiffness than the coal pillar (Clark and Boyd, 1998). However, even very compressible fills will eventually develop passive resistance to surrounding deformations based solely on volumetric constraint. As mentioned earlier, there is difficulty in assessing the amount of surcharge loading caused by vertical displacement of the roof (Fig 5.1 II). Initially, when the SAM cut is made, the overburden load above the cut will be transferred to the adjacent pillars. Over time the pillars will deform, “shedding” the overburden load, and the roof will begin to “sag.” The amount of loading, or the percentage of the total overburden weight, associated with the sagging roof must be quantified in order to calculate the horizontal pressure within the fill. Because the amount of fracturing and caving of roof material above the fill will be dependent on the geologic conditions, the simplest approach is to use a range of surcharge loads. In order to examine the structural capacity of backfill by itself, the minimum surcharge will be equal to zero. Any horizontal pressure exerted at that point

will be a direct result of the backfill properties. Since the SAM cuts are so narrow, the unbroken strata above the fractured zone, in some instances, can be assumed to be bridged by side abutments located above the pillars. Based on this phenomenon, the maximum surcharge load will probably never reach the total weight of the overburden. Therefore a maximum surcharge equal to $0.25 s_z$, where s_z is the overburden pressure, will be used.

The actual procedure for incorporating the additional strength provided by the fill is based on Equation 2.16. Substituting s_h from Equation 5.1 into Equation 2.16 yields:

$$s_1' = s_1 + [(g_f \times h_f + p) \times K_{pf} + 2 \times c \times \sqrt{K_{pf}}] \times K_{pp} \quad [5.2]$$

where,

s_1' = pillar strength with fill (lb/in²)

s_1 = pillar strength without fill (lb/in²)

K_{pp} = coefficient of passive earth pressure of the pillar

In this instance the pillar strength without fill, s_1 , is equal to 900 psi. The “new” pillar strength, s_1' , can be readily substituted into the Mark-Bieniawski pillar strength formula (Eqn.4.2) and used in conjunction with the *Mathcad 4.0* pillar program described in the previous section.

5.2 Design of Backfill Supported Pillars

It can be seen from Equation 5.2 that the inclusion of backfill in the pillar calculations introduces a new set of variables. The density of the fill, the angle of internal friction of the fill and of the coal pillar, the cohesion of the fill, and the surcharge load. Because of the numerous variables, design charts were developed which can be used to ultimately determine the pillar widths based on a range of fill properties and underground conditions. In order to determine the additional strength added to the pillar, the total horizontal pressure exerted by the fill was broken into three components:

$$s_{h-T} = s_{h-f} + s_{h-c} + s_{h-sl} \quad [5.3]$$

where,

s_{h-T} = the total horizontal pressure (lb/in²)

s_{h-f} = the horizontal pressure due to the weight of the fill (lb/in²)

s_{h-c} = the horizontal pressure due the cohesion of the fill (lb/in²)

s_{h-sl} = the horizontal pressure due to surcharge loading of the fill (lb/in²)

Each component has a corresponding design chart from which their magnitude can be determined. The sum of those values can then be used to find the “new” pillar strength from a fourth design chart. Finally, the pillar widths associated with the new strength can be read from a design chart like the ones presented in section 4.3.1.

The first design chart can be used to determine the horizontal pressure exerted by the weight of the fill only. Since it is assumed that the backfill occupies the entire cut, there is one of these design charts for each seam height (Fig 5.2a-c). The two other components of the fill pressure are independent of seam height; thus the charts pertain to any underground conditions (Fig 5.3-5.4). As with the initial pillar design charts, the ones including backfill are dependent on seam height and cut width. Only the charts for a 7’ cut width are presented here (Fig 5.6-5.8).

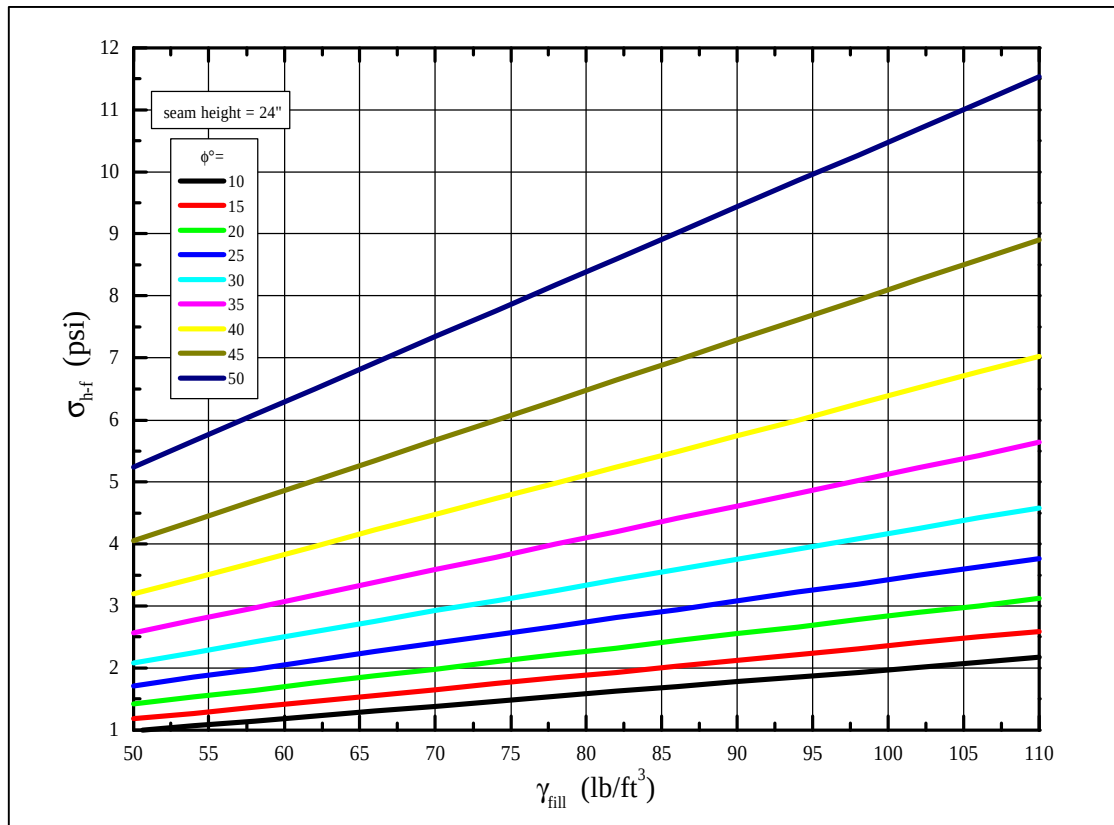


Figure 5.2a – Lateral Pressure Due to Fill Weight (24” Seam)

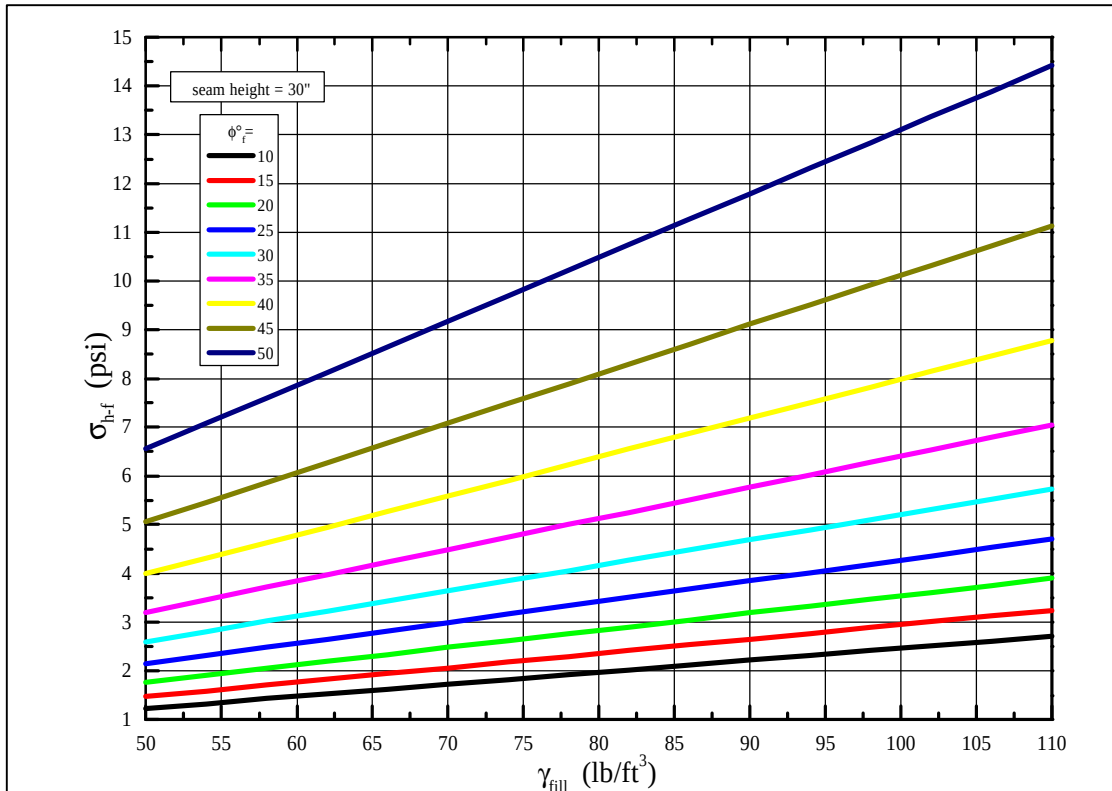


Figure 5.2b – Lateral Pressure Due to Fill Weight (30" Seam)

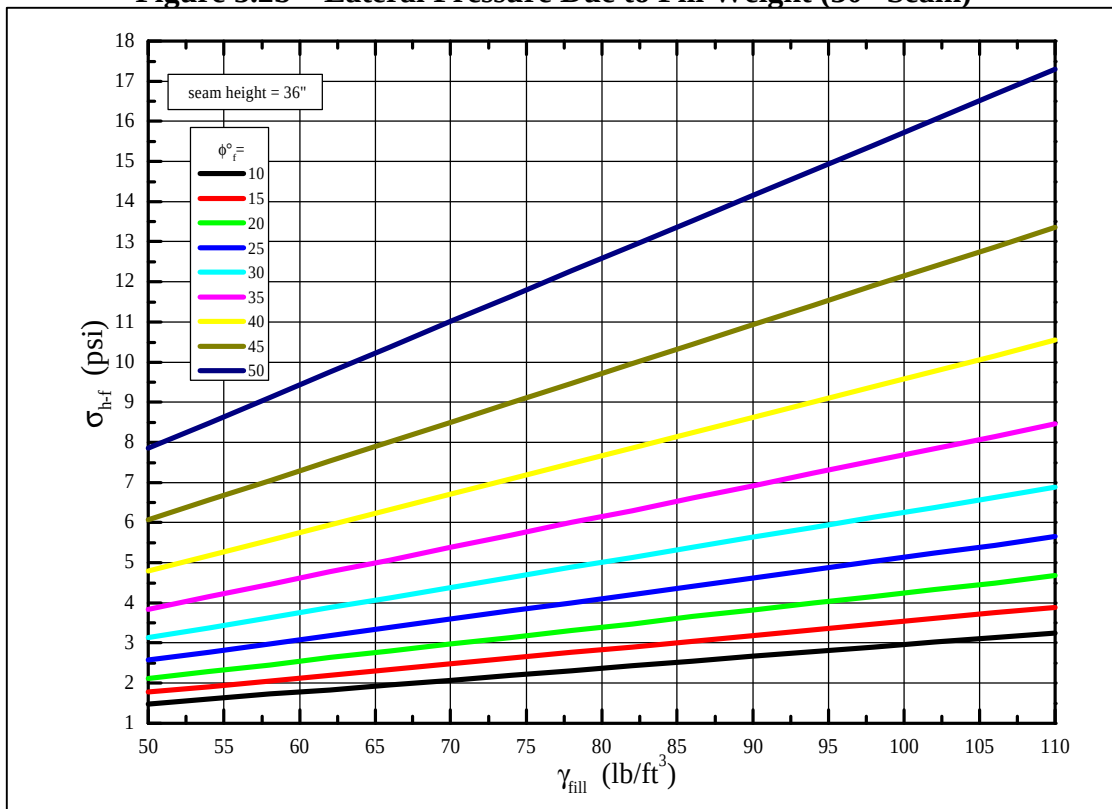


Figure 5.2c – Lateral Pressure Due to Fill Weight (36" Seam)

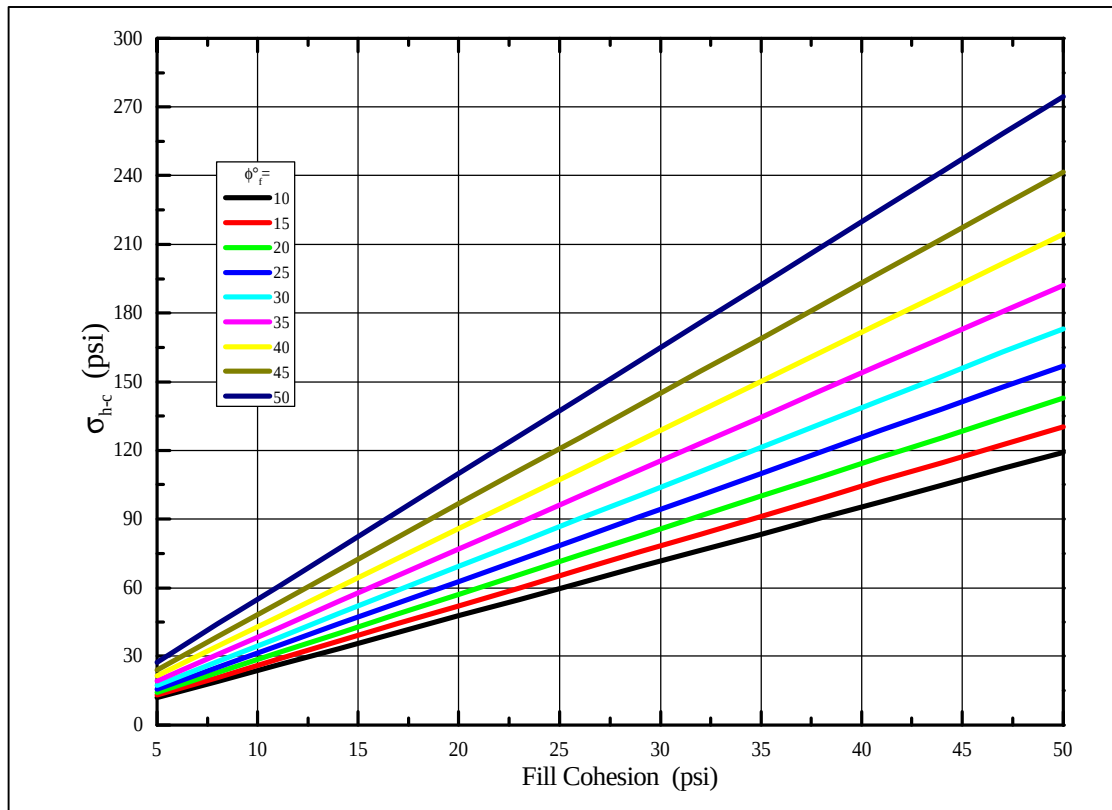


Figure 5.3 – Lateral Pressure Due to Fill Cohesion

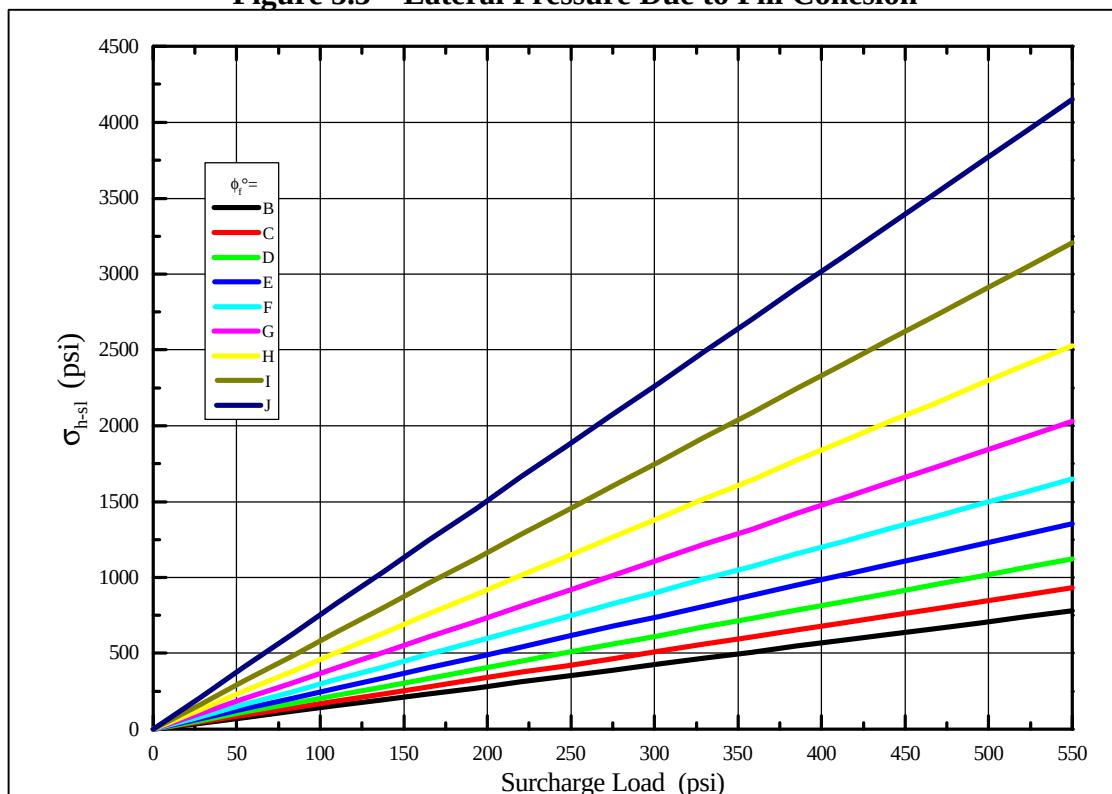


Figure 5.4 – Lateral Pressure Due to Surcharge Loading

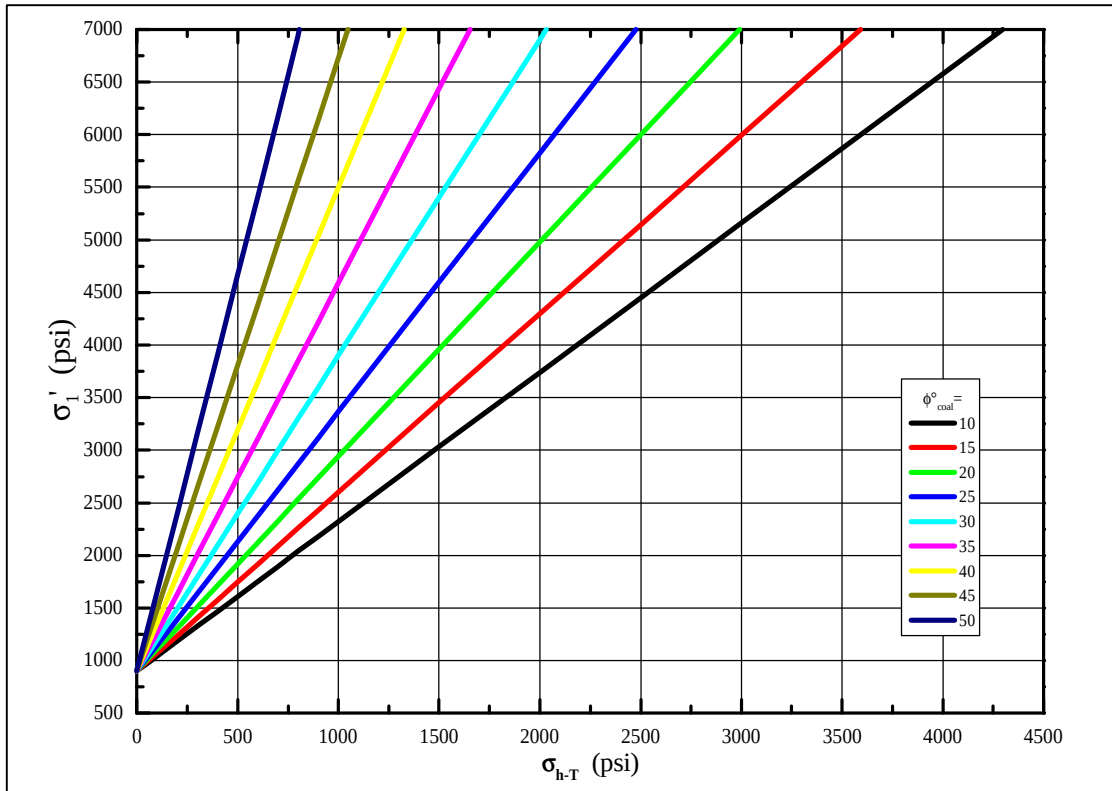


Figure 5.5 – Strength of Backfill Supported Pillars

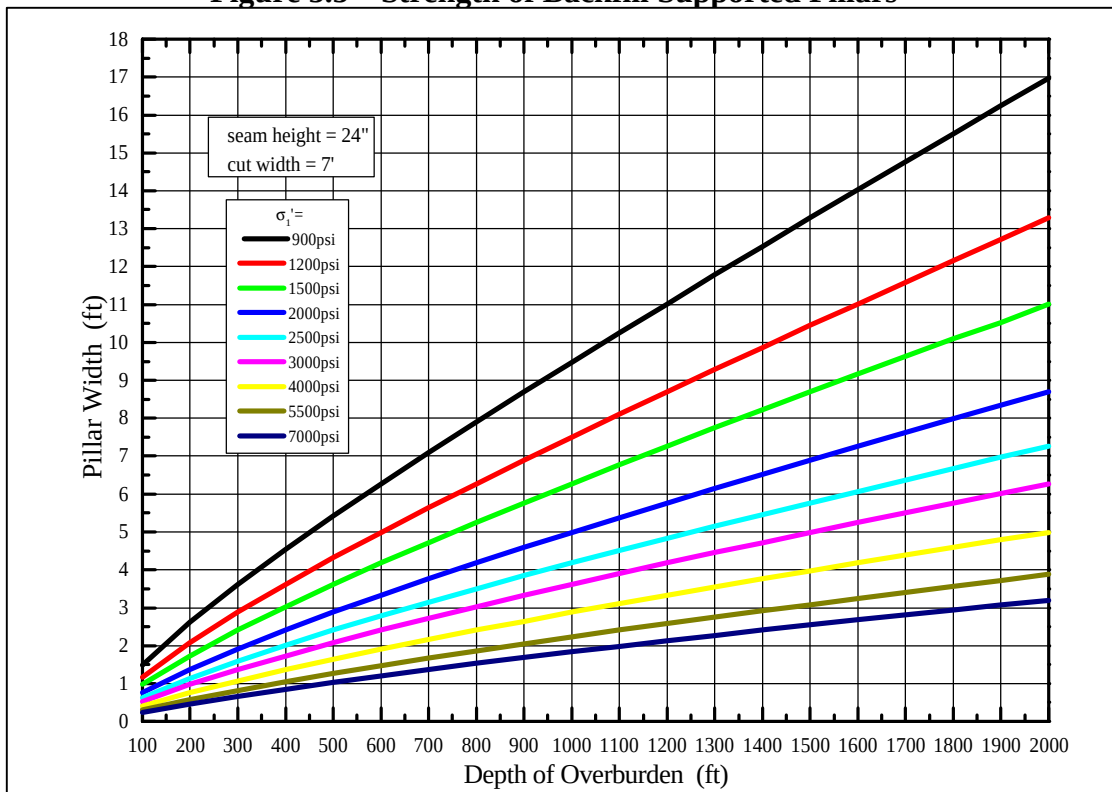


Figure 5.6 – Widths of Backfill Supported Pillars (24" Seam)

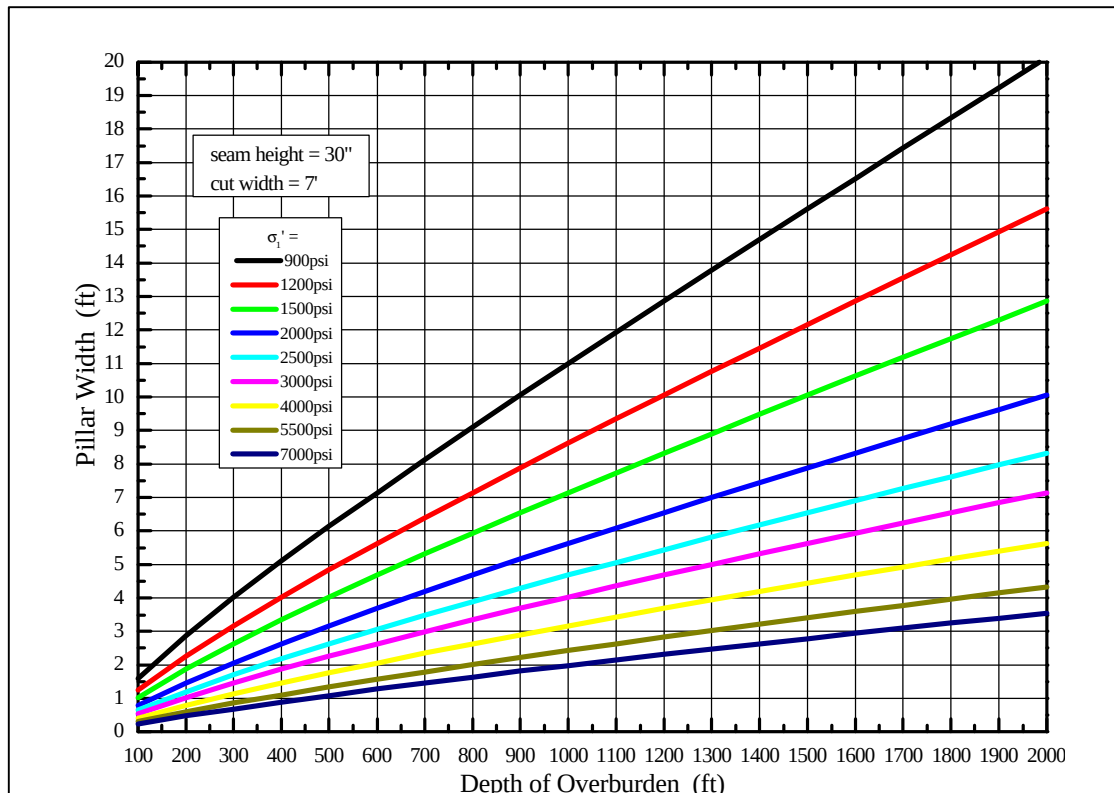


Figure 5.7 – Widths of Backfilled Supported Pillars (30” Seam)

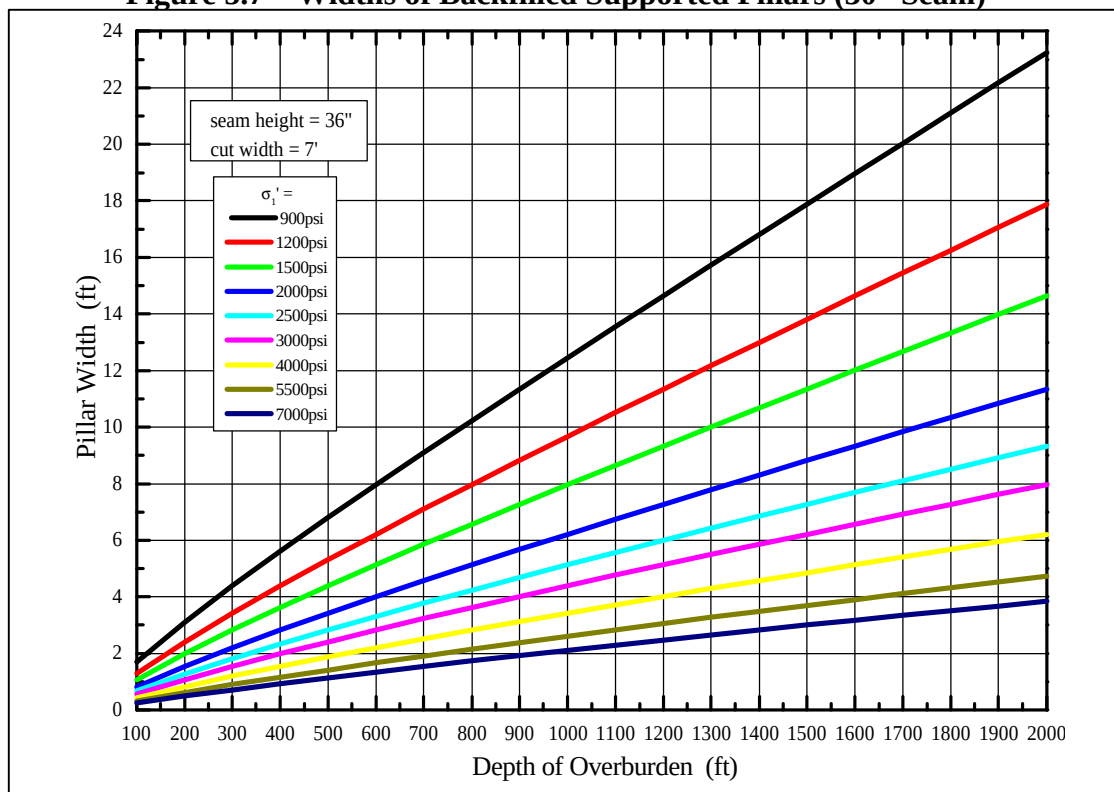


Figure 5.8 – Widths of Backfilled Supported Pillars (36” Seam)

Figures 5.4 and 5.5 indicate that the lateral pressure developed in the fill due to surcharge loading has the most influence on gains in pillar strength. A maximum allowable surcharge load of $0.25 s_z$ was previously set because of a lack of understanding of the behavior fill underground. In addition, the maximum increase in pillar strength given in Figures 5.5 through 5.8 is approximately 6000 psi ($s_1' \sim 7000$ psi). Although larger increases in pillar strength may be possible, the effectiveness of fill as a ground support method needs to be quantified further (Hume and Searle, 1998). Therefore, conservatism was used to set the maximum amount of fill support. However, the ability of fill to reduce the widths of the pillars, even for small increases in strength, is illustrated by Figures 5.6, 5.7, and 5.8. The purpose of Figures 5.6-5.8 is not only to show the smaller pillar widths possible due to backfill. They can also be used to determine a range of conditions that will be necessary to satisfy a “predetermined” pillar width. This will be helpful in the design of a suitable backfill (i.e., density, compressibility, cohesion).

5.3 Recovery Rates From Backfilled Panels

As in section 4.4, the widths determined in the previous section were used to calculate the theoretical recovery rates for a singular, backfilled panel. Recovery rates were calculated for each increasing increment of pillar strength due to backfilling (Appendix C). The rates are shown in Figures 5.9 through 5.11 for the three seam heights and for a cut width of 7'. As expected, the rate of recovery increases with increasing backfill support. The greatest increases in recovery occur as the depth increases and the magnitude of support increases, mainly because the “non-backfilled” recovery is so poor at greater depths. The overall range of improvement is from a minimum of 3% to a maximum of 32%. As with Figures 5.6-5.8, these charts can be used in conjunction with Figures 5.2-5.5 to determine certain properties that will satisfy a predetermined rate of recovery or range of recoveries.

It is obvious that backfill can improve recovery. However, the increases in recovery may not validate the cost of the backfilling operation. The cost of backfilling, and other concerns of backfilling, are covered in the following section.

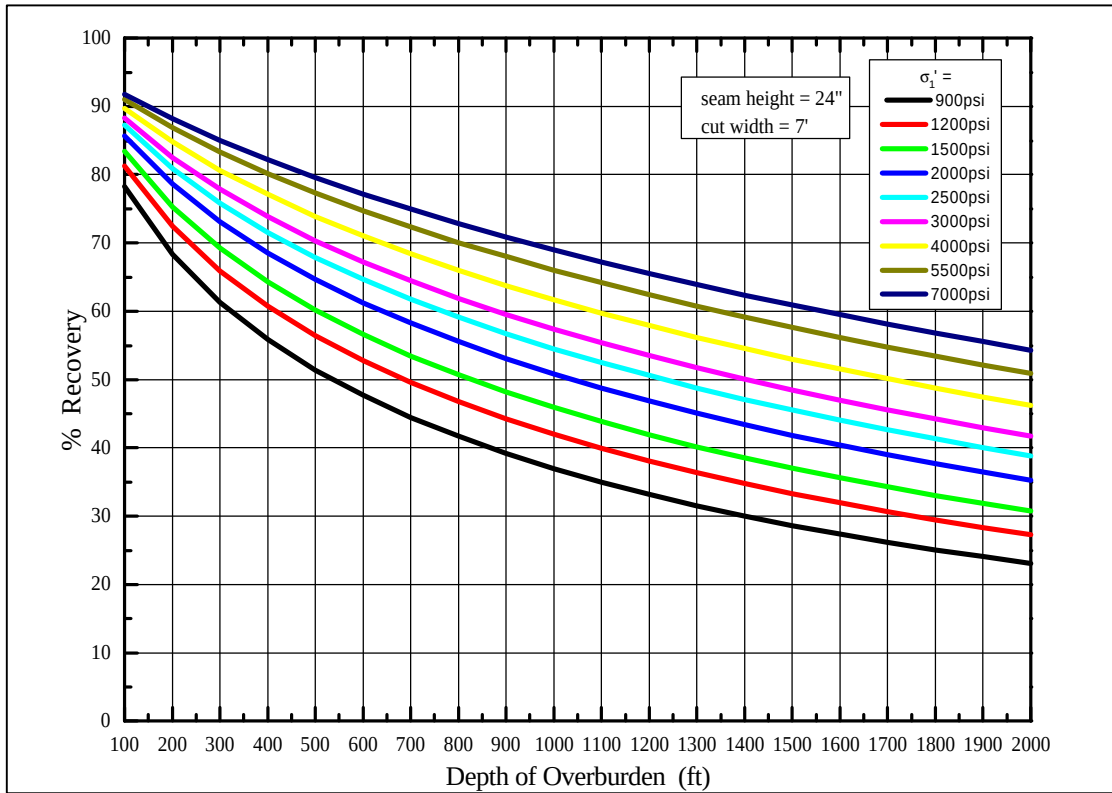


Figure 5.9 – Recovery with Backfill vs. Depth (24” Seam)

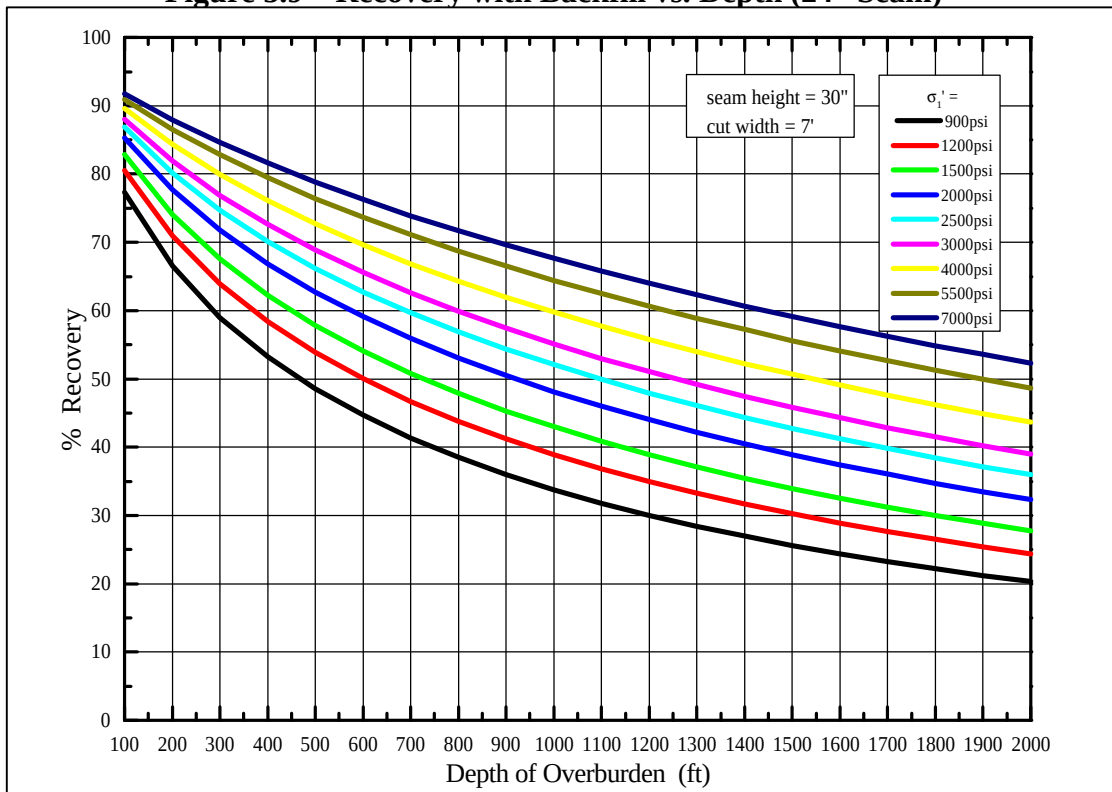


Figure 5.10 – Recovery with Backfill vs. Depth (30” Seam)

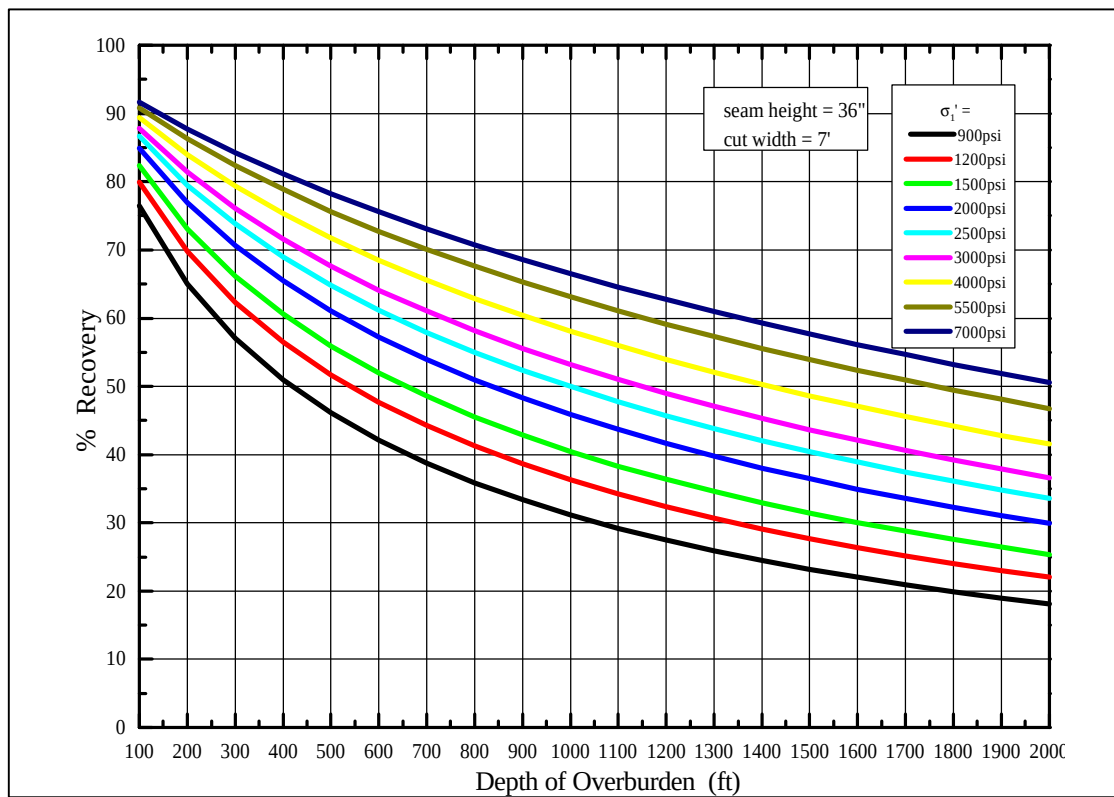


Figure 5.11 – Recovery with Backfill vs. Depth (36” Seam)

CHAPTER 6. BACKFILLING COSTS AND OTHER CONSIDERATIONS

The improvement in recovery created by the use of backfill may not outweigh the increase in costs incurred by the installation and operation of the backfilling system. A cost analysis must be carried out to determine the economic feasibility of backfill. In addition to cost there are other issues concerning backfill that can affect its potential. Backfill material availability, material properties, and possible alterations to the mining operation must also be considered. Based upon that information, conclusions regarding the overall potential of backfill in thin-seam coal mining operations can be drawn.

6.1 Cost Analysis of Backfill

The most importance cost that must be determined is not the cost of the fill itself but rather the cost of the fill required to produce an additional ton of coal (Hume and Searle, 1998). The approach to calculating the cost of fill on this basis is very simple. Hume and Searle used it to determine the costs of different backfilling systems for thin-seam highwall mining. Since thin-seam highwall mining is the closest type of existing mining method to the proposed Self-Advancing Miner operation, the same approach was taken here. The cost of fill required to produce an additional ton of coal is equal to the cost per ton of fill multiplied by the ratio of the recovery due to backfilling to the additional recovery. Written in equation form:

$$Cost = Cost\ of\ Fill\ per\ ton \times \frac{Recovery\ with\ Fill}{(Recovery\ with\ Fill - Recovery\ no\ Fill)}$$

The cost of fill per ton is dependent on many things including material availability and delivery, material properties, and filling method. Since an actual method of placement and a fill material have not yet been designated for the thin-seam technology, three possible methods and their costs are considered (Table 6.1). Several assumptions were made in order to determine these costs. Among the more important were:

1. no additives, such as cement, are to be used
2. one material option was priced for each placement method
3. fill density was set at $1.4\ kg/m^3$ ($\sim 87\ lb/ft^3$)

4. 100% filling of the voids

For a more detailed explanation of the filling costs and the filling methods, refer to Hume and Searle (1998).

Table 6.1 – Backfilling Methods and Costs

Filling Method	Capital Costs		Oper. Costs		Total	
	\$/yd ³	\$/ton	\$/yd ³	\$/ton	\$/yd ³	\$/ton
1. Hydraulic Delivery	0.17	0.15	2.40	2.04	2.57	2.19
2. Pneumatic Delivery	0.27	0.23	2.28	1.94	2.55	2.17
3. Filling on retreat using the miner	0.66	0.56	1.64	1.40	2.30	1.96

The costs in Table 6.1 are given in \$/yd³ so that if a different fill density is required, the cost per ton can be adjusted. The cost per ton in Table 6.1 assumes a fill density of about 87 lb/ft³. This falls within the density range used in section 5.2 and will be used in the first stage of the cost analysis. The sensitivity of the cost to changes in fill density will be analyzed in the second stage. Using the aforementioned formula and costs, Figures 6.1-6.3 were developed and illustrate the cost of fill for every extra ton of

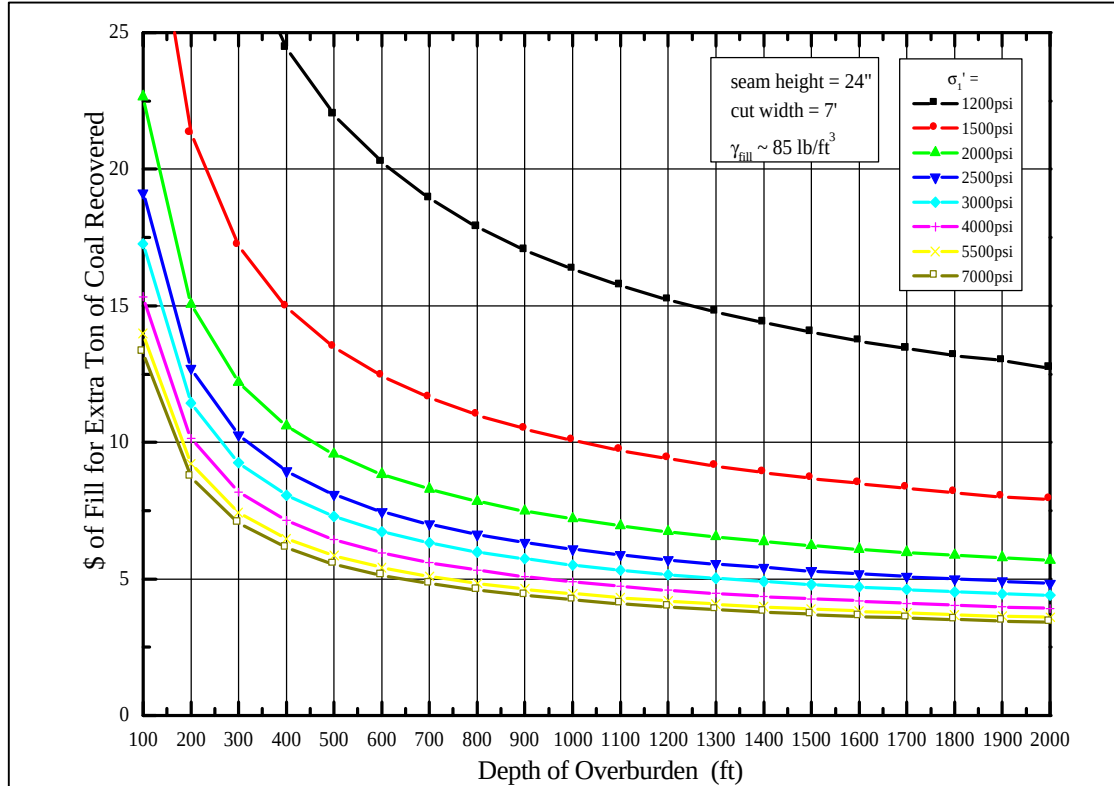


Figure 6.1 – Fill Cost vs. Depth (24” Seam)

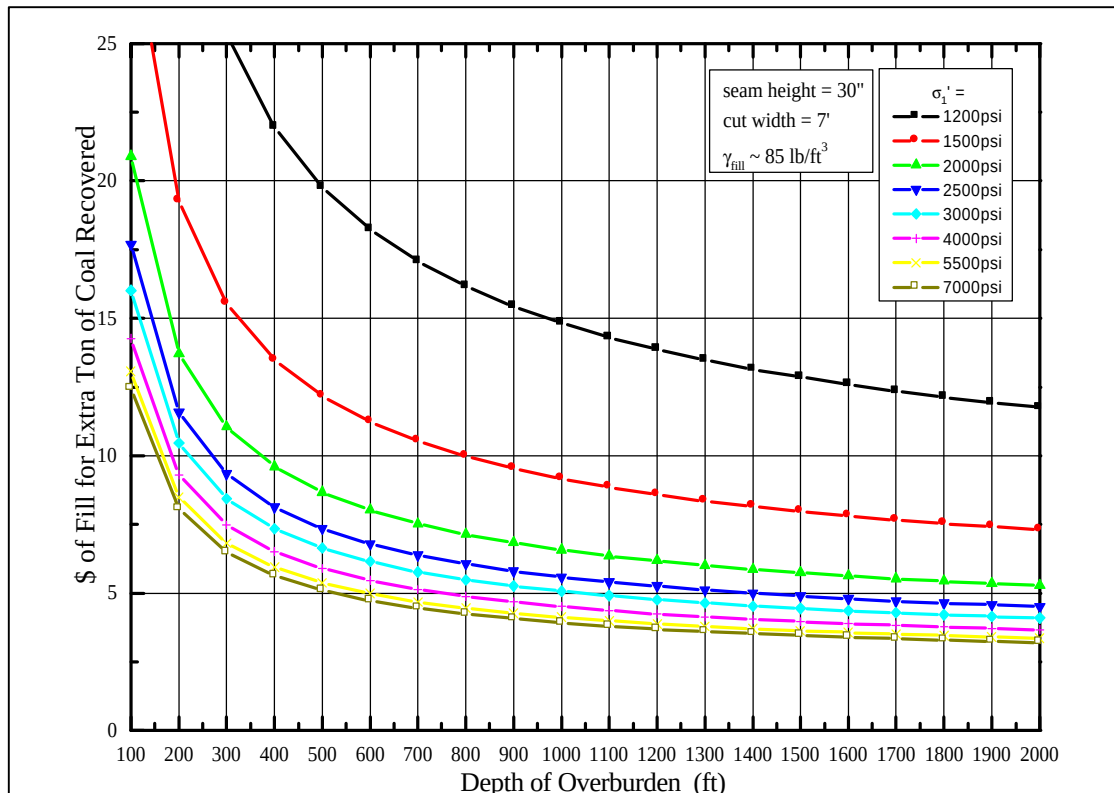


Figure 6.2 – Fill Cost vs. Depth (30" Seam)

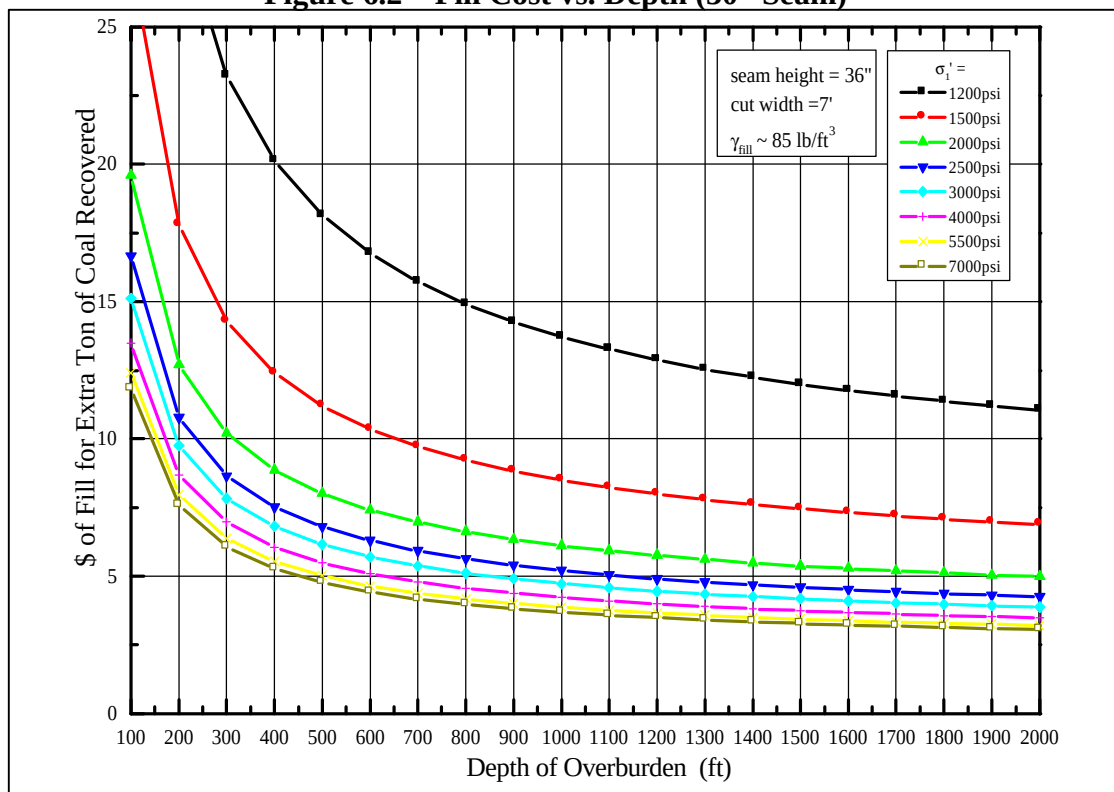


Figure 6.3 – Fill Cost vs. Depth (36" Seam)

coal recovered versus depth, and for the same range of pillar strengths used previously. The total cost taken from Table 6.1 was \$1.96, given for the placement method of filling on retreat. This method seems viable for incorporation and use with the technology proposed for the Self-Advancing Miner. The general trend illustrated by Figures 6.1-6.3 would not change if one of the other two costs were used. The higher costs associated with those methods of placement would affect the overall fill cost only when there is little increase in recovery due to backfilling. However, when that occurs it seems that filling is not economical regardless of the cost of placement. Figures 6.1-6.3 indicate that the ratio of the recovery with fill to the increase in recovery is the most critical factor in fill cost. Thus the cost of fill decreases with depth and increased pillar strength because filling has a greater effect on recovery under these circumstances.

One of the problems with the cost analysis, and Figures 6.1-6.3, is that one value of density is used for the entire range of pillar strengths. Although the supporting effect of fill has not been quantified definitively, it can be concluded that only very dense fills will be able to increase the pillar strength to values approaching 7000 psi. Additionally, Table 6.1 indicates that as the density of the fill increases, the cost of the fill per ton decreases. Thus a denser fill will provide more support at a lower cost (Figure 6.4). This is not completely accurate since improvements made in fill density will incur costs due to activities such as material blending and quality control. However, Figure 6.4 reveals that low density fills may not be economic. Therefore, a certain range of fill densities can be eliminated from possible use based on cost rather than theoretical performance.

With the cost of fill per ton of extra coal recovered displayed over a range of backfilled pillar strengths and fill densities, a cut-off point for the cost must be determined. Again, since the Self-Advancing Miner and corresponding thin-seam technology is a design stage, a thorough understanding of the mining costs does not exist. Because the fill costs were developed per ton of extra coal recovered, a simple presumption is that the filling cost should not exceed the average profit made per ton of extracted coal. Assuming an average profit of no more than \$5.00 per ton of coal, at the mine level, the implementation and operation of a backfilling system should not exceed \$5.00. Figures 6.1-6.3 reveal that there are circumstances in which backfill can be applied at costs equal to or less than \$5.00. From the charts it can be seen that the fill

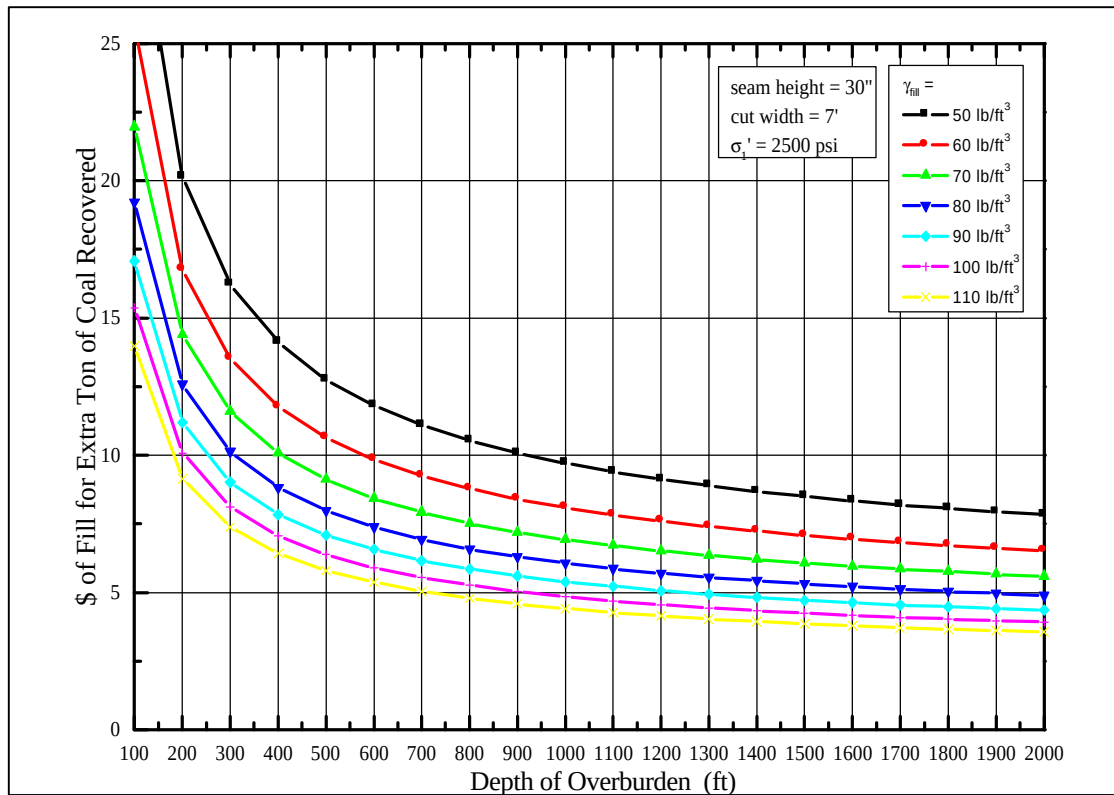


Figure 6.4 – Sensitivity of Cost to Fill Density

must provide at least 1000 psi of additional support ($s_1' = 2000$ psi) for it to become economic. The charts also indicate that backfill will be economical at depths greater than 500'. Again, this is due to the combination of poor initial recoveries (w/o fill) and large increases in those recoveries due to filling. With the maximum cost set, Figure 6.4 indicates that the fill density probably has to be at least 70 lb/ft³. Although slightly less dense fills (~60 lb/ft³) would probably "graph economically" for greater values of s_1' (3000 to 7000 psi), there is little chance that they could provide the necessary amount of support to reach those values of s_1' .

Backfilling is economically feasible under certain conditions. At a depth of 500' filling becomes economic but it is dependent on the amount of lateral support provided by the fill. The minimum amount of lateral restraint provided by the fill must be able to increase the pillar strength to at least 2000 psi. Backfilling will also be more economic as the seam height and cut width are increased. Additionally the density of the fill must be at least 70 lb/ft³. Fills below this density do not meet economic or structural requirements. It can be concluded that in order to meet economic constraints, backfill

applied in thin-seam coal mining must have a density of 70 lb/ft³ or greater and it must be able to increase the pillar strength to at least 2000 psi.

6.2 Viable Backfill Materials

The fill costs in Table 6.1 were developed using one material option per placement method. The cost analysis was carried out using placement method 3, filling on retreat using the miner. The tabulated costs are based on the proposition that a dry or damp aggregate would be used. A wet fill, namely a hydraulic fill, may not be compatible with the placement method because the miner may become flooded if it is retreating on a downward dip. This would seem to eliminate the use of tailings reject as a fill material. However, if further dewatering of the reject was carried out using additional thickeners the washery refuse may be feasible as a backfill material. Although plant reject would only account for approximately 5% to 15% of the total fill volume needed, more material can be obtained from other spoil piles. This application may offset some of the other material costs perhaps in the range of \$0.11 to \$0.29 per ton of extra coal recovered (Hume and Searle, 1998).

Because of the low density of coal waste it will not be possible to completely fill the voids with a backfill completely comprised of reject material. Although the use of cement or other pozzolans will increase the bulk density of reject, the costs in the previous section were calculated assuming no additives. A mixture of coal waste and crushed aggregate seems to be the most feasible material option. Material availability is the main concern of this option. The fill costs were determined assuming aggregate would be available on site and included costs for crushing and screening equipment. But that was specifically for highwall mine operating at a depth of about 500', where enough material would be excavated to completely fill all voids. In the thin-seam coal mine it has been assumed that each panel will be developed using three-entry systems with entry widths equal to 20'. There is little chance that development work will create enough material to completely fill all voids. This is also true if the coal waste volume is included. The amount of fill material needed is dependent on the total recovery which itself is dependent on the lateral restraint provided by the fill. As the pillar strength

increases due to increasing confinement, the pillars become thinner, more cuts are made, thus more material is required (Figure 6.5). Although Figures 6.1-6.3 indicated that fill costs decrease with increasing confinement, it was assumed that there was enough material on site for 100% filling. This probably will not be true for underground thin-seam coal mining, using a high strength fill may actually increase costs since additional “outside” materials would have to be acquired. Figure 6.5 shows that, depending on

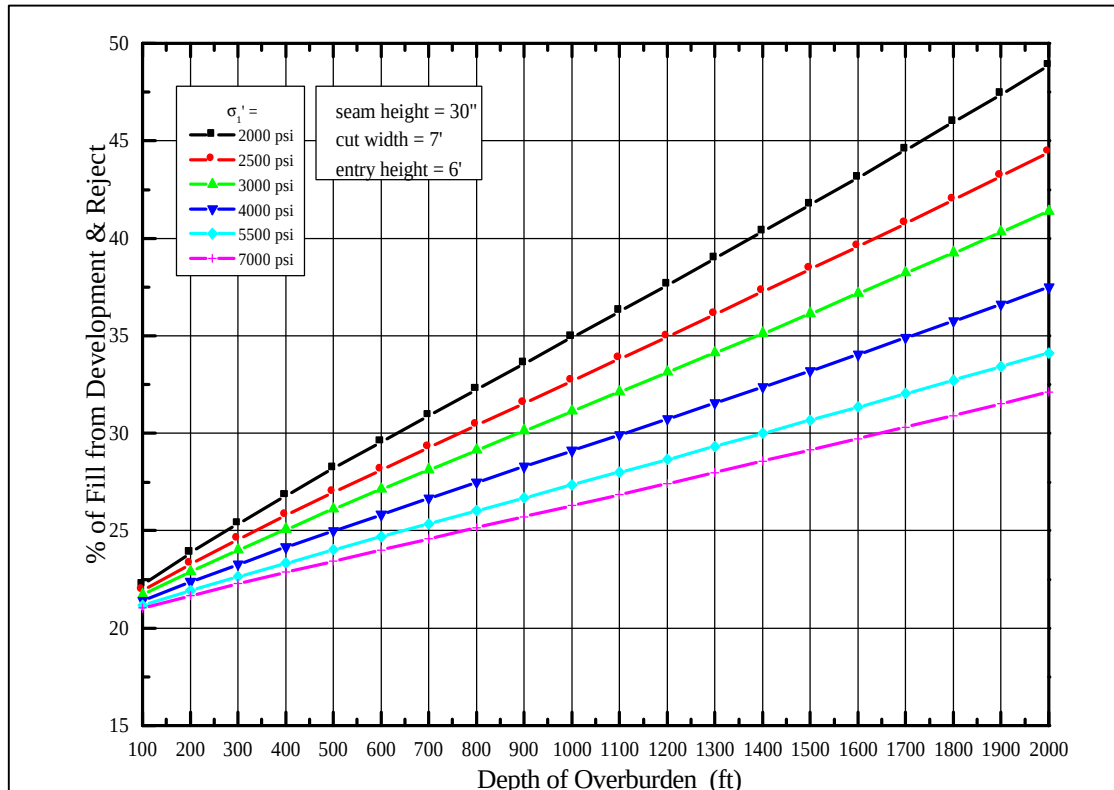


Figure 6.5 – Percentage of Total Fill Material “On-Hand”

the backfilled pillar strength, up to 47% of the fill material can be generated from the development and cleaning operations (assuming 5% reject from raw coal and 30% swell). On average though, only 25% to 35% of the total material can be generated from these processes and the rest must be solicited from other sources.

It may be necessary to purchase crushed stone or screened sand in order to completely fill the voids. An alternative may be, if possible, to cut “higher” entries. Depending on the geologic conditions and any effects on the cutting cycle, the entry height can be increased in order to generate more on-site fill material. The increase in the volume of fill material due to a change in entry height is shown in Figure 6.6.

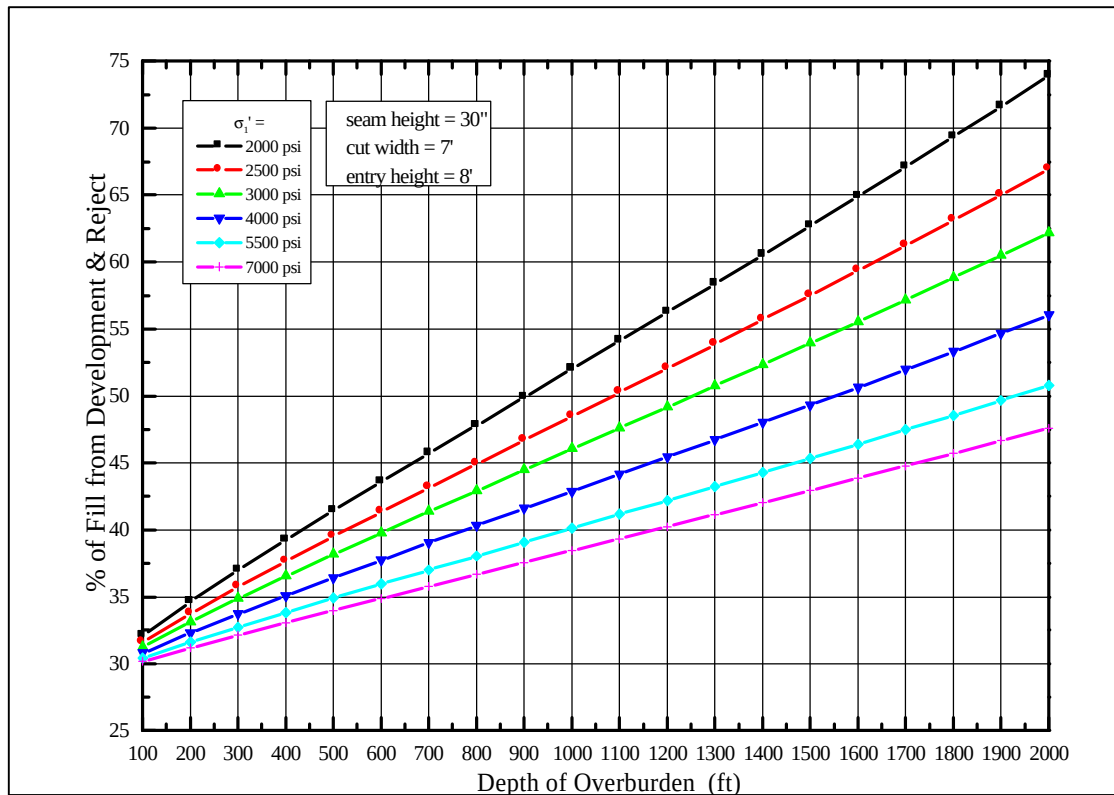


Figure 6.6 – Increase In Available Fill Material

It can be seen that a substantial increase in the available volume of material can take place by increasing the entry height. On average, a 15% to 20% increase is possible. A more thorough examination of the benefits and disadvantages of this alternative is necessary and will be site specific in terms of geologic conditions.

Assuming, generously, that 50% of the total fill volume can be generated from the mining and coal preparation operations, a combination of other materials will be necessary to ensure 100% filling. A combination of sand, fly ash, or other pozzolanic slags, and cement is one possibility. Research has shown that river sands work well in mixtures with coal refuse and pozzolanic material (section 2.5). Mixing 25% (by dry weight) of sand with cemented coal waste has been shown to make the waste more workable while reducing the water requirements (Knissel and Helms, 1983). Of course the amount of cement would have to be kept to a minimum. However, as mentioned earlier, the use of waste materials can have an off-setting effect on the fill cost.

Employing fly ash or other furnace slags obtained from power plants will qualify as a cheap cementing agent and possibly offset some of increase in cost due to the use of

portland cement. The weight content of the fly ash should be at least 6.5% but may be limited by availability. The cement content should be kept around 3% in order to curtail costs and because at that content the use of other pozzolans becomes increasingly beneficial.

If the aforementioned mixture of materials does not account for 100% of the necessary volume, additional aggregate can be acquired. If the costs increase to a point where backfilling becomes uneconomic, partial filling of the voids should be considered. Although it has not been examined or discussed in this case, partial filling of the voids can be used, to some extent, to increase pillar stability and overall recovery.

6.3 Material Properties and Requirements

It was already concluded that the density of the fill, regardless of material type, should be greater than 70 lb/ft³. Blending the available materials together will have to be carried out with this requirement in mind. Of the materials proposed for use in the backfill mixture, the coal refuse will have the lowest density. The percentage of coarse and fine refuse will determine the overall density of the coal waste. Coarse coal refuse has an approximate average dry density of 98 lb/ft³ while the dry density of fine refuse is around 52 lb/ft³ (Cheng and Usman, 1983). Although the density of the fine refuse is quite low, its influence on the overall fill density will not be great. If a maximum of 15% of the total fill material is coal waste and all of that is in the form of fine refuse, the density of the other materials will have more of an effect on the overall density. For instance, the crushed aggregate generated during development may make up to 45% of the total volume and have an average density of about 140 lb/ft³. Additional aggregate, sand, and cementing agents should help keep the overall density around 90 lb/ft³ to 100 lb/ft³.

In order to optimize the density of the fill, blending will have to be undertaken with care. The denser the fill is upon placement, the sooner it will begin to resist pillar deformations. For the fill to provide adequate support it must have similar or greater initial stiffness than the coal pillar (Clark and Boyd, 1998). Proper sizing of the aggregate and overall mixture will ensure that void space is kept to a minimum. The use

of cement and fly ash will also increase the initial stiffness of the fill. Aside from proper blending of the materials before filling, compacting the material upon placement may also increase the density and stiffness of the backfill.

The water content of the fill mixture should be kept at the minimum required for proper mixing of the cementing agents. Knissel and Helms' (1983) work indicated that the strength of cemented coal backfill can be optimized at water contents less than 12% and, on average around 9%. The water content of the coal refuse portion may have to be controlled with thickeners or other additives in addition to the sand. If the water content is too high, flooding of the miner is possible. Therefore the fill should maintain a "pasty" consistency. Paste backfills, mixtures of dewatered tailings and cement, have been used successfully in other mining operations. The consistency of paste backfills is similar to that of medium-slump concrete. The addition of aggregates, as in this case, is possible without significantly affecting the characteristics of the paste (Brackebusch, 1994). In addition to the advantage of reduced dewatering, paste fill also has lower labor and materials costs, reduces tailings disposal requirements, and has competent support properties.

As discussed in sections 2.2 through 2.5, extensive testing of the fill materials will be necessary in order to assess the material properties. Standard soil testing and classification of the individual materials and the mixture as a whole will be important in determining optimum characteristics, behavior, and mix design. A system of quality control measures should be developed and maintained so that poorly engineered backfill and inconsistent in situ performance of the fill is avoided. For a more complete list of recommended standards refer to Archibald (et al., 1993).

6.4 Affect of Backfill on the Mining Sequence

The entire pillar design process was carried out ignoring all types of abutment loading. Although there are no plans for yielding of any pillars or the development of gob areas, there may be some abutment loading caused by the mining sequence and the backfilling operation. The two options to consider when examining the mining sequence are:

1. mining cuts sequentially leaving the minimum increment between cuts
2. leaving larger pillars between cuts before returning to recover the coal left in the larger pillar.

Both options assume that filling will be performed by the miner upon its retreat from each cut, but they present opposing circumstances in terms of abutment loading.

Figure 6.7 illustrates each sequence option and the effects of abutment loading.

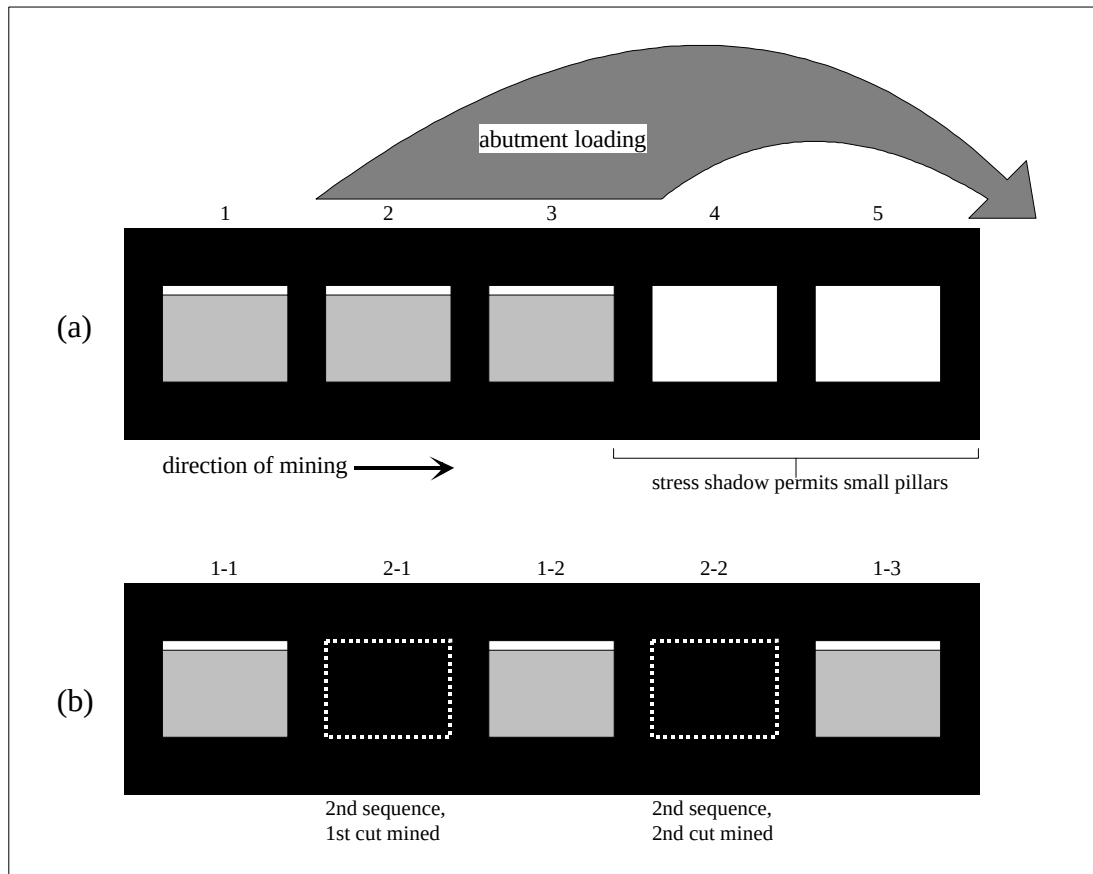


Figure 6.7 – Alternative Mining Sequences

(a) mining inside forward abutment

(b) mining outside forward abutment

(After Clark and Boyd, 1998)

Alternative 1, with sequential mining and immediate filling, takes place inside a forward abutment load. Thus the last formed pillar falls within a stress arch. This will give the freshly placed backfill a chance to set-up and stiffen before loading from the overburden is redistributed throughout the pillar. Alternative 2 works in the same manner. From Figure 6.7a, after cut 1-3 is made and filled, the miner moves back down the face and commences with cut 2-1. The fill in cuts 1-1 and 1-2 have now had an extended period

of time to set and stiffen without having been subjected to high initial loads because the coal pillar left between them is oversized and stable.

Although Alternative 2 provides more time for the backfill to settle and stiffen it would probably be less efficient than Alternative 1. Moving the miner and conveyors back and forth along the face would delay the entire operation. Therefore leaving the minimum spacing between sequential cuts is the best option assuming immediate filling of the cut as the miner withdraws. The operation is more efficient and safety is maintained by working inside the abutment created by the previously backfilled cuts.

CHAPTER 7. CONCLUSIONS AND RECOMMENDATIONS

7.1 Conclusions

As the percentage of coal found in thick seams decreases, underground thin-seam coal mining technology must be developed in order to recover reserves located in seams less than 36". The technology and design involved is somewhat analogous to current trends in highwall coal mining. One such technology has already been proposed in which a self-advancing miner, cutting only to the height of the seam, proceeds 600' into the coal seam at an angle of approximately 45°. Before the Self-Advancing Miner is employed the coal seam will be exposed by developing a series of panels. The panels will be approximately 4000' long and 850' wide. As the SAM progresses along the length of the panel pillars will be left between the individual cuts. The opposite side of the panel will be mined accordingly with a "separation" pillar left running through the center of the panel.

The pillars left within the panel have been analyzed and designed to meet safety requirements. Three main types of pillars were defined, analyzed, and sized according to function and stability. Based on the pillar design, the recovery rates of the Self-Advancing Miner were calculated for different seam heights, cut widths, and overburden depths. The range of theoretical recovery rates, without backfill, varies from a minimum of 20% at a depth of 2000' to a maximum of 80% at 100'. There is a slight improvement in recovery if wider cuts are taken in thinner seams. A more reasonable range of recoveries, for depths between 400' and 1200', is 60% to 30%. In order to increase the recovery from the thin-seams, backfilling of the SAM cuts has been proposed.

There is little doubt that backfill has the ability to provide support to surrounding pillars. However, quantifying the magnitude of that support has proven to be quite difficult. Although the support is somewhat dependent on the fill's properties, it was found that the deformation of the roof and surrounding pillars has the most effect on the magnitude of restraint provided by the backfill. Using classical earth pressure theory, the magnitude of fill support can be incorporated into the design procedure used to calculate the initial pillar widths. A "new" pillar strength due to backfilling can be calculated by

adding the original pillar strength to the amount of horizontal pressure exerted by the fill multiplied by the angle of internal friction of the coal pillar. Design charts were developed and can be used to determine the “new” pillar strength for a range of fill densities, fill cohesions, fill angles of internal friction, surcharge loads on the fill, and coal angles of internal friction. The increase in recovery due to backfill increased from a minimum of 3% to a maximum of 32% dependent on the amount of pressure exerted by the fill. If the backfill is capable of doubling the strength of the fill, an increase in recovery of 15% is possible at any depth.

Although large increases in recovery are possible with backfill, the cost of the filling operation may offset those gains. When determining the cost of backfill, the cost of fill per extra ton of coal recovered is more critical than the actual per ton cost of the fill. Using this approach, costs of filling were calculated for varying fill pressures and overburden depths. The cost of backfilling decreased dramatically with increasing depth and magnitude of the horizontal fill pressure since large increases in recovery were possible under these circumstances. Based on the cost data it was determined that backfilling is not economically feasible at depths less than 500’ or when the fill fails to provide enough support to increase the pillar strength beyond 2000 psi.

Additional considerations were made in regards to the backfill. The cost analysis was carried out assuming that the method of placement would consist of the miner filling each cut as it withdraws from the cut. This system of filling is the most viable when considering the self-advancing miner technology. The actual fill material will consist of a combination of waste rock generated during panel development, tailings from the coal washing process, screened sand, fly ash or other pozzolanic waste, and a small portion of cement. It was determined that a maximum of 50% of the fill material can be obtained from the mining operation (development and preparation). The additional materials will have to be obtained off-site, which may affect the overall cost. However, using waste materials such as fly ash may off-set the additional costs. Material testing and quality control of the mixing and placement operations must be carried out in order to ensure proper in situ performance of the backfill.

Using backfill to increase recovery from underground thin-seam coal mines is feasible. Its ability to increase pillar strength allows for the design of smaller pillars,

resulting in increased coal recovery. From an economic standpoint, the increase in recovery must be large enough to offset the capital and operating costs of backfill. This type of situation is possible and indicates that backfill is economically feasible. The actual design of a backfill mix and placement operation for thin-seam mining is contingent upon a more thorough examination of certain aspects of backfilling.

7.2 Recommendations

Although this study indicates that backfill is feasible, more detail is required before concluding that backfill is the best option for increasing recovery in thin-seam coal mines.

The entire thin-seam coal mining operation, including mine design and layout, system operations, mining sequence, adaptability to backfilling, and all other aspects of the mining method must be finalized before the implementation of a filling strategy is undertaken. The overall costs of the mining operation must also be determined in order to calculate the exact cost of fill per extra ton of coal recovered.

A more precise way of quantifying the additional support provided by the fill is necessary in order to estimate possible improvements in recovery. Laboratory studies of the backfill mix, particularly its interaction with the surrounding pillars are needed in order to examine the fill's behavior. In situ testing of backfill in similar conditions, such as in highwall mining, may aid in these studies.

Testing of cheap, alternative backfill materials should be performed. In addition to coal waste and fly ash other solid wastes, such as smelter slag and municipal waste, should be considered as material options and subjected to strength and durability analyses.

When information regarding a specific backfill (i.e., materials, mix design, magnitude of support, placement, etc.) is obtained, a feasibility study that focuses solely on that backfill should be conducted so that improvements in recovery and ground support can be determined per individual backfill design.

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APPENDIX A:

OUTPUT FROM PILLAR DESIGN PROGRAMS

A.1 - GEOMETRIC PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
 Tributary Theory
 Mark-Bieniawski Pillar Strength Formula
 Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
 B = angle of SAM cut (deg)
 L = length of cut (ft)
 H = coal seam thickness (ft)
 C = cut width of SAM (ft)
 E = width of entry (ft)
 Z = range of depths (ft) [minimum, increment..maximum]
 W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
 W = calculated pillar widths (ft)

S = 900
 B = 45
 L = 600
 H = 2
 C = 7
 E = 20
 Z = 100,200..2000
 W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{ W + C \} \right\}, W \right]$$

Depth	Width
100	1.551
200	2.718
300	3.751
400	4.712
500	5.626
600	6.508
700	7.366
800	8.207
900	9.033
1000	9.848
1100	10.653
1200	11.451
1300	12.242
1400	13.027
1500	13.808
1600	14.584
1700	15.357
1800	16.128
1900	16.895
2000	17.660

A.2a - RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
Rib Pillar Design
Mark-Bieniawski Pillar Strength Formula
Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
L = length of cut (ft)
H = coal seam thickness (ft)
C = cut width of SAM (ft)
Z = range of depths (ft) [minimum, increment..maximum]
W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
W = calculated pillar widths (ft)

S = 900
L = 600
H = 2
C = 7
Z = 100,200..2000
W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{ W + C \} \right\}, W \right]$$

Depth	Width
100	1.493
200	2.620
300	3.617
400	4.543
500	5.423
600	6.272
700	7.098
800	7.906
900	8.700
1000	9.482
1100	10.255
1200	11.021
1300	11.779
1400	12.533
1500	13.281
1600	14.025
1700	14.766
1800	15.504
1900	16.224
2000	16.971

A.2b - RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
Rib Pillar Design
Mark-Bieniawski Pillar Strength Formula
Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
L = length of cut (ft)
H = coal seam thickness (ft)
C = cut width of SAM (ft)
Z = range of depths (ft) [minimum, increment..maximum]
W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
W = calculated pillar widths (ft)

S = 900
L = 600
H = 2
C = 10
Z = 100,200..2000
W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{ W + C \} \right\}, W \right]$$

Depth	Width
100	1.895
200	3.216
300	4.355
400	5.396
500	6.374
600	7.308
700	8.210
800	9.086
900	9.942
1000	10.782
1100	11.608
1200	12.423
1300	13.227
1400	14.024
1500	14.814
1600	15.595
1700	16.372
1800	17.144
1900	17.912
2000	18.675

A.2c - RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
 Rib Pillar Design
 Mark-Bieniawski Pillar Strength Formula
 Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
 L = length of cut (ft)
 H = coal seam thickness (ft)
 C = cut width of SAM (ft)
 Z = range of depths (ft) [minimum, increment..maximum]
 W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
 W = calculated pillar widths (ft)

S = 900
 L = 600
 H = 2
 C = 12
 Z = 100,200..2000
 W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{ W + C \} \right\}, W \right]$$

Depth	Width
100	2.132
200	3.565
300	4.786
400	5.894
500	6.930
600	7.915
700	8.862
800	9.780
900	10.674
1000	11.549
1100	12.408
1200	13.253
1300	14.087
1400	14.910
1500	15.725
1600	16.532
1700	17.332
1800	18.126
1900	18.915
2000	19.698

A.3a - RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
 Rib Pillar Design
 Mark-Bieniawski Pillar Strength Formula
 Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
 L = length of cut (ft)
 H = coal seam thickness (ft)
 C = cut width of SAM (ft)
 Z = range of depths (ft) [minimum, increment..maximum]
 W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
 W = calculated pillar widths (ft)

S = 900
 L = 600
 H = 2.5
 C = 7
 Z = 100,200..2000
 W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{ W + C \} \right\}, W \right]$$

Depth	Width
100	1.600
200	2.874
300	4.023
400	5.103
500	6.140
600	7.146
700	8.130
800	9.098
900	10.052
1000	10.995
1100	11.931
1200	12.859
1300	13.781
1400	14.699
1500	15.612
1600	16.522
1700	17.430
1800	18.335
1900	19.237
2000	20.138

A.3b - RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
Rib Pillar Design
Mark-Bieniawski Pillar Strength Formula
Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
L = length of cut (ft)
H = coal seam thickness (ft)
C = cut width of SAM (ft)
Z = range of depths (ft) [minimum, increment..maximum]
W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
W = calculated pillar widths (ft)

S = 900
L = 600
H = 2.5
C = 10
Z = 100,200..2000
W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{ W + C \} \right\}, W \right]$$

Depth	Width
100	2.043
200	3.539
300	4.848
400	6.057
500	7.202
600	8.302
700	9.369
800	10.410
900	11.431
1000	12.436
1100	13.427
1200	14.407
1300	15.378
1400	16.340
1500	17.296
1600	18.245
1700	19.190
1800	20.130
1900	21.066
2000	21.998

A.3c - RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
 Rib Pillar Design
 Mark-Bieniawski Pillar Strength Formula
 Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
 L = length of cut (ft)
 H = coal seam thickness (ft)
 C = cut width of SAM (ft)
 Z = range of depths (ft) [minimum, increment..maximum]
 W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
 W = calculated pillar widths (ft)

S = 900
 L = 600
 H = 2.5
 C = 12
 Z = 100,200..2000
 W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{ W + C \} \right\}, W \right]$$

Depth	Width
100	2.306
200	3.928
300	5.331
400	6.615
500	7.824
600	8.980
700	10.097
800	11.183
900	12.245
1000	13.288
1100	14.314
1200	15.327
1300	16.328
1400	17.319
1500	18.302
1600	19.277
1700	20.246
1800	21.209
1900	22.167
2000	23.120

A.4a - RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
Rib Pillar Design
Mark-Bieniawski Pillar Strength Formula
Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
L = length of cut (ft)
H = coal seam thickness (ft)
C = cut width of SAM (ft)
Z = range of depths (ft) [minimum, increment..maximum]
W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
W = calculated pillar widths (ft)

S = 900
L = 600
H = 3
C = 7
Z = 100,200..2000
W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{ W + C \} \right\}, W \right]$$

Depth	Width
100	1.688
200	3.095
300	4.387
400	5.615
500	6.803
600	7.963
700	9.103
800	10.228
900	11.341
1000	12.445
1100	13.542
1200	14.633
1300	15.719
1400	16.801
1500	17.881
1600	18.958
1700	20.033
1800	21.106
1900	22.178
2000	23.248

A.4b - RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
 Rib Pillar Design
 Mark-Bieniawski Pillar Strength Formula
 Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
 L = length of cut (ft)
 H = coal seam thickness (ft)
 C = cut width of SAM (ft)
 Z = range of depths (ft) [minimum, increment..maximum]
 W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
 W = calculated pillar widths (ft)

S = 900
 L = 600
 H = 3
 C = 10
 Z = 100,200..2000
 W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{ W + C \} \right\}, W \right]$$

Depth	Width
100	2.167
200	3.821
300	5.291
400	6.660
500	7.966
600	9.226
700	10.454
800	11.658
900	12.841
1000	14.009
1100	15.164
1200	16.309
1300	17.444
1400	18.573
1500	19.695
1600	20.812
1700	21.924
1800	23.032
1900	24.137
2000	25.240

A.4c - RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
 Rib Pillar Design
 Mark-Bieniawski Pillar Strength Formula
 Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
 L = length of cut (ft)
 H = coal seam thickness (ft)
 C = cut width of SAM (ft)
 Z = range of depths (ft) [minimum, increment..maximum]
 W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
 W = calculated pillar widths (ft)

S = 900
 L = 600
 H = 3
 C = 12
 Z = 100,200..2000
 W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{ W + C \} \right\}, W \right]$$

Depth	Width
100	2.452
200	4.247
300	5.819
400	7.271
500	8.646
600	9.968
700	11.250
800	12.501
900	13.728
1000	14.936
1100	16.128
1200	17.307
1300	18.475
1400	19.633
1500	20.783
1600	21.927
1700	23.064
1800	24.196
1900	25.323
2000	26.446

A.5 - TYPE 3 RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
 Rib Pillar Design
 Mark-Bieniawski Pillar Strength Formula
 Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
 L = length of cut (ft)
 H = coal seam thickness (ft)
 C = cut width of SAM (ft)
 Z = range of depths (ft) [minimum, increment..maximum]
 W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
 W = calculated pillar widths (ft)

S = 900
 L = 600
 H = 2
 C = 50
 Z = 100,200..2000
 W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{L \times W\} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{W + C\} \right\}, W \right]$$

Depth	Width
100	5.043
200	7.752
300	9.931
400	11.833
500	13.559
600	15.160
700	16.666
800	18.097
900	19.468
1000	20.778
1100	22.065
1200	23.305
1300	24.512
1400	25.691
1500	26.845
1600	27.976
1700	29.086
1800	30.178
1900	31.253
2000	32.311

A.6 - TYPE 3 RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
 Rib Pillar Design
 Mark-Bieniawski Pillar Strength Formula
 Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
 L = length of cut (ft)
 H = coal seam thickness (ft)
 C = cut width of SAM (ft)
 Z = range of depths (ft) [minimum, increment..maximum]
 W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
 W = calculated pillar widths (ft)

S = 900
 L = 600
 H = 2.5
 C = 50
 Z = 100,200..2000
 W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{ L \times W \} - 1.65 \times Z \times \left\{ L + \frac{E}{\cos(B \times P / 180)} \right\} \times \{ W + C \} \right], W$$

Depth	Width
100	5.544
200	8.605
300	11.082
400	13.254
500	15.231
600	17.070
700	18.805
800	20.458
900	22.043
1000	23.574
1100	25.057
1200	26.499
1300	27.906
1400	29.282
1500	30.630
1600	31.953
1700	33.254
1800	34.534
1900	35.796
2000	37.041

A.7 - TYPE 3 RIB PILLAR DESIGN PROGRAM

ASSUMPTIONS:

Stability Factor = 1.5
 Rib Pillar Design
 Mark-Bieniawski Pillar Strength Formula
 Overburden pressure increases 1.1 psi per 100 feet

USER INPUTS:

S = coal strength (psi)
 L = length of cut (ft)
 H = coal seam thickness (ft)
 C = cut width of SAM (ft)
 Z = range of depths (ft) [minimum, increment..maximum]
 W = initial guess for pillar width (ft) [if negative number is returned increase guess]

OUTPUT:

Z = input range of depths (ft)
 W = calculated pillar widths (ft)

S = 900
 L = 600
 H = 3
 C = 50
 Z = 100,200..2000
 W = 60

$$width = root \left[S \times \left\{ 0.64 + 0.54 \times \left(\frac{W}{H} \right) - 0.18 \times \left(\frac{W^2}{L \times H} \right) \right\} \times \{L \times W\} - 1.65 \times Z \times \left\{ \left(L + \frac{E}{\cos(B \times P / 180)} \right) \times \{W + C\} \right\}, W \right]$$

Depth	Width
100	5.981
200	9.365
300	12.119
400	14.534
500	16.756
600	18.820
700	20.772
800	22.635
900	24.425
1000	26.156
1100	27.837
1200	29.473
1300	31.072
1400	32.637
1500	34.173
1600	35.682
1700	37.167
1800	38.631
1900	40.075
2000	41.051

APPENDIX B:

**SPREADSHEETS FOR PANEL RECOVERY RATES
WITHOUT BACKFILL**

TABLE B.1A - Panel Extraction and Recovery

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	Type 1 Width (ft)	Type 2 Width (ft)	Type 3 Width (ft)	Coal Extracted (cu. ft)	Recovery (%)
24	100	4000	600	7	45	1.5	55	5.0	5345952	78.28803
	200					2.6	65	7.8	4684941	68.39089
	300					3.6	75	9.9	4213534	61.35315
	400					4.5	85	11.8	3846578	55.88610
	500					5.4	95	13.6	3547212	51.43348
	600					6.3	105	15.2	3295130	47.68982
	700					7.1	115	16.7	3078375	44.47521
	800					7.9	125	18.1	2889098	41.67167
	900					8.7	135	19.5	2721710	39.19531
	1000					9.5	145	20.8	2572310	36.98797
	1100					10.3	155	22.1	2437715	35.00078
	1200					11.0	165	23.3	2315561	33.19961
	1300					11.8	175	24.5	2204307	31.56080
	1400					12.5	185	25.7	2102117	30.05707
	1500					13.3	195	26.8	2008116	28.67515
	1600					14.0	205	28.0	1921169	27.39817
	1700					14.8	215	29.1	1840418	26.21337
	1800					15.5	225	30.2	1765219	25.11106
	1900					16.2	235	31.3	1695015	24.08290
	2000					17.0	245	32.3	1629319	23.12169

TABLE B.1B - Panel Extraction and Recovery

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	Type 1 Width (ft)	Type 2 Width (ft)	Type 3 Width (ft)	Coal Extracted (cu. ft)	Recovery (%)
24	100	4000	600	10	45	1.9	55	5.0	5439140	79.65270
	200					3.2	65	7.8	4859456	70.93846
	300					4.4	75	9.9	4440722	64.66123
	400					5.4	85	11.8	4109546	59.70670
	500					6.4	95	13.6	3835017	55.60657
	600					7.3	105	15.2	3600564	52.11031
	700					8.2	115	16.7	3396077	49.06525
	800					9.1	125	18.1	3215266	46.37625
	900					9.9	135	19.5	3053383	43.97173
	1000					10.8	145	20.8	2907062	41.80147
	1100					11.6	155	22.1	2773906	39.82782
	1200					12.4	165	23.3	2651856	38.02127
	1300					13.2	175	24.5	2539569	36.36100
	1400					14.0	185	25.7	2435505	34.82400
	1500					14.8	195	26.8	2338783	33.39695
	1600					15.6	205	28.0	2248821	32.07088
	1700					16.4	215	29.1	2164514	30.82952
	1800					17.1	225	30.2	2085417	29.66602
	1900					17.9	235	31.3	2010983	28.57221
	2000					18.7	245	32.3	1940874	27.54297

TABLE B.1C - Panel Extraction and Recovery

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	Type 1 Width (ft)	Type 2 Width (ft)	Type 3 Width (ft)	Coal Extracted (cu. ft)	Recovery (%)
24	100	4000	600	12	45	2.1	55	5.0	5484559	80.31784
	200					3.6	65	7.8	4942985	72.15782
	300					4.8	75	9.9	4549465	66.24463
	400					5.9	85	11.8	4235893	61.54237
	500					6.9	95	13.6	3973948	57.62103
	600					7.9	105	15.2	3748762	54.25514
	700					8.9	115	16.7	3551258	51.30725
	800					9.8	125	18.1	3375396	48.68592
	900					10.7	135	19.5	3217159	46.33027
	1000					11.5	145	20.8	3073406	44.19338
	1100					12.4	155	22.1	2941879	42.23959
	1200					13.3	165	23.3	2820859	40.44437
	1300					14.1	175	24.5	2708817	38.78426
	1400					14.9	185	25.7	2604782	37.24440
	1500					15.7	195	26.8	2507644	35.80822
	1600					16.5	205	28.0	2416732	34.46549
	1700					17.3	215	29.1	2331377	33.20618
	1800					18.1	225	30.2	2251003	32.02155
	1900					18.9	235	31.3	2175108	30.90411
	2000					19.7	245	32.3	2103389	29.84923

TABLE B.2A - Panel Extraction and Recovery

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	Type 1 Width (ft)	Type 2 Width (ft)	Type 3 Width (ft)	Coal Extracted (cu. ft)	Recovery (%)
30	100	4000	600	7	45	1.6	55	5.5	6599298	77.26862
	200					2.9	65	8.6	5705531	66.56528
	300					4.0	75	11.1	5072926	59.01426
	400					5.1	85	13.3	4585748	53.21239
	500					6.1	95	15.2	4192067	48.53283
	600					7.1	105	17.1	3864429	44.64461
	700					8.1	115	18.8	3585503	41.33940
	800					9.1	125	20.5	3343963	38.48120
	900					10.1	135	22.0	3132393	35.98089
	1000					11.0	145	23.6	2945041	33.76945
	1100					11.9	155	25.1	2777374	31.79282
	1200					12.9	165	26.5	2626563	30.01693
	1300					13.8	175	27.9	2489936	28.40985
	1400					14.7	185	29.3	2365354	26.94608
	1500					15.6	195	30.6	2251383	25.60839
	1600					16.5	205	32.0	2146531	24.37907
	1700					17.4	215	33.3	2049659	23.24450
	1800					18.3	225	34.5	1959961	22.19505
	1900					19.2	235	35.8	1876665	21.22146
	2000					20.1	245	37.0	1798972	20.31431

TABLE B.2B - Panel Extraction and Recovery

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	Type 1 Width (ft)	Type 2 Width (ft)	Type 3 Width (ft)	Coal Extracted (cu. ft)	Recovery (%)
30	100	4000	600	10	45	2.0	55	5.5	6715371	78.62767
	200					3.5	65	8.6	5929404	69.17717
	300					4.8	75	11.1	5366595	62.43057
	400					6.1	85	13.3	4925466	57.15442
	500					7.2	95	15.2	4563028	52.82755
	600					8.3	105	17.1	4256267	49.17140
	700					9.4	115	18.8	3991079	46.01552
	800					10.4	125	20.5	3758364	43.24999
	900					11.4	135	22.0	3551547	40.79560
	1000					12.4	145	23.6	3365939	38.59570
	1100					13.4	155	25.1	3198156	36.60955
	1200					14.4	165	26.5	3045364	34.80308
	1300					15.4	175	27.9	2905399	33.15022
	1400					16.3	185	29.3	2776697	31.63209
	1500					17.3	195	30.6	2657650	30.22948
	1600					18.2	205	32.0	2547290	28.93066
	1700					19.2	215	33.3	2444440	27.72159
	1800					20.1	225	34.5	2348430	26.59417
	1900					21.1	235	35.8	2258521	25.53951
	2000					22.0	245	37.0	2174142	24.55079

TABLE B.2C - Panel Extraction and Recovery

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	Type 1 Width (ft)	Type 2 Width (ft)	Type 3 Width (ft)	Coal Extracted (cu. ft)	Recovery (%)
30	100	4000	600	12	45	2.3	55	5.5	6772315	79.29442
	200					3.9	65	8.6	6037917	70.44316
	300					5.3	75	11.1	5508000	64.07556
	400					6.6	85	13.3	5089785	59.06115
	500					7.8	95	15.2	4743419	54.91599
	600					9.0	105	17.1	4448081	51.38737
	700					10.1	115	18.8	4190974	48.32023
	800					11.2	125	20.5	3963902	45.61525
	900					12.2	135	22.0	3760872	43.20005
	1000					13.3	145	23.6	3577568	41.02235
	1100					14.3	155	25.1	3410988	39.04586
	1200					15.3	165	26.5	3258460	37.23839
	1300					16.3	175	27.9	3118157	35.57776
	1400					17.3	185	29.3	2988449	34.04437
	1500					18.3	195	30.6	2867980	32.62189
	1600					19.3	205	32.0	2755787	31.29865
	1700					20.2	215	33.3	2650870	30.06264
	1800					21.2	225	34.5	2552535	28.90550
	1900					22.2	235	35.8	2460103	27.81902
	2000					23.1	245	37.0	2373051	26.79690

TABLE B.3A - Panel Extraction and Recovery

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	Type 1 Width (ft)	Type 2 Width (ft)	Type 3 Width (ft)	Coal Extracted (cu. ft)	Recovery (%)
36	100	4000	600	7	45	1.7	55	6.0	7838945	76.44686
	200					3.1	65	9.4	6696750	65.05035
	300					4.4	75	12.1	5892916	57.05896
	400					5.6	85	14.5	5279554	50.97696
	500					6.8	95	16.8	4788851	46.12022
	600					8.0	105	18.8	4384111	42.12179
	700					9.1	115	20.8	4042625	38.75364
	800					10.2	125	22.6	3749555	35.86733
	900					11.3	135	24.4	3494699	33.36089
	1000					12.4	145	26.2	3270518	31.15903
	1100					13.5	155	27.8	3071471	29.20654
	1200					14.6	165	29.5	2893408	27.46208
	1300					15.7	175	31.1	2733044	25.89287
	1400					16.8	185	32.6	2587747	24.47278
	1500					17.9	195	34.2	2455284	23.17964
	1600					19.0	205	35.7	2334111	21.99808
	1700					20.0	215	37.2	2222757	20.91349
	1800					21.1	225	38.6	2120072	19.91442
	1900					22.2	235	40.1	2025008	18.99055
	2000					23.2	245	41.5	1936810	18.13433

TABLE B.3B - Panel Extraction and Recovery

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	Type 1 Width (ft)	Type 2 Width (ft)	Type 3 Width (ft)	Coal Extracted (cu. ft)	Recovery (%)
36	100	4000	600	10	45	2.2	55	6.0	7976317	77.78654
	200					3.8	65	9.4	6970107	67.70566
	300					5.3	75	12.1	6253342	60.54883
	400					6.7	85	14.5	5696630	55.00405
	500					8.0	95	16.8	5242783	50.49193
	600					9.2	105	18.8	4862054	46.71379
	700					10.5	115	20.8	4535242	43.47599
	800					11.7	125	22.6	4250155	40.65594
	900					12.8	135	24.4	3998767	38.17280
	1000					14.0	145	26.2	3774495	35.96055
	1100					15.2	155	27.8	3572876	33.97438
	1200					16.3	165	29.5	3390241	32.17764
	1300					17.4	175	31.1	3224014	30.54432
	1400					18.6	185	32.6	3071636	29.04900
	1500					19.7	195	34.2	2931532	27.67577
	1600					20.8	205	35.7	2802085	26.40855
	1700					21.9	215	37.2	2682115	25.23550
	1800					23.0	225	38.6	2570533	24.14573
	1900					24.1	235	40.1	2466410	23.13003
	2000					25.2	245	41.1	2368951	22.19169

TABLE B.3C - Panel Extraction and Recovery

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	Type 1 Width (ft)	Type 2 Width (ft)	Type 3 Width (ft)	Coal Extracted (cu. ft)	Recovery (%)
36	100	4000	600	12	45	2.5	55	6.0	8044678	78.45321
	200					4.2	65	9.4	7103239	68.99888
	300					5.8	75	12.1	6428586	62.24566
	400					7.3	85	14.5	5899829	56.96605
	500					8.6	95	16.8	5465478	52.63664
	600					10.0	105	18.8	5097637	48.97723
	700					11.3	115	20.8	4779765	45.82005
	800					12.5	125	22.6	4500803	43.05358
	900					13.7	135	24.4	4252908	40.59886
	1000					14.9	145	26.2	4030422	38.39883
	1100					16.1	155	27.8	3829213	36.41188
	1200					17.3	165	29.5	3645980	34.60493
	1300					18.5	175	31.1	3478175	32.95223
	1400					19.6	185	32.6	3323808	31.43383
	1500					20.8	195	34.2	3181120	30.03206
	1600					21.9	205	35.7	3048642	28.73226
	1700					23.1	215	37.2	2925392	27.52444
	1800					24.2	225	38.6	2810271	26.39766
	1900					25.3	235	40.1	2702495	25.34404
	2000					26.4	245	41.1	2601307	24.36833

TABLE B.4A - % Coal Left in Pillars

Seam Height	(in)	24																			
Cross-Cut Height	(ft)	8																			
Cut Width	(ft)	7																			
Cut Length	(ft)	600																			
Cut Angle	(deg)	45																			
Panel Length	(ft)	4000																			
Depth	(ft)	100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
Type 1 Width	(ft)	1.493	2.62	3.617	4.543	5.423	6.272	7.098	7.906	8.7	9.482	10.255	11.021	11.779	12.533	13.281	14.025	14.766	15.504	16.239	16.971
Type 2 Width	(ft)	55	65	75	85	95	105	115	125	135	145	155	165	175	185	195	205	215	225	235	245
Type 3 Width	(ft)	5.043	7.752	9.931	11.833	13.559	15.16	16.666	18.097	19.468	20.778	22.065	23.305	24.512	25.691	26.845	27.976	29.086	30.178	31.253	32.311
#Type 1 Pillars		640	561	505	461	425	395	369	346	326	308	292	277	264	252	240	230	220	211	203	195
#Type 2 Pillars		4																			
#Type 3 Pillars		1																			
Vol. Type 1	(ft3)	1792	3144	4340	5452	6508	7526	8518	9487	10440	11378	12306	13225	14135	15040	15937	16830	17719	18605	19487	20365
Vol. Type 2	(ft3)	66000	78000	90000	102000	114000	126000	138000	150000	162000	174000	186000	198000	210000	222000	234000	246000	258000	270000	282000	294000
Vol. Type 3	(ft3)	40344	62016	79448	94664	108472	121280	133328	144776	155744	166224	176520	186440	196096	205528	214760	223808	232688	241424	250024	258488
Total Vol. In Pillars	(ft3)	1482617	2165300	2654139	3036311	3349485	3614375	3843178	4043903	4222259	4382139	4527030	4659104	4780014	4891636	4994869	5090864	5180495	5264430	5343234	5417394
Vol. of Panel	(ft3)	6828569	6850241	6867673	6882889	6896697	6909505	6921553	6933001	6943969	6954449	6964745	6974665	6984321	6993753	7002985	7012033	7020913	7029649	7038249	7046713
% Type 1		79.4725	82.7268	83.4429	83.4449	83.1475	82.7002	82.1677	81.5827	80.9641	80.3241	79.6661	78.9994	78.3244	77.6449	76.9612	76.275	75.5875	74.899	74.2099	73.5207
% Type 2		17.8063	14.4091	13.5637	13.4374	13.614	13.9443	14.3631	14.8371	15.3472	15.8827	16.4346	16.999	17.5732	18.1534	18.7392	19.3287	19.9209	20.515	21.1108	21.7079
% Type 3		2.72113	2.86408	2.99336	3.11773	3.23847	3.35549	3.46921	3.58011	3.68864	3.79322	3.89924	4.00163	4.10242	4.20162	4.29961	4.39627	4.49162	4.58595	4.67926	4.77145

TABLE B.4B - % Coal Left in Pillars

Seam Height	(in)	24																			
Cross-Cut Height	(ft)	8																			
Cut Width	(ft)	10																			
Cut Length	(ft)	600																			
Cut Angle	(deg)	45																			
Panel Length	(ft)	4000																			
Depth	(ft)	100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
Type 1 Width	(ft)	1.895	3.216	4.355	5.396	6.374	7.308	8.21	9.086	9.942	10.782	11.608	12.423	13.227	14.024	14.814	15.595	16.372	17.144	17.912	18.675
Type 2 Width	(ft)	55	65	75	85	95	105	115	125	135	145	155	165	175	185	195	205	215	225	235	245
Type 3 Width	(ft)	5.043	7.752	9.931	11.833	13.559	15.16	16.666	18.097	19.468	20.778	22.065	23.305	24.512	25.691	26.845	27.976	29.086	30.178	31.253	32.311
#Type 1 Pillars		457	408	373	345	322	303	285	270	257	244	233	223	213	205	197	189	182	175	169	163
#Type 2 Pillars		4																			
#Type 3 Pillars		1																			
Vol. Type 1	(ft3)	2274	3859	5226	6475	7649	8770	9852	10903	11930	12938	13930	14908	15872	16829	17777	18714	19646	20573	21494	22410
Vol. Type 2	(ft3)	66000	78000	90000	102000	114000	126000	138000	150000	162000	174000	186000	198000	210000	222000	234000	246000	258000	270000	282000	294000
Vol. Type 3	(ft3)	40344	62016	79448	94664	108472	121280	133328	144776	155744	166224	176520	186440	196096	205528	214760	223808	232688	241424	250024	258488
Total Vol. In Pillars	(ft3)	1389429	1990785	2426951	2773343	3061680	3308941	3525476	3717735	3890586	4047387	4190839	4322809	4444752	4558248	4664202	4763212	4856399	4944232	5027266	5105839
Vol. of Panel	(ft3)	6828569	6850241	6867673	6882889	6896697	6909505	6921553	6933001	6943969	6954449	6964745	6974665	6984321	6993753	7002985	7012033	7020913	7029649	7038249	7046713
% Type 1		78.0958	81.2126	81.893	81.8752	81.5633	81.1033	80.5607	79.9669	79.3413	78.6968	78.0349	77.3656	76.6895	76.0099	75.3278	74.643	73.9583	73.2734	72.589	71.905
% Type 2		19.0006	15.6722	14.8334	14.7115	14.8938	15.2315	15.6575	16.1389	16.6556	17.1963	17.753	18.3214	18.8987	19.4812	20.0677	20.6583	21.2503	21.8436	22.4376	23.0325
% Type 3		2.90364	3.11515	3.27357	3.41335	3.54289	3.66522	3.78184	3.8942	4.0031	4.10695	4.21204	4.31294	4.41185	4.50892	4.60443	4.69868	4.79137	4.88294	4.97336	5.0626

TABLE B.4C - % Coal Left in Pillars

Seam Height	(in)	24																			
Cross-Cut Height	(ft)	8																			
Cut Width	(ft)	12																			
Cut Length	(ft)	600																			
Cut Angle	(deg)	45																			
Panel Length	(ft)	4000																			
Depth	(ft)	100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
Type 1 Width	(ft)	2.132	3.565	4.786	5.894	6.93	7.915	8.862	9.78	10.674	11.549	12.408	13.253	14.087	14.91	15.725	16.532	17.332	18.126	18.915	19.698
Type 2 Width	(ft)	55	65	75	85	95	105	115	125	135	145	155	165	175	185	195	205	215	225	235	245
Type 3 Width	(ft)	5.043	7.752	9.931	11.833	13.559	15.16	16.666	18.097	19.468	20.778	22.065	23.305	24.512	25.691	26.845	27.976	29.086	30.178	31.253	32.311
#Type 1 Pillars		385	347	319	297	279	263	249	237	226	216	206	198	190	183	176	170	164	158	153	148
#Type 2 Pillars		4																			
#Type 3 Pillars		1																			
Vol. Type 1	(ft3)	2558	4278	5743	7073	8316	9498	10634	11736	12809	13859	14890	15904	16904	17892	18870	19838	20798	21751	22698	23638
Vol. Type 2	(ft3)	66000	78000	90000	102000	114000	126000	138000	150000	162000	174000	186000	198000	210000	222000	234000	246000	258000	270000	282000	294000
Vol. Type 3	(ft3)	40344	62016	79448	94664	108472	121280	133328	144776	155744	166224	176520	186440	196096	205528	214760	223808	232688	241424	250024	258488
Total Vol. In Pillars	(ft3)	1344010	1907257	2318208	2646996	2922749	3160743	3370295	3557606	3726810	3881043	4022866	4153806	4275504	4388971	4495341	4595301	4689536	4778646	4863141	4943324
Vol. of Panel	(ft3)	6828569	6850241	6867673	6882889	6896697	6909505	6921553	6933001	6943969	6954449	6964745	6974665	6984321	6993753	7002985	7012033	7020913	7029649	7038249	7046713
% Type 1		77.3555	80.3898	81.0436	81.01	80.687	80.2173	79.6656	79.0652	78.4335	77.7837	77.1178	76.4447	75.7667	75.0846	74.4011	73.7165	73.0317	72.3473	71.6639	70.9813
% Type 2		19.6427	16.3586	15.5292	15.4137	15.6017	15.9456	16.3784	16.8653	17.3875	17.9333	18.4943	19.0669	19.6468	20.2325	20.8216	21.4132	22.0064	22.6005	23.1949	23.7897
% Type 3		3.00176	3.25158	3.42713	3.57628	3.7113	3.83707	3.95597	4.06948	4.17902	4.28297	4.38792	4.48841	4.5865	4.68283	4.77739	4.87037	4.96186	5.05214	5.1412	5.22903

TABLE B.5A - % Coal Left in Pillars

Seam Height	(in)	30																			
Cross-Cut Height	(ft)	8																			
Cut Width	(ft)	7																			
Cut Length	(ft)	600																			
Cut Angle	(deg)	45																			
Panel Length	(ft)	4000																			
Depth	(ft)	100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
Type 1 Width	(ft)	1.6	2.874	4.023	5.103	6.14	7.146	8.13	9.098	10.052	10.995	11.931	12.859	13.781	14.699	15.612	16.522	17.43	18.335	19.237	20.138
Type 2 Width	(ft)	55	65	75	85	95	105	115	125	135	145	155	165	175	185	195	205	215	225	235	245
Type 3 Width	(ft)	5.544	8.605	11.082	13.254	15.231	17.07	18.805	20.458	22.043	23.574	25.057	26.499	27.906	29.282	30.63	31.953	33.254	34.534	35.796	37.041
#Type 1 Pillars		632	547	486	439	402	370	343	320	300	282	266	252	239	227	216	206	196	188	180	172
#Type 2 Pillars		4																			
#Type 3 Pillars		1																			
Vol. Type 1	(ft3)	2400	4311	6035	7655	9210	10719	12195	13647	15078	16493	17897	19289	20672	22049	23418	24783	26145	27503	28856	30207
Vol. Type 2	(ft3)	82500	97500	112500	127500	142500	157500	172500	187500	202500	217500	232500	247500	262500	277500	292500	307500	322500	337500	352500	367500
Vol. Type 3	(ft3)	55440	86050	110820	132540	152310	170700	188050	204580	220430	235740	250570	264990	279060	292820	306300	319530	332540	345340	357960	370410
Total Vol. In Pillars	(ft3)	1941423	2865800	3523175	4032073	4445524	4791552	5087828	5345899	5573319	5775980	5958478	6123708	6274405	6412748	6540199	6658280	6768163	6870660	6966576	7056719
Vol. of Panel	(ft3)	8540721	8571331	8596101	8617821	8637591	8655981	8673331	8689861	8705711	8721021	8735851	8750271	8764341	8778101	8791581	8804811	8817821	8830621	8843241	8855691
% Type 1		80.1465	83.3886	84.082	84.0643	83.752	83.2893	82.7421	82.1437	81.5114	80.8562	80.1867	79.506	78.8178	78.1245	77.4273	76.7278	76.0269	75.3249	74.6223	73.9197
% Type 2		16.9978	13.6088	12.7726	12.6486	12.8219	13.1481	13.5618	14.0294	14.5335	15.0624	15.608	16.1667	16.7347	17.3093	17.8894	18.4732	19.0598	19.6488	20.2395	20.8312
% Type 3		2.85564	3.00265	3.14546	3.28714	3.42614	3.56252	3.69608	3.82686	3.95509	4.08139	4.20527	4.32728	4.44759	4.56622	4.68334	4.79899	4.9133	5.0263	5.13825	5.24904

TABLE B.5B - % Coal Left in Pillars

Seam Height	(in)	30																			
Cross-Cut Height	(ft)	8																			
Cut Width	(ft)	10																			
Cut Length	(ft)	600																			
Cut Angle	(deg)	45																			
Panel Length	(ft)	4000																			
Depth	(ft)	100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
Type 1 Width	(ft)	2.043	3.539	4.848	6.057	7.202	8.302	9.369	10.41	11.431	12.436	13.427	14.407	15.378	16.34	17.296	18.245	19.19	20.13	21.066	21.998
Type 2 Width	(ft)	55	65	75	85	95	105	115	125	135	145	155	165	175	185	195	205	215	225	235	245
Type 3 Width	(ft)	5.544	8.605	11.082	13.254	15.231	17.07	18.805	20.458	22.043	23.574	25.057	26.499	27.906	29.282	30.63	31.953	33.254	34.534	35.796	37.041
#Type 1 Pillars		451	399	361	331	307	286	268	253	239	226	215	205	195	187	179	171	164	158	152	146
#Type 2 Pillars		4																			
#Type 3 Pillars		1																			
Vol. Type 1	(ft3)	3065	5309	7272	9086	10803	12453	14054	15615	17147	18654	20141	21611	23067	24510	25944	27368	28785	30195	31599	32997
Vol. Type 2	(ft3)	82500	97500	112500	127500	142500	157500	172500	187500	202500	217500	232500	247500	262500	277500	292500	307500	322500	337500	352500	367500
Vol. Type 3	(ft3)	55440	86050	110820	132540	152310	170700	188050	204580	220430	235740	250570	264990	279060	292820	306300	319530	332540	345340	357960	370410
Total Vol. In Pillars	(ft3)	1825351	2641927	3229506	3692355	4074563	4399714	4682252	4931497	5154164	5355082	5537695	5704907	5858943	6001404	6133932	6257521	6373381	6482191	6584721	6681549
Vol. of Panel	(ft3)	8540721	8571331	8596101	8617821	8637591	8655981	8673331	8689861	8705711	8721021	8735851	8750271	8764341	8778101	8791581	8804811	8817821	8830621	8843241	8855691
% Type 1		78.8841	81.981	82.6345	82.5981	82.2727	81.8011	81.2473	80.6432	80.0078	79.3516	78.6812	78.0016	77.3157	76.6251	75.9322	75.2373	74.5419	73.8462	73.1506	72.4553
% Type 2		18.0787	14.762	13.934	13.8123	13.9892	14.3191	14.7365	15.2084	15.7154	16.2462	16.794	17.3535	17.9213	18.4957	19.0742	19.6563	20.2404	20.8263	21.4132	22.0009
% Type 3		3.03722	3.25709	3.43148	3.58958	3.73807	3.8798	4.01623	4.14844	4.27674	4.40217	4.52481	4.64495	4.76298	4.87919	4.99353	5.10633	5.21764	5.32752	5.43622	5.54377

TABLE B.5C - % Coal Left in Pillars

Seam Height	(in)	30																			
Cross-Cut Height	(ft)	8																			
Cut Width	(ft)	12																			
Cut Length	(ft)	600																			
Cut Angle	(deg)	45																			
Panel Length	(ft)	4000																			
Depth	(ft)	100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
Type 1 Width	(ft)	2.306	3.928	5.331	6.615	7.824	8.98	10.097	11.183	12.245	13.288	14.314	15.327	16.328	17.319	18.302	19.277	20.246	21.209	22.167	23.12
Type 2 Width	(ft)	55	65	75	85	95	105	115	125	135	145	155	165	175	185	195	205	215	225	235	245
Type 3 Width	(ft)	5.544	8.605	11.082	13.254	15.231	17.07	18.805	20.458	22.043	23.574	25.057	26.499	27.906	29.282	30.63	31.953	33.254	34.534	35.796	37.041
#Type 1 Pillars		380	339	309	286	266	250	235	222	211	201	191	183	175	168	161	155	149	143	138	133
#Type 2 Pillars		4																			
#Type 3 Pillars		1																			
Vol. Type 1	(ft3)	3459	5892	7997	9923	11736	13470	15146	16775	18368	19932	21471	22991	24492	25979	27453	28916	30369	31814	33251	34680
Vol. Type 2	(ft3)	82500	97500	112500	127500	142500	157500	172500	187500	202500	217500	232500	247500	262500	277500	292500	307500	322500	337500	352500	367500
Vol. Type 3	(ft3)	55440	86050	110820	132540	152310	170700	188050	204580	220430	235740	250570	264990	279060	292820	306300	319530	332540	345340	357960	370410
Total Vol. In Pillars	(ft3)	1768406	2533414	3088102	3528037	3894172	4207900	4482358	4725959	4944839	5143453	5324863	5491811	5646185	5789652	5923601	6049025	6166952	6278086	6383138	6482641
Vol. of Panel	(ft3)	8540721	8571331	8596101	8617821	8637591	8655981	8673331	8689861	8705711	8721021	8735851	8750271	8764341	8778101	8791581	8804811	8817821	8830621	8843241	8855691
% Type 1		78.2041	81.2092	81.8393	81.7876	81.4515	80.9715	80.411	79.8013	79.1615	78.502	77.8291	77.148	76.4609	75.7702	75.0777	74.3838	73.6898	72.9959	72.3027	71.6102
% Type 2		18.6609	15.3942	14.5721	14.4556	14.6373	14.9718	15.3937	15.8698	16.3807	16.9147	17.4652	18.0268	18.5966	19.1721	19.7515	20.3339	20.918	21.5034	22.0894	22.6759
% Type 3		3.13503	3.3966	3.58861	3.75676	3.91123	4.05666	4.19534	4.32886	4.45778	4.5833	4.70566	4.82518	4.94245	5.05764	5.17084	5.28234	5.39229	5.50072	5.6079	5.71388

TABLE B.6A - % Coal Left in Pillars

Seam Height	(in)	36																			
Cross-Cut Height	(ft)	8																			
Cut Width	(ft)	7																			
Cut Length	(ft)	600																			
Cut Angle	(deg)	45																			
Panel Length	(ft)	4000																			
Depth	(ft)	100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
Type 1 Width	(ft)	1.688	3.095	4.387	5.615	6.803	7.963	9.103	10.228	11.341	12.445	13.542	14.633	15.719	16.801	17.881	18.958	20.033	21.106	22.178	23.248
Type 2 Width	(ft)	55	65	75	85	95	105	115	125	135	145	155	165	175	185	195	205	215	225	235	245
Type 3 Width	(ft)	5.981	9.365	12.119	14.534	16.756	18.82	20.772	22.635	24.425	26.156	27.837	29.473	31.072	32.637	34.173	35.682	37.167	38.631	40.075	41.501
#Type 1 Pillars		626	535	470	421	382	350	323	299	279	261	245	231	218	207	196	186	177	169	162	155
#Type 2 Pillars	4																				
#Type 3 Pillars	1																				
Vol. Type 1	(ft3)	3038	5571	7897	10107	12245	14333	16385	18410	20414	22401	24376	26339	28294	30242	32186	34124	36059	37991	39920	41846
Vol. Type 2	(ft3)	99000	117000	135000	153000	171000	189000	207000	225000	243000	261000	279000	297000	315000	333000	351000	369000	387000	405000	423000	441000
Vol. Type 3	(ft3)	71772	112380	145428	174408	201072	225840	249264	271620	293100	313872	334044	353676	372864	391644	410076	428184	446004	463572	480900	498012
Total Vol. In Pillars	(ft3)	2415165	3597967	4434850	5077192	5594558	6024067	6388976	6704403	6980739	7225692	7444911	7642606	7822157	7986234	8137130	8276411	8405584	8525838	8638230	8743540
Vol. of Panel	(ft3)	10254110	10294718	10327766	10356746	10383410	10408178	10431602	10453958	10475438	10496210	10516382	10536014	10555202	10573982	10592414	10610522	10628342	10645910	10663238	10680350
% Type 1		80.63188	83.86923	84.5445	84.51096	84.17977	83.70138	83.13871	82.52462	81.87728	81.20772	80.52302	79.82787	79.12514	78.41731	77.70619	76.99263	76.27763	75.56167	74.84554	74.12933
% Type 2		16.3964	13.00734	12.17629	12.05391	12.22617	12.54966	12.95982	13.42401	13.92403	14.44844	14.99011	15.54444	16.10809	16.6787	17.25424	17.83382	18.41633	19.00106	19.58735	20.17489
% Type 3		2.971723	3.12343	3.279209	3.435127	3.594064	3.748963	3.90147	4.051368	4.198696	4.343833	4.486877	4.627689	4.766767	4.903988	5.039566	5.173547	5.306044	5.43726	5.567113	5.695771

TABLE B.6B - % Coal Left in Pillars

Seam Height	(in)	36																			
Cross-Cut Height	(ft)	8																			
Cut Width	(ft)	10																			
Cut Length	(ft)	600																			
Cut Angle	(deg)	45																			
Panel Length	(ft)	4000																			
Depth	(ft)	100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
Type 1 Width	(ft)	2.167	3.821	5.291	6.66	7.966	9.226	10.454	11.658	12.841	14.009	15.164	16.309	17.444	18.573	19.695	20.812	21.924	23.032	24.137	25.24
Type 2 Width	(ft)	55	65	75	85	95	105	115	125	135	145	155	165	175	185	195	205	215	225	235	245
Type 3 Width	(ft)	5.981	9.365	12.119	14.534	16.756	18.82	20.772	22.635	24.425	26.156	27.837	29.473	31.072	32.637	34.173	35.682	37.167	38.631	40.075	41.051
#Type 1 Pillars		447	390	350	319	294	272	254	238	224	211	200	190	181	172	164	157	150	144	138	133
#Type 2 Pillars		4																			
#Type 3 Pillars		1																			
Vol. Type 1	(ft3)	3901	6878	9524	11988	14339	16607	18817	20984	23114	25216	27295	29356	31399	33431	35451	37462	39463	41458	43447	45432
Vol. Type 2	(ft3)	99000	117000	135000	153000	171000	189000	207000	225000	243000	261000	279000	297000	315000	333000	351000	369000	387000	405000	423000	441000
Vol. Type 3	(ft3)	71772	112380	145428	174408	201072	225840	249264	271620	293100	313872	334044	353676	372864	391644	410076	428184	446004	463572	480900	492612
Total Vol. In Pillars	(ft3)	2277793	3324611	4074424	4660116	5140626	5546124	5896359	6203803	6476670	6721715	6943506	7145773	7331187	7502346	7660881	7808436	7946226	8075377	8196827	8305998
Vol. of Panel	(ft3)	10254110	10294718	10327766	10356746	10383410	10408178	10431602	10453958	10475438	10496210	10516382	10536014	10555202	10573982	10592414	10610522	10628342	10645910	10663238	10674950
% Type 1		79.4638	82.54292	83.1773	83.12471	82.7828	82.29682	81.73001	81.11449	80.46681	79.79873	79.11654	78.42534	77.72715	77.02527	76.32027	75.61376	74.90628	74.19845	73.49096	72.83154
% Type 2		17.38525	14.07684	13.25341	13.13272	13.30577	13.63114	14.04256	14.50723	15.00771	15.53175	16.07257	16.62521	17.18685	17.75445	18.32687	18.90263	19.48095	20.06098	20.64213	21.23766
% Type 3		3.150945	3.380245	3.56929	3.742568	3.91143	4.072033	4.227422	4.378282	4.525474	4.669523	4.810884	4.949444	5.085997	5.220287	5.352857	5.483607	5.612777	5.740562	5.866904	5.930798

TABLE B.6C - % Coal Left in Pillars

Seam Height	(in)	36																			
Cross-Cut Height	(ft)	8																			
Cut Width	(ft)	12																			
Cut Length	(ft)	600																			
Cut Angle	(deg)	45																			
Panel Length	(ft)	4000																			
Depth	(ft)	100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
Type 1 Width	(ft)	2.452	4.247	5.819	7.271	8.646	9.968	11.25	12.501	13.728	14.936	16.128	17.307	18.475	19.633	20.783	21.927	23.064	24.196	25.323	26.446
Type 2 Width	(ft)	55	65	75	85	95	105	115	125	135	145	155	165	175	185	195	205	215	225	235	245
Type 3 Width	(ft)	5.981	9.365	12.119	14.534	16.756	18.82	20.772	22.635	24.425	26.156	27.837	29.473	31.072	32.637	34.173	35.682	37.167	38.631	40.075	41.051
#Type 1 Pillars		376	332	301	276	256	238	224	210	199	188	179	171	163	155	149	143	137	131	126	122
#Type 2 Pillars		4																			
#Type 3 Pillars		1																			
Vol. Type 1	(ft3)	4414	7645	10474	13088	15563	17942	20250	22502	24710	26885	29030	31153	33255	35339	37409	39469	41515	43553	45581	47603
Vol. Type 2	(ft3)	99000	117000	135000	153000	171000	189000	207000	225000	243000	261000	279000	297000	315000	333000	351000	369000	387000	405000	423000	441000
Vol. Type 3	(ft3)	71772	112380	145428	174408	201072	225840	249264	271620	293100	313872	334044	353676	372864	391644	410076	428184	446004	463572	480900	492612
Total Vol. In Pillars	(ft3)	2209431	3191478	3899180	4456916	4917932	5310541	5651837	5953155	6222530	6465787	6687169	6890034	7077027	7250174	7411294	7561879	7702950	7835639	7960743	8073643
Vol. of Panel	(ft3)	10254110	10294718	10327766	10356746	10383410	10408178	10431602	10453958	10475438	10496210	10516382	10536014	10555202	10573982	10592414	10610522	10628342	10645910	10663238	10674950
% Type 1		78.8284	81.8147	82.42123	82.35533	82.00317	81.51149	80.93958	80.31934	79.66904	78.99912	78.31603	77.62455	76.92726	76.22617	75.52282	74.81864	74.11376	73.40903	72.70481	72.04964
% Type 2		17.92316	14.66405	13.84907	13.73147	13.90828	14.23584	14.6501	15.11803	15.62066	16.14653	16.68868	17.2423	17.80409	18.37197	18.94406	19.51896	20.0962	20.67477	21.2543	21.84887
% Type 3		3.248438	3.521252	3.729707	3.913199	4.088548	4.252674	4.410318	4.562623	4.710303	4.854351	4.995298	5.133153	5.268653	5.401857	5.533123	5.662402	5.790041	5.916199	6.040893	6.101484

APPENDIX C:

**SPREADSHEETS FOR PILLAR WIDTHS AND RECOVERY RATES
WITH BACKFILL**

TABLE C.1 - Pillar Widths with Backfill

Seam Height (ft)	s_1' (psi)	Depth of Overburden (ft)																			
		100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
		pillar widths (ft)																			
2	900	1.49	2.62	3.62	4.54	5.42	6.27	7.10	7.91	8.70	9.48	10.26	11.02	11.78	12.53	13.28	14.03	14.77	15.50	16.24	16.97
	1200	1.18	2.08	2.88	3.62	4.32	4.99	5.64	6.27	6.89	7.50	8.11	8.70	9.29	9.87	10.45	11.02	11.59	12.16	12.72	13.28
	1500	0.97	1.74	2.41	3.03	3.62	4.18	4.72	5.25	5.77	6.27	6.77	7.26	7.75	8.23	8.70	9.17	9.64	10.10	10.53	11.02
	2000	0.76	1.37	1.91	2.41	2.88	3.33	3.76	4.18	4.59	4.99	5.38	5.77	6.15	6.52	6.89	7.26	7.63	7.99	8.34	8.70
	2500	0.62	1.14	1.59	2.01	2.41	2.79	3.15	3.50	3.84	4.18	4.51	4.83	5.15	5.46	5.77	6.07	6.37	6.67	6.97	7.26
	3000	0.53	0.97	1.37	1.74	2.08	2.41	2.72	3.03	3.33	3.62	3.90	4.18	4.45	4.72	4.99	5.25	5.51	5.77	6.02	6.27
	4000	0.41	0.76	1.07	1.37	1.65	1.91	2.16	2.41	2.65	2.88	3.11	3.33	3.55	3.76	3.97	4.18	4.38	4.59	4.79	4.99
	5500	0.30	0.57	0.82	1.05	1.26	1.47	1.67	1.86	2.05	2.23	2.41	2.58	2.75	2.92	3.08	3.25	3.41	3.56	3.72	3.88
	7000	0.24	0.46	0.66	0.85	1.03	1.20	1.37	1.53	1.68	1.84	1.98	2.13	2.27	2.41	2.55	2.68	2.81	2.94	3.07	3.20
	8500	0.20	0.39	0.56	0.72	0.87	1.02	1.16	1.30	1.44	1.57	1.69	1.82	1.94	2.06	2.18	2.29	2.41	2.52	2.63	2.74
	10000	0.17	0.33	0.48	0.62	0.76	0.89	1.01	1.14	1.25	1.37	1.48	1.59	1.70	1.81	1.91	2.01	2.11	2.21	2.31	2.41

TABLE C.2 - Pillar Widths with Backfill

Seam Height (ft)	s_1' (psi)	Depth of Overburden (ft)																			
		100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
		pillar widths (ft)																			
2.5	900	1.60	2.87	4.02	5.10	6.14	7.15	8.13	9.10	10.05	11.00	11.93	12.86	13.78	14.70	15.61	16.52	17.43	18.34	19.24	20.14
	1200	1.25	2.26	3.17	4.02	4.84	5.63	6.39	7.15	7.89	8.62	9.34	10.05	10.76	11.46	12.16	12.86	13.55	14.24	14.93	15.61
	1500	1.03	1.87	2.63	3.34	4.02	4.68	5.31	5.94	6.55	7.15	7.74	8.33	8.91	9.48	10.05	10.62	11.18	11.74	12.30	12.86
	2000	0.79	1.46	2.07	2.63	3.17	3.69	4.19	4.68	5.16	5.63	6.09	6.55	7.00	7.44	7.89	8.33	8.76	9.19	9.62	10.05
	2500	0.65	1.20	1.71	2.18	2.63	3.06	3.48	3.89	4.29	4.68	5.06	5.44	5.81	6.18	6.55	6.91	7.27	7.62	7.97	8.33
	3000	0.55	1.03	1.46	1.87	2.26	2.63	2.99	3.34	3.69	4.02	4.35	4.68	5.00	5.31	5.63	5.94	6.24	6.55	6.85	7.15
	4000	0.42	0.79	1.14	1.46	1.77	2.07	2.35	2.63	2.90	3.17	3.43	3.69	3.94	4.19	4.43	4.68	4.92	5.16	5.39	5.63
	5500	0.31	0.59	0.86	1.11	1.35	1.58	1.80	2.01	2.22	2.43	2.63	2.83	3.03	3.22	3.41	3.59	3.78	3.96	4.14	4.32
	7000	0.25	0.48	0.69	0.89	1.09	1.28	1.46	1.64	1.81	1.98	2.15	2.31	2.47	2.63	2.79	2.94	3.09	3.25	3.39	3.54

TABLE C.3 - Pillar Widths with Backfill

Seam Height (ft)	s_1' (psi)	Depth of Overburden (ft)																			
		100	200	300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000
		pillar widths (ft)																			
3	900	1.69	3.10	4.39	5.62	6.80	7.96	9.10	10.23	11.34	12.45	13.54	14.63	15.72	16.80	17.88	18.96	20.03	21.11	22.18	23.25
	1200	1.31	2.41	3.43	4.39	5.31	6.21	7.10	7.96	8.82	9.67	10.51	11.34	12.17	12.99	13.82	14.63	15.45	16.26	17.07	17.88
	1500	1.07	1.98	2.83	3.62	4.39	5.13	5.86	6.57	7.27	7.96	8.65	9.33	10.00	10.67	11.34	12.00	12.67	13.32	13.98	14.63
	2000	0.82	1.54	2.20	2.83	3.43	4.01	4.57	5.13	5.68	6.21	6.74	7.27	7.79	8.31	8.82	9.33	9.84	10.34	10.84	11.34
	2500	0.67	1.26	1.81	2.33	2.83	3.31	3.78	4.24	4.69	5.13	5.57	6.00	6.43	6.85	7.27	7.69	8.10	8.51	8.92	9.33
	3000	0.56	1.07	1.54	1.98	2.41	2.83	3.23	3.62	4.01	4.39	4.76	5.13	5.49	5.86	6.21	6.57	6.92	7.27	7.62	7.96
	4000	0.43	0.82	1.19	1.54	1.87	2.20	2.52	2.83	3.13	3.43	3.72	4.01	4.29	4.57	4.85	5.13	5.40	5.68	5.95	6.21
	5500	0.32	0.61	0.89	1.15	1.41	1.66	1.90	2.14	2.37	2.60	2.83	3.05	3.26	3.48	3.69	3.90	4.11	4.32	4.52	4.73
	7000	0.25	0.49	0.71	0.93	1.14	1.34	1.54	1.73	1.92	2.11	2.29	2.47	2.65	2.83	3.00	3.17	3.34	3.51	3.68	3.84

TABLE C.4 - Panel Extraction and Recovery with Fill

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	% Recovery for each σ_1'								
						900	1200	1500	2000	2500	3000	4000	5500	7000
24	100	4000	600	7	45	78.29	81.33	83.40	85.71	87.22	88.31	89.77	91.04	91.81
	200					68.39	72.47	75.32	78.62	80.88	82.53	84.81	86.89	88.19
	300					61.35	65.94	69.24	73.12	75.82	77.84	80.68	83.32	85.02
	400					55.89	60.76	64.32	68.57	71.58	73.85	77.09	80.17	82.17
	500					51.43	56.47	60.18	64.68	67.92	70.38	73.91	77.32	79.56
	600					47.69	52.80	56.62	61.29	64.68	67.28	71.04	74.72	77.16
	700					44.48	49.61	53.49	58.28	61.78	64.48	68.43	72.32	74.93
	800					41.67	46.80	50.71	55.56	59.15	61.93	66.02	70.09	72.84
	900					39.20	44.29	48.20	53.10	56.75	59.59	63.79	68.00	70.86
	1000					36.99	42.03	45.93	50.85	54.53	57.42	61.72	66.04	69.00
	1100					35.00	39.98	43.86	48.78	52.49	55.41	59.77	64.19	67.23
	1200					33.20	38.11	41.95	46.87	50.58	53.52	57.93	62.44	65.55
	1300					31.56	36.39	40.19	45.08	48.80	51.75	56.20	60.78	63.94
	1400					30.06	34.80	38.56	43.42	47.13	50.09	54.57	59.18	62.40
	1500					28.68	33.33	37.04	41.86	45.56	48.51	53.01	57.67	60.93
	1600					27.40	31.97	35.62	40.39	44.07	47.02	51.53	56.22	59.51
	1700					26.21	30.69	34.29	39.01	42.67	45.61	50.12	54.83	58.15
	1800					25.11	29.50	33.04	37.71	41.34	44.27	48.77	53.49	56.83
	1900					24.10	28.38	31.93	36.47	40.07	42.98	47.47	52.21	55.56
	2000					23.12	27.33	30.76	35.30	38.86	41.76	46.23	50.97	54.34

TABLE C.5 - Panel Extraction and Recovery with Fill

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	% Recovery for each σ_1'								
						900	1200	1500	2000	2500	3000	4000	5500	7000
30	100	4000	600	7	45	77.27	80.58	82.81	85.27	86.90	88.05	89.57	90.90	91.71
	200					66.57	70.99	74.10	77.68	80.13	81.90	84.34	86.56	87.93
	300					59.01	63.96	67.54	71.75	74.69	76.88	79.95	82.79	84.60
	400					53.21	58.43	62.26	66.86	70.14	72.61	76.12	79.44	81.58
	500					48.53	53.87	57.85	62.71	66.21	68.88	72.72	76.42	78.84
	600					44.64	50.02	54.08	59.09	62.75	65.57	69.65	73.65	76.29
	700					41.34	46.70	50.79	55.90	59.67	62.59	66.87	71.10	73.92
	800					38.48	43.79	47.89	53.05	56.89	59.89	64.31	68.73	71.71
	900					35.98	41.22	45.29	50.47	54.36	57.41	61.95	66.52	69.62
	1000					33.77	38.91	42.96	48.13	52.04	55.13	59.75	64.44	67.65
	1100					31.79	36.84	40.84	45.98	49.90	53.01	57.70	62.49	65.79
	1200					30.02	34.96	38.90	44.01	47.92	51.05	55.78	60.64	64.01
	1300					28.41	33.24	37.12	42.18	46.08	49.21	53.97	58.89	62.32
	1400					26.95	31.67	35.48	40.48	44.36	47.48	52.26	57.22	60.70
	1500					25.61	30.22	33.96	38.90	42.75	45.86	50.64	55.64	59.16
	1600					24.38	28.88	32.55	37.42	41.23	44.33	49.11	54.13	57.68
	1700					23.24	27.63	31.23	36.03	39.81	42.88	47.65	52.68	56.26
	1800					22.20	26.47	30.00	34.72	38.46	41.51	46.26	51.30	54.89
	1900					21.22	25.39	28.84	33.49	37.18	40.21	44.93	49.96	53.57
	2000					20.31	24.38	27.76	32.33	35.97	38.97	43.66	48.69	52.30

TABLE C.6 - Panel Extraction and Recovery with Fill

Seam Height (in)	Overburden (ft)	Panel Length (ft)	Cut Length (ft)	Cut Width (ft)	Cut Angle (deg)	% Recovery for each σ_1'								
						900	1200	1500	2000	2500	3000	4000	5500	7000
30	100	4000	600	7	45	76.45	79.97	82.33	84.94	86.64	87.84	89.43	90.80	91.62
	200					65.05	69.78	73.10	76.92	79.52	81.40	83.99	86.30	87.73
	300					57.06	62.32	66.13	70.63	73.77	76.11	79.36	82.37	84.27
	400					50.98	56.47	60.55	65.45	68.95	71.59	75.33	78.87	81.13
	500					46.12	51.71	55.91	61.06	64.79	67.64	71.75	75.68	78.24
	600					42.12	47.70	51.96	57.26	61.15	64.15	68.51	72.77	75.58
	700					38.75	44.27	48.55	53.92	57.91	61.01	65.58	70.09	73.10
	800					35.87	41.30	45.54	50.95	54.99	58.18	62.89	67.60	70.77
	900					33.36	38.68	42.88	48.27	52.36	55.59	60.41	65.28	68.59
	1000					31.16	36.35	40.49	45.86	49.95	53.21	58.09	63.11	66.53
	1100					29.21	34.27	38.34	43.65	47.74	51.02	55.98	61.06	64.58
	1200					27.46	32.39	36.38	41.63	45.70	48.98	53.97	59.14	62.73
	1300					25.89	30.69	34.60	39.77	43.81	47.08	52.10	57.31	60.97
	1400					24.47	29.13	32.96	38.05	42.06	45.31	50.33	55.59	59.29
	1500					23.18	27.71	31.45	36.46	40.42	43.65	48.66	53.94	57.68
	1600					22.00	26.40	30.05	34.97	38.88	42.09	47.08	52.37	56.14
	1700					20.91	25.19	28.75	33.58	37.44	40.61	45.58	50.88	54.67
	1800					19.91	24.06	27.54	32.28	36.08	39.22	44.16	49.45	53.26
	1900					18.99	23.02	26.41	31.06	34.80	37.91	42.80	48.09	51.90
	2000					18.13	22.05	25.36	29.91	33.59	36.66	41.51	46.77	50.59

APPENDIX D:

SPREADSHEETS FOR BACKFILLING COSTS

TABLE D.1 - Cost of Fill for Every Extra Ton of Coal Recovered

		900 psi		1200 psi		1500 psi		2000 psi		2500 psi		3000 psi		4000 psi		5500 psi		7000 psi	
Seam	Depth	Recov.	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	
(ft)	(ft)	(%)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	
2	100	78.29	81.33	52.35	83.40	31.95	85.71	22.65	87.22	19.13	88.31	17.27	89.77	15.33	91.04	13.99	91.81	13.31	
	200	68.39	72.47	34.85	75.32	21.31	78.62	15.06	80.88	12.70	82.53	11.44	84.81	10.13	86.89	9.21	88.19	8.73	
	300	61.35	65.94	28.16	69.24	17.21	73.12	12.18	75.82	10.27	77.84	9.25	80.68	8.18	83.32	7.43	85.02	7.04	
	400	55.89	60.76	24.43	64.32	14.95	68.57	10.60	71.58	8.94	73.85	8.06	77.09	7.13	80.17	6.47	82.17	6.13	
	500	51.43	56.47	22.00	60.18	13.48	64.68	9.57	67.92	8.08	70.38	7.28	73.91	6.44	77.32	5.85	79.56	5.54	
	600	47.69	52.80	20.26	56.62	12.43	61.29	8.83	64.68	7.46	67.28	6.73	71.04	5.96	74.72	5.42	77.16	5.13	
	700	44.48	49.61	18.93	53.49	11.63	58.28	8.28	61.78	7.00	64.48	6.32	68.43	5.60	72.32	5.09	74.93	4.82	
	800	41.67	46.80	17.88	50.71	11.00	55.56	7.84	59.15	6.63	61.93	5.99	66.02	5.31	70.09	4.83	72.84	4.58	
	900	39.20	44.29	17.04	48.20	10.49	53.10	7.48	56.75	6.34	59.59	5.73	63.79	5.08	68.00	4.63	70.86	4.39	
	1000	36.99	42.03	16.33	45.93	10.06	50.85	7.19	54.53	6.09	57.42	5.51	61.72	4.89	66.04	4.46	69.00	4.22	
	1100	35.00	39.98	15.74	43.86	9.70	48.78	6.94	52.49	5.88	55.41	5.32	59.77	4.73	64.19	4.31	67.23	4.09	
	1200	33.20	38.11	15.22	41.95	9.39	46.87	6.72	50.58	5.70	53.52	5.16	57.93	4.59	62.44	4.19	65.55	3.97	
	1300	31.56	36.39	14.78	40.19	9.13	45.08	6.54	48.80	5.55	51.75	5.02	56.20	4.47	60.78	4.08	63.94	3.87	
	1400	30.06	34.80	14.38	38.56	8.89	43.42	6.37	47.13	5.41	50.09	4.90	54.57	4.36	59.18	3.98	62.40	3.78	
	1500	28.68	33.33	14.03	37.04	8.68	41.86	6.22	45.56	5.29	48.51	4.79	53.01	4.27	57.67	3.90	60.93	3.70	
	1600	27.40	31.97	13.72	35.62	8.49	40.39	6.09	44.07	5.18	47.02	4.70	51.53	4.19	56.22	3.82	59.51	3.63	
	1700	26.21	30.69	13.43	34.29	8.32	39.01	5.97	42.67	5.08	45.61	4.61	50.12	4.11	54.83	3.76	58.15	3.57	
	1800	25.11	29.50	13.18	33.04	8.16	37.71	5.87	41.34	4.99	44.27	4.53	48.77	4.04	53.49	3.69	56.83	3.51	
	1900	24.10	28.38	12.99	31.93	7.99	36.47	5.78	40.07	4.92	42.98	4.46	47.47	3.98	52.21	3.64	55.56	3.46	
	2000	23.12	27.33	12.73	30.76	7.90	35.30	5.68	38.86	4.84	41.76	4.39	46.23	3.92	50.97	3.59	54.34	3.41	

TABLE D.2 - Cost of Fill for Every Extra Ton of Coal Recovered

		900 psi	1200 psi		1500 psi		2000 psi		2500 psi		3000 psi		4000 psi		5500 psi		7000 psi	
Seam	Depth	Recov.	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost
(ft)	(ft)	(%)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)
2.5	100	77.27	80.58	47.75	82.81	29.31	85.27	20.89	86.90	17.69	88.05	16.01	89.57	14.27	90.90	13.07	91.71	12.45
	200	66.57	70.99	31.47	74.10	19.28	77.68	13.70	80.13	11.58	81.90	10.47	84.34	9.30	86.56	8.48	87.93	8.07
	300	59.01	63.96	25.33	67.54	15.53	71.75	11.05	74.69	9.34	76.88	8.43	79.95	7.49	82.79	6.82	84.60	6.48
	400	53.21	58.43	21.96	62.26	13.49	66.86	9.60	70.14	8.12	72.61	7.34	76.12	6.51	79.44	5.94	81.58	5.64
	500	48.53	53.87	19.78	57.85	12.17	62.71	8.67	66.21	7.34	68.88	6.64	72.72	5.89	76.42	5.37	78.84	5.10
	600	44.64	50.02	18.24	54.08	11.23	59.09	8.02	62.75	6.79	65.57	6.14	69.65	5.46	73.65	4.98	76.29	4.72
	700	41.34	46.70	17.08	50.79	10.53	55.90	7.52	59.67	6.38	62.59	5.77	66.87	5.13	71.10	4.68	73.92	4.45
	800	38.48	43.79	16.16	47.89	9.98	53.05	7.14	56.89	6.06	59.89	5.48	64.31	4.88	68.73	4.45	71.71	4.23
	900	35.98	41.22	15.43	45.29	9.53	50.47	6.83	54.36	5.80	57.41	5.25	61.95	4.68	66.52	4.27	69.62	4.06
	1000	33.77	38.91	14.83	42.96	9.16	48.13	6.57	52.04	5.58	55.13	5.06	59.75	4.51	64.44	4.12	67.65	3.91
	1100	31.79	36.84	14.30	40.84	8.85	45.98	6.35	49.90	5.40	53.01	4.90	57.70	4.37	62.49	3.99	65.79	3.79
	1200	30.02	34.96	13.87	38.90	8.58	44.01	6.17	47.92	5.25	51.05	4.76	55.78	4.24	60.64	3.88	64.01	3.69
	1300	28.41	33.24	13.48	37.12	8.35	42.18	6.00	46.08	5.11	49.21	4.64	53.97	4.14	58.89	3.79	62.32	3.60
	1400	26.95	31.67	13.15	35.48	8.15	40.48	5.86	44.36	4.99	47.48	4.53	52.26	4.05	57.22	3.70	60.70	3.52
	1500	25.61	30.22	12.85	33.96	7.97	38.90	5.74	42.75	4.89	45.86	4.44	50.64	3.96	55.64	3.63	59.16	3.46
	1600	24.38	28.88	12.59	32.55	7.81	37.42	5.62	41.23	4.79	44.33	4.35	49.11	3.89	54.13	3.57	57.68	3.39
	1700	23.24	27.63	12.34	31.23	7.66	36.03	5.52	39.81	4.71	42.88	4.28	47.65	3.83	52.68	3.51	56.26	3.34
	1800	22.20	26.47	12.13	30.00	7.53	34.72	5.43	38.46	4.63	41.51	4.21	46.26	3.77	51.30	3.45	54.89	3.29
	1900	21.22	25.39	11.93	28.84	7.42	33.49	5.35	37.18	4.57	40.21	4.15	44.93	3.71	49.96	3.41	53.57	3.25
	2000	20.31	24.38	11.75	27.76	7.31	32.33	5.27	35.97	4.50	38.97	4.09	43.66	3.67	48.69	3.36	52.30	3.20

TABLE D.3 - Cost of Fill for Every Extra Ton of Coal Recovered

		900 psi		1200 psi		1500 psi		2000 psi		2500 psi		3000 psi		4000 psi		5500 psi		7000 psi	
Seam	Depth	Recov.	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	Recov.	Cost	
(ft)	(ft)	(%)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	(%)	(\$/ton coal)	
3	100	76.45	79.97	44.46	82.33	27.42	84.94	19.60	86.64	16.66	87.84	15.11	89.43	13.50	90.80	12.40	91.62	11.83	
	200	65.05	69.78	28.93	73.10	17.79	76.92	12.70	79.52	10.77	81.40	9.76	83.99	8.69	86.30	7.96	87.73	7.58	
	300	57.06	62.32	23.22	66.13	14.29	70.63	10.20	73.77	8.65	76.11	7.83	79.36	6.97	82.37	6.38	84.27	6.07	
	400	50.98	56.47	20.13	60.55	12.40	65.45	8.86	68.95	7.52	71.59	6.81	75.33	6.06	78.87	5.54	81.13	5.27	
	500	46.12	51.71	18.14	55.91	11.20	61.06	8.01	64.79	6.80	67.64	6.16	71.75	5.49	75.68	5.02	78.24	4.77	
	600	42.12	47.70	16.76	51.96	10.35	57.26	7.41	61.15	6.30	64.15	5.71	68.51	5.09	72.77	4.65	75.58	4.43	
	700	38.75	44.27	15.72	48.55	9.72	53.92	6.97	57.91	5.93	61.01	5.37	65.58	4.79	70.09	4.38	73.10	4.17	
	800	35.87	41.30	14.91	45.54	9.23	50.95	6.62	54.99	5.64	58.18	5.11	62.89	4.56	67.60	4.18	70.77	3.97	
	900	33.36	38.68	14.25	42.88	8.83	48.27	6.34	52.36	5.40	55.59	4.90	60.41	4.38	65.28	4.01	68.59	3.82	
	1000	31.16	36.35	13.72	40.49	8.50	45.86	6.12	49.95	5.21	53.21	4.73	58.09	4.23	63.11	3.87	66.53	3.69	
	1100	29.21	34.27	13.27	38.34	8.23	43.65	5.92	47.74	5.05	51.02	4.58	55.98	4.10	61.06	3.76	64.58	3.58	
	1200	27.46	32.39	12.88	36.38	7.99	41.63	5.76	45.70	4.91	48.98	4.46	53.97	3.99	59.14	3.66	62.73	3.49	
	1300	25.89	30.69	12.54	34.60	7.79	39.77	5.62	43.81	4.79	47.08	4.35	52.10	3.90	57.31	3.58	60.97	3.41	
	1400	24.47	29.13	12.25	32.96	7.61	38.05	5.49	42.06	4.69	45.31	4.26	50.33	3.82	55.59	3.50	59.29	3.34	
	1500	23.18	27.71	11.99	31.45	7.46	36.46	5.38	40.42	4.60	43.65	4.18	48.66	3.74	53.94	3.44	57.68	3.28	
	1600	22.00	26.40	11.76	30.05	7.32	34.97	5.28	38.88	4.51	42.09	4.11	47.08	3.68	52.37	3.38	56.14	3.22	
	1700	20.91	25.19	11.56	28.75	7.19	33.58	5.20	37.44	4.44	40.61	4.04	45.58	3.62	50.88	3.33	54.67	3.17	
	1800	19.91	24.06	11.37	27.54	7.08	32.28	5.12	36.08	4.37	39.22	3.98	44.16	3.57	49.45	3.28	53.26	3.13	
	1900	18.99	23.02	11.20	26.41	6.98	31.06	5.04	34.80	4.31	37.91	3.93	42.80	3.52	48.09	3.24	51.90	3.09	
	2000	18.13	22.05	11.05	25.36	6.88	29.91	4.98	33.59	4.26	36.66	3.88	41.51	3.48	46.77	3.20	50.59	3.06	

TABLE D.4 - Sensitivity of Cost to Fill Density

Seam	Fill Cost	Depth	Recov.	Recov.	γ_{fill} (lb/ft ³)						
					50	60	70	80	90	100	110
			900 psi	2500 psi	Cost						
(ft)	(\$/yd ³)	(ft)	(%)	(%)	(\$/ton coal)	(\$/ton coal)	(\$/ton coal)	(\$/ton coal)	(\$/ton coal)	(\$/ton coal)	(\$/ton coal)
2.5	2.30	100	77.27	86.90	30.75	25.62	21.96	19.22	17.08	15.37	13.98
		200	66.57	80.13	20.13	16.78	14.38	12.58	11.19	10.07	9.15
		300	59.01	74.69	16.23	13.53	11.59	10.14	9.02	8.12	7.38
		400	53.21	70.14	14.12	11.77	10.08	8.82	7.84	7.06	6.42
		500	48.53	66.21	12.76	10.64	9.12	7.98	7.09	6.38	5.80
		600	44.64	62.75	11.81	9.84	8.43	7.38	6.56	5.90	5.37
		700	41.34	59.67	11.09	9.24	7.92	6.93	6.16	5.55	5.04
		800	38.48	56.89	10.53	8.78	7.52	6.58	5.85	5.27	4.79
		900	35.98	54.36	10.08	8.40	7.20	6.30	5.60	5.04	4.58
		1000	33.77	52.04	9.71	8.09	6.93	6.07	5.39	4.85	4.41
		1100	31.79	49.90	9.39	7.82	6.71	5.87	5.22	4.69	4.27
		1200	30.02	47.92	9.12	7.60	6.51	5.70	5.07	4.56	4.15
		1300	28.41	46.08	8.89	7.40	6.35	5.55	4.94	4.44	4.04
		1400	26.95	44.36	8.68	7.23	6.20	5.43	4.82	4.34	3.95
		1500	25.61	42.75	8.50	7.08	6.07	5.31	4.72	4.25	3.86
		1600	24.38	41.23	8.34	6.95	5.95	5.21	4.63	4.17	3.79
		1700	23.24	39.81	8.19	6.82	5.85	5.12	4.55	4.09	3.72
		1800	22.20	38.46	8.06	6.71	5.76	5.04	4.48	4.03	3.66
		1900	21.22	37.18	7.94	6.61	5.67	4.96	4.41	3.97	3.61
		2000	20.31	35.97	7.83	6.52	5.59	4.89	4.35	3.91	3.56