AN ECONOMIC EVALUATION OF BACKFILLING IN VIRGINIA'S COAL MINES

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(ABSTRACT)

Backfilling is a technique used to place material in the voids created by underground mining. This technique has been used extensively abroad to reduce surface subsidence, fire hazards, and temperatures in underground workings. It can also help to eliminate environmental problems caused by coal waste dumps, improve coal recovery, and improve mine ventilation. However, backfilling is an additional operation in the coal mining cycle that can reduce productivity and increase production costs.

This thesis identifies the backfilling methods applicable for coal mines in southwest Virginia. Following identification of variables that can affect the selection of a backfilling method, a multi-attribute decision analysis

technique is developed. This technique is applied to a mine model which is representative for the mining conditions in southwest Virginia. An evaluation of the two most suitable backfilling methods using coal refuse as fill material is then made and compared to the surface disposal of coal refuse.

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I - INTRODUCTION

Backfilling is the filling of voids created by mining and is used extensively in metallic underground mining. It has been used in alleviating technical problems in coal mines in Europe. These technical problems include surface subsidence, that may cause serious damage to surface structures and flooding of mines. In multiple-seam mining, backfilling of voids reduces ground disturbance in the vicinity of mining operations and facilitates extraction of multiple seams.

It is generally recognized that ground support is the most important function of backfilling. Support of underground openings for mining operations is accomplished by room-and-pillar partial extraction methods. This mining method is predominantly used in southwest Virginia's underground coal mines. However, as mining depth increases, the percentage of coal that must remain unmined to achieve the needed support becomes prohibitively large and the method uneconomical. In Europe this limit was reached around the 1930's, and the longwall mining system became popular (Fairhurst, 1974). This mining system is designed to extract virtually all of the coal in the seam allowing the overlying rock to collapse into the gob. Ground disturbances are consequently more severe, and inevitably involve surface subsidence as well as damage to the intervening strata. The amount of damage and subsidence is directly related to the volume of the mined out area into which the roof collapse takes place. Backfilling reduces this volume and consequently reduces subsidence and its effects.

Increased resource recovery, enhanced ventilation control and minimization of underground coal mine fires are other potential advantages of backfilling. The coal recovery rate can be increased by using backfilling because fewer coal pillars will then be required to support the overburden, because the fill will also provide support. This may increase mineable coal reserves substantially. Coal mine ventilation can be enhanced as a result of improved ground support. Substantial air leakage in underground coal mines usually occurs, especially when stoppings, overcasts, and other ventilation control devices experience damage from ground settlements. Backfilling will reduce ground movement and provide an efficient seal, which will in turn decrease the need for large quantities of air in the mine. This sealing function will also reduce spontaneous combustion of coal and resulting coal mine fires.

Another potential benefit of backfilling is the disposal of wastes. The disposal of waste material inevitably requires planning and control in order to minimize the environmental impact of mining. Presently, this waste is disposed of on the surface. This disposal practice results in non-productive use of the area, potential air and water pollution, possible failure of waste embankments and loss of aesthetic value of the area. By disposing of coal mine refuse underground, these problems can be eliminated.

There are several backfilling techniques each with its associated advantages and disadvantages. The backfilling system for any operation is site-specific in its use of available technology, its flexibility for change, the cost of its component system, the local community pressures and the institutional and legal framework in which backfilling is carried out.

It is therefore important to identify the different backfilling methods, their advantages and disadvantages, and the site conditions that prevail, in order to justify a backfilling system that is technically and economically feasible for southwest Virginia.

1.1 - Statement and purpose of the research

This research attempts to identify the most suitable backfilling techniques for the coal mines of southwest Virginia. In order to achieve this objective, this research was conducted as outlined below:

1. Literature review of backfilling techniques used in the coal mining industry.

2. Determination of variables effecting the selection of a backfilling method.

3. Review of underground coal mining operations in southwest Virginia.

4. A mine model that assumes average mining conditions for southwest Virginia was used to determine the most suitable backfilling methods.

5. Evaluation of the costs associated with those backfilling methods.

1.2 - Organization of the thesis

This thesis is divided into seven chapters. Chapter 2 reviews the different backfilling techniques used in underground coal mines. In addition, it contains an evaluation of different fill materials. Chapter 3 reviews surface disposal practices and their associated costs. Chapter 4 develops a framework for selecting a backfilling method, this framework is then applied to a hypothetical mine model, for southwest Virginia, in order to determine the most suitable backfilling methods. A cost estimate for the best and second best backfilling method is given in Chapter 5 and Chapter 6 respectively, while Chapter 7 summarizes the results of this thesis.

II - BACKFILLING IN COAL MINING

2.1 - Backfilling techniques

Backfilling has been utilized extensively in Europe and India, particularly for thick, deep, steep or multiple seams and for workings under urbanized areas (Cooley, 1978). Backfilling has a number of advantages; however, some disadvantages have restricted its use in the United States.

The advantages of backfilling can be listed as follows:

- Elimination of environmental problems caused by coal waste dumps.
- Reduction of surface subsidence.
- Potential to improve recovery.
- Reduction of temperatures in underground workings.
- Reduction of fire hazards.
- Replacement of pillars of uncertain strength with material of known strength.
- Improvement of ventilation.

The most important disadvantages of backfilling are as follows:

- Addition of a new operation to the mining cycle.

However, this may be alleviated if the fill can be utilized to improve recovery, reduce surface damage and reduce problematic environmental effects.

- Possibility of reduced production rate caused by problems associated with backfilling.

Backfilling systems are characterized and identified by the mechanisms used for transport and emplacement of the fill material. The overall characteristics, adaptabilities and limitations of each system depend on the individual transporting and stowing elements of the system. Backfilling systems can be classified as gravity stowing, hand stowing, mechanical stowing, pneumatic stowing and hydraulic stowing.

2.1.1 - Gravity stowing

Gravity stowing is used in European coal mines and is restricted to seams dipping at gradients of at least 45-50 degrees, where fill material can be fed down into the gob by gravitational force. However, when gravity stowing is performed through special pipes, the gradient can be as little as 30 degrees (Smoldyrev, 1978).
The system requires that the lowering of fill is accompanied by continuous unloading at the bottom via some discharge-feeder mechanism. This will avoid clogging of pipes or chutes. It is also common to let some material

accumulate at the end of the pipe to form a cushion. Two variations of gravitational backfilling exist, one using mechanical surface transport and the other using mechanical in-mine transport.

In systems using mechanical surface transport, the fill material is transported on the surface to boreholes or to small-diameter shafts and fed directly into the gob by gravity. This method is used in steeply dipping, thick coal seams under shallow cover.

In systems using mechanical in-mine transport, the fill material is transported to the underground working panel by any combination of mechanical transport devices and then fed into the gob by a belt. The method is commonly used in steeply dipping coalbeds. In this system an independent haulage system for the fill material is used. A typical gravity-stowing technique is shown in Figure 1.

Production rates of 150 cubic yards per hour in inclined coalbeds and 260 cubic yards per hour in steeply dipping coalbeds have been achieved (Smoldyrev, 1978). The productivity of each worker servicing the stowing system is usually 20-30 cubic yards per man shift and the labor requirement is 25-30 man shifts per 1,000 tons of material emplaced (Bucek et al., 1979).

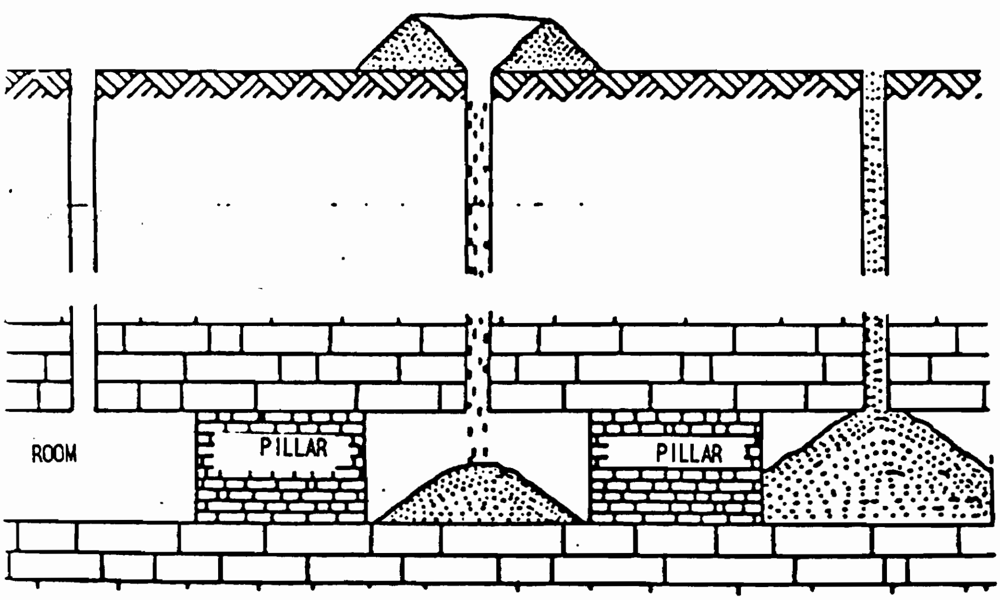

Figure 1: A typical gravity-stowing technique
(Source: Huck et al., 1982)

Gravity stowing is the least expensive backfilling method and can achieve high filling rates, depending upon the method of transportation used within the mine. The major disadvantage of this stowing method is the low density of the stowed mass, which makes spontaneous combustion possible if the fill material is not inert. Another problem with this stowing method is that suitable surface access must be provided when vertical boreholes are employed. This method can only be used if the gradient is away from the face.

For gravity stowing, the moisture content of backfill material should be such that it is not sticky, and the diameter of its individual particles should not exceed 10 to 12 inches in order to avoid problems of impact against support structures. The predominant fractions of the stowing material should consist of lump rock from 2.5 to 6 inches in diameter (Smoldyrev, 1978).

2.1.2 - Hand stowing

Hand stowing was originally used in coal mines. It involved the construction of box-like structures underground into which rocks and debris from roof falls were placed. These structures supported the roof over caved areas and provided a means of disposing of refuse in the mine. Hand stowing is extremely labor-intensive and therefore it is unsuited for most current mining practices. Because of today's highly mechanized mining techniques and the current cost of labor, it has become a method of the past.

2.1.3 - Mechanical stowing

Mechanical stowing techniques were developed with the introduction of conveyors into mining operations. Now seldom used in coal mines, mechanical stowing methods are occasionally still used for constructing packwalls in advancing longwall faces. Scrapers, and slinger-type machines were at one time used for mechanical stowing operations. Scrapers were used in mines working somewhat thin, gently dipping seams. During placement of fill material into the void, initial compaction was obtained by ramming the material with the scraper bucket. In the case of slinger-type machines, fill material was thrown into the gob and the initial compaction of the fill resulted from the high-velocity impact of the thrown material. A maximum capacity of 50 to 60 tons per hour could be reached with a belt thrower (Bucek et al., 1979), although this was a dusty operation and the fill material that was emplaced by this device could not be compacted well.

The ideal moisture content of the material used for mechanical stowing was fairly low, with no specific shape preferences for the particles. Mechanical transport in conveyors limited the maximum size of feed material to 8 inches, but sizes of less than 6 inches were preferred in order to prevent spillage (Mitchell, 1968). For slinger or throwing-type mechanical stowers, fill material between 2 and 3 inches was recommended (Luckie and Spicer, 1966). For scraper-type mechanical stowers, various bulk materials including large lump rocks of up to 12 inches in diameter could be used (Smoldyrev, 1978). Fine-fraction material could also be used if not too wet.

While mechanical stowing permitted higher emplacement rates and required less labor than did hand stowing, it never gained wide popularity. Now traditional methods of mechanical backfilling are rarely employed as means of solid stowing, due to the low capacity of the machines, the incomplete filling of the gob that results, the insufficient density of the stowed mass and the congestion often caused in the working area. Moreover, when mining on a longwall, this method cannot be used.

There are no significant health, safety, or adverse environmental hazards that can be associated specifically with mechanical backfilling systems that employ conventional mining machinery. However, the likelihood of fire could possibly increase depending upon the characteristics and the compaction of the stowed material.

2.1.4 - Pneumatic stowing

Pneumatic stowing is highly developed in the German and French mining industries (Cooley, 1978). Pneumatic stowing is an established technique that is most appropriate with seam thicknesses over 2.5 meters and, in particular with dips of more than 30 degrees (Rauer and Voss, 1983). Although pneumatic stowing can be adapted to any active mining system, it has been used exclusively with longwall mining. Rauer and Voss (1983) indicated a convergence of 50 percent with this type of backfilling.

With this method, fill material is injected into the gob by the energy of a jet of compressed air supplied by a compressor or blower. Prior to commencing the backfilling operation, packwalls are built on both sides of the area to be stowed.

The equipment used for pneumatic stowing is identical to that used for pneumatic transport. A stower can be installed at the surface or in the underground panel that is being filled, usually in the tailgate of an advancing longwall panel. There are two methods of pneumatic stowing used with longwall mining: conventional stowing and lateral-discharge stowing.

With conventional stowing, the fill material is discharged from the end of a pipe running parallel to the longwall face. Stowing commences at the main gate and retreats along the length of the face to the tailgate as areas become filled. Brattice-cloth supported on timber props or a heavy mobile stowing shield is used to contain the fill on the face-side. However, more often, a moveable heavy stowing shield is used instead of the brattice-cloth. The shield is moved by a small compressed air-winch in step with the stowing process. As each pipe length of stowing is completed the operation stops and a pipe length is uncoupled and advanced to the position at which the next stowing sequence will begin. The stowing shield is moved along the face for one pipe length and then stowing resumes. The process is repeated until stowing is completed along the entire face. Upon completion of the face length and with the advance of the face, the stowing pipe at the tailgate is extended, usually by the introduction of a new pipe length. This continues until the total length of pipe exceeds the ability of the stower to maintain flow against the pressure losses. At this time the stower and associated haulage supply system must also be advanced. A conventional pneumatic stowing layout for a longwall face is shown in Figure 2.

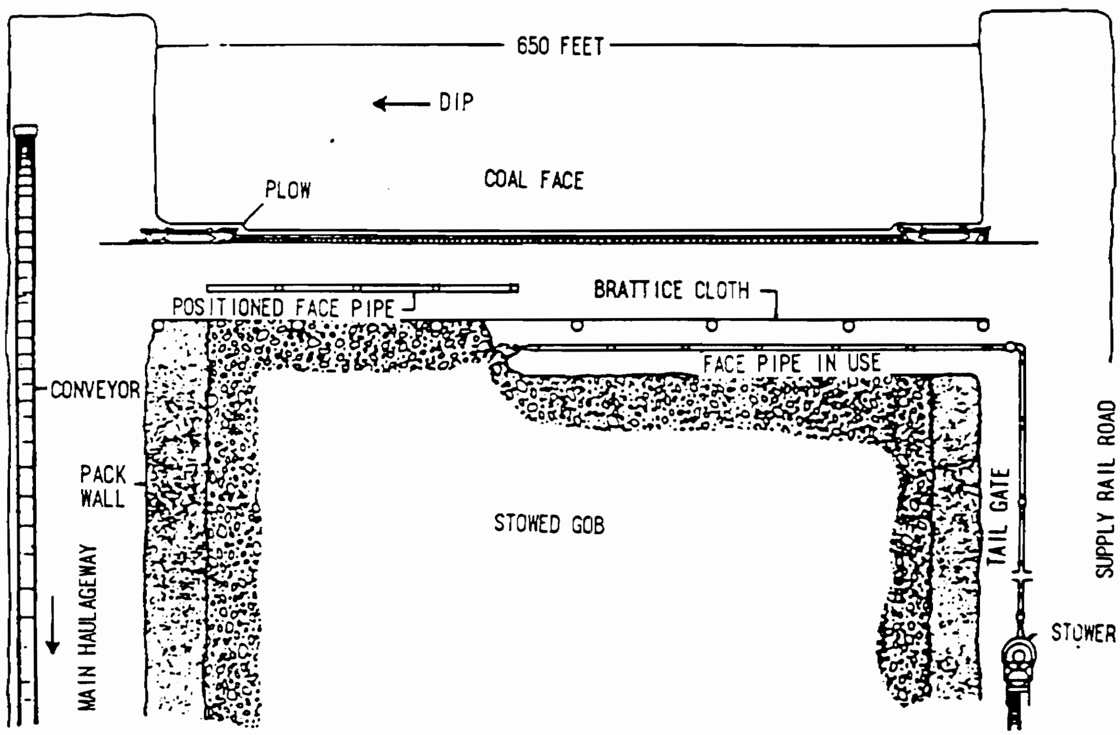

Figure 2: Conventional pneumatic-stowing layout for a longwall face
(Source: National Academy of Sciences, 1975 a)

During stowing operations using the lateral-discharge method, the gob is filled progressively along the face by first operating the distal deflector and thereby filling the first section. Once this section is filled the first deflector is closed and the next simultaneously opened without interruption of the flow of fill material. Advancement of the section of pipe in the tailgate is achieved by the incorporation of telescopic pipes that permit a more or less continuous advance when the telescopic sections are extended. A lateral pneumatic stowing layout for a longwall face is shown in Figure 3.

The conventional stowing system is not suited for highly productive longwall mining, where power loaders and power supports are used for rapidly advancing continuous faces (Bassier and Sander, 1978). Consequently, the conventional stowing system is being replaced by the lateral-discharge system, which is capable of keeping pace with rapidly advancing mechanized longwall faces.

The lateral-discharge system is fully operational for horizontal coal seams, has a high performance rate and lowers the shift outlay (Bassier and Sander, 1978). The decrease in manpower requirements results mainly from elimination of the need for the physically demanding and accident-prone work of manually shifting the stowing pipes,

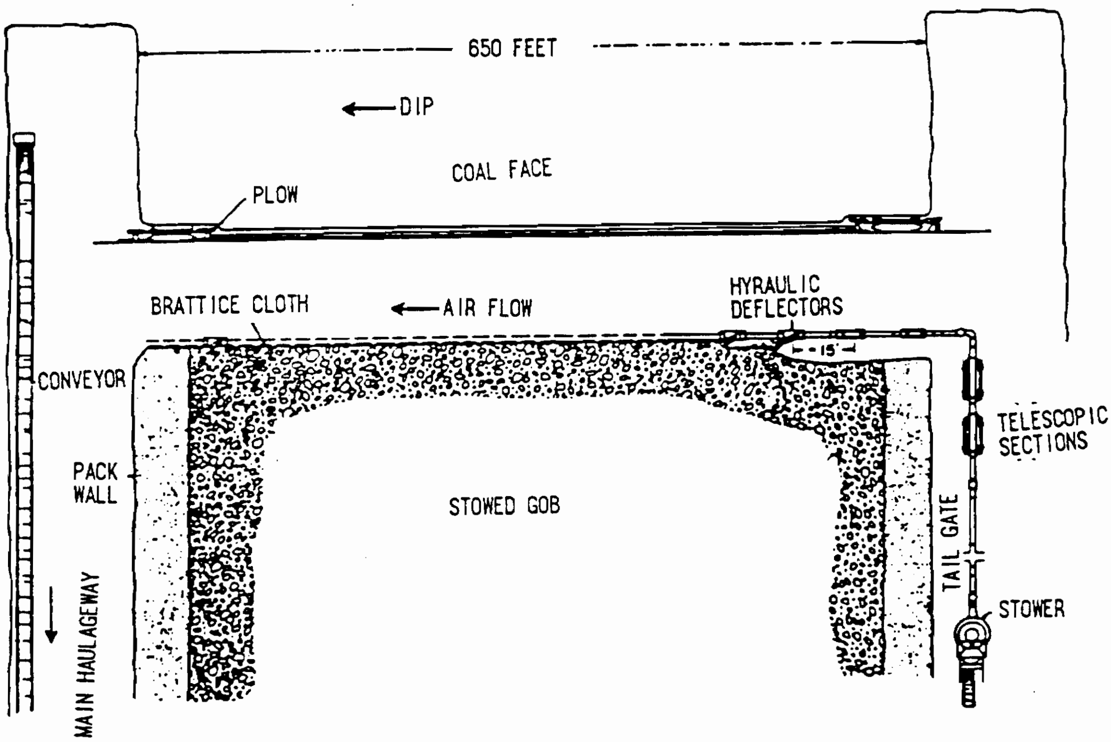

Figure 3: Lateral pneumatic-stowing layout for a longwall face
(Source: National Academy of Sciences, 1975 b)

and for the need to provide timber supports and brattice cloth.

Only two persons are required for the stowing operation: the face stower, who operates the lateral discharge units, and the machine operator, who operates the stower and the remote-controlled conveyor system that transports fill material from the bunker to the stower. Several variations of pneumatic backfilling systems exist. Each utilizes different surface and or underground transportation methods. They are distinguished as pneumatic backfilling with mechanical surface transport, pneumatic backfilling with mechanical in-mine transport and as hydro-pneumatic backfilling.

In pneumatic backfilling with mechanical surface transport, a pneumatic stower is installed on the surface and prepared fill material is brought to the stower by mechanical means. This method is used in Belgium, and has been utilized mainly in mines working steeply dipping seams because the horizontal distribution of the pneumatic transport is limited to a maximum of 2,000 feet before resuspension into the air stream is required. This method does not interfere with the underground coal-transport system (Bucek et al., 1979).

In pneumatic backfilling with mechanical in-mine transport, the prepared fill material is brought to the surface-storage bunker by mechanical means. From there the material is transported underground either by mechanical means or by gravity transport through a shaft or borehole and then by a conveyor or locomotive to the stower. This transport system is separate from that used for coal haulage. However, sometimes the mine cars employed for hauling the coal out of the mine are used to transport fill material from the surface-storage bunker to the stower.

In hydro-pneumatic backfilling a pneumatic stower is installed underground and the fill material is transported to the stower by hydraulic means. This method has been used in Czechoslovakia for stowing the gob of a thick coal seam mined under surface structures (Vorobjev and Deshminkh, 1966). A similar approach has been used in Poland, where clay balls were transported hydraulically and stowed pneumatically (Luckie and Spicer, 1966).

Pneumatic stowing machines have capacities ranging from 20-to-350 cubic meter per hour with an air consumption of 100 cubic meter per cubic meter of stowing material. Pneumatic transport systems handling stowing material are limited in length and confined to the immediate vicinity of the face being stowed (Rauer and Voss, 1983).

Moisture content, shape and size of the stowing material are important factors in pneumatic stowing. The moisture content of the material is important, because it affects dust formation, transport-pipe clogging and compaction of the stowed mass. Typically, for coal waste, the maximum moisture content of the stowing feed is 7 percent, although moisture contents as high as 12 percent have also been reported (Rauer and Voss, 1983). Size requirements of the feed for pneumatic stowing vary with the type of the pneumatic device used. Maximum size of the stowing material particles should not exceed one-third to one-half of the pipe's diameter (Jerabek and Harman, 1966). Generally, a size of 3 inches is suitable for pneumatic stowing, provided that it does not contain excess moisture and fines (Bucek et al., 1979).

In pneumatic stowing, significant amounts of respirable dust can be created when fine particles of fill material are projected by a high-velocity air stream. This float dust, particularly particles smaller than 5 microns, can constitute a health hazard in the immediate working area. Existing methods for reducing dust include: injecting water into the discharge stream of fill material, wetting down the material before it is placed into the stowing hopper, and minimizing the airflow for transporting the material through the stowing pipe.

Pneumatic stowing does not affect mine ventilation to any significant extent (Courtney and Singh, 1972). However, if the ventilation system is leaking, considerable amounts of dust may be blown across the coal face. Dust control is the most serious problem associated with pneumatic stowing, particularly when stowing is done in proximity to sections that are being mined.

Noise produced by most pneumatic stowers typically exceeds the desired sound level. Traditionally, underground coal miners have used the sounds created by falling rocks as warning signals of impending roof falls. It has been stated that the additional noise created by stowing equipment could diminish miner's ability to hear such sounds. Typical, modern mining machinery, however, produces noise far in excess of that generated by the pneumatic stower; therefore, generally roof noises are not detectable in proximity to mining operations regardless of the presence of a stower.

Voss and Sielaff (1977) have stated that complete backfilling, before caving has occurred, will create conditions in which dangerous levels of methane gas could accumulate in the gob area. In addition, because of the resistance to air flow provided by the stowed material, gasses will be restricted in their movement to the bleeders. Whether or not this condition is real and does in fact create a safety hazard has not been resolved. However, if it appears at any time that ventilation in a stowed area will be impeded, explosion-proof seals will be required.

2.1.5 - Hydraulic stowing

The most advanced technology for hydraulic stowing known is used in Europe. This method is used in conjunction with longwall mining, although it is also adaptable to room-and-pillar mining, as practiced in India. Presently, hydraulic stowing is more commonly used than any other method. This method of stowing is suitable for mines working coal seams that are deep and susceptible to bumps, for thick seams worked in multiple lifts, and for steeply to gently dipping seams. European experience indicates that, with hydraulic stowing, a convergence of 20-to-30 percent is achieved (Rauer and Voss, 1983).

Prior to backfilling, the stowing area is sealed off from the face cavity by packwalls. The fill material is mixed with water and placed in the gob, this is done using either a gravity head created by a difference in elevation or an artificial head created by slurry pumps. In most hydraulic backfill operations, a gravity head is used for transporting the slurry from the surface to the gob. The most favorable ratio of vertical-to-horizontal distances for slurry lines is 1:6, although a ratio as high as 1:12 has been used with a low-density sand slurry (Smoldyrev, 1978). This ratio depends on the density of the slurry and frictional resistance of the pipe lines. The scheme of a typical hydraulic stowing installation is shown in Figure 4.

The prepared fill material flows from the surface bunker to the mixing flume, where it is flushed by a water jet from a hydro-monitor nozzle to form a slurry. Feeding from the bunker to the mixing flume is often accomplished by a belt feeder that controls the water-to-solid ratio in the slurry. The slurry flows through the gob under the hydraulic head created by the difference in elevations. At the working face, the slurry is projected into the gob, behind a temporarily constructed stowage retaining dam or barricade along the face. This barricade contains the slurry until the water drains off, leaving the material that fills the gob and forms the stowing massif.

Longwall faces using hydraulic backfill are worked either parallel to the strike or at a slight angle to the dip, advancing up-dip to allow gravity drainage of the stowed mass into the tailgate entry. In any hydraulic stowing system, the major problem is the high volume of water to be dealt with underground. The orientation of the working face of the mine to the dip should be such that gravity drainage of water occurs away from the face. In this way drainage can

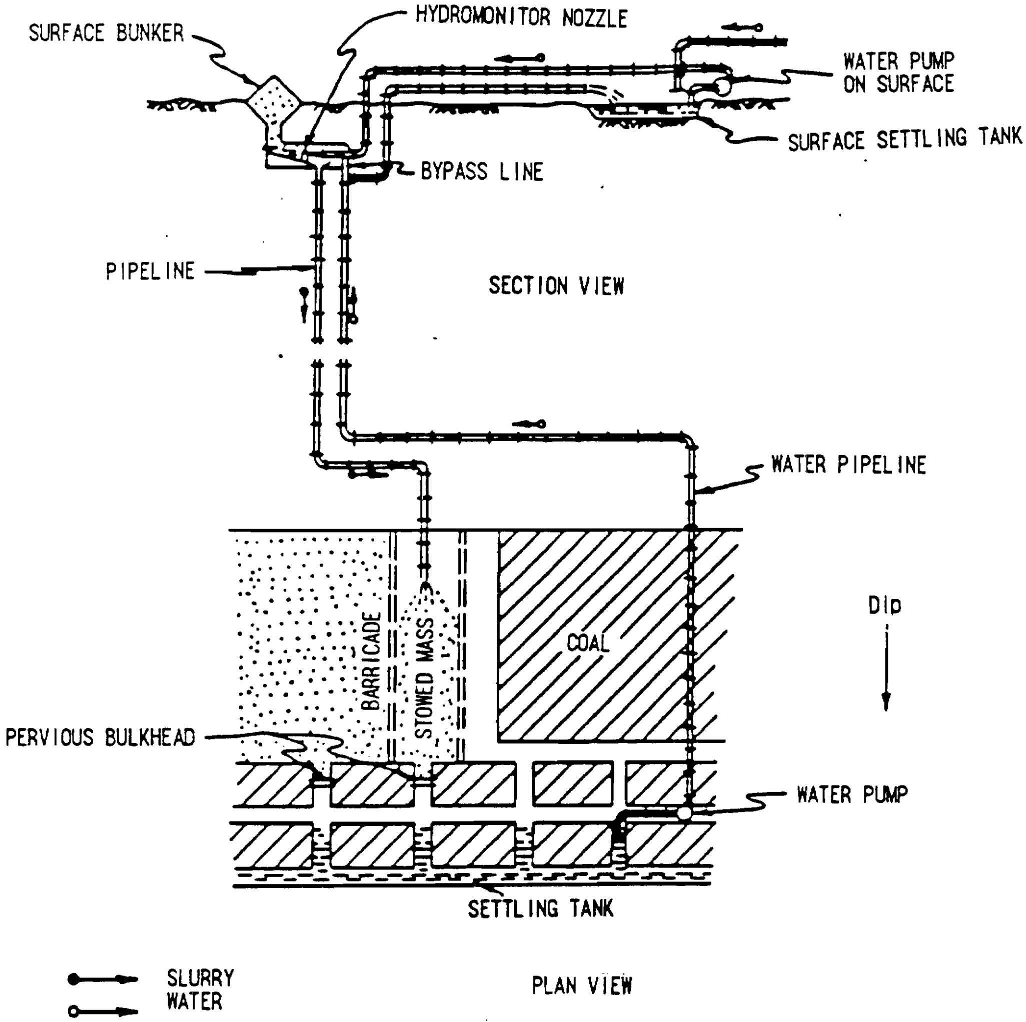

Figure 4: Scheme of a typical hydraulic-stowing installation
(Source : Bucek et al., 1979 a)

be controlled and the water collected in a suitably positioned settling area, from which it can be pumped to the surface for recycling. Some water is left behind in the fill and it is therefore necessary for an additional water supply to be available.

The slurry is laterally discharged by being projected through branch nozzles spaced along the face stowing pipeline. The pipe is suspended from the powered roof supports and advances with them. The stowage-front dam also advances automatically with the mechanized roof support and considerably reduces the labor requirement necessary with conventional dams.

The most extensively used fill material is sand, which has the advantage of being fairly consistent in its characteristics (Munjeri, 1987). Hydraulic stowing operations can emplace up to 215 cubic yards of solids per hour, and the density of the fill is greater than that of all other stowing methods (Smoldyrev, 1978).

Large-scale hydraulic stowing systems have relatively small space requirements, as well as the advantage of ease of operation, when compared to mechanical systems. In addition when compared to pneumatic stowing methods, the potential of increased dust concentrations within the mine atmosphere is eliminated. Hydraulic transport and stowage also do not interfere with the mine haulage system and, as such, are flexible with respect to the location of the stowing equipment. Several variations of hydraulic backfilling systems exist. They are classified as direct hydraulic backfilling, hydraulic backfilling with mechanical surface transport and hydraulic backfilling with the pump located at the stowing panel (Bucek et al., 1979). In the case of direct hydraulic backfilling, material is transported and stowed solely by slurry pumps.

In hydraulic backfilling with mechanical surface transport, material is transported to a central storage bunker on the surface by a conveyor, aerial tramway, truck or rail system. The hydraulic stowing equipment is installed near the central bunker and the slurry is sluiced down a shaft or borehole and through the workings to the gob. The stowing plant is located permanently on the surface, and the slurry is distributed primarily under the gravity head resulting from the difference in elevation between the surface and the mine level. Booster pumps are often used underground to distribute the slurry when the workings are extensive. Hydraulic stowing can also be performed through several boreholes, as was done in controlled flushing projects. However, this system is suitable only for areas where the surface topography allows trucks to transport fill to the boreholes and permits boreholes to be drilled easily, and where a water source is readily available near the stowing area.

In hydraulic backfilling with the pump at the stowing panel, material is transported to an underground stowing machine through a borehole or shaft, then by a coal car, and finally rotary dumped onto a conveyor feeding the hopper of the hydraulic stower. This method has been used successfully in Poland (Kinder, 1978).

Important feed-material requirements for hydraulic stowing or transport relate to the shape, size and size distribution of the particles in the feed, because these factors govern the flow characteristics of the slurry and determine the efficiency of the operation and the drainage characteristics of the stowed mass (Smoldyrev, 1978). The feed material should not exceed one-third of the internal diameter of the pipe. If slurry pumps are used for pressurized hydro-transport, the maximum diameter should not exceed 0.8 times the diameter of aperture of the pump (Smoldyrev, 1978). A maximum particle size of 2 to 2.5 inches is normally acceptable for short distances, but a maximum particle size of 1.5 inches is recommended for long-distance hydro-transport (Dierks, 1931). These requirements will avoid clogging and excessive wear on pipes and pumps. Limitations are placed on the fines, because they affect the filtration and drainage characteristics of the fill (Munjeri, 1987).

Moisture associated with hydraulic backfilling can contribute to the deterioration of pillars, roof and floor, especially to those floors containing expansive clays. These underclays swell with the addition of water and lose strength. Pillars resting on these weakened clays tend to punch into the floor which, in turn, contributes to roof failures. The continual wetting and drying of pillars can causes clays to become slippery, a condition that reduces the maneuverability of men and machinery. Water can also lubricate slippage planes in the roof, promote movement of rock strata and thus contribute to roof failure. With the effective use of water control measures such as ditches, pipes, sumps and pumps, the water problems associated with hydraulic stowing are no worse than those encountered in a typical wet mine.

2.2 - Transportation of fill material

Regardless of the method used for stowing, the material must be transported underground. Depending on the location of the stower, three transport stages may be required: surface transport, transport from the surface underground and underground transport. At each stage, different transport systems or modes may be used. The selection of a particular mode is determined by the local operating conditions, such as the topography, coalbed geology, mine layout, mining system, and the physical characteristics of the fill material.

2.2.1 - Surface transport systems

Mechanical transport systems. Mechanical systems used for surface transport include conveyors, tramways, trucks and railroads.

Conveyor transport systems have a very high capacity and are efficient and quiet in operation. The system has good adverse gradient capability with a maximum grade of as high as 20 degrees provided that the material is fairly dry and contains no large lumps (Mitchell, 1968). The system is capable of transporting large tonnages at high hourly rates over grades that would reduce wheeled vehicle efficiency or that would require an excessively lengthy vehicle haulage time if routed to obtain workable grades. The system has a high initial cost, low manpower requirements and is easy to maintain. Cost per ton-mile is low, provided the utilization is high.

Aerial tramway transport systems are most suitable for rugged terrain, because of their long span and good adverse gradient capabilities. Such systems are often used for transporting coal wastes or for stowing materials over slopes in excess of 30 degrees (Mitchell, 1968), where the use of trucks or belt conveyors would be impossible. Aerial tramways are completely automatic except for inspection and lubrication. They require a loading bin with an automatic loading facility at the loading terminal. At the discharge they require a tripper to trip the tramcar over the storage bunker. The system can handle a capacity ranging from 200 to 400 tons per hour. The system has a high initial cost but low operating costs and a high efficiency (Bucek et al., 1979).

Truck transport systems are mobile, flexible and most effective where good roads are available or can be constructed without excessive grades. Truck transport systems require a loading bin at the receiving end and an unloading hopper at the discharge end with a short feeder conveyor to discharge the prepared fill material into the storage bunker. The system is low in initial and replacement costs. Cost per ton-mile decreases with increased length of haulage route. Where public roads are used, the system is subject to adverse public opinion because of traffic hazards, noise, wear and tear upon roads and the fouling of the roads due to spillage.

Railroad systems able to move fill material to mine site are presently limited to those using high-capacity, wide-gage rail cars which dump their loads through the bottom or through the side walls. Such systems are suitable only when the terrain is level, in which case they can be the least expensive type of transportation available. A railroad system requires loading facilities and dumping facilities with a short feeder conveyor at the storage bunker end. A mine using rail transport for hauling coal from its underground workings to a distant preparation plant can also use the return of empty mine cars to haul fill to an underground stower.

Pneumatic transport systems. Pneumatic conveying uses the energy of a jet of compressed air to move the fill material. With the use of an airtight feeder, waste material is introduced into a stream of compressed air moving with a high speed through the pipe. The operating pressure for a pneumatic system ranges from 1 to 4 times the ambient atmospheric pressure (Whetton and Sinha, 1948). For the combination of pneumatic stowing and powered supports, face pipes with wear bushings of 20 mm thick hard metal casting surrounded by a 5 mm thick shell of metal sheet have proved to be ideal. Since the pipes are not shifted manually, weight does not matter so that thick walls can be selected (Rauer and Voss, 1983).

<u>Hydraulic transport systems.</u> In hydraulic transport systems, the fill is moved in the form of a slurry of a predetermined density by pumps, or by gravity when the available hydraulic head is sufficient. The typical components of this transport system include a mixing installation for making a controlled slurry, a slurry pump, a pipeline installation for pumping and distributing the slurry and a dewatering installation at the end of the transport line to dewater the slurry. For hydraulic transport of coarse coal a 7 mm wear per million tons of coal transported has been reported (Duckworth et al., 1983). The Bureau of Mines conducted laboratory tests to determine the slurry transport and deposition of coal waste. Slurries with up to 60 percent solids were tested. The friction-pressure gradient was 0.24 foot of water per foot of 4-inch standard steel pipe. Deposition tests showed that without the use of a flocculant, the slurries would not dewater when left to stand for a week (McKibbin, 1983).

The mixing installation includes a mixing tank into which the prepared fill is fed and mixed with water to form an evenly mixed slurry. The mixing installation usually has an automatic feed control which maintains a specified solids percentage. The usual percentage of solids varies from 30 to 50 percent, although concentrations as low as 10 and as high as 75 percent have been used by ore-mining companies

(Charmbury et al., 1968). The percentage of solids depends primarily on the particle size, specific gravity and the diameter of the slurry pipe.

Water sources for hydraulic transport can be either pumped-out mine water or the water supply used by the coal preparation plant. Proper mixing of the slurry to reduce excess amounts of water is necessary.

Slurry pumps used for hydraulic transport can be either centrifugal, dredge service-type pumps or positive-displacement high-pressure multiplex plunger or piston-type mud pumps, depending on the job requirements. The positive displacement-type pumps are limited to a maximum particle size of about 0.125 inch, but have a high discharge head, discharging at a pressure up to 2,000 psi, from a single stage (Cummins and Given, 1973). Consequently, these types of pumps are used for low discharge high volumes requiring high heads. Centrifugal pumps are capable of handling coarser solids at a high volume, but with a low discharge head. Positive displacement-type pumps are simple, high in efficiency, low in maintenance and require minimum operating attention, but have a high initial cost. Centrifugal pumps have a low initial cost, low maintenance and a low operating cost. Dewatering facilities at the end of the slurry transport line on the surface usually consist of a bank of cyclones and filters, although other methods of dewatering can be used.

2.2.2 - Surface to underground transport

For transporting fill from the surface to the underground mine level, gravity or mechanical methods are usually employed. Pneumatic or hydraulic transport methods are employed only when the same transport mode also serves as the underground stowing system.

Gravity transport systems. The principle of free-fall is used to move fill from the surface to the underground mine level. Vertical or inclined chutes, cased boreholes or special pipes installed in a shaft are the routes through which material is lowered.

Gravity lowering of material through a pipe installed in a shaft is presently done to depths of 1,000 to 1,600 feet, although the most economical depth range is considered to be 800 to 1,000 feet (Smoldyrev, 1978). In all methods of gravity transport, lowering of the backfill is generally accompanied by continuous unloading at the bottom through a short apron or pan feeder in order to avoid the clogging of pipes or chutes by the material.

Mechanical transport systems. Depending on the depth of the mine, various mechanical means are used for transporting fill from the surface storage bunker to the underground mine level. In Germany, vertical spiral conveyors are used when the vertical lowering depth is not great (Carr and Charlton, 1973). However, this type of conveyor has high wear rates, high maintenance costs and a high likelihood of blockage. When depths are greater than 1,000 to 1,500 feet, fill can be lowered into the mine by mine cars or skip cars through a shaft. Automatic loading and unloading facilities must be provided at the surface and at the shaft bottom when skip cars are used. The use of coal-hauling mine cars to carry fill material on their return to the underground mine may cause scheduling problems, particularly in high-production mines. Often, separate belt conveyors are used for transporting fill from the surface storage bunkers to the underground mine level when the mines are worked by slope or adit entries.

2.2.3 - Underground transport systems

Separate in-mine transportation of fill material must be provided where direct borehole transport from the surface to the stowage area is not possible. Underground transport by gravity is not feasible except in steeply pitching seams. In-mine pneumatic or hydraulic transport is generally used, when it is also the stowing method. Therefore, mechanical means are normally used to transport backfill material in the mine.

Underground transport has two components: the transport system through the main entries and the transport system through the panel entries to the stower hopper. Transportation of backfill material through the main entries is usually performed by locomotives and dump cars of drop-bottom, side-discharge, or sometimes rotary-dump type. Transportation in the panel to the stower is usually performed by a feeder belt or by chain conveyors. A tipple and a storage dump for transferring the material from the dump cars to the feeder conveyor is located at the transfer station. A rail siding is provided near the tipple to accommodate at least one train of backfill-loaded dump cars. The tipple can be designed for either side dumping or, where there is sufficient head room rotary dumping. Sometimes continuous conveyor systems are used for transporting backfill material from the point where the material is lowered to the stower hopper. Most high-production, mechanized mines transport backfill material by using lines apart from those that transport coal.

2.3 - Materials used for backfilling

Materials used for underground stowing must be environmentally acceptable and meet the physical requirements specified by their purpose. Other factors that need to be considered in choosing material for underground stowing are availability, cost and the stowing system being employed.

Backfilling may be used to provide ground support or a working platform, and thus to permit maximum ore recovery. Filling can also be used as a method of waste disposal. In the performance of each function, fill may respond in a number of modes depending on fill type, mining method, and stress environment (Aitchison et al., 1973). A backfilling design must aim at achieving the best compromise between cost and performance (Thomas, 1973).

2.3.1 - Selection of fill material

Since backfill material can be used for several reasons, Scoble and Piciacchia (1986) have proposed an approach to the selection and preparation of fill materials, they attempt to minimize cementing agent addition, while at the same time consideration is given to both materials handling and drainage objectives. In order to achieve these multiple goals, a number of design equations can be employed to evaluate certain parameters of interest when determining the ideal physical properties of the material. Where the same goal may be achieved by more than one alternative, the final decision may be based on economic considerations. Thus, a number of associated costs must be accounted for in approaching a final design. In the case where backfill is solely used as a form of waste disposal, conditions detrimental to the mine environment must be considered.

In order to assess the ability of fill materials to function as ground support, their short-term and long-term mechanical behavior must be determined. Deformability, strength, compressibility, creep, slake durability and permeability are properties that must be known, as must their potential environmental effects. Creep, compressibility and slake durability are long-term properties that may play an important role in assessment of the performance of a given fill material over time. These properties will greatly depend on the mineralogical composition of the fill material. The best material is rigid, free-draining and possesses a high apparent angle of internal friction so as to provide maximum strength and allow minimum deformation (Aylmer, 1973).

The permeability depends upon the void ratio and gradation. Fines must be removed in order to increase fill permeability and to speed drainage (Singh, 1976). If the water does not drain properly, excess pore pressure may build up in the fill. This will cause a reduction in shear strength.

Other factors which affect strength development of cemented fill include void ratio, grain shape, mineralogical composition, grain size and gradation of the aggregate being used. Also important are curing time, moisture content and curing conditions of the cemented fill (Dhar et al., 1983).

Void ratio is perhaps the most important single parameter that affects strength and deformability. Generally speaking, the lower the void ratio, the higher the shear strength, and the lower the deformability.

An aggregate material with a rough surface texture will produce a stronger cemented fill than will a material with a smooth surface, all other factors being equal. This is due to the better bonding which occurs between the aggregate and the cement.

Strength properties are increased by using an aggregate with a high crushing strength that will stop propagation of failure cracks when the fill is highly stressed (Neville, 1987). Fill strength will be diminished in the presence of deleterious substances in the aggregate. These substances may interfere with the process of cement hydration, prevent the development of a good cement bond, or disrupt the cement bond after formation.

The maximum grain size for fill material is important when considering its use as an aggregate. For concrete mixtures with a high water-to-cement ratio, the use of a larger grain size in the aggregate will improve the strength properties of the fill. When using a mixture with a low water-to-cement ratio, the maximum particle size of the aggregate should not exceed 40 mm (Neville, 1987).

The shear strength and compressive strength of cemented fill will be improved by using an aggregate with a low void ratio, a well-graded distribution of grain sizes and an angular particle shape. These factors contribute to better grain-to-grain contact for improved shear strength and better grain interlocking for improved compressive strength. By using an aggregate with these properties, the quantity of cement required is reduced while the strength properties for the mixture are maintained, thus saving on fill costs.

Both shear strength and compressive strength are increased with curing time of cemented hydraulic fill (Barrett, 1973).

Such increases occur due to the increase in the degree of hydration of the Portland cement.

The percentage of solids in the fill mixture should be kept as high as possible since small increases in pulp density result in significant increases in the compressive strength of cemented fill. However, there are limitations on the percentage solids that can be pumped without causing excessive damage to the piping system. Curing conditions also affect the strength of the cemented fill. Factors such as curing temperature, curing humidity and the pH of the water that comes in contact with the fill during curing are important to its final strength (Thomas and Vance, 1969).

2.3.2 - Fill materials

Industrial waste, mill tailings, sand, gravel, crushed rock, waste from refuse piles, residential waste, cementing agents and other additives may be used as fill material. Sometimes a mixture of materials must be used to meet all requirements set for its purpose.

In situations where fill is required to be free-standing, have high strength, superior stiffness and improved long-term performance, the use of a cementing material is needed. Portland cement is the most common cementing agent.

The high cost of Portland cement has led to investigations of the feasibility of using different additives to replace or partially replace this substance. Partial replacement of Portland cement with pozzolans, fly-ash, and slag has been investigated. Pozzolans are siliceous or aluminum silicious materials which have little or no cementitious value, but will in finely ground form react with alkali and alkaline earth hydroxides at ordinary temperatures to form or assist in forming compounds possessing cementitious properties (Nieminen and Seppanen, 1983).

Fly-ash is generated continuously at thermal power plants and coal-based chemical plants. Fly-ash, when mixed with lime, has pozzolanic properties which result in a slow chemical binding action. A detailed knowledge of the mechanical properties of fly-ash is required in order to take advantage of its potential value as a fill material. Tests have shown that fly-ash undergoes a cementation reaction. Studies on the applicability of fly-ash as backfill material have been done and the results can be summarized as follows:

- Grain-size distribution, specific surface area and free lime content significantly influence the quality of the fill.
- The slurry concentration is the most important parameter controlling final strength and thus the quality of

hydraulically placed fly-ash fill.

- The addition of flocculants to the fine fly-ash slurries results in decreased strength, modulus of deformation and final moisture content.
- A high-humidity depositional environment results in a decrease in strength and modulus of deformation and an increase in final moisture content.
- An increase in temperature of the depositional environment results in an increase in strength and modulus of deformation, and a decrease in final moisture content (Fauconnier and Kersten, 1982).

Other researchers have investigated the use of fly-ash as a substitute for Portland cement. Results have indicated that, when replacing up to one third of Portland cement with fly-ash in a mixture of Portland cement and sand, no reduction in uniaxial compressive strength occurs (Thomas, 1973). Thomas has shown that, in mixtures of fly-ash, Portland cement and mill tailings, fly-ash percentages of higher than 20 percent result in significant losses in compressive strength of the material.

Mixtures of slag and pyrrhotite tailings have also been tested as fill material in underground mines. The results are as follows:

- Cementation takes place in various degrees; the higher the tailing content in the samples, the stronger the cementing bond between the slag fragments.

- The strength improves with time, with good strength being achieved after 28 days.
- Based on its properties a slag-pyrrhotite mixture can be used as backfill material (Nantel and Lecuyer, 1983).

Other cementing materials which have been tested include lime and sulfides (Arioglu et al., 1985). Self-cementing sulfide fill has been used as a cementing agent for waste rock (Lukaszewski, 1973). However, problems have been experienced with this type of fill due to the inherent tendency of sulfides to oxidize and give off great quantities of heat. Subsequently, its use has not been applied widely. In addition to this heating action, sulfide fills cause acid drainage water and a reduction in oxygen content in stopes (Bayah et al., 1983). However, strength tests on fills using tailings with 80-to-90 percent (by weight) ferruginous sulfide pyrrhotite have shown strength values even higher than those of tailings which have been strengthened with Portland cement, although controlled comparisons are not available (Lukaszewski, 1973).

Coal mine waste is used for backfilling in active and abandoned European coal mines. In the United States, where this material is readily available, it has only been used for backfilling in abandoned coal mines. When used for this purpose, coal mine refuse generated by preparation plants is separated in coarse and fine refuse. Coarse refuse (+0.6 mm) generally consists of run-of-mine material arising from driving headings and coarse material separated in the washing plant. Fine refuse (-0.6 mm) consists of fines remaining in suspension in the water after the washing process and fines rejected from the froth flotation process used for cleaning fine coal.

Coal mine refuse, if used as a fill material, may eventually degrade after placement and may not have the characteristics appropriate to a stand-alone ground control material. Therefore, stabilization with some cementitious material, performed in order to increase strength and stiffness, may be appropriate. A mixture of 55 percent coal refuse and a high early type III Portland cement can be made to form a cemented backfill that will reach a uniaxial strength of 500 psi in one day. Other tests also indicate that coal refuse, bottom ash, fly-ash, and flue gas desulfurization sludge mixed with Portland cement in a 4 : 1 ratio gives a good strength. A mixture of coal waste and Portland cement in a 2 : 1 ratio failed to give a strength higher than 50 psi. This was thought to be associated with clay and other fine particles in the coal waste. However, coal waste that is unsuitable for cement can be placed as loose backfill (Bur, 1982).

Laboratory test on the engineering properties of coal wastes conducted by Stewart and Atkins (1982) had the following results:

- For non-slaky materials, maximum laboratory densities were highest and permeabilities lowest when the samples contained between 30 and 40 percent finer 4.9 mm.
- Shear strength increased rapidly when the proportion of material finer than 4.9 mm was increased from 20 to 40 percent, but then increased only slightly when the percentage was increased to 60 percent. See Figure 5.
- Optimum moisture content and permeability were the physical properties most affected by the addition of fines. Optimum moisture content increased as much as 3 percent, and permeabilities decreased up to three orders of magnitude. See Figures 6 and 7.

Other researchers have studied the use of coal waste reinforced with horizontal steel mesh or spirally wound high-tensile steel wire. They have concluded that these are very effective artificial support methods for shallow mining and that their viability will depend on local costs and circumstances (Hahn et al., 1983).

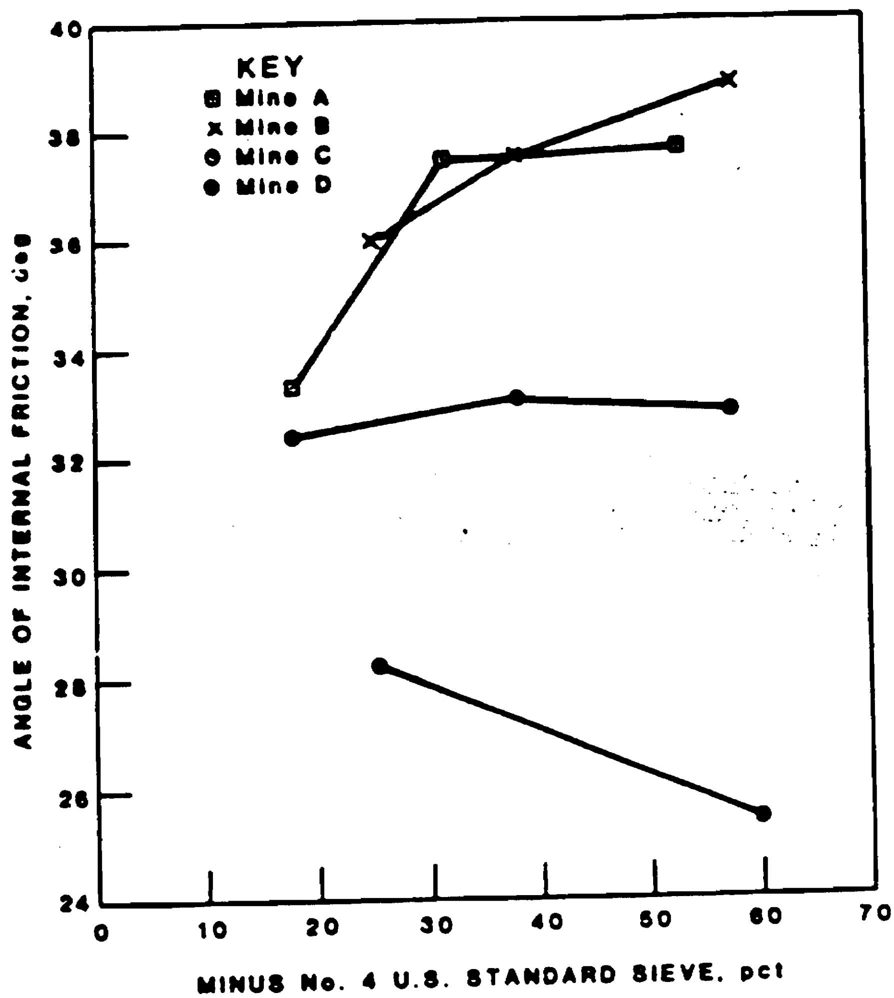

Figure 5: Effects on angle of internal friction of coal waste with increasing percentage material finer 4.9 mm
(Source: Stewart and Atkins, 1982 a)

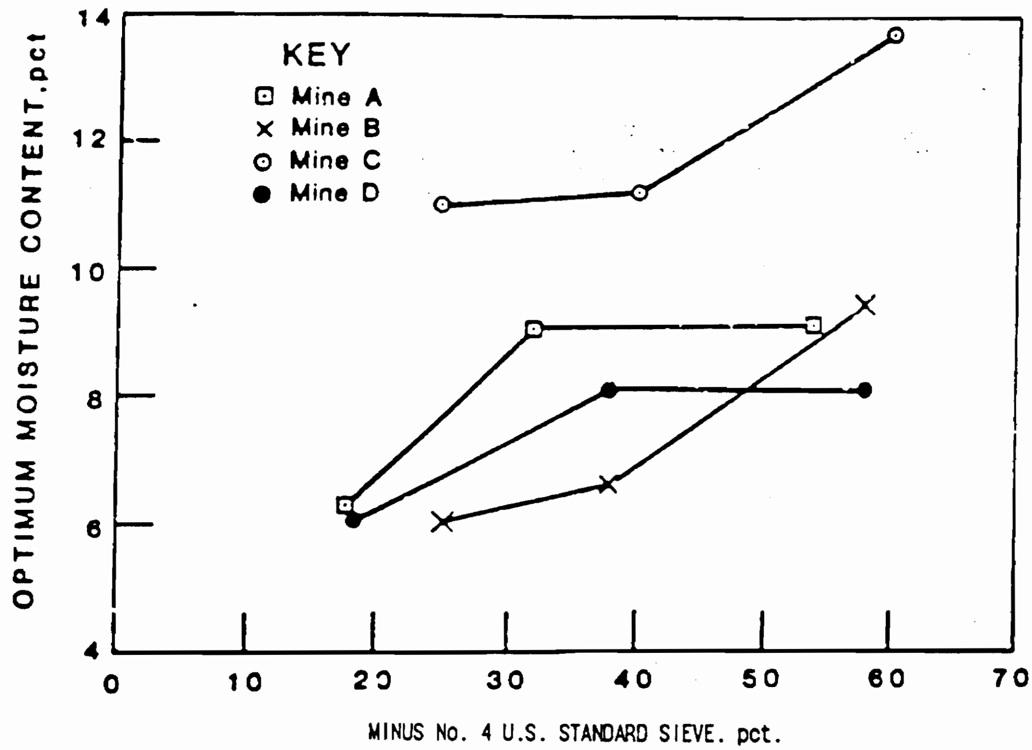

Figure 6: Effects on optimum moisture content of coal waste with increasing percentage material finer 4.9 mm (Source: Stewart and Atkins, 1982 b)

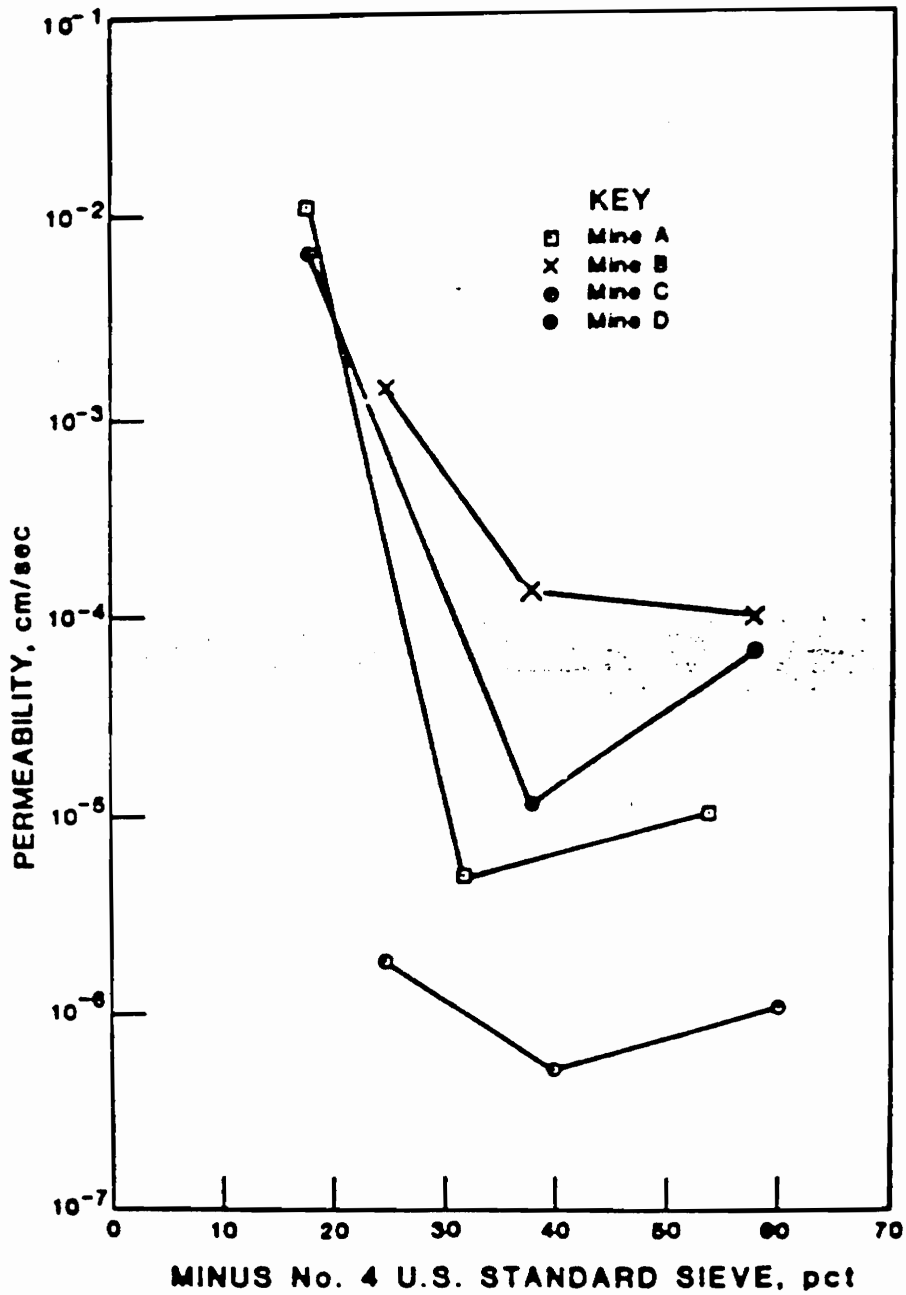

Figure 7: Effects on permeability of coal waste with increasing percentage material finer 4.9 mm (Source: Stewart and Atkins, 1982 c)

The effects of binding additives, fly-ash, bentonite and cement on the strength of coal refuse have been investigated. It has been found that both fly-ash and bentonite increase the strength of fill material when used in amounts equal to about 2-4 percent by volume.

When Portland cement is used as a binding additive, it has been found to affect the coal refuse in such a way that a linear increase in strength is established. The internal angle of friction has been found to increase to a value of 35 degrees. However, the cemented coal refuse has a much lower strength than would be expected from a standard concrete mixture with a comparable concentration of cement. This is due to the much lower strength of the coal refuse when compared to standard concrete aggregates. It must be noted that cemented backfill material does not necessarily have to be as strong as standard construction-grade concrete, as long as its support capabilities are adequate for effective subsidence control (Carlson and Saperstein, 1989).

Residential waste should also be considered as a possible alternative for underground fill. The costs of disposing of this waste in the traditional manner are between $16-$110 (Roanoke Times and World News, April 15, 1990). However, the costs may increase as a result of an increased number of constraints imposed by Federal and State agencies. Research into the use of selected forms of residential waste as reinforcement in cemented fills in underground mines looks promising (Mitchell and Stone, 1987).

Sand, gravel and crushed rock are well known for their use as aggregates in construction cement. Because of their properties they are an excellent material for use as backfill. However, the cost constraints with which they are associated might cause them to be economically infeasible backfill materials. In India, however, sand has been hydraulically emplaced in underground coal mines (Dhar et al., 1983). Selected forms of residential waste can also be used to fill up the gob.

2.4 - Impact of regulations

Regulations have grown out of environmental concern and lessons learned following major accidents. One of the first environmentally-related facets of refuse disposal to be attacked involved burning refuse banks, which created a major air pollution problem in several eastern states until the early 1960's. Pennsylvania was among the first to introduce successful regulations to eliminate this.

Another refuse-disposal problem involved the fact that fine refuse slurry from preparation plants had traditionally been discharged into nearby streams. This problem was addressed in the 1950's and 1960's by many states, by prohibiting this practice. Unfortunately, the regulations, unlike Pennsylvania's air pollution control restrictions, had a hidden danger. Mining companies began to impound slurried fines behind dams constructed of coarse waste. Most companies in the mining industry did not realize the danger presented by this practice until 1972 when a dam along Buffalo Creek in West Virginia failed with a resultant loss of 116 lives (National Academy of Sciences, 1975). After this disaster, many regulations were issued by various Federal and State agencies. All were designed to prevent the recurrence of such an event.

Since 1972, regulatory trends have re-emphasized previous concerns for environmental as well as safety issues. The regulations pending under the Surface Mining Control and Reclamation Act of 1977 are the most stringent issued to date.

Past regulations have altered disposal operations. Stability requirements for impounding structures have changed procedures at numerous locations. Many old coarse-refuse impounding methods have been abandoned in favor of more costly settling-pond disposal systems or combined refuse deposition systems involving the mechanical dewatering of fines in the preparation plant. The outslopes of many old structures have been terraced, covered and planted in order to minimize erosion, control fines and improve stability. End dumping has been replaced by selective placement of coarse refuse in compacted lifts. Water pollution controls have also been added where they previously did not exist. Sedimentation ponds and neutralization facilities have been incorporated into disposal systems in response to Federal and State water pollution regulations. Few requirements have been introduced that lack technical solutions, and resultant costs amortized over the life of disposal sites have generally been quite low. An exception is the cost of dealing with impounding embankments, which have required major reshaping in order to conform to new stability requirements.

The actual impact of regulations can vary greatly, according to conditions that exist at any given site. Many regulations give the regulatory authority power to approve or deny permit applications based on rather vague or general standards. Under these conditions site-specific requirements can vary somewhat, even among neighboring mines. Non-specific regulations give the regulatory agency authority to tailor performance requirements to actual conditions present at a given site.

Of concern to mine operations throughout the country are refuse-related regulations to be issued in the near future by the Office of Surface Mining (OSM). They contain some requirements that may not be technically feasible. For instance, all refuse piles under OSM regulations are to be covered by a four-foot layer of non-toxic incombustible material. This is in excess of nearly all current regulations (Kaufman et al., 1978).

In addition to technical and economic implications of refuse disposal regulations, voids and overlaps have also been researched. With the advent of national regulations imposed by the Mine Safety and Health Administration (MSHA) and recently, as interim regulations by the Office of Surface Mining, voids seem to have been eliminated. Based on this review of regulations, it appears that all essential safety and environmental aspects of refuse disposal have been covered. Overlaps, however, are plentiful (Kaufman et al., 1978).

Future trends in refuse-disposal regulations will be shaped by the same factors that have driven their development in the past: environmental concerns and accidents. It appears that present regulations, particularly those imposed by MSHA, are sufficient to minimize the danger of future catastrophic occurrences and therefore will have little effect the development of future refuse-disposal regulations. It is, at this time, the regulations already imposed by the Surface Mining and Reclamation Act of 1977, that have an impact upon environmentally undesirable aspects of coal waste disposal. Under the act of 1977, the requirements for returning coal processing waste underground are reported (Rubin et al., 1981) to be as follows:

- Each plan shall describe the design, operation and maintenance of any proposed coal processing waste disposal facility for approval by the regulatory authority and the mine safety and health administration.
- Each plan shall describe the source and quality of waste to be filled; method of constructing underground retaining walls, influence of the backfilling operation on active underground mine operation, surface area to be supported by the backfill, and anticipated occurrence of surface effects after backfilling.
- The applicant shall describe the source of the hydraulic transport mediums, methods of dewatering the placed backfill, retainment of water underground and treatment of water if released to surface streams, including the effect on the hydraulic regime.
- The plan shall describe each permanent monitoring well to be located in the backfilled area, the stratum underlying the mined coal and the gradient of the

backfilled area.

- Where applicable the preceding requirements shall also apply to pneumatic backfilling operations.

Underground waste disposal presently plays an insignificant part in overall refuse-disposal schemes. Nor do current regulations specifically encourage these methods; in fact, several agencies require permit applications before such techniques can be employed. In the future, however, as coal production increases, refuse production will also increase. With this gain in refuse quantity may come legislative pressure to backstow refuse underground (Kaufman et al., 1978).

III - ECONOMICS OF SURFACE DISPOSAL

3.1 - Surface disposal practices

As soon as excavation begins at an underground coal mine, waste is generated. The initial waste material usually comes from slope, shaft or tunnel development and consists mainly of rock material, shales, sandstones or limestones encountered as development proceeds toward the coal seam. Once production has commenced, waste will continue to be produced from development and production works underground, as well as coal-preparation plants and coal-powered plants.

By far the greatest source of waste material from underground mining has been in the form of preparation plant refuse. When coal was mined manually, much of the coal was separated from the waste by miners underground. With mechanical systems, coal and any rock partings or other impurities are removed through the use of machines. It is the task of the preparation plant to separate the impurities and to thus produce a saleable product.

The degree to which impurities are removed by preparation plants depends on market specifications, plant capabilities and on the inherent characteristics of the coal. Consequently, there can be a considerable range of physical properties for refuse. Rock materials contributing to the refuse can be classified as clay, stone, siltstone or shale and, in lesser amounts, of sandstone and limestone. Fine coal, high in sulfur and ash content, also contributes to the total refuse volume.

Coal refuse is divided into two size ranges. The portion considered coarse is that which is greater than 28 mesh. Material of less than 28 mesh is classified as fine refuse. Each size range has traditionally been handled separately in the disposal process. When all coal washing processes have been completed, the coarse refuse remaining is screened and routed to a surge bin. From there it is transported to a dump site by any of several methods that might include aerial trams, belt conveyors, trucks or scrapers.

Once at the final disposal site, the coarse refuse is distributed by a bulldozer or front-end loader. In the past, refuse was simply end-dumped. Today coarse refuse is deposited in layers and compacted. The resulting structure is usually a large pile on the surface. These dumps have been characterized (Kaufman et al., 1978) as:

1. Valley fills;
2. Cross-valley deposits;
3. Side-hill deposits;

4. Ridge deposits; or
5. Waste heaps.

Fine refuse is also contained in the wash water at the end of the processing cycle. Until the early 1960's this slurry was usually discharged into nearby streams. With the advent of water-pollution-control regulations which banned this activity, mining companies began to impound the slurried fines behind dams, which were often composed of coarse refuse. Behind such impoundments, fines settled and any water discharged was free of its refuse load. This was a popular method of disposal until 1972 when more stringent regulations for the impounding structures built from coarse mine refuse were introduced. While coarse refuse is still being used in impoundment construction, stability is much more critically reviewed (Kaufman et al., 1978).

Other forms of fines disposal are also becoming increasingly popular. Earthen impoundments used as settling basins are one example. When fines have built up in these basins, an excavator is used to pull the fines from the pond for transport to the coarse-disposal site. Mechanical dewatering of fines in the preparation plant is also gaining favor. In these systems, dewatered fines are mixed directly with coarse refuse and delivered to the disposal site.

Like coarse dumps, impoundments have been given recognizable classifications (Kaufman et al., 1978):

1. Cross-valley;
2. Side-hill;
3. Diked pond; and
4. Incised pond.

3.2 - Economic analysis of surface disposal

In 1989 there were 36 active coal preparation plants in Virginia according to a survey by the Virginia Coal Research Center. Table I provides production data of the coal preparation plants. Because of the great number of preparation plants and variations in the conditions, several assumptions are needed to analyze the cost of surface disposal.

In this analysis it is assumed that the preparation plant, with a clean coal capacity of 2,025,000 tons per year, serves several mines located in the region and generates 675,000 tons of coal refuse annually. The fine coal refuse which makes up 10% of the total coal refuse is dewatered, mixed with the coarse refuse and stored in a bin from where it is transported by trucks for 1 mile to the surface disposal facility. This facility has diversion ditches,

Table I: Capacity of Coal Preparation Plants Operating in Virginia.

County	Coal Type*	Daily ** Capacity	1988 Clean Tonnage
Wise	M/S	3,000	NA
Dickenson	M/S	15,000	NA
Dickenson	M/S	10,000	NA
Russell	M/S	22,000	NA
Buchanan	M	10,500	2,594,593
Tazewell	M	2,800	0
Buchanan	M	7,500	1,681,552
Buchanan	M/S	4,000	890,698
Buchanan	S	NA	0
Buchanan	M	7,500	1,750,000
Buchanan	M	7,500	1,500,000
Buchanan	M	5,500	723,926
Buchanan	M	8,200	2,050,000
Buchanan	M	3,400	848,167
Dickenson	M/S	3,500	NA
Wise	M/S	900/hour	1,027,338
Buchanan	M	4,800	1,180,000
Lee	M	6,000	NA
Buchanan	M/S	4,400	1,150,000
Buchanan	M	1,600	149,002
Buchanan	M	1,600/shift	640,067
Dickenson	M	2,000/shift	500,000
Wise	M/S	6,000	1,200,000
Lee	M/S	4,000	NA
Buchanan	-	3,200	0
Tazewell	M	6,500	1,500,000
Tazewell	M/S	6,000	1,397,000
Buchanan	M/S	6,500	1,666,000
Buchanan	M/S	7,500	1,945,000
Buchanan	M/S	4,500	550,000
Buchanan	M/S	NA	NA
Wise	M/S	10,000	1,750,000
Buchanan	M	7,500	1,650,000
Wise	S	10,200	2,230,095
Wise	S	2,160	39,392
Wise	S	5,700	1,134,815

* S = Steam; M = Metallurgical
** Clean coal

(Source: U.S. Mine Safety & Health Administration, District #5, Norton, and VCCER Virginia Coal Preparation Plant Survey, May, 1989) underdrains, collectors and holding ponds.

To estimate the surface disposal cost, it is assumed that the preparation plant operates 220 days per year in 2 shifts. A disposal area of 160 acres will be sufficient for 20 years. The cost of surface disposal of coal refuse include costs of land, facility design, site preparation, equipment, labor, fuel, supplies, maintenance, power, and miscellaneous. All dollar values are adjusted to the third quarter of 1989 and the salvage value of all items is assumed to be negligible.

The costs are estimated to be as follows:

Land purchase 160 acres @ 1,700 $/acre	= $	272,000
Disposal facility design (25% of purchase price)	= $	68,000
Initial site preparation	= $	1,200,000

Equipment

1	Cat 773 (50 ton) Truck	=	$ 450,000
1	Cat D8N Bulldozer	=	$ 300,000
1	Rock-bin (250 ton)	=	$ 81,000
	Filter equipment	=	$ 650,000
2	Vacuum pumps & motors (500 HP)	=	$ 65,000
	Installation of equipment	=	$ 569,000

Labor

There are 3 persons required for each shift for surface disposal. One to operate the truck, one for the bulldozer, one to operate the filter equipment and rock-bin. A detailed breakdown of labor cost estimation is given in Appendix A together with the assumptions. There are three employees for each shift, therefore the total cost for labor is:

6 @ $ 49,721 = $ 298,326

Fuel, supplies, maintenance, and power:

1 Truck @ 65 $/hr @ 3,190 hr/yr	= $ 207,350
1 D8N Bulldozer @ 56 $/hr @ 3,190 hr/yr	= $ 178,640

Power:

450 Kw @ 3,520 hr/yr @ 0.0226 $/kwh	= $ 35,798
450 Kw @ 108 $/Kw/yr	= $ 48,600

Miscellaneous costs

Underdrains: 300 ft/yr @ 32 $/ft	= $ 10,000
New roads: 700 ft/yr @ 48 $/ft	= $ 34,000
Diversion ditches: 250 ft/yr @ 8 $/ft	= $ 2,000
Clearance: 8 acres @ 3,000 $/acre	= $ 24,000
Reclaiming: 8 acres @ 4,800 $/acre	= $ 38,000
Engineering, reports, surveying	= $ 65,000
Equipment insurance, taxes, licences ($ 1,546,000 @ 2.5%)	= $ 38,650

A summary of the capital and operation cost are presented in Tables II through VI. The capital costs include costs for purchase of land, equipment, facility design, site preparation, and installation as shown in Table II, while Table III presents the depreciation schedule for the capital cost of the surface disposal system with assumed interest rates of 0 and 15 percent. Table IV presents the manpower requirements, and Table V presents the annual operation costs which include labor, overhead and administration,

Table II : Summary of Capital Investment for Surface Disposal

Item	Quantity	Cost
Land purchase		$ 272,000
Facility design		$ 68,000
Site preparation		$ 1,200,000
Truck	1	$ 450,000
Bulldozer	1	$ 300,000
Rock-bin	1	$ 81,000
Filter equipment	1	$ 650,000
Vacuum pumps	2	$ 65,000
Installation		$ 569,000
Total		$ 3,655,000

Table III: Straight-line Depreciation Schedule for
Capital Cost of Surface Disposal System
at Interest Rates of 0 and 15 Percent

Item	Depreciation (years)	Annual Cost (0%)	(15%)
Land purchase	20	$ 13,600	$ 43,466
Facility design	20	$ 3,400	$ 10,866
Site preparation	20	$ 60,000	$ 191,760
Truck	5	$ 90,348	$ 134,755
Bulldozer	5	$ 60,000	$ 89,489
Rock-bin	20	$ 4,000	$ 12,944
Filter equipment	20	$ 32,000	$ 103,870
Vacuum pumps	5	$ 13,000	$ 19,390
Installation	20	$ 28,450	$ 90,926
Total		$ 304,798	$ 697,466

Table IV: Manpower Requirement for Surface Disposal

Description	Number	Annual cost
Truck operator	2	$ 99,442
Bulldozer operator	2	$ 99,442
Preparation plant	2	$ 99,442

Table V : Annual Operating Cost for Surface Disposal

Description	Annual costs
Labor	$ 298,326
Overhead and administration (40% labor)	$ 119,330
Miscellaneous	$ 211,650
Fuel, supplies, maintenance, and power	$ 470,388
Total	$ 1,099,694

miscellaneous, fuel, supplies, maintenance, and power. Table VI summarizes the surface disposal cost at assumed interest rates of 0 and 15 percent. The surface disposal cost at an interest rate of 0 percent is $ 2.08 per ton of waste, while at an interest rate of 15 percent the surface disposal cost is $ 2.66.

3.2.1 - Sensitivity analysis

A sensitivity analysis has been performed by varying the cost items in order to determine the effect of the different cost items on the cost of surface disposal. Table VII presents results from this analysis. As can be seen from the table the surface disposal cost is more sensitive to the variations in fuel, supplies, maintenance, and power costs.

3.3 - Case study of a surface disposal operation

In this case study information from a preparation plant operating in southwest Virginia is provided and described below. The preparation plant handles 3,000,000 tons of run of mine coal and produces 1,800,000 tons clean coal. The remaining 1,200,000 tons is divided into coarse and fine refuse which make up 80 and 20 percent respectively of the refuse. The coarse refuse is transported by belt over a

Table VI: Summary of Surface Disposal Cost at Interest Rates of 0 and 15 Percent

Cost of capital 0 percent:	
Annual capital costs:	$ 304,798
Annual operating costs:	$ 1,099,694
Total annual costs:	$ 1,404,492
Disposal cost per ton of waste:	$ 2.08
Cost of capital 15 percent:	
Annual capital costs:	$ 697,466
Annual operating costs:	$ 1,099,694
Total annual costs:	$ 1,797,160
Disposal cost per ton of waste:	$ 2.66

Note: Annual tonnage 675,000

Table VII: Sensitivity Analysis of Surface Disposal with Interest Rates of 0 and 15 Percent

Variable	Change in cost items (%)	Cost per ton of refuse ($) (0%)	(15%)
Site preparation cost	+30	2.11	2.77
	0	2.08	2.66
	-30	2.05	2.55
Equipment cost	+30	2.18	2.86
	0	2.08	2.66
	-30	1.98	2.46
Labor cost	+30	2.27	2.85
	0	2.08	2.66
	-30	1.89	2.47
Fuel, supplies, maintenance, power cost	+30	2.29	2.87
	0	2.08	2.66
	-30	1.87	2.45
Miscellaneous cost	+30	2.18	2.76
	0	2.08	2.66
	-30	1.98	2.56

distance of 2,500 feet horizontally and 400 feet vertically by using two 36-inch conveyor belts to a 500 ton storage bin. The refuse is then hauled between 2,000 and 3,000 feet by using two 30 ton haulers. A D-7 bull dozer spreads and partially compacts the coarse refuse to form a dam. Loaded haulers traverse the fill to perform the final compaction of the coarse refuse. The fine refuse is pumped 4,000 feet horizontally and 500 feet vertically by utilizing two 150 HP pumps. The fine refuse consist of minus 100 mesh material. The slurry impoundment has a design life of twenty years. The preparation plant receives its raw material from several mines located in the area.

The cost associated with this operation are 0.89 $/ton of coal refuse disposed. In the case when the fine refuse is dewatered and than transported together with the coarse refuse to the disposal site the cost were 1.52 $/ton.

IV - SELECTION OF STOWING METHOD

4.1 - Considerations in selection of stowing method

When selecting and designing an underground stowing method, overall compatibility between the mining system and stowing method is essential. In addition, equipment and technology necessary to introduce the system must be readily available. It is also important that the stowing method cause minimal change and disruption in mining operations and have a minimum effect on health and safety of production personnel as well as personnel engaged in the stowing operation. Moreover, the stowing system should provide maximum associated benefits. Equally important for the design of an underground stowing system are the geologic and hydrologic conditions of the mine and the volume and characteristics of the backfill material being used (GEX, Colorado Inc., 1981). To satisfy or meet all those requirements, a multi-attribute decision analysis technique can be used to analyze which backfilling system will best suit a certain mining operation.

4.2 - Evaluation framework

Munjeri (1987), and Rauer and Voss (1983) came up with technical characteristics of the backfilling methods and desired properties of fill material. In order to evaluate which backfilling system will best suit a certain mine, an evaluation framework has been developed. This framework can be explained with the flowchart presented in Figure 8.

To create this evaluation framework, the backfilling systems and mining operation should be considered. Consequently, attributes can be selected by which the relative merits and applicability of three different backfilling systems can be compared.

One restriction in identifying the attributes is that they be independent of one another, even though this is difficult to achieve. A great number of attributes can be identified that may be important. However, for practical reasons the number of attributes selected should be kept within manageable limits. It should be recognized that too few means important attributes have been ignored, while too many may result in an abundance of futile details. Therefore, attributes should be selected such that each will supply a different information for the selection of a backfilling system.

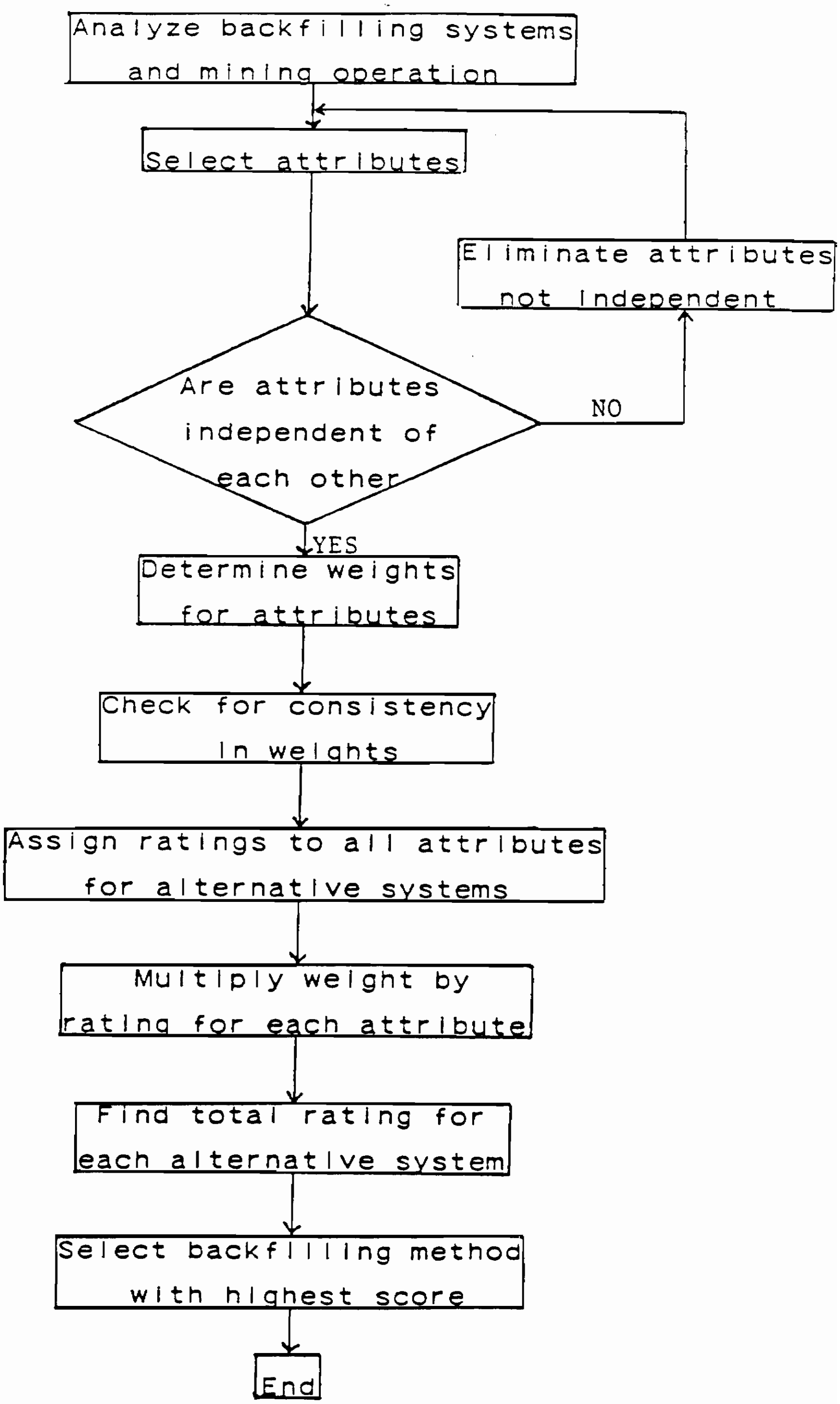

Figure 8: Flowchart of evaluation framework

Once the attributes are selected, weights are assigned to them, this is based on assumptions that the weights are additive and that it must be possible to consider and judge the relative weight of any combination of attributes. The weights assigned are then checked for consistency. Numerical values are assigned regarding the degree to which each backfilling system satisfies each attribute. The weight of each attribute is multiplied by the evaluation rating for that attribute. This is called weighted evaluation. Consequently, the weighted scores for all attributes are summed up for each alternative system. The system with the highest score should be selected.

4.3 - Coal mining operations in southwest Virginia

Information regarding the geology, hydrology and mining operations in this region was gathered in order to select a typical mining operation and relevant attributes to evaluate if backfilling is technically feasible for this region.

The Pennsylvanian strata, the main coal-bearing strata in Virginia, also consist of sandstone, siltstone, and shale. The coal is essentially horizontal except for upturned beds along Pine Mountain to the northwest, Stone and Powell Mountains to the southeast, and Hunter Valley fault near the southeastern edge. Three commercially important coal producing formations are: the Lee, Norton, and Wise Formations (Gwin and Henderson Jr., 1984). A stratigraphic column of formations in the Big Stone Gap region is presented in Figure 9, while Table VIII provides 1987 coal production data in Virginia by County and coal seam.

The Lee Formation includes the coal-bearing strata lying above the Bluestone Formation. In eastern Lee County, the formation is composed of three quartzarenite members separated by intervals of shale, siltstone, and two to six coal beds. Usually characterized by cross-bedding, the resistant quartzarenites form ridges with rugged terrain and cliffs.

The Norton Formation includes all strata lying between the uppermost quartzarenite member of the Lee Formation and the of the overlying Gladeville Sandstone. Composed mainly of medium to fine-grained sandstone, the formation also contains nine coal beds (Gwin and Henderson Jr., 1984).

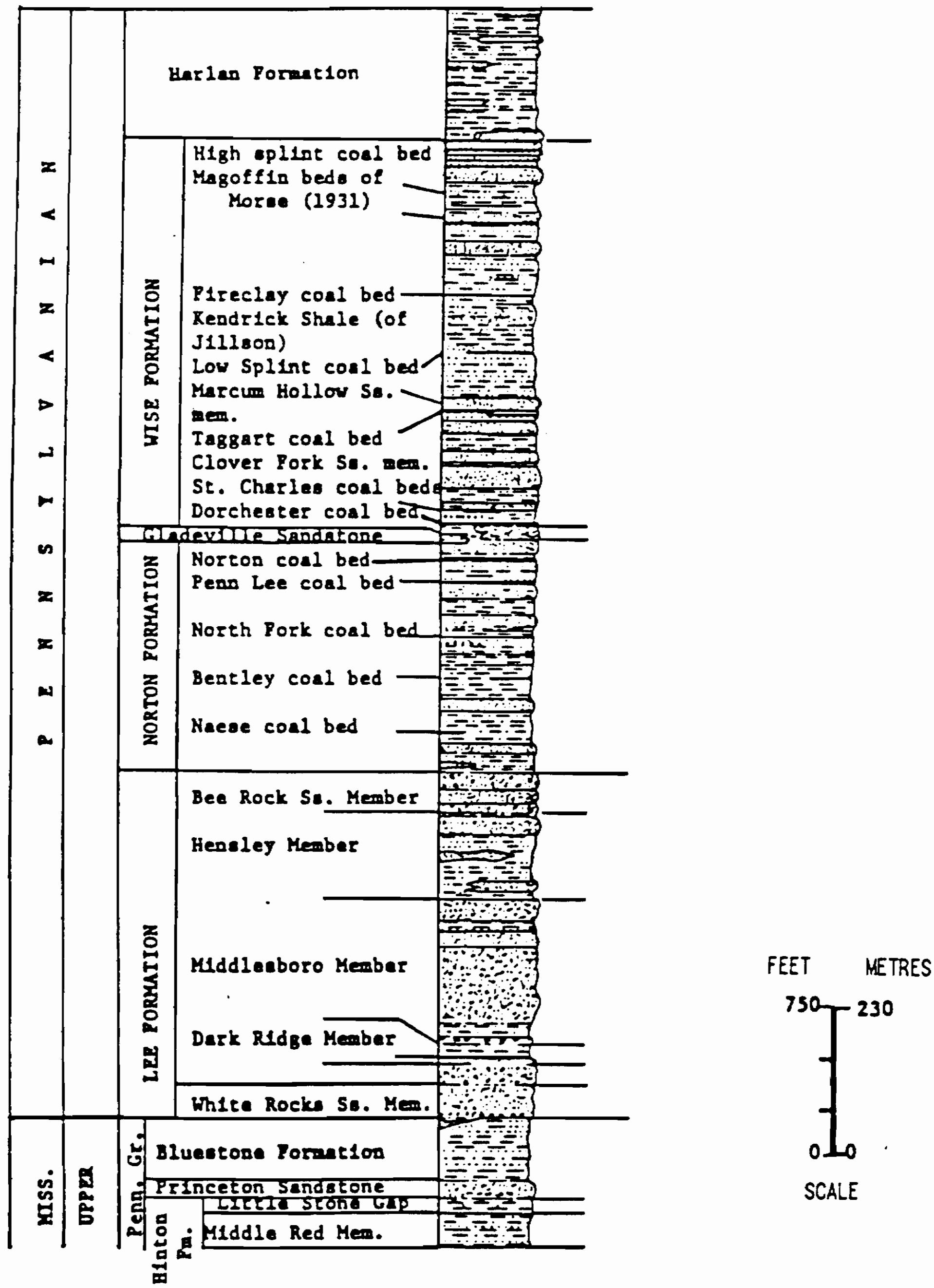

Figure 9: A stratigraphic column of upper formations in the Big Stone Gap region, Virginia (Source: Miller and Englund, 1975)

Table VIII: 1987 Coal Production in Virginia by County and Coal Seam (s. tons)

	Buchanan	Dickenson	Lee	Russell	Scott	Tazewell	Wise
Aily	0	0	0	0	0	0	60,540
Big Fork	0	0	0	28,573	0	0	0
Blair	1,541,269	976	0	0	0	0	406,544
Campbell Creek	37,268	8,231	0	0	0	0	707,663
Cedar Grove	283,929	0	0	0	0	0	123,027
Clintwood	88,013	717,708	7,224	0	0	0	1,868,892
Cove Creek	0	0	0	0	13,350	0	0
Dorchester	529,417	755,674	0	0	0	0	3,810,765
Eagle	739,788	48,580	0	0	0	0	0
Greasy Creek	0	0	0	0	0	442,562	0
Hagy	742,543	977	0	0	0	0	0
High Splint	0	0	0	0	0	0	293,301
Jawbone	2,403,348	1,279,039	0	177,515	0	0	228,745
Kelly	0	0	0	0	0	0	1,110,121
Kennedy	1,193,772	1,654	0	41,073	101,801	0	10,246
Kirk	0	0	0	0	0	0	0
Lower Banner	34,281	1,559,177	0	0	0	0	0
Lower Horsepen	0	0	0	0	0	107,376	0
Low Splint	0	0	129,780	0	0	0	682,478
Lower Seaboard	0	0	0	0	0	554,741	0
Lower St. Charles	0	0	145,223	0	0	0	2,700
Lyons	0	24,665	0	0	0	0	400,786
Middle Horsepen	0	0	0	0	0	1,360	0
Morris	0	0	0	0	0	0	251,516
Middle Seaboard	0	0	0	0	0	61,068	0
Pardee	0	0	392,725	0	0	0	870,929
Phillips	0	0	2,000	0	0	0	196,783
Pinhook	0	0	0	0	0	0	9,310
Pocahontas #3	7,585,324	0	0	0	0	326,901	0
Pocahontas #5	0	0	0	0	0	39,263	0
Pocahontas #8	0	0	0	0	0	101,290	0
Raven	891,245	847,171	0	101,420	0	97,426	0
Splashdam	2,398,516	995,151	0	0	0	0	121,322
Taggart	0	0	1,521,508	0	0	0	498,218
Taggart Marker	0	0	0	0	0	0	91,537
Tiller	364,603	178,705	0	114,401	0	78,280	0
Upper Banner	0	1,915,762	0	20,561	0	0	292,694
Upper Horsepen	0	0	0	0	0	469,904	0
Upper Standiford	0	0	377,771	0	0	0	635,751
Upper St. Charles	0	0	4,036	0	0	0	0
Wax	0	0	538,171	0	0	0	0

(Source: Virginia Division of Mineral Resources, Publication 95, 1989)

The Wise Formation lies above the Gladeville Sandstone and below the Harlan Sandstone. Lithologically similar to the Norton Formation, the Wise Formation consists of sandstone, siltstone, shale, minor limestone, and 18 named coalbeds. The sandstones, which constitute approximately one third of the formation, are light gray, fine to medium-grained, thin to massive-bedded, micaceous, feldspathic, and contain about 50 to 60 percent quartz (Miller and Englund, 1975).

The thickness of the mined seams in this area varies between 22 and 84 inches (Virginia Coal Mining Directory, 1988). Mine roofs in the region consist of shales eroded in places by sandstone washouts. Their stability changes with local variations in the lithology and deteriorates especially in sections where sandstone washouts or rolls occur in the immediate mine roof. Moderate to severe mine roof problems occur in less than 25 percent of the mined areas. Mine floors were reported to be firm to somewhat soft; when these are wet for prolonged periods of time, they become plastic and rutted (Bucek et al., 1979).

The hydrologic conditions of each mine depend on the permeability of the overlying strata and infiltration rates through the surrounding rocks. The mines are damp to very wet with shallow water pooling, and most of the mines require pumping rates in excess of 5,000 gallons per day

(Bucek et al., 1979).

In this region most of the land is covered by forests, and virtually all housing and industrial developments are located on floodplains. Railroads and highways are predominant modes of coal haulage. Trucks are used most commonly for coal hauling and refuse disposal (Bucek et al., 1979).

According to the Virginia Department of Mines, Minerals and Energy, southwest Virginia produced a total of 46,359,000 tons of coal for 1988. Table IX provides coal production data for 1988 in Virginia by County and mining method.

The 1988 longwall census reported 13 operating longwall mines, constituting 21.8 percent of the total production and 26.3 percent of the total underground production (Virginia Department of Mines, Minerals and Energy, Division of Mines, 1989). Table X provides production data for Virginia's top underground coal mines in 1988 with the related mining method. In southwest Virginia 545 coal mines were in operation for 1988 (Virginia Department of Mines, Minerals and Energy, Division of Mines, 1989).

Table IX: 1988 Coal Production (s. tons) in Virginia by
County and Mining Method

County	Mining Method				Total
	Surface	Underground			
		Continuous	Longwall	Conventional	
Buchanan	1,344,955	11,010,763	6,382,678	1,030,841	19,769,236
Dickenson	1,174,044	4,665,115	1,668,955	492,372	8,000,486
Lee	171,601	1,251,044	1,126,776	16,949	2,566,370
Russell	80,333	274,997	-----	5,317	360,648
Scott	-----	121,902	-----	-----	121,902
Tazewell	-----	2,739,785	-----	414,044	3,153,830
Wise	5,166,154	6,270,512	942,894	7,062	12,386,622
Total	7,937,087	26,334,118	10,121,303	1,966,585	46,359,093

(Source: Virginia Department of Mines, Minerals and Energy; Division of Mines, Big Stone Gap, Virginia, 1989)

Table X: 1988 Coal Production of Major Underground Coal Mines in Virginia by Mining Method

Name of Mine	Mining method	Production (s. tons)
Buchanan #1	Longwall	2,594,593
VP #1	Longwall	1,733,143
VP #6	Longwall	1,681,552
VP #5	Longwall	1,632,244
VP #3	Longwall	1,470,118
Holton	Longwall	1,190,929
Bullitt	Longwall	1,091,662
Mc Clure #1	Longwall	907,549
Splashdam #1	Longwall	723,388
Lambert Fork #2	Longwall	706,578
#2	Room-and-pillar	495,801
Big Creek #2	Room-and-pillar	454,402
Greenbrier #1	Room-and-pillar	407,710
#1	Room-and-pillar	355,568
Prescott #2	Room-and-pillar	347,218
Youngs Br. #15	Room-and-pillar	338,176
Wentz #1	Room-and-pillar	320,593
Winston #10	Room-and-pillar	314,395

(Source: Virginia Department of Mines, Big Stone Gap, Virginia, 1989)

Drift mines are predominant in this region (EPRI, 1977). The recovery for room-and-pillar mining varies between 50 and 80 percent. For longwall mining the recovery averages around 70 percent (Crickmer and Zegeer, 1981).

4.4 - General description of mine model

Because of the large number of coal mines operating in southwest Virginia and the variation in mining method, geologic and hydrologic conditions a hypothetical model mine typical for this region was developed by Bucek et al.(1979).

This model is updated and then used to demonstrate how a stowing method can be selected for a mine, by using the evaluation framework described earlier.

It is assumed that the hypothetical model mine operates two shifts per day, 220 days per year, and has a 20-year life. Out of five working sections, only four sections are productive at any time. The fifth section is kept for contingency. The four sections produce 1,181 tons of coal per shift.

Assuming 1,800 tons of clean coal per acre-foot with 4 feet of mining height and using a 60-percent mining recovery factor, a 1,806 acre resource will be required to produce 390,000 tons of clean coal per year for 20 years. Given a rejection rate of 25 percent, 520,000 tons of run of mine coal will be produced. Thus generating 130,000 tons of coal refuse per year.

The average seam depth is 700 feet with main entries driven up-dip at three percent and a three percent dip of the seam. The production sections have six entries. Entries and crosscuts are 20 feet wide, while pillars are usually square, 60 x 60 feet. In the lower sections of the mine special drainage measures include sumps, drains, and pumps. Water is pumped out of the mine at a rate of 5,000 gallons per day. The mine floor is soft when wet and firm when dry.

Each mining unit will consist of a continuous miner, two shuttle cars, and a roof bolter. The shuttle cars will dump into a ratio feeder at the tailpiece of the unit belt conveyor. Unit manpower consists of nine men. Coal is transported to the surface by belt conveyors. Main heading, production heading and panel belts are 36 inches wide. Trucks are used to transport the coal from the mine to a preparation plant three miles away. Tables XI, XII, and XIII provide details of the mining operation.

Table XI: Geologic and Hydrologic Parameters for Mine Model in Southwest Virginia

Seam name	Pocahantas
Average seam thickness	48 inches
Average seam depth	700 feet
Seam dip	3 percent
Changes in seam dip	gradual
Mine roof lithology	shale, siltstone
Mine area with roof problems	< 25 percent
Mine floor Characteristics	firm to soft when wet
Coal rank	bituminous
Coal sulfur content	1.2 percent
Coal ash content	9.9 percent
Hardgrove grindability index	30
Heat value	13,379 Btu/lb
Methane emissions	never stops miner
Water quality problems	none
Special mine drainage measures	sumps, drains, pumps
Pumping rates	> 5,000 gallons/day

Table XII: Mining Parameters for Mine Model

Mine type	drift
Mining method	room-and-pillar, continuous
Number of shifts	2
Mine life	20 years
Production/shift	1,181 s. tons
Annual clean coal production	390,000 s. tons
Annual run of coal production	520,000 s. tons
Mining area	1,806 acres
Percent extraction	60
Number of sections	5
Personnel/section	9
Number of entries per section	6
Entry and crosscut width	20 feet
Pillars (length x width)	60 x 60 feet
Average face haulage distance	250 feet
Floor gradient	main entry driven up-dip 3 percent
Face haulage	(2) 4-ton cable reel shuttle cars
Intermediate haulage	36-inch belt
Average intermediate distance	1750 feet
Main haulage	36-inch belt (2000-foot sections (4))
Average primary haul distance	8,000 feet
Portal to preparation plant	3 miles
Surface transportation	(3) 42-ton trucks

Table XIII: Preparation Plant Parameters for Mine Model

Annual clean coal production	2,025,000 s. tons
Annual refuse production	675,000 s. tons
Reject (%)	25

Note: This preparation plant serves several coal mines in the region

4.5 - Application of evaluation framework

For this evaluation it is assumed that coal waste generated from the preparation plant will be used as backfill material, since it is readily available, costly to dispose of on the surface, and causes environmental problems if disposed of on the surface.

Besides the cost of fill material, several other attributes need to be considered before selecting a backfilling system. Several authors Munjeri (1987), and Rauer and Voss (1983) provided information about desirable features of attributes. Table XIV lists the attributes that should be evaluated during the selection process.

To assign weights to the attributes selected, they were ranked in a decreasing order of importance. Subsequently, the attribute considered most important was given a weight of 10 and the other attributes was given a weight relative to the weight of the first attribute. This was accomplished by comparing the attributes to each other and determining their relative importance. The weights were then normalized to sum up to 10. This was done by multiplying each weight by 10 and dividing it by the sum of the weights assigned to all attributes.

Table XIV: List of Attributes Evaluated During Selection Process

Cost of fill material
Availability of fill material
Effects on productivity
Pre-treatment of fill material
Availability of water
Equipment cost
Capacity
Number of operators
Availability of equipment
Flexibility
Seam thickness
Dip of seam
Mining method
Mine floor
Mine roof
Health effects on stowing personnel
Health effects on production personnel
Effect on groundwater
Convergence

To select a backfilling system for this particular mine, a heavier weight has been placed on attributes related to the cost and the technical aspects of the backfilling method. So that a backfilling system will be selected that is relative inexpensive and technically feasible.

In the next step numerical ratings were assigned to each attribute depending upon how each alternative backfilling system satisfies each attribute. A rating of 5 was given if an attribute is found completely satisfactory, while 1 was given if an attribute is not satisfactory. A value between 5 and 1 was given to the attributes whose conditions were in between. Table XV presents a format according to which the attributes of the alternatives were evaluated, while Table XVI presents the ratings given to the attributes of the three backfilling systems. Table XVII presents the weighted evaluation of the three backfilling systems with their total score. The alternative with the highest total score is the one that will best meet the priorities set. In this case hydraulic backfilling is selected since it has the highest score. The second best alternative is mechanical backfilling, while pneumatic backfilling is the least desirable method. The cost and design concepts of both hydraulic and mechanical backfilling are further evaluated and are presented in Chapter 5 and 6 respectively.

Table XV: Format to Assign Ratings to Attributes of Backfilling Systems

Attribute	Evaluation rating	
	5	1
Cost of fill material	Minimal	Expensive
Availability of fill material	Does not influence operation	Restrict the operation
Effects on productivity	No effect	Decreases heavily
Pre-treatment of fill material	Not needed	Screening, Crushing, Blending, Dewatering
Availability of water	Necessary available or not necessary	Necessary not available
Equipment cost	Minimal	Expensive
Capacity	High	Low
Number of operators	<=6	>=14
Availability of equipment	Available	Needs to be developed
Flexibility	Very flexible	Inflexible
Seam thickness	Does not influence operation	Has negative effect on operation
Dip of seam	Does not influence operation	Has negative effect on operation
Mining method	Has no effect on operation	Has negative effect on operation
Health effects on stowing personnel	Nothing	Dust, Damp, Noise
Health effects on production personnel	Nothing	Dust, Damp, Noise
Effect on groundwater	Less than with surface disposal	More than with surface disposal
Convergence	20 %	50 %
Mine floor	Does not influence operation	Has negative effect on operation
Mine roof	Does not influence operation	Has negative effect on operation

Table XVI: Ratings of Attributes for Three Backfilling Methods

Attribute	Backfilling method		
	Hydraulic	Pneumatic	Mechanical
Cost of fill material	5	5	5
Availability of fill material	5	5	5
Effects on productivity	4	2	3
Pre-treatment of fill material	3	1	4
Availability of water	5	4	4
Equipment costs	5	1	3
Capacity	5	5	3
Number of operators	3	1	5
Availability of equipment	3	1	5
Flexibility	5	3	4
Seam thickness	5	1	5
Dip of seam	5	5	1
Mining method	4	4	5
Mine floor	4	5	5
Mine roof	4	4	4
Health effects on stowing personnel	4	2	3
Health effects on production personnel	5	5	5
Effect on groundwater	3	5	5
Convergence	5	3	1

Table XVII: Weighted Evaluation of Three Backfilling Methods with Total Score

Attribute	Backfilling method		
	Hydraulic	Pneumatic	Mechanical
Cost of fill material	4.0	4.0	4.0
Availability of fill material	4.0	4.0	4.0
Effects on productivity	3.2	1.6	2.4
Pre-treatment of fill material	2.1	0.7	2.8
Availability of water	3.5	2.8	2.8
Equipment costs	3.0	0.6	1.8
Capacity	3.0	3.0	1.8
Number of operators	1.8	0.6	3.0
Availability of equipment	1.5	0.5	2.5
Flexibility	2.5	1.5	2.0
Seam thickness	2.5	0.5	2.5
Dip of seam	2.5	2.5	0.5
Mining method	1.6	1.6	2.0
Mine floor	1.6	2.0	2.0
Mine roof	1.6	1.6	1.6
Health effects on stowing personnel	1.2	0.6	0.9
Health effects on production personnel	1.5	1.5	1.5
Effect on groundwater	0.9	1.5	1.5
Convergence	1.5	0.9	0.3
Total score	43.5	32.0	39.9

V - ECONOMIC FEASIBILITY OF HYDRAULIC BACKFILLING

5.1 - Design concept of backfilling system

The size of this backfilling operation is based on a model mine described earlier that operates 220 days per year and produces 520,000 tons of run of mine coal and 130,000 tons coal refuse annually. This by-product will occupy 25 percent of the voids created by this mining operation.

Since the mine operates 220 days per year, 590 tons of refuse are produced per day. The refuse consists of 334 tons of material larger than 3/8 inch and 256 tons of material that is finer than 3/8 inch. This amount of coal refuse can be stowed in one shift operating five hours of the eight available. When backfilling is in progress, five sections will be worked; four sections for coal production and one for backfilling. The sections designated for hydraulic backfilling should be laid out in the direction of maximum dip starting off the sub-mains. This facilitates the handling of spent water from the backfilling area.

The coal refuse will be transported from the preparation plant to the mine site by the same trucks that transport the run of mine coal to the preparation plant three miles away. Once at the mine site the refuse will be screened, crushed to -3/8 inch and stored in a silo prior to being mixed to a slurry containing 60 percent solids and pumped into the mine, where it will be retained in an area that is surrounded by pervious barricades. The water drained off from the fill can then flow to the lower section of the mine from where it is pumped out to the water reservoir.

Figure 10 provides a conceptual layout of the system that is sub-divided into a surface transport system, a material preparation system and an operation system underground. The elements of this system are summarized below, while information supporting the data is included in Appendix B.

5.1.1 - Surface transport

Refuse produced in the preparation plant is hauled by trucks from the refuse load-out bin at the preparation plant to a waste processing facility near the mine site over a distance of three miles. The cycle time analysis of the trucks for the assumed length and grades of haul road, shown in Appendix B, indicates that the existing three coal hauling trucks for the model mine are adequate to haul 300 tons of refuse per shift on their return trip from the preparation plant to the mine. Therefore no additional trucks or drivers are required for surface transport of refuse. However, the operating cost and roadway maintenance will

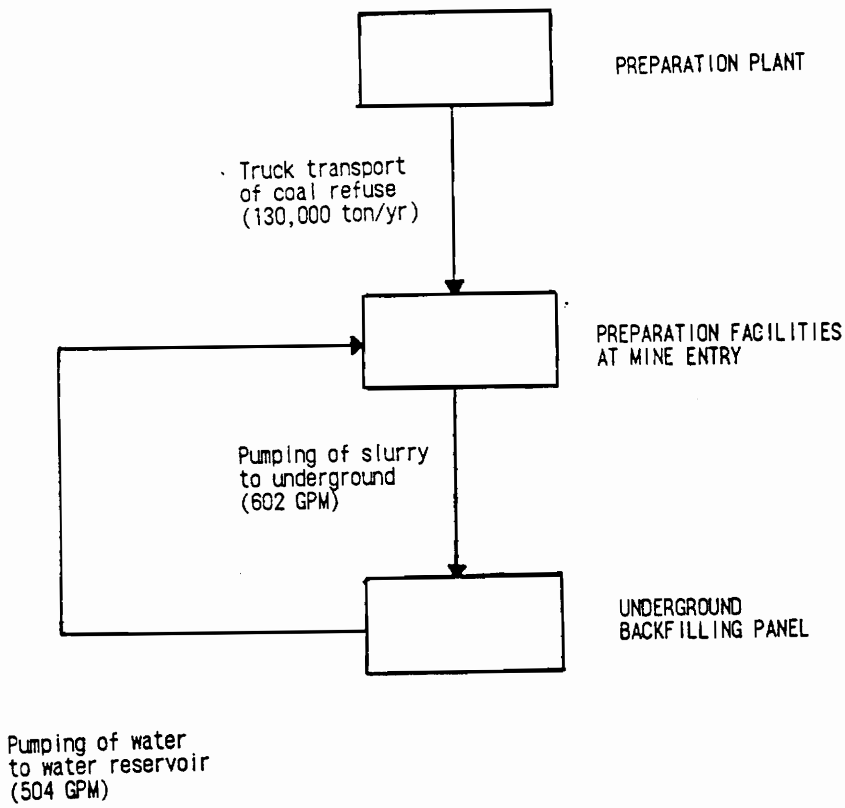

Figure 10: Conceptual layout of hydraulic backfilling system

increase because of additional fuel and lubricant consumption as a result of the longer travel distance of trucks with load.

5.1.2 - Material preparation

Refuse coming from the preparation plant to the fill material preparation facilities located at the mine entry is screened with a 3/8 inch aperture vibrating screen and the coarse fraction is transported to an impact crusher where it is crushed to -3/8 inch. This material than goes back to the screen. The -3/8 inch fraction is stored in a silo from where it is transported to a mixing tank prior to being pumped to the backfilling section. The silo is capable of storing 900 tons of refuse. In the mixing tank a slurry is formed with 60 percent solids (by weight). Water required for the slurry preparation is obtained from a water reservoir located at the mine entry as shown in Figure 11.

From the mixing tank the slurry is directly pumped into a designated backfilling panel by three centrifugal slurry pumps working in series to develop the required pressure head. The slurry and water line are laid along the service haulage road of the mine to facilitate handling, replacement and repair of the pipelines. The layout of the slurry and water pipeline are shown in Figure 11. From the reservoir

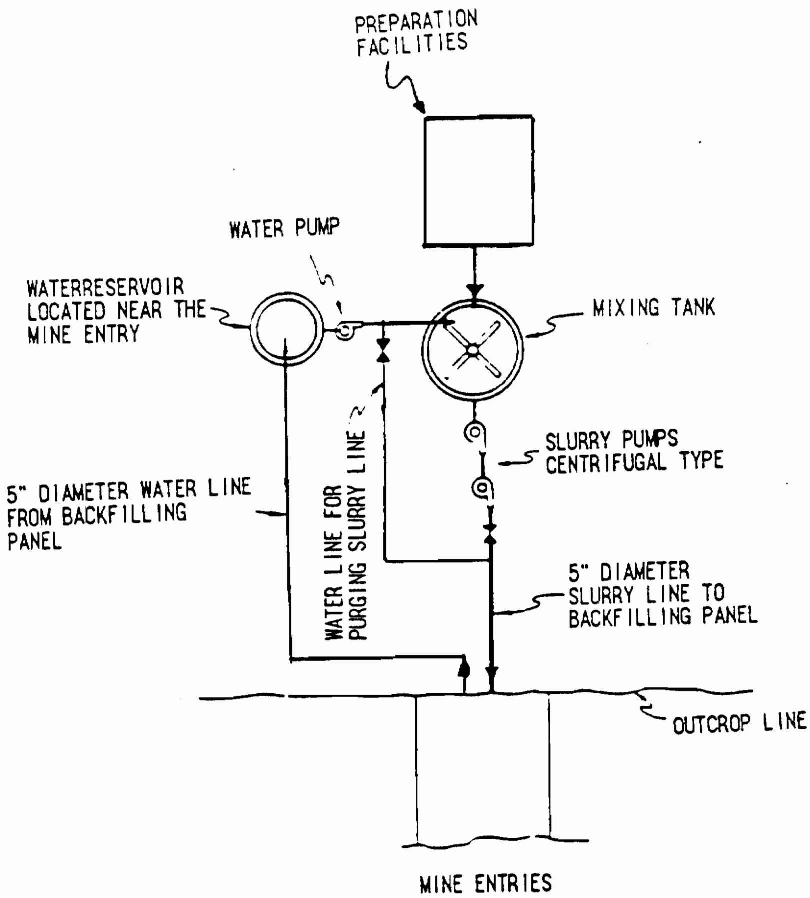

Figure 11: Slurry and water handling arrangements

located near the mine, water is pumped to the mixing tank for recirculation. The slurry line within the backfilling panel is made of lighter weight and shorter pipes with quick release couplings.

Equipment and manpower required at this stage are as follows:

- One preparation building that will contain the crusher, screen, mixing tank and slurry pumps.
- One hopper with a 50 tons capacity.
- One impact crusher for crushing coarse material to -3/8 inch, with a capacity of 75 tons per hour.
- One silo with a 900 ton capacity.
- Three 24-inch by 200 feet feeder belts (one from the vibrating screen to the crusher, one from the crusher to the vibrating screen and one from the screen to the silo, all with a capacity of 100 tph.
- One 5-foot by 14 foot single deck vibrating screen with a Capacity of 50 tph.
- One 24-inch by 150-foot variable speed feeder belt from the silo to the mixing tank fitted for measurement of the feed rate with a capacity of 150 tph.
- Three slurry pumps of centrifugal type, installed in series for pumping the slurry to the backfilling panel with the necessary discharge valves and pressure gauges with a capacity of 800 GPM.

- Ten thousand feet of 5-inch diameter, abrasion-resistant steel pipe slurry line, five thousand feet of epoxy-coated fiberglass slurry line for the backfilling panel.
- Slurry mixing tank with a capacity of 100 cubic foot.
- Agitator.
- One 50 GPM centrifugal pump with density control meter.
- One reclaim water storage reservoir with a capacity of 16,000 cubic foot
- One 700 GPM centrifugal pump to pump water from reservoir to mixing tank.

The manpower required at this stage is two: one at the screening and crushing plant and one person to operate the slurry pumps and monitor the feed to the mixing tank. This person has also direct communication with the person controlling the backfilling operation underground.

5.1.3 - Operation underground

The fill material in slurry form is emplaced into the mined-out area of the backfilling panel in stages starting from the lower end of the panel and progressing toward the entry as shown in Figure 12. Before backfilling, pervious barricades, made of timber posts, lagging boards, and burlap, are erected at the lower end of the backfilling panel to create a section sump. Barricades are also placed

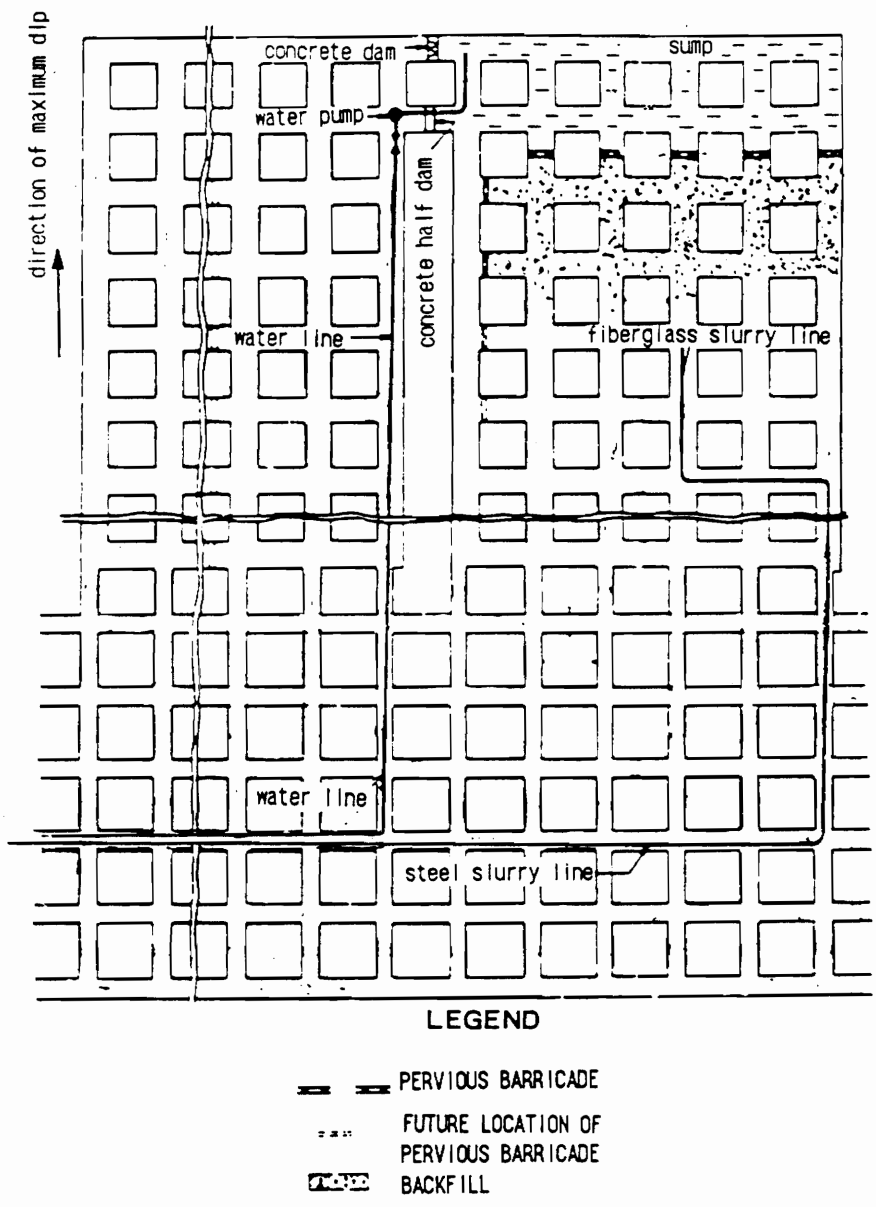

Figure 12: Hydraulic backfilling in progress

in stages along the sides, as shown in Figure 12. These pervious barricades retain the waste material but permit water to drain to the sump from where it is pumped out of the mine. The sump has the capacity to contain at least the amount of water introduced into the backfilling panel per day. Before starting backfilling, sump openings are connected from the adjacent panels planned for subsequent backfilling. The lowermost connection is provided with a full dam made of precast concrete blocks. The water pump for handling spent water of the backfill is always placed in a panel adjacent to the panel being filled as shown in Figure 12. This will allow ventilation of the pump and sump area.

As backfilling progresses, the slurry line in the panel is dismantled in sections and moved to an adjacent panel that is prepared for backfilling. Prior to backfilling, cribs and posts used for temporary support during retreat operations are removed.

Equipment and manpower required at this stage is as follows:

- One 700 GPM centrifugal-type water pump to pump out spent water from the backfilling panel, along with necessary valves
- Fifteen thousand feet of 5-inch diameter water line from the water pump in the backfilling panel to the water reservoir.

Three persons are required during a backfilling shift to erect barricades, dismantle and lay slurry lines, and control the backfilling operations. The water pump is activated by the rise in water level in the sump. Table XVIII summarizes the equipment and materials necessary to accomplish this operation.

5.2 - Cost estimation of hydraulic backfilling

All equipment, construction, operation, and maintenance cost are based on prices and costs are collected from different sources including the Canadian Institute of Mining and Metallurgy, 1982, A. A. Mathews, Inc., 1977, GEX, Colorado Inc., 1981, Bucek et al., 1979, U.S. BUMINES, IC-8689, 1975, and NUS Corporation, 1977. It was necessary to update the cost to the third quarter of 1989 so that a comparison can be made on the same base year between the cost of surface disposal and hydraulic disposal underground. The Marshall and Swift cost index (M&S), is used to update the equipment, operational, and maintenance costs to 1989 U.S. dollar values by using the following equation:

Cost now = (Cost then) x (Index now)/(Index then)

Table XVIII: Summary of Equipment and Materials Necessary for Hydraulic Backfilling

Refuse load-out bin
Preparation building
Hopper
5 x 14 Single deck vibrating screen
Impact crusher
Conveyor feeder belts (3)
Refuse storage silo
Feed belt from silo to mixing tank
Slurry mixing tank
Agitator
Centrifugal pump for density control
Centrifugal pumps for slurry transport (3)
5-inch steel-pipe (10,000 feet)
5-inch fiberglass pipe (5,000 feet)
Reclaim water storage reservoir
Centrifugal water pumps (2)
5-inch steel-pipe (15,000 feet)
Water monitoring wells (2)

The Marshall and Swift cost index (M&S) has several values. The all-industry equipment index is the average of the indices calculated for 47 different industries. Other M&S index values are calculated according to the type of industry, one of these being an M&S index for the mining and milling industry. This latter index is employed for updating the data. M&S indices for mining and milling are presented in Table XIX.

Power consumption was calculated from the motor horse power converted to kilowatts and multiplied by the time that the motor must be operated per year. All equipment is assumed to have a 10-year life unless otherwise stated.

Capital and operation cost of the hydraulic backfilling system are presented in Tables XX, XXI, XXII, XXIII, XXIV, and XXV. Table XX summarizes the capital investment for the hydraulic backfilling system. Table XXI presents a straight-line depreciation schedule for the capital costs of this hydraulic backfilling system for 0% and 15% interest rates. Table XXII presents the manpower requirements for hydraulic backfilling. Table XXIII presents the power consumption for the equipment necessary. Table XXIV presents the annual operating costs for the hydraulic backfilling system, while Table XXV summarizes the cost for hydraulic

Table XIX: Marshall & Swift Indices for Mining & Milling from 1975 - 1989 (third quarter). 1926 = 100

Year	Index
1975	451
1976	483
1977	521
1978	565
1979	619
1980	684
1981	740
1982	784
1983	799
1984	817
1985	823
1986	827
1987	837
1988	874
1989	918

(Source: Chemical Engineering, various years)

Table XX: Summary of Capital Investment for Hydraulic Backfilling System

Item	Quantity	Cost ($)
Refuse load-out bin	1	10,000
Preparation building	1	13,500
Hopper	1	9,000
5 x 14 Single deck vibrating screen	1	29,800
Impact crusher	1	37,000
Conveyor feeder belts	3	198,500
Refuse storage silo	1	157,600
Feed belt from silo to mixing tank	1	49,600
Slurry mixing tank	1	34,700
Agitator	1	22,200
Centrifugal pump for density control	1	1,700
Centrifugal pumps for slurry transport	3	47,900
5-inch steel-pipe (10,000 feet)	1	77,700
5-inch fiberglass pipe (5,000 feet)	1	47,900
Reclaim water storage reservoir	1	45,200
Centrifugal water pump	2	31,600
5-inch steel-pipe (15,000 feet)	1	58,000
Water monitoring wells	2	30,000
Total direct costs		901,900
Field indirect costs (2% of direct)		18,000
Total construction		919,000
Engineering (2% of construction)		18,400
Overhead and administration (5% of construction)		46,000
Sub-total		983,400
Contingency (15 % of sub-total)		147,500
Development cost		60,100
Total estimate of capital investment		1,191,000

Table XXI: Straight-line Depreciation Schedule for Capital Cost of Hydraulic Backfilling System at Interest Rates of 0 and 15 Percent

Item	Depreciation (years)	Annual Charge ($) (at 0%)	(at 15%)
Refuse load-out bin	20	500	1,600
Preparation building	20	675	2,160
Hopper	20	450	1,450
Single deck vibrating screen	10	2,980	5,950
Impact crusher	10	3,700	7,375
Conveyor feeder belts	10	19,850	39,460
Refuse storage silo	20	7,880	25,190
Feed belt	10	4,960	9,890
Slurry mixing tank	10	3,470	6,920
Agitator	5	4,440	6,620
Centrifugal pump (density control)	10	170	340
Centrifugal slurry pumps	10	4,925	9,550
5-inch steel-pipe (10,000 feet)	5	15,540	23,180
5-inch fiberglass pipe (5,000 feet)	5	9,580	14,290
Reclaim water storage reservoir	20	2,260	7,220
Centrifugal water pumps	10	3,160	6,300
5-inch steel-pipe (15,000 feet)	10	5,800	11,560
Water monitoring wells	20	1,500	4,800
Engineering, indirect cost, contingency, development, overhead	20	14,500	46,260
Total		106,340	230,115

Table XXII: Manpower Requirements for Hydraulic Backfilling

Description	Total Number	Total Annual Salary
Refuse preparation	1	$ 49,721
Slurry preparation	1	$ 49,721
Underground operators	3	$ 149,163
Total	5	$ 248,605

Table XXIII: Power Consumption of Backfilling Equipment

Item	HP	Hrs/day	Kwh/yr
5 x 14 Single deck vibrating screen	10	5	8,200
Impact crusher	100	5	82,000
Conveyor feeder belts	30	5	24,600
Feed belt from silo to mixing tank	10	5	8,200
Agitator	7.5	5	6,150
Centrifugal pump for density control	5	5	4,100
Centrifugal pumps for slurry transport	900	5	738,240
Centrifugal water pumps	520	5	426,540
Water monitoring wells	100	5	82,000
Total	1,682.5		1,383,030

Table XXIV: Annual Operating Costs for Hydraulic Backfilling System

Item	Annual costs
Labor	$ 248,605
Overhead (40% of labor)	$ 99,442
Supplies, Materials, and Maintenance (92.4% of labor)	$ 229,711
Power	$ 169,323
Insurance (2.5% of equipment cost)	$ 22,600
Total	$ 769,681

Table XXV: Summary of Costs for Hydraulic Backfilling at Interest Rates of 0 and 15 percent

<u>Interest rate 0 percent:</u>	
Annual capital costs:	$ 106,340
Annual operating costs:	$ 769,681
Total annual costs:	$ 876,021
Disposal costs per ton of waste:	$ 6.74
<u>Interest rate 15 percent:</u>	
Annual capital costs:	$ 230,115
Annual operating costs:	$ 769,681
Total annual costs:	$ 999,796
Disposal costs per ton of waste:	$ 7.69

Note: Annual tonnage 130,000

backfilling at interest rates of 0 and 15 percent.

The capital costs include costs for construction, engineering, overhead and administration, and contingency. It is assumed that construction cost are two percent of the equipment costs and that costs of engineering are two percent of the equipment and construction cost. Overhead and administration cost are five percent of the equipment and construction cost, while contingency costs are fifteen percent of the cost of equipment, construction, engineering, and overhead and administration.

The operation costs include costs for labor, overhead, supplies, materials, maintenance, power, and insurance. It is assumed that the costs for supplies, maintenance, materials are 0.924 times labor cost (NUS Corporation, 1977). Power costs were calculated based on the power consumption of the equipment installed and energy consumed. Overhead costs are assumed to be 40 percent of the costs for direct labor. Depreciation cost are based on a straight-line depreciation of the equipment. Table XXVI presents results of an sensitivity analysis where the capital and operating costs involved in the hydraulic backfilling operation are varied between +30% and -30%. As can be seen from this table the cost of hydraulic backfilling is more sensitive to the operating costs. Table XXVII summarizes the disposal costs

Table XXVI: Sensitivity Analysis of Hydraulic Backfilling with Interest Rates of 0 and 15 Percent

Variable	Change in Cost (%)	Cost per ton of refuse ($) (0%)	Cost per ton of refuse ($) (15%)	Change in Cost per ton of refuse (%) (0%)	Change in Cost per ton of refuse (%) (15%)
Operating Cost	+30	8.52	9.47	26	23
	+20	7.92	8.88	18	15
	+10	7.33	8.28	9	8
	0	6.74	7.69	0	0
	-10	6.15	7.10	-9	-8
	-20	5.56	6.51	-18	-15
	-30	4.96	5.92	-26	-23
Capital Cost	+30	6.98	8.22	4	7
	+20	6.90	8.04	2	5
	+10	6.82	7.87	1	2
	0	6.74	7.69	0	0
	-10	6.66	7.51	-1	-2
	-20	6.58	7.34	-2	-5
	-30	6.49	7.16	-4	-7

Table XXVII: Summary of Disposal Cost Per Ton of Refuse at Interest Rates of 0 and 15 Percent

Method	Cost Per ton of Refuse	
	(0%)	(15%)
Surface disposal	$ 2.08	$ 2.66
Hydraulic backfilling	$ 6.74	$ 7.69

for coal refuse placed on the surface and hydraulically underground. According to the calculations the costs of disposing the coal waste hydraulically underground is more expensive than surface disposal on a per ton basis.

VI - ECONOMIC FEASIBILITY OF MECHANICAL BACKFILLING

6.1 - Design concept of backfilling system

Bucek et al. (1979) described a mechanical backfilling system that can be utilized in the described coal mine. This system involves the mechanical transport of coal refuse to a service shaft, the introduction of the refuse by gravity feed to the mine level, and the mechanical transport and stowage of the coal refuse at the face. A schematic view of this system is presented in Figure 13. It is assumed that a service shaft is available that will facilitate the use of a conveyor system to transport the fill material into the mine.

Six hundred tons of refuse will be backfilled per day. This can be done in one shift at a rate of 120 ton per hour. When backfilling is in progress five panels will be worked: four panels for coal production and one for backfilling.

All backfilling will be performed in the partially extracted panels that may be laid out in any one area of the mine, but the area should be close to the service shaft in order to reduce the distance of in-mine refuse transport. Panels will be laid out adjacent to each other and on either side of sub-mains in this area. The extend of this area is

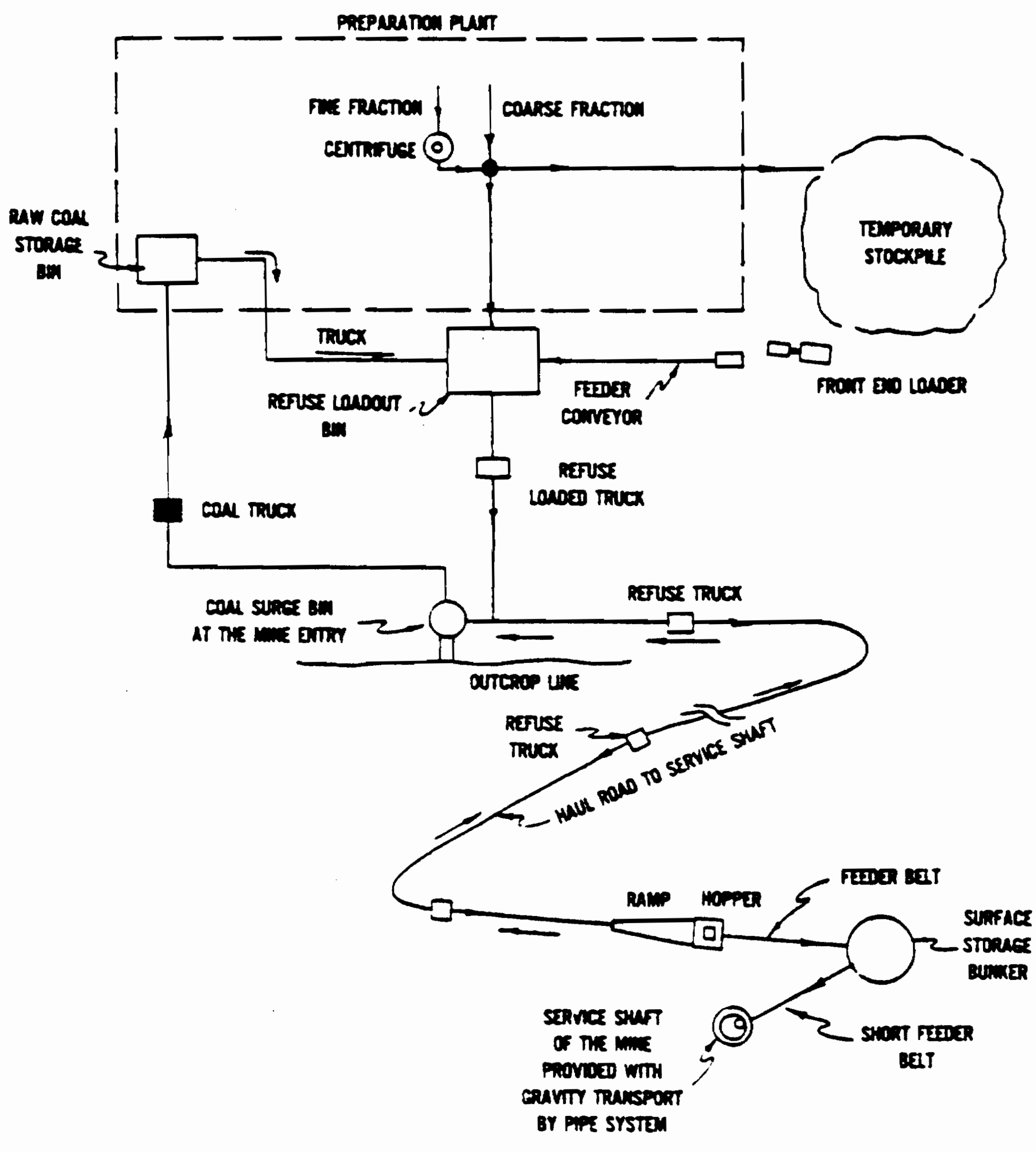

Figure 13: Schematic view of mechanical backfilling system

(Source: Bucek et al., 1979)

determined from an estimate of the number of backfilling panels required for the total amount of refuse likely to be disposed of over the life of the mine.

For the mine, the maximum possible distance of backfilling panel from the service shaft bottom is not greater than 4,000 feet along the sub-main. The backfilling will be conducted in one panel at the time and will be performed shortly after the pillars left during development are partially mined in the retreat operation.

Simultaneous extraction of coal and backfilling in the same panel will not be practiced. This will minimize capital expenditure and labor costs, since the rate of coal extraction would otherwise have to be reduced during the retreat operation to keep pace with the low volume of backfilling. Furthermore, to compensate for this reduced production, an extra section would have to be worked. Sub-mains are assumed to be at level grade whereas the backfilling panels are driven both up dip and down dip off the sub-mains.

Each backfilling panel, measuring 2,000 feet by 480 feet center to center of the barrier pillar, is first developed in the same way as in the other working sections of the mine. However, unlike in other sections, the width of the barrier pillars between the backfilling panels are not maintained, but reduced to 50 feet during retreat mining. A panel is worked with 60 by 60-foot square pillars and 20-foot wide entries and crosscuts; the barrier pillars are initially 60 feet wide. During retreat prior to backfilling, pillars are reduced from 60 by 60 foot to 50 by 50 foot by taking a 10-foot wide slice off each of two adjacent sides as shown in Figure 14. The room span after the reduction of pillars is 30 feet, which is acceptable when the roof is adequately supported by two different methods of artificial support. Roof bolts and timber posts with cribs are assumed to be used presently in depillaring operations in other sections of the mine. Thus, no extra cost is incurred in providing support in the backfilling panel (Bucek et al., 1979)

For the panels designated for backfilling, a refuse conveyor is installed in the fourth entry and a coal conveyor is installed in the third entry. During the retreat operation, as the coal conveyor is shortened, ventilation stoppings will be re-erected to make the fourth entry a neutral entry for the refuse belt. The refuse belt is installed during retreat by laying the conveyor structures and belt pieces in the fourth entry of the next panel to be backfilled as they are obtained by shortening the refuse belt in the panel where backfilling is in progress. This will prevent the

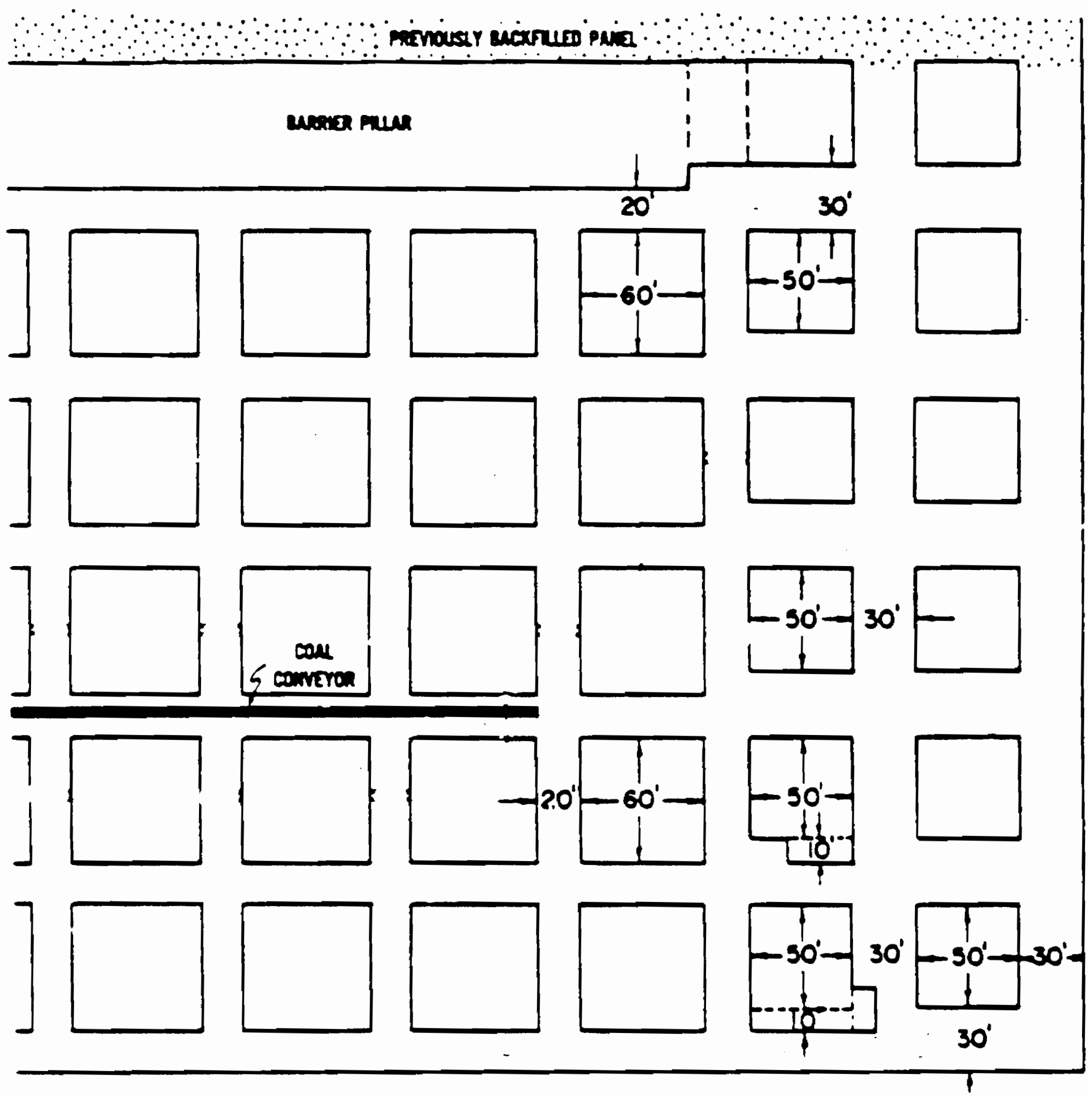

Figure 14: Retreat mining of the developed panel

(Source: Bucek et al., 1979)

temporary stoppage of the backfilling operation during the changeover from one panel to the other. Only the drive head and tail end sections of the refuse belt will have to be moved to the new panel. This can be accomplished during off-shift periods.

The mine has a service shaft for ventilation and for power feeder cables or pumping mains about 2.5 miles from its portal and that the elevation of the top of the shaft is about 700 feet higher than the seam elevation. There is an existing road to the service shaft; however, this road requires improvement for transporting the coal refuse. The calculations for the backfilling operation are sufficiently conservative to compensate for the temporary stoppage of backfilling during minor breakdowns. Some of the design calculations are shown in Appendix C.

6.1.1 - Surface transport

The coal refuse is dewatered and transported by truck to the refuse storage bin at the service shaft. This will be done by the same trucks that transport the coal from the mine to the preparation plant. The surface storage bin is loaded during the production shifts, but unloaded only during the backfilling shift.

Equipment and manpower required at this stage are as follows:

- Centrifuge for dewatering purposes
- Refuse load-out bin with chute and hydraulically operated gate for truck loading
- Upgrading of road from mine entry to surface shaft
- Extra maintenance of road section from mine entry to surface shaft

No extra manpower is required for these operations.

6.1.2 - Transport from surface to underground

The pipeline will be supported at regular vertical intervals in the service shaft by clamping the pipeline to transverse steel I-beams which are assumed to be existing for supporting pumping mains and electric power cables. The discharge end of the pipeline leads into a velocity damping chamber that is designed to absorb the impact energy of the falling refuse. In order to avoid the formation of air plugs and to provide convenient installation and servicing, the pipeline is sectionalized with the a funnel at the top of each section. Refuse material from the surface storage bunker is fed to the top funnel of the pipeline by a short feeder belt at the rate of 120 tons per hour during the backfilling operation. The material drops through the pipeline into the velocity damping chamber and is continuously removed by an apron feeder and fed to the underground refuse transport belt.

Equipment and manpower required at this stage are as follows:

- Hopper with ramp
- One 30-inch by 150-foot feed belt conveyor from the hopper to the refuse storage bin
- One refuse bin with a 900-ton capacity
- One 24-inch by 50-foot feeder belt conveyor from the silo to the funnel of the pipe conveying system
- 7 sections of 24-inch diameter by 100 foot, 1/4 inch thick steel pipe
- One velocity damping chamber
- One apron feeder with a capacity of 300 tons per hour

One man is required at this stage to control the feed to the chute.

6.1.3 - Underground transport

The in-mine transport of the backfill material projected for the mine utilizes a system of belt conveyors. The main refuse conveyor system between the apron feeder and the refuse conveyor in the backfilling panel consists of two 2,000-foot long belt conveyors supported by wire rope suspended from the roof. The backfilling panel belt is an extensible belt system capable of being advanced or retracted for 200 feet without requiring the addition or removal of belting. All refuse belt conveyors, the apron feeder, and the discharge gate below the velocity damping chamber can be controlled remotely from a single location and can be started and stopped sequentially by incorporating a sequence control relay mechanism into the electrical the circuit breakers of the system. The electrical circuits of the conveyor system is designed to stop all of the conveyors in sequence in the event of abnormal conditions or system malfunctions.

Equipment and manpower required at this stage are as follows:

- Two refuse transport belts along the sub-main size 24-inch x 2,000-foot
- Conveyor within the panel, an extensible belt system capable of being advanced or retracted by 200 feet without stopping to add or remove belting

One operator is required to operate the belt system and to load the scoop cars.

6.1.4 - Backfilling process

Refuse is backfilled by battery-powered scooped cars as backfilling will progress from the inby end limit of the panel until the whole panel is completely filled as shown in Figure 15. As the backfilling progresses, the extensible panel belt is retracted. Belt pieces and conveyor support structures obtained from retracting the belt are then transferred to the next panel being prepared for backfilling. This will avoid interrupting backfilling operations during the changeover from one panel to another since the final remaining portion of the extensible conveyor can be transferred to the new panel during the off-shift period. It is important that a new panel be ready for backfilling before backfilling in one panel is completed. This should not be difficult to achieve, however, since at least four extraction panels will be worked at any time, two of which can be prepared for backfilling. Since coal extraction and backfilling are carried out independently, no production loss is anticipated in backfilling.

Equipment and manpower required at this stage are as follows:

- Two scoop cars with a capacity of 5 tons
- Two sets of batteries for each scoop car

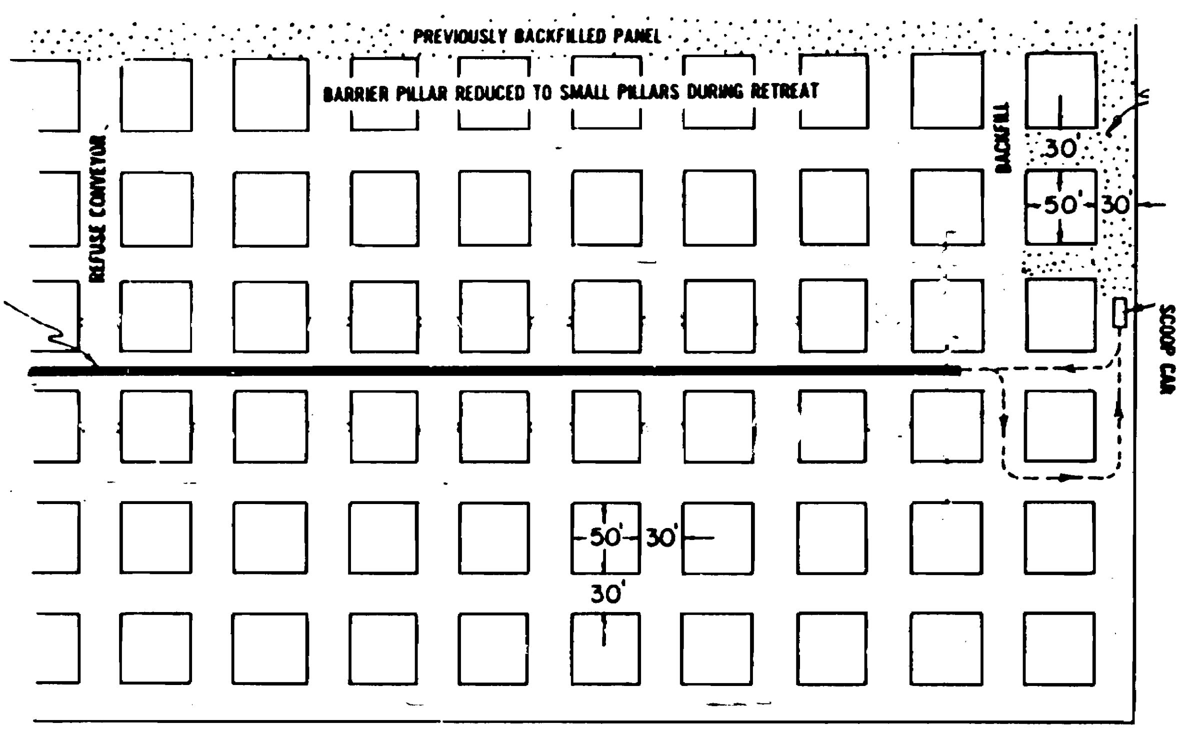

Figure 15: Mechanical backfilling in progress

(Source: Bucek et al., 1979)

Manpower required at this stage is four, two to operate the scoop cars and two to extend brattice for ventilation control and to remove temporary supports.

Table XXVIII provides a summary of the equipment and materials necessary for this mechanical backfilling operation.

6.2 - Cost estimation of mechanical backfilling

All equipment, construction operation, and maintenance cost are updated to the costs of the third quarter of 1989. The Marshall and Swift cost index (M&S), is used to update the equipment, operational and maintenance cost.

Capital and operation cost of mechanical backfilling system are presented in Table XXIX, XXX, XXXI, XXXII, XXIII, and XXXIV. Table XXIX summarizes the capital investment for the mechanical backfilling system. Table XXX presents a straight-line depreciation schedule for the capital costs of the mechanical backfilling system. Table XXXI presents the manpower requirements for the mechanical backfilling system. Table XXXII presents the power consumption of mechanical backfilling equipment. Table XXXIII presents the annual operating costs for the mechanical backfilling system, while Table XXXIV summarizes the cost for mechanical backfilling.

Table XXVIII: Summary of Equipment and Materials Necessary for Mechanical Backfilling

Refuse load-out bin
Road upgrading
Hopper
30" x 150' conveyor belt
Refuse storage bin
24" x 50' conveyor belt
24" x 150' conveyor belt
(7) pipes 100'x 24" x 1/4"
Velocity damping chamber
Apron feeder
(2) 24" x 2,000' conveyor belts
Extensible conveyor belt
(2) scoop cars
(2) batteries
(2) water monitoring wells

Table XXIX: Summary of Capital Investment for Mechanical Backfilling System

Item	Quantity	Cost($)
Refuse load-out bin	1	10,000
Road upgrading		266,000
Hopper	1	9,000
30" x 150' conveyor belt	1	52,200
Refuse storage bin	1	157,600
24" x 50' conveyor belt	1	15,100
24" x 150' conveyor belt	1	45,300
Pipes 100' x 24" x 1/4"	7	34,200
Velocity damping chamber	1	1,600
Apron feeder	1	3,600
24" x 2,000' conveyor belt	2	1,207,500
Extensible conveyor belt system	1	236,400
Scoop cars	2	158,000
Batteries	2	62,700
Water monitoring wells	2	15,000
Total direct costs		2,274,200
Field indirect costs (2% of direct)		45,500
Total construction		2,319,700
Engineering (2% of construction)		46,400
Overhead and administration (5% of construction)		116,000
Sub-total		2,482,100
Contingency (15% of sub-total)		372,300
Development cost		60,100
Total estimate of capital investment		2,914,500

Table XXX: Straight-line Depreciation Schedule for Capital Cost of Mechanical Backfilling System at Interest Rates of 0 and 15 Percent

Item	Depreciation (years)	Annual Charge ($) (at 0%)	(at 15%)
Refuse load-out bin	10	1,000	2,000
Road upgrading	10	26,600	53,010
Hopper	10	900	1,800
30" x 150' conveyor belt	10	5,220	10,400
Refuse storage bin	10	15,760	31,410
24" x 50' conveyor belt	10	5,100	10,170
24" x 150' conveyor belt	10	4,530	9,030
Pipes 100' x 24" x 1/4"	5	6,840	10,200
Velocity damping chamber	5	320	480
Apron feeder	5	720	1,080
24" x 2,000' conveyor belt	10	120,750	240,660
Extensible conveyor belt system	10	23,640	47,120
Scoop cars	10	15,800	31,490
Batteries	10	6,270	12,490
Water monitoring wells	20	750	2,400
Total		234,200	463,740

Table XXXI: Manpower Requirements for Mechanical Backfilling

Description	Total Number	Annual Salary
Scoop car operators	2	$ 99,442
Brattice men	2	$ 99,442
Belt operators	2	$ 99,442
Total	6	$ 298,326

Table XXXII: Power Consumption of Mechanical Backfilling Equipment

Item	HP	Hr/day	Kwh/yr
24" x 150' conveyor	15	3	7,376
30" x 150' conveyor	45	10	73,755
24" x 50' conveyor	5	5	4,098
Apron feeder	15	5	12,293
24" x 2,000' conveyor	35	5	28,683
Extensible conveyor system	55	5	45,073
Total	170		171,278

Table XXXIII: Annual Operating Costs for Mechanical Backfilling System

Item	Annual cost ($)
Labor	298,326
Overhead (40% of labor)	119,330
Supplies, Materials, and Maintenance (92.4% of labor)	275,653
Power	17,586
Insurance (2.5% of equipment cost)	56,855
Total	767,750

Table XXXIV: Summary of Costs for Mechanical Backfilling at Interest Rates of 0 and 15 Percent

Interest rate 0 percent:	
Annual capital costs:	$ 234,200
Annual operating costs:	$ 767,750
Total annual costs:	$ 1,001,950
Disposal costs per ton of waste:	$ 7.71
Interest rate 15 percent:	
Annual capital costs:	$ 463,740
Annual operating costs:	$ 767,750
Total annual costs:	$ 1,231,490
Disposal costs per ton of waste:	$ 9.48

Note: Annual tonnage 130,000

The capital cost include costs for construction, engineering, overhead and administration, and contingency. It is assumed that construction cost are two percent of the equipment cost and that costs of engineering are two of the equipment and construction costs. Overhead and administration costs are five percent of the equipment and construction cost, while contingency costs are fifteen percent of equipment, construction engineering, and overhead and administration.

The operation cost include costs for labor, overhead, supplies, materials, maintenance, power, and insurance. It is assumed that the costs for supplies, maintenance, materials are 0.924 times costs (NUS Corporation, 1977). Power costs were calculated based on power consumption of the equipment installed and energy consumed. Overhead costs are assumed to be 40 percent of the costs for direct labor. Depreciation cost are based on a straight-line depreciation of the equipment.

Table XXXV summarizes the costs of coal refuse disposed on the surface, placed hydraulically and mechanically underground. The cost of placing material mechanically underground is more expensive than hydraulic backfilling which in turn is more expensive than surface disposal.

Table XXXV: Comparison of Disposal Costs of Coal Refuse Using Different Methods at Interest Rates of 0 and 15 Percent

Method	Cost per ton of Refuse (0%)	(15%)
Surface Disposal	$ 2.08	$ 2.66
Hydraulic backfilling	$ 6.74	$ 7.69
Mechanical backfilling	$ 7.71	$ 9.48

VII - CONCLUSIONS AND RECOMMENDATIONS

In southwest Virginia the predominant underground coal mining methods are room-and-pillar mining and longwall mining. The literature survey indicates that backfilling techniques are technically feasible for both mining methods.

Hydraulic and mechanical backfilling are the most promising techniques for the majority of southwest Virginia's underground coal mines. The most suitable technique for a particular mine however, will depend on the weights and ratings assigned to the attributes affecting the selection of a particular backfilling method. Those weights, as well as ratings, can differ for each individual mining operation.

In order to estimate the costs of backfilling, an evaluation has been made for both hydraulic and mechanical backfilling methods. This study has been performed for an room-and-pillar mine, that utilizes coal refuse as fill material. The study indicates that mechanical backfilling is more capital intensive than hydraulic backfilling. However, the operating costs are about the same for both methods. Both methods of underground disposal are more expensive than surface disposal with mechanical disposing being the most expensive method.

The cost estimates, however, can only be considered as preliminary, and may be subject to an increase or decrease of up to 30%, since they are obtained by extrapolation from historical data. Furthermore, in estimating the costs of hydraulic and mechanical backfilling, a number of factors have been kept optimal for the mine; therefore, the costs estimated can be considered as valid estimates under the most suitable conditions, where no losses in production or declines in productivity are taken into account. On the other hand benefits associated with backfilling are also not taken into consideration, because they might differ significantly between mine operators, and there are no reliable data available for an accurate estimate.

Coal refuse has been assumed as fill material because surface disposal of coal refuse is an additional cost item for the mine operator, causing increasing environmental concerns. The cost of coal refuse disposal on the surface varies depending on the disposal method and disposal site.

Backfilling with coal refuse will represent an additional cost for the mine operators, who are using existing surface disposal sites. At present the surface disposal costs of some industrial waste has been reported more expensive than the underground disposal of coal refuse. In some mining operations the use of this waste as backfilling material may provide an economic incentive for backfilling. However, an extensive study on the properties and qualities of the industrial waste together with an evaluation of their surface disposal costs should be made in order to determine the materials that can be used as fill in the underground coal mines.

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APPENDIX A. Detailed breakdown of labor cost

This detailed breakdown is made by using the United Mine Workers contract of 1988. The laborers involved in this operation are classified in grade 1 in the contract as truck drivers, preparation plant operators, and equipment operators. In order to estimate the annual cost of such a worker the following assumptions were necessary. No overtime work or service on Saturday is required for this operation. The operators have an average experience of 5 years and work in 2 shifts for 220 days per year. The coal production is 15 ton/man/shift, and insurance is paid to an independent insurance carrier at a rate of $350 month/man.

Base wage = 15.957 $/hr

Shift differential[1] = .150 $/hr

Hourly wages = 16.107 $/hr

Base wage per day = 7.25 hr/day @ 16.107 $/hr = 117 $/day

Annual wages:

Normal days:	220 day/yr @ 117 $/day =	$ 25,740
Holidays[2] :	33 day/yr @ 117 $/day =	$ 3,861
Total annual wage	=	$ 29,601

<u>Benefits and pension trust:</u>

Benefit trust:

2.52 $/hr @ 7.25 hr/day @ 220 day/yr = $ 4,019

Pension trust:

.97 $/ton @ 15 ton/day @ 220 day/yr = $ 3,201

Total annual benefits and pension trust = $ 7,220

<u>Taxes:</u>

Compensation 20 % @ $ 29,601	= $ 5,920
Unemployment 6 % @ $ 6,000	= $ 360
Social security 7.57% @ $ 29,601	= $ 2,240
Total annual taxes	= $ 8,520

<u>Annual cost per employee:</u>

Annual wages of employee		= $29,601
Expenses covered by employer:		
Taxes	= $ 8,520	
Benefits and pension trust	= $ 7,220	
Clothing	= $ 180	
Insurance (350 $/month @ 12 month/yr)	= $ 4,200	
Total		= $49,721

[1] Average for the day and afternoon shifts

[2] Breakdown of holidays

Regular vacation: 12 day/yr

Work on birthday: 2 day/yr

Personal and sick leave: 5 day/yr

Floating vacation: 4 day/yr

Holidays: 10 day/yr

Total: 33 day/yr

APPENDIX B. Design calculations for hydraulic backfilling operation

Assumptions:

Bank weight of coal: 1,900 lbs/yd^3

Swelled weight of coal waste: 1,900 lbs/yd^3

Bank weight coal refuse: 2,900 lbs/yd^3

Determination of volume needed for backfilling:

Run of mine coal production: 520,000 tons

Coal waste generated: 130,000 tons

Volume coal waste:

(130,000 tons @ 2,000 lbs/ton)/1,900 lbs/yd^3 = 136,842 yd^3

Volume gob:

(520,000 tons @ 2,000 lbs/ton)/1,900 lbs/yd^3 = 547,368 yd^3

Percentage of gob needed for backfilling purposes:

136,842 yd^3 / 547,368 yd^3 @ 100 % = 25 %

Determination of slurry volume:

Hydraulic backfilling will be conducted in one shift only. The effective stowing time is 5 hours. Thus the stowing rate is: 130,000 tons/yr/(220 days/yr * 5 hrs/day) = 120 tph

The slurry will be pumped at a concentration rate 60 percent by weight of solid. Therefore, for each 60 tons of solids 40 tons of water is required. The total weight of the slurry is 200 tph.

120 tons solids per hour is equal to:

240,000 (lbs/hr) ÷ 2,900 (lbs/yd^3) = 83yd^3/hr

80 tons water per hour = 94yd^3/hr

Total volume per hour = 177yd^3/hr

Therefore 600 GPM slurry needs to be pumped.

Determination of pump size for slurry transport:

Assumptions:

The slurry is transported over a distance that is equal to 15,000 feet of straight pipeline. The discharge pressure for the slurry pump from mine entry to backfilling panel must be sufficient to overcome friction losses only, because it is assumed that the upgrade portion of the slurry line is in balance with the downgrade portion of the line. The maximum pressure for a centrifugal pump installation is limited to 600 psi.

The minimum operating velocity for this coal slurry is assumed to be 10 feet per second to avoid a depositional flow of the slurry. Therefore the inside pipe diameter must be 5 inches. The estimated friction loss in the pipe is assumed to be 717 Pa/ft.

Therefore, 15,000 (ft) @ 717 Pa (Pa/ft)= 10,775,000 Pa = 1560 psi will be required to overcome the friction losses. Since one slurry pump can create only 600 psi, three slurry pumps in series will be sufficient to handle this slurry. Assuming a pump efficiency of 80 percent a centrifugal pump with a capacity of (600)+(.80) = 750 GPM ≈ 800 GPM will be required.

Assuming the mechanical efficiency of the pump to be 65 percent the required horsepower required will be:

1,560 (psi) × 600 (GPM)+(1,716 × 0.65) = 842; use: 900 HP.

<u>Determination of spent water transport:</u>

Total volume of water pumped into the mine with the slurry per day is 400 tons @ 264.2 gal/ton = 105,680 gallons. It is assumed that all the water will be returned.

Amount of water introduced into the mine for flushing the slurry line is assumed to be 700 GPM for 20 minutes each time with a maximum of four times per shift = 56,000 gallons. Therefore the total volume of water to be pumped out is: 105,680 + 56,000 = 161,680 gallons.

Assuming pumping time to be 5 hours:

volume to be pumped: 161,680+(5 × 60) = 538.9 GPM.

Assuming a pump efficiency of 80 percent a pump with a capacity of 700 GPM will be required.

<u>Time study analysis of trucks:</u>

Operation	Time (minutes)
1. loading refuse at preparation plant	1.5
2. Hauling refuse from preparation plant to mine portal	7.3
3. Unloading refuse	1.3
4. Move to coal silo at mine portal	3.3
5. Spotting and loading of coal	1.5
6. Hauling coal to preparation plant	7.6
7. Turning, spotting and unloading at coal storage bin at prep. plant	1.3
Total cycle time	23.8

Assuming 50 minutes of production per hour for the truck, the average number of trips per hour per truck:

50 (min/hr) + 23.8 (min/trip) = 2.1 (trip/hr)

Therefore, total number of trips per truck per shift, assuming 6 hours of effective time per shift:

2.1 (trip/hr) @ 6 (hr/shift) = 12.6 (trip/shift)

The production of coal per shift = 1,200 tons

Payload for existing trucks = 42 tons

<u>Amount of coal hauled per truck per shift:</u>

42 (ton/trip/truck) @ 12.6 (trip/shift) = 529

(ton/shift/truck)

Number of trucks required per shift for 1,200 tons of coal:

1,200 (ton/shift) ÷ 529 (ton/shift/truck) = 2.27 trucks; therefore, assume three trucks.

The calculation of three trucks takes into account the hauling of refuse from the preparation plant to the refuse storage bunker by each of the three trucks in all the trips in a shift. Thus, three trucks can theoretically haul 1,600 tons of coal from the mine to the preparation plant and haul 1,600 tons of coal refuse back on their return journey to the refuse storage bunker per shift.

The projected backfilling system requires hauling of only 300 tons of coal refuse per shift, therefore, the requirement of three trucks for coal hauling has adequate spare capacity for additional coal refuse hauling in any shift.

Determination of conveyor belt:

From tables of a conveyor belt manufacturer, it is found that a 30-inch belt can easily handle 156 tons of coal waste per hour with a belt speed of 100 fpm.

Horsepower requirement:

It is assumed that the conveyor belt is 250 feet long, the average load on the conveyor is 100 tph and the belt speed is 100 fpm and the height the load is raised is 40 feet.

Horsepower required to overcome friction and lift the load will be:

Friction Horsepower = (0.085W + 3.92T+S)(L+1,000)(S+100)

where:

W = Width of belt in inches

T = Average load in tph

S = Speed of belt in fpm

L = Ultimate length of conveyor in feet

Gravity Horsepower = (T x H)+840

where:

T = Average load in tph

H = Height in feet that the load is raised

By applying these two formulas the horsepower required is:

Friction Horsepower:

(0.085 x 30 + 3.92 x 100+100)(250+1000)(100+100) = 1.62 HP

Gravity Horsepower:

(100 x 40)+840 = 4.76 HP

Total Horsepower required = 6.38 HP

Assuming a drive efficiency of 95 percent:

HP = 6.38 + 0.95 = 6.71

Allowing at least 25 percent for starting ; use: 10 HP

APPENDIX C. Design calculations for mechanical backfilling operation

Time study analysis of trucks that haul coal refuse and run of mine coal:

Operation	Time(minutes)
1. Loading refuse at prep. plant	1.5
2. Hauling refuse from prep. plant to silo	22.8
3. Unloading refuse	1.3
4. Return time from silo to mine entry	7.3
5. Spotting and loading of coal	1.5
6. Hauling coal to prep. plant	7.6
7. Turning, spotting and unloading at coal storage bin at prep. plant	1.3
Total cycle time	43.3

<u>Time study analysis of trucks that haul run of mine coal:</u>

Operation	Time(minutes)
1. Spotting and loading of coal	1.5
2. Hauling coal to prep. plant	7.6
3. Turning, spotting and unloading at coal storage bin at prep. plant	1.3
4. Returning empty to mine portal	<u>5.0</u>
Total cycle time	15.4

To transport 600 tons of coal refuse from preparation plant to silo 6 trips will be required per shift. In those 6 trips 252 tons of run of mine coal will also be transported to the preparation plant. Since 900 tons of run of mine coal is produced per shift an additional 16 trips of transport of solely run of mine coal to the preparation plant are required. The time required for those operations is:

coal and coal refuse transport:

6 (trips) x 43.3 (min/trip) = 259.8 min

coal transport:

16 (trips) x 15.4 (min/trip) = 246.4 min

Total time for operation: 506.2 min

Assuming 50 minutes of production per hour and 6 hours of effective time per truck the time a truck is available per shift is 300 minutes.

Therefore, 506.2(min) + 300(min/truck) = 1.7 trucks.

Therefore the three trucks for this operation can transport the coal refuse to the silo without requiring additional trucks.

<u>Time study analysis of scoop car:</u>

Operation	Time(seconds)
1. Positioning scoop at conveyor	10
2. Loading time for scoop	60
3. Travel time (loaded)	80
4. Positioning scoop at backfill face	10
5. Dumping time	20
6. Travel time (empty)	<u>60</u>
Total cycle time	240

Scoop car requirement:

Hourly trips per scoop car = 15

Backfilling rate per hour per scoop car = 75 tons

Backfilling rate 120 tons per hour

Therefore 1.6 scoop car is required, therefore use 2.

VITA

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