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HIGH ANGLE MINING SYSTEM

Contract J0-133925
ESD Corporation

BUREAU OF MINES
UNITED STATES DEPARTMENT OF THE INTERIOR

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Under Bureau of Mines Contract J0-133925, ESD Corporation developed the High Angle Mining System (HAMS) as a method of extracting coal from seams varying widely in thickness and pitch. The system would normally be employed to extract coal lying beyond the economic limit of a strip or open-pit highwall. It is a surface-controlled operation involving no underground personnel. The HAMS can significantly increase the reserve base of coal that can be economically surface mind.

A number of open-pit or surface mines which have moderately or steeply pitched seams were visited and studied. Existing technology was reviewed to identify systems or system components which merited further consideration for the development of the HAMS concept. Conventional and innovative approaches based on augering and two approaches based on slackline excavation were developed in sufficient detail to enable comparison on the basis of operational limitation, recovery ratio, productivity, and economy. The slackline excavator concept employing a projectile bucket was selected as the best approach. The capability of economically mining pitched seams of coal with the HAMS was demonstrated.

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FOREWARD

This report was prepared by ESD Corporation, 600 Meridian Avenue, San Jose, California, under Bureau of Mines Contract Number J0-133925. The contract was initiated under the technical direction of Bruceton Research Center, with Richard Farrar acting as Technical Project Officer. Oliver Snyder was the contract administrator for the Bureau of Mines. This report is a summary of the work recently completed as a part of the contract during the period September 1980 to June 1983. This report was submitted by the author in June 1983.

The author wishes to acknowledge the assistance of John Corra and Al Rangitsch, FMC Corporation, Skull Point Mine, and Fred Hahn, ESCO Corporation, in the performance of this contract.

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EXECUTIVE SUMMARY

In many areas where pitched coal seams are being mined by surface methods, a large reserve of coal lies below the economical mining depth. A method of mining a portion of this large coal reserve before the pit is backfilled and landscaped has potential for significant cost benefit.

Under Bureau of Mines Contract J0-133925, ESD Corporation developed the High Angle Mining System (HAMS) as a method of extracting coal from seams varying widely in thickness and pitch. The system would normally be employed to extract coal lying beyond the economic limit of a strip or open-pit highwall. It is a surface-controlled operation involving no underground personnel. The High Angle Mining System can significantly increase the reserve base of coal that can be economically surface mined. It employs few moving parts, is reasonably economical to fabricate, is simple to operate, and has excellent potential for safe, economical coal mining.

The program's direction was to make a thorough study of the recovery problem, review existing technology, consider alternatives, and develop a prototype of the concept considered to have the best potential for coal recovery.

A number of open-pit or surface mines which have moderately or steeply pitched seams were visited and studied. Existing technology was reviewed to identify systems or components of systems which merited further consideration for the development of the HAMS concept.

Conventional approaches and innovations based on augering and two approaches based on slackline excavation were developed in sufficient detail to enable comparison on the bases of operational limitations, recovery ratio, productivity, and economy. The slackline excavator concept employing a projectile bucket was selected as the best approach. This approach also gained the concurrence of the demonstration mine that agreed to cost-share its development. This demonstration mine was the FMC Skull Point Mine near Kemmerer, Wyoming.

The system consists of a slackline excavator system with a modified dragline projectile bucket suspended from a track cable on low-friction sheaves. The track cable is anchored to the bottom of a small pilot hole drilled down into the coal seam near and parallel to the roof. It is anchored by inserting PVC plastic pipe alongside the cable, pumping in cement grout, and removing the pipe. Guiding the bucket by means of the track cable ensures mining in a predetermined direction with good control of the bucket behavior. The bucket is controlled from the surface to excavate a vertical slot of a width approximately equal to the bucket and a height approaching the seam thickness. The face of the excavation is advanced by the gravity-powered projectile impact of the bucket into the coal. The projectile face of the bucket contains an array of sharply pointed conical teeth for this purpose. The floor of the excavation is ripped during the hoist cycle by ripping shanks attached to the underside of the bucket. Ripped coal is strewn behind the bucket while the loose coal ripped earlier is scooped up through the front end of the bucket.

A prototype High Angle Mining System was designed, fabricated, and tested. It was demonstrated that, using readily available equipment, a guidance system can be constructed to provide a means for coal recovery beyond the economical stripping limit. Holes were drilled parallel to the top of the seam to depths of up to 365 feet. Steel cables were inserted into the holes and grouted in place to provide anchor strengths equal to or greater than the cable strength. The cable was used as a trolley system to suspend and guide the projectile bucket. After being properly adjusted, the bucket impact fragmented enough coal to advance the bucket at economically acceptable rates. The ripping and loading action fully loaded the bucket as it was retrieved.

As a result of work performed under this contract, it can properly be concluded that the High Angle Mining System, as conceived and tested by ESD Corporation, is a feasible method for the mining of coal from seams varying widely in thickness and pitch. The system should be employed to extract coal lying beyond the present economic limit of a strip mine or open-pit highwall, thus allowing the capture of large coal reserves which are otherwise not economical or practical to surface mine.

It is further concluded that, while the projectile bucket mines coal at a reasonable rate, the incorporation into the system of a motorized bucket head will increase production rates and make production more predictable. Guiding the motorized bucket by the preinstalled track cable will give positive control of mining direction and will ensure the desired spacing between excavations.

ESD recommends that, as funding becomes available, a full-scale High Angle Mining System be fabricated and incorporated into the normal production of a mine for a 1-year evaluation period. Such a program should include both the motorized bucket for driving the initial excavation to the bottom of the track cable and the dragline bucket to remove the coal in a slot shape to the bottom of the coal seam.

The system should also include a conveyor system and a storage hopper for rapid removal of the mined coal. This will allow a realistic evaluation free of delays caused by coal piling up and preventing normal bucket dumping.

1.0 INTRODUCTION

Aside from augering, which is practical only with a narrow range of seam pitch (downward pitch of no more than approximately 10 degrees) , there has been virtually no effort to mine coal from exposed dipping seams beyond the economic limit of surface mining. Because augering is not applicable to moderately and steeply pitched seams, a new method was sought for mining coal lying beyond the highwall at the economic stripping limit.

Under Bureau of Mines Contract J0-133925, ESD Corporation has developed the High Angle Mining System (HAMS) as a safe, efficient, and profitable method of extracting coal from seams varying widely in thickness and pitch. The system would normally be employed to extract coal lying beyond the economic limit of a strip or open-pit highwall. It is a surface-controlled operation involving no underground personnel. The High Angle Mining System can significantly increase the reserve base of coal that can be economically surface mined.

The program's direction was to make a thorough study of the recovery problem, review existing technology, consider alternatives, and develop a prototype of the concept considered to have the best potential for coal recovery. During a 1-year study, concepts were formulated and evaluated. The most promising concept was submitted to the demonstration mine and the Government for approval. The preliminary design and economic analysis of the approved approach were completed during the balance of the study.

Following the study, a prototype High Angle Mining System was designed, fabricated, and demonstrated to be a feasible method of mining pitched seams of coal which lie below the economic stripping limits of open-pit mining.

2.0 SUMMARY

The scope of this contract was to investigate the possibility of developing a High Angle Mining System (HAMS) to mine pitched-seam coal reserves.

A number of open-pit or surface mines which have moderately or steeply pitching seams were visited and studied. Existing technology was reviewed to identify systems or components of systems which merited further consideration for the development of the HAMS concept.

A number of conventional approaches and innovations based on augering and two approaches based on slackline excavation were developed in sufficient detail to enable comparison on the bases of operational limitations, recovery ratio, productivity, and economy.

The slackline excavator concept employing a projectile bucket was selected as the best approach. This approach also gained the support of the FMC Skull Point Mine management near Kemmerer, Wyoming, which agreed to cost share in the development.

The slackline excavator concept has few moving parts, is reasonably inexpensive to fabricate, is simple to operate, and has excellent potential for safe, economical coal mining. The system consists of a slackline excavator system with a modified dragline projectile bucket suspended from a track cable on low-friction sheaves. The track cable is anchored to the bottom of a small pilot hole drilled down into the coal seam near and parallel to the roof. It is anchored by inserting PVC plastic pipe alongside the cable, pumping in cement grout, and removing the pipe. Guiding the bucket by means of the track cable ensures mining in a predetermined direction with good control of the bucket behavior. The bucket is controlled from the surface to excavate a vertical slot of a width approximately equal to the bucket and a height approaching the seam thickness. The face of the excavation is advanced by the gravity-powered projectile impact of the bucket into the coal. The projectile face of the bucket contains an array of sharply pointed conical teeth for this purpose. The floor of the excavation is ripped during the hoist cycle by ripping shanks attached to the underside of the bucket. Ripped coal is strewn behind the bucket while the loose coal ripped earlier is scooped up through the front end of the bucket.

Although during the demonstration test several delays resulted from equipment failures and necessary modifications, the capability of economically mining pitched seams of coal with the High Angle Mining System was shown to be feasible, with limitations discussed in this report.

The inclusion of a motorized cutting head on the HAMS bucket will improve the bucket advancement and make performance more predictable.

3.0 TECHNICAL DISCUSSION

3.1 BACKGROUND

Large reserves of pitched seam coal are located in 19 of the 50 states in the United States. These reserves are both moderately pitched seams (8 to 25 degrees) and steeply pitched seams (25 to 90 degrees). The most common modern method of mining coal from pitched seams is by open-pit or stripping methods. When the pitched seams are mined below a certain depth, or the overburden becomes too thick because of topography, it becomes uneconomical to remove the overburden to mine the coal. When that depth is reached, the mining for that area is stopped. The pit must then be backfilled and the landscape returned to a condition similar to the original terrain. Therefore, the coal reserve lying below the economically feasible stripping depth is lost or unmined until such time as underground mining can be justified. New methods are needed to reclaim these large coal reserves before access to them is backfilled.

Several methods of mining these lower coal reserves have been used with some degree of success, the most common being the standard auger method. A summary of some of the concepts which have been tried is given in Subsection 3.1.3.

3.1.1 Estimated Reserve Base

A Bureau of Mines-sponsored report[1] in 1975 estimates the augerable reserve base in the United States to be 5 billion tons for a depth of 200 feet. Based on a 25-percent recovery rate, the augerable reserve is estimated at 1.25 billion tons.

This reserve base was limited to coal residing in seams from 2 to 8 feet thick and less than 10 degrees in pitch. The amount was noted to be nearly equally split between the eastern and western coal regions with a trace falling in the central region. The practice of augering has been limited almost entirely to the eastern region.

3.1.1.1. Estimate For Moderately Pitching Coal Seams (8 to 25 Degrees)

A 1978 report prepared for the Bureau of Mines identifies the location of moderately pitching seams (8 to 25 degrees) within the continental United States.[2] Seams within this range were found to reside in 16 states as shown in Figure 1. Volume II of that report contains detailed information regarding strippable reserves which are estimated at 23 billion tons.

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1. Ford, Bacon, and Davis, "Technology of Auger Mining," Bureau of Mines OFR 108-76.
2. Skelly and Loy, "Development of Concepts for Surface Mining Moderately Pitching Coal Seams," Bureau of Mines OFR 61(1)79 and 61(2)79.

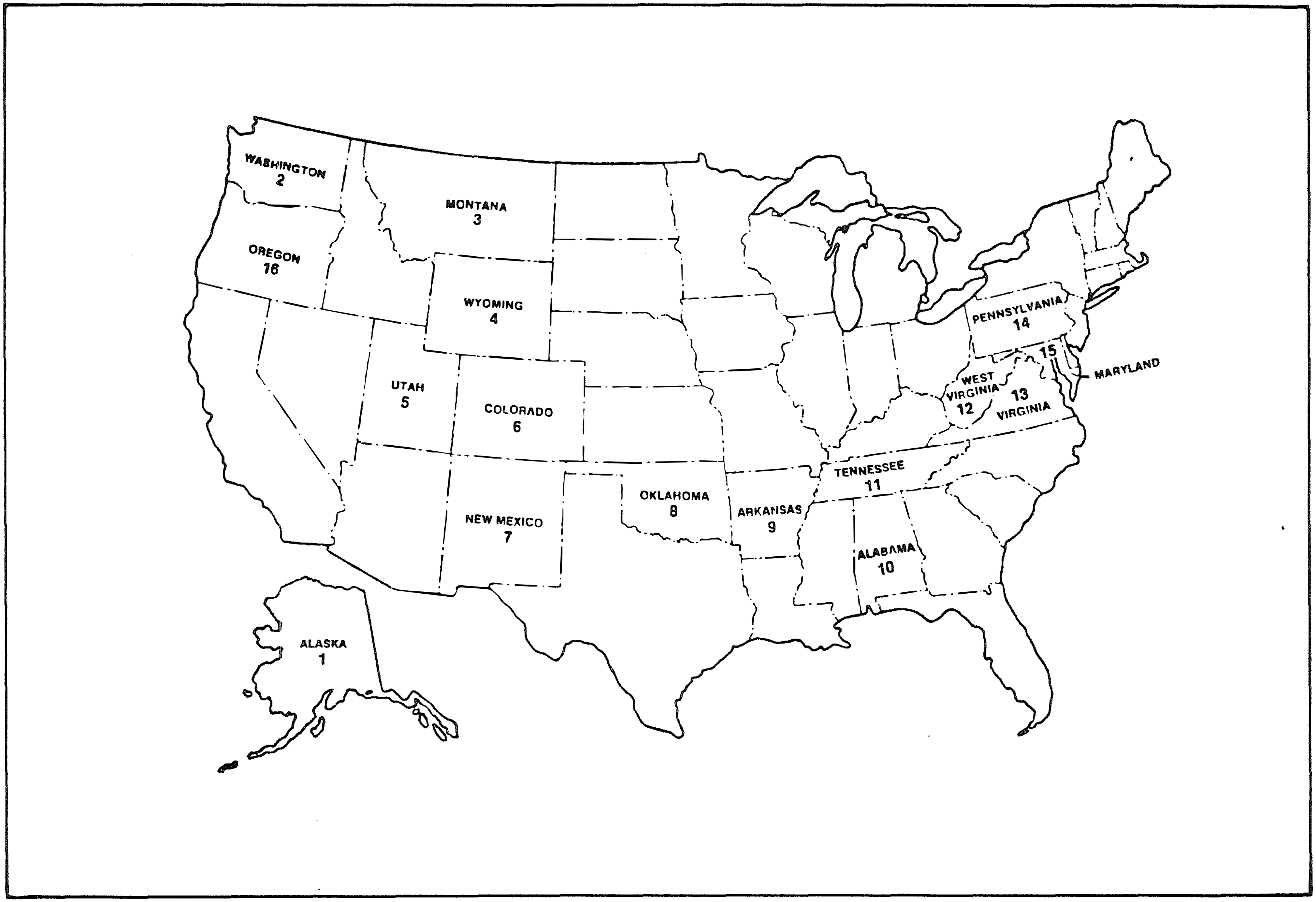


Figure 1 LOCATION OF MODERATELY PITCHING SEAMS (8 TO 25 DEGREES)

The information is tabulated by coal region, field, seam nomenclature, county, strippable reserves, pitch, thickness, quality, overburden characteristics, and areal extent. The subject of a HAMS reserve base was not addressed.

3.1.1.2 Estimate For Steeply Pitching Coal Seams (25 to 90 Degrees)

A 1979 study,[3] prepared for the U.S. Department of Energy by the same authors as the above report, surveyed reserves for steeply pitching seams (25 to 90 degrees). Seams within this range were found to be located within 18 states, as shown in Figure 2. The 1979 report, like the 1978 report, contains a detailed description of seams. It also contains for each state detailed maps that identify the boundaries of steeply pitching seams for each coal region. The total strippable reserves lying in steeply pitching seams are estimated at 67 billion tons. A HAMS reserve base is estimated at approximately 235 million tons, as shown in Table 1 taken from the same report. The calculated values were obtained from the estimated seam outcropping lengths and seam heights for a penetration depth of 150 feet. The extrapolated values, marked by an asterisk, were based on a judgmental estimate of steeply pitching seams as a percentage of a state's strippable reserve base.

Table 1 ESTIMATE OF RESERVE BASE FOR STEEPLY PITCHING SEAMS (25 TO 90 DEGREES) BY STATE

State	Calculated reserve base
Alabama	* 46.4×10^6
Alaska	* 19.5×10^6
California	4.3×10^6
Colorado	* 19.5×10^6
Kentucky	* 4.2×10^6
Montana	18.7×10^6
New Mexico	9.7×10^6
North Carolina	2.1×10^6
Oklahoma	30.8×10^6
Oregon	* 1.2×10^6
Pennsylvania	* 12.4×10^6
Rhode Island	Trace
Tennessee	* 0.4×10^6
Texas	0.4×10^6
Utah	2.3×10^6
Virginia	27.0×10^6
Washington	* 21.0×10^6
Wyoming	14.7×10^6
Total	234.6×10^6 tons

* Extrapolated.

3. Skelly and Loy, "Evaluation of High Angle Auger Systems," Draft Final Report for U.S. Department of Energy, Contract ET-77-C01-8914, Task Order 0071, April 1979.

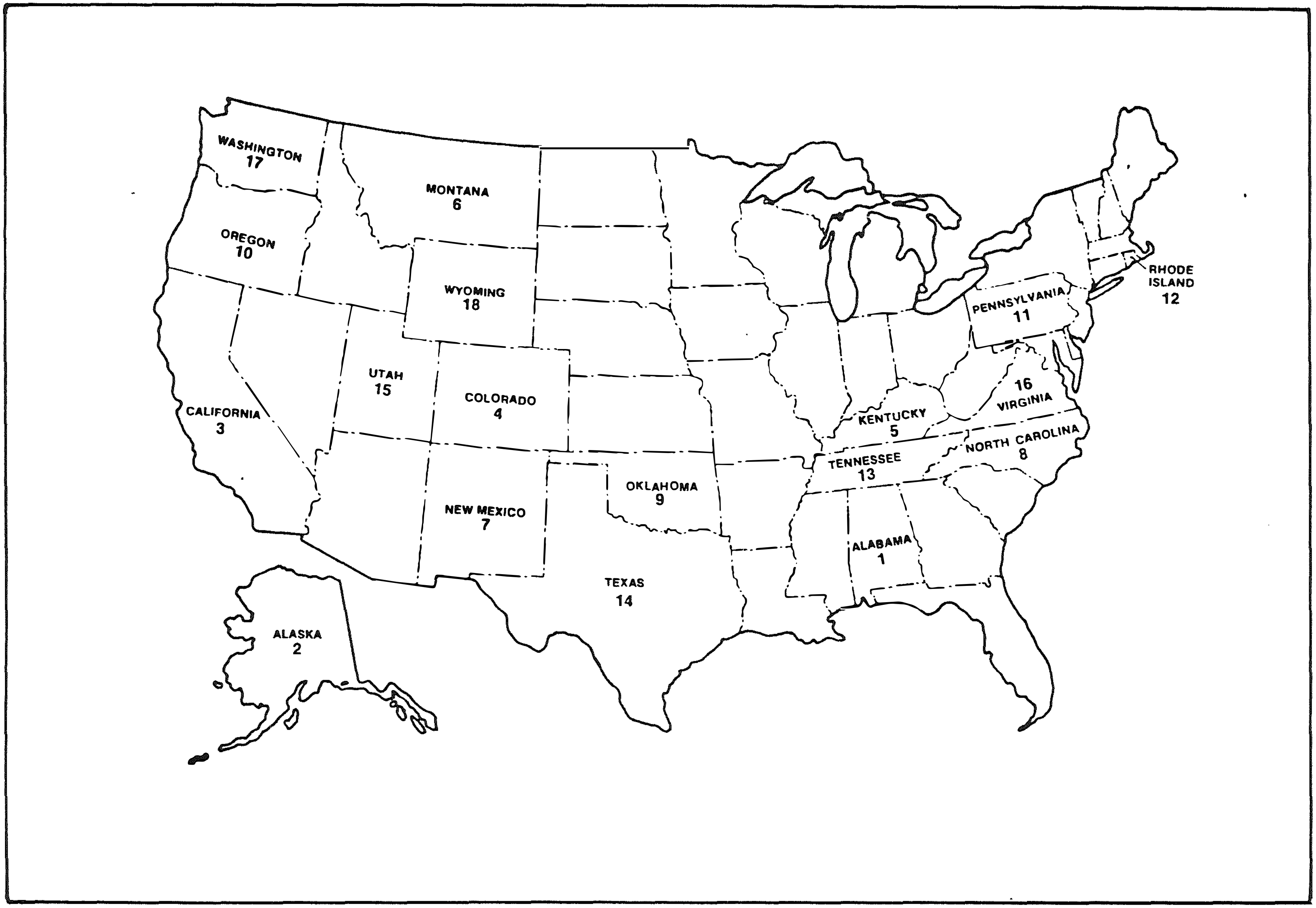


Figure 2 LOCATION OF STEEPLY PITCHING SEAMS (25 TO 90 DEGREES)

3.1.2 Pitched-Seam Mining

3.1.2.1 Underground Pitched-Seam Mining

For many years pitched-seam mining was a hand-mining operation which was made more difficult by the pitch of the coal seam. Very little open-pit mining was used in the early 1900's, but in recent years a larger and larger percent of all coal mining has been by surface or open-pit mining. For example, in 1974 46 percent was underground mining and 54 percent was surface mining. By 1978, 37.1 percent was underground mining and 62.9 percent was surface mining, an increase of 9 percent in surface mining in just 5 years. The cost of open-pit mining is much less per ton than underground mining and will therefore continue to increase as long as it is economically feasible to remove the overburden to obtain the coal. However, the coal reserve of underground-minable coal is understandably much larger than what is considered to be surface-minable coal.

3.1.2.2 Surface or Open-Pit Mining

Surface mining in pitched-seam coal fields becomes more expensive as the seams become deeper. The maximum depth to which it is economically feasible to remove the overburden and mine the coal depends upon the number and thickness of the coal seams, the thickness of the parting seams, and the degree of seam pitch. When the cost of removing the overburden to obtain the coal becomes too high, the mine is filled in and returned to the original or similar landscape.

3.1.2.3 Mining Beyond Surface or Open-Pit Mining

Since the large coal reserves lying below the feasible open-pit mining methods have been generally lost to production, great interest has been shown in methods of recovering this lost coal before the pit is again filled in for relandscaping.

Several types of machines have been designed and tested in an attempt to recover a greater amount of coal before the open pits are again filled in. The following subsection addresses these machines as they were examined for applicability in a high angle mining system.

3.1.3 Alternate Methods Examined for Applicability to HAMS

The ESD Corporation project team examined a considerable number of existing systems for possible applications to the High Angle Mining System. These systems included the following:

- Auger systems
- RSV Thin Seam Mining System
- Eckenrobe Auger
- New Edna Miner.

Also examined were cutting systems, including drum, ripper, and boring types, as well as their hybrid variations. These included such systems as the following:

- Continuous miners
- Tunnel-boring machines
- NCB-DOSCO In-Seam Miner
- Boom heading machines
- Dredge-type rotating cutterheads.

These several alternate methods, which are described in Appendix A, all had shortcomings which precluded their application to HAMS. Thus, it was determined that ESD would proceed with the development of a slackline excavator concept with projectile bucket.

3.2 EQUIPMENT DESCRIPTION AND OPERATION

The slackline excavator with projectile bucket concept selected for the High Angle Mining System (HAMS) consists of a specially designed bucket and a surface machine. The dragline-like bucket is suspended from a track cable on low-friction sheaves. The track cable is anchored to the bottom of a small pilot hole drilled down into the coal seam near and parallel to the roof. The bucket is controlled from the surface to excavate a vertical slot of a width approximately equal to the width of the bucket and a height approaching the seam thickness.

The face of the excavation is advanced by gravity-powered projectile impact of the bucket with the coal. The projectile face of the bucket contains an array of sharply pointed conical teeth for this purpose. The floor of the excavation is ripped during the hoist cycle by ripping shanks attached to the underside of the bucket. Ripped coal is strewn behind the bucket, while loose coal that has been previously ripped is scooped up as the bucket is pulled along the floor of the excavation by the hoist cable.

A complete bucket cycle is illustrated in Figure 3. From position (A) at the surface, the bucket, suspended from the tensioned track cable, begins its gravity-powered descent at a controlled speed. At position (B) the bucket is allowed to accelerate until impact at position (C). Coal broken loose during impact is trapped in the bucket. The track cable is then relaxed and the hoist cable drags the bucket across the floor of the excavation, gathering loose coal and leaving behind it a layer of newly ripped coal. At position (D) the track cable is retensioned to lift the loaded bucket clear of the floor to position (E), from which it is quickly hoisted to the surface and dumped.

The surface machine, shown in Figure 4, consists of the surface structure, track cable system, hoist cable system, and off-conveyor system. (Numbers in parentheses in the following discussion refer to Figure 4.)

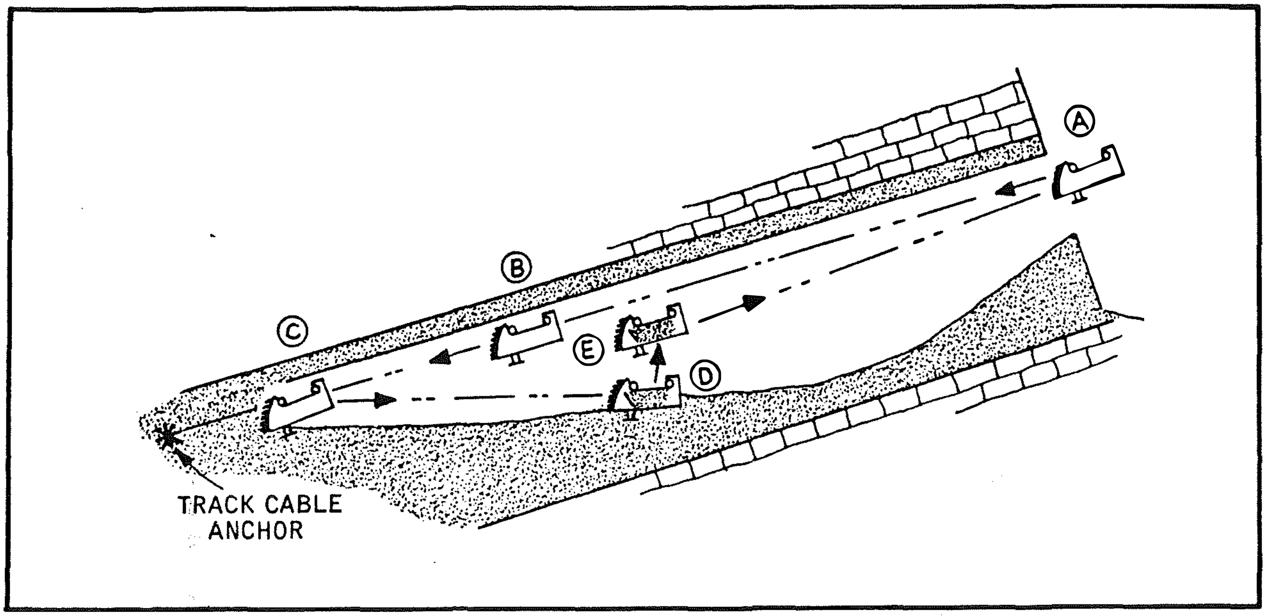


Figure 3 BUCKET CYCLE

The surface structure consists of tubular members jointed together by weldments at the node points. It is designed to be highway transportable in a disassembled state. The structure features 3-point load reaction while resting on the pit floor or working bench. Operational working loads of the two independent cable systems are reacted directly by the highwall face, also on a 3-point support pattern.

The upper deck contains the operator's station (1) which is heavily armored to provide protection from falling rock. A heavy grille-work (not shown) protects personnel from the cable systems in the event of failure. Figure 5 shows a view from behind the operator's console.

The surface machine is dragged into place by a tractor. The single ground lock leg (2) is actuated to rotate the machine towards the highwall face until the upper pad (3) bears against it.

The track cable system consists of a hydraulic tensioning cylinder (4) and a single heavy wire rope anchored by grouting to the bottom of a small-diameter pilot hole. A leaf chain carries the tensioning load between the track cable and the cylinder over a small-diameter pulley. Using a light cable, the free end of the pre-anchored track cable is threaded through the bucket and joined to the leaf chain with a wedge-type cable clamp. By preloading the track cable above normal working tension, the machine is snugged against the highwall face and the integrity of the track cable, anchor, cable clamp, and immediate highwall area is verified.

The hoist system consists of a modern divided-drum hoist (5) featuring solid-state control circuitry and both disk-type mechanical and motor-controlled dynamic braking. The DC shunt-wound motor is powered by rectified AC power from the mine's high-voltage system. The two-part hoist cable allows a smaller diameter drum hoist and head sheave. Hoist cable tension is equalized at the bucket through a whiffle tree arrangement. The hoist drum is grooved to accommodate a single layer of wire rope.

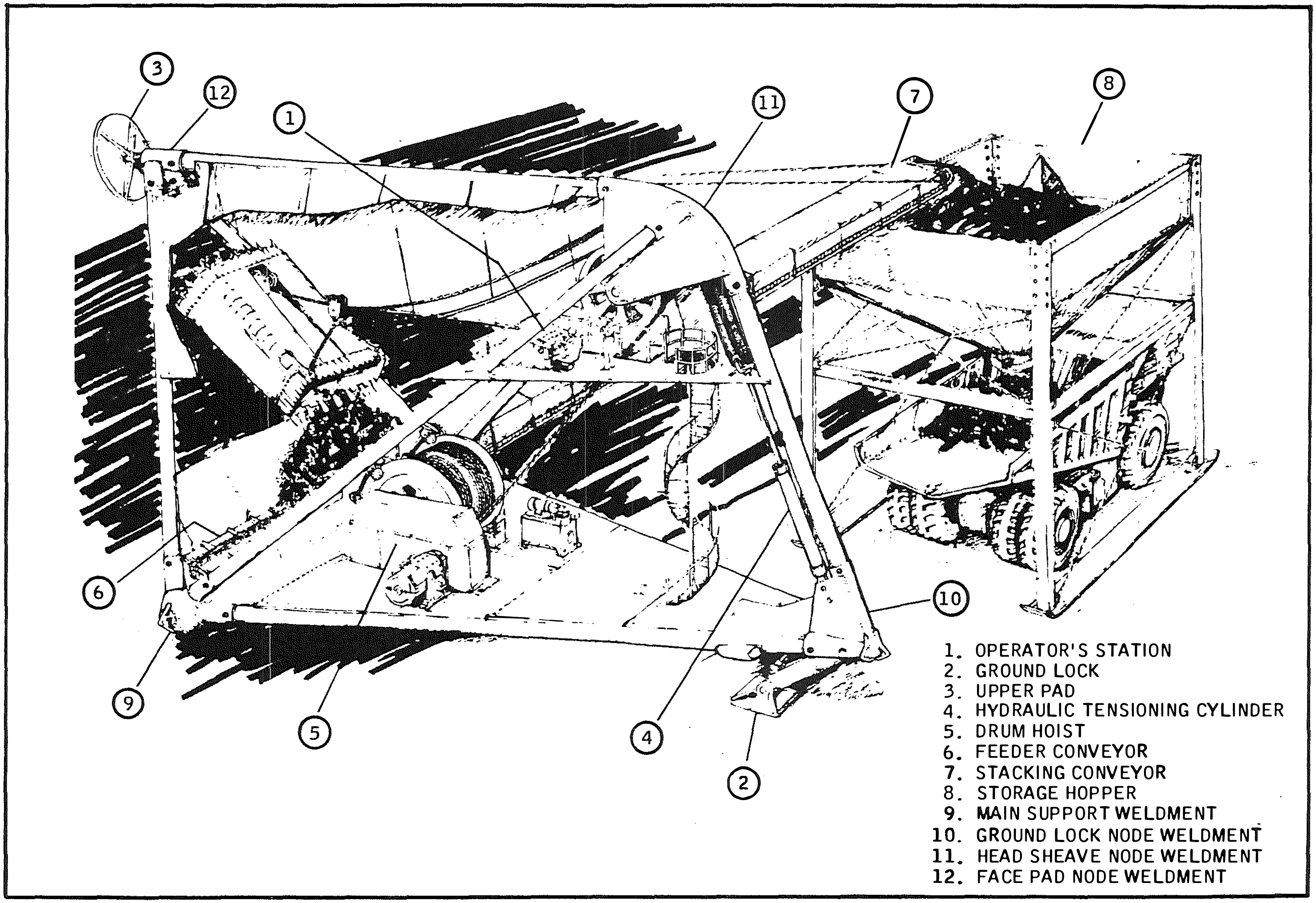


Figure 4 SURFACE MACHINE AND BUCKET

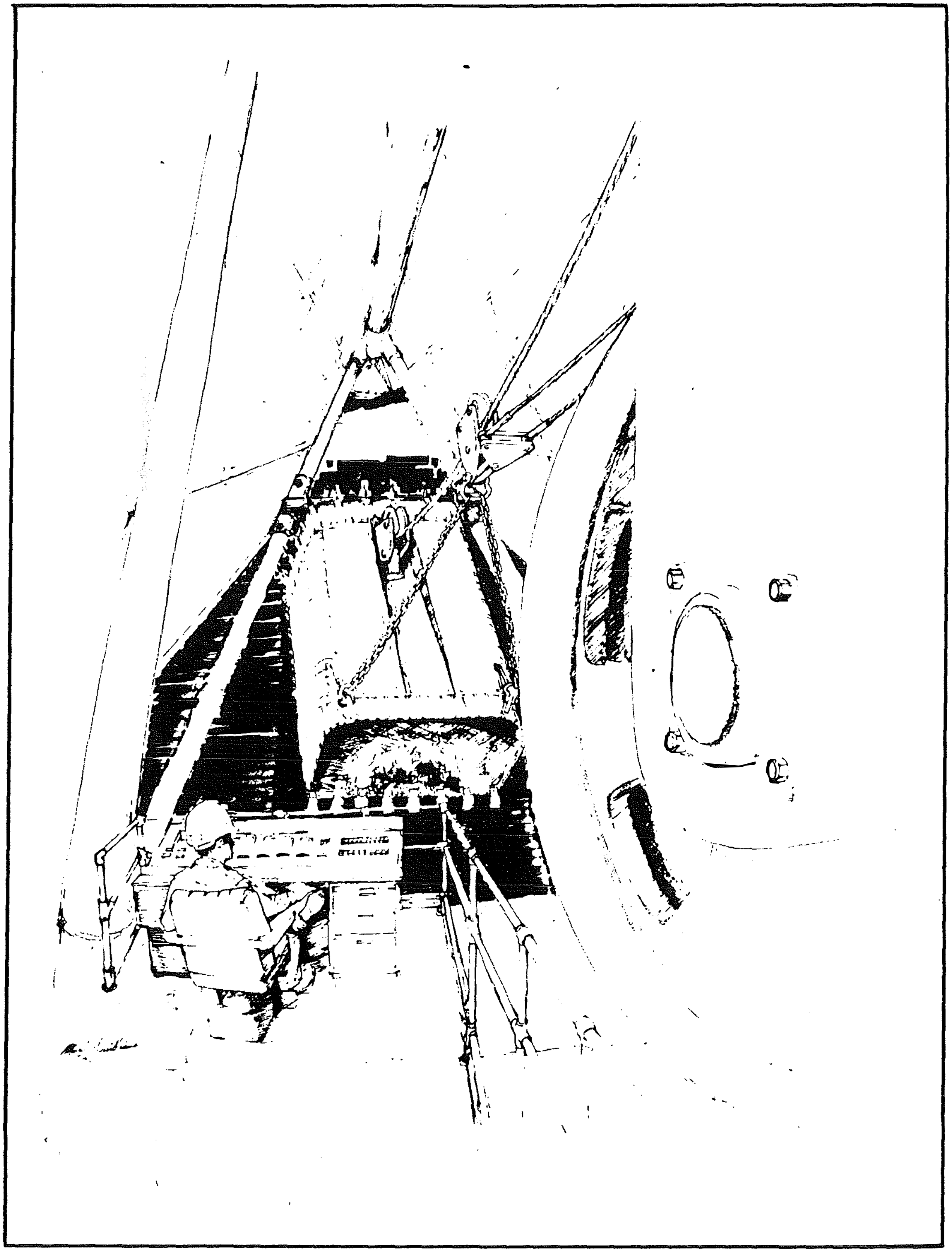


Figure 5 OPERATOR'S STATION

The off-conveyor system consists of a chain conveyor section (6) which feeds a pocketed-belt stacking conveyor (7). The stacking conveyor is slewable in azimuth and controllable in elevation. Also shown is an optional storage hopper (8) capable of quickly loading the surface haulage trucks. This improves economy by speeding surface haulage and eliminating a front-end loader and the attendant potential for product contamination.

3.2.1 Bucket Design

The overall dimensions of the bucket, shown in Figure 6 (in the ripping mode), are approximately 6 feet high, 8 feet wide and 12 to 16 feet long. The empty weight is about 10 tons. The payload capacity is 9 cubic yards, or about 6 tons of coal at a density of 50 pounds per cubic foot. About 75 percent of the bucket weight is carried on the two-wheeled truck pinned to the bucket near its projectile face.

The bucket is heavily constructed of steel plate with reinforcements at the corners, at load points, and at high-wear areas. The end frames and projectile grating are about 1-1/2 inches thick. The balance of the plating is 1/2 inch thick.

3.2.1.1 Projectile Face Design

The projectile face of the bucket contains an array of conical teeth designed to maximize penetration and coal fracturing between teeth. The peripheral teeth are inclined outward to create clearance for the bucket structure within the excavation. Practical experience would indicate an optimum cutting plane clearance angle (between the outside of the tooth and direction of motion) of about 5 degrees.

The upper peripheral teeth shape the roof of the excavation. These are arranged to produce a slight arch with rounded corners.

Each interior tooth is mounted with its nominal axis parallel to the bucket and set back slightly from the peripheral teeth. Based on the limited testing of a single projectile tooth, the best estimate for tooth spacing is 18 inches.

The impact print of the tooth array on the excavation face can be raised and lowered within limits by varying the tension in the track cable. Through the employment of this technique, the volume of coal broken loose by each impact may be increased significantly.

The efficiency of the projectile impact mode of operation is determined largely by the bucket's ability to capture and retain the coal broken loose. This could be referred to as "face cleaning." Significant amounts of coal left behind will absorb some bucket energy and contribute to the creation of more fines during each subsequent impact cycle.

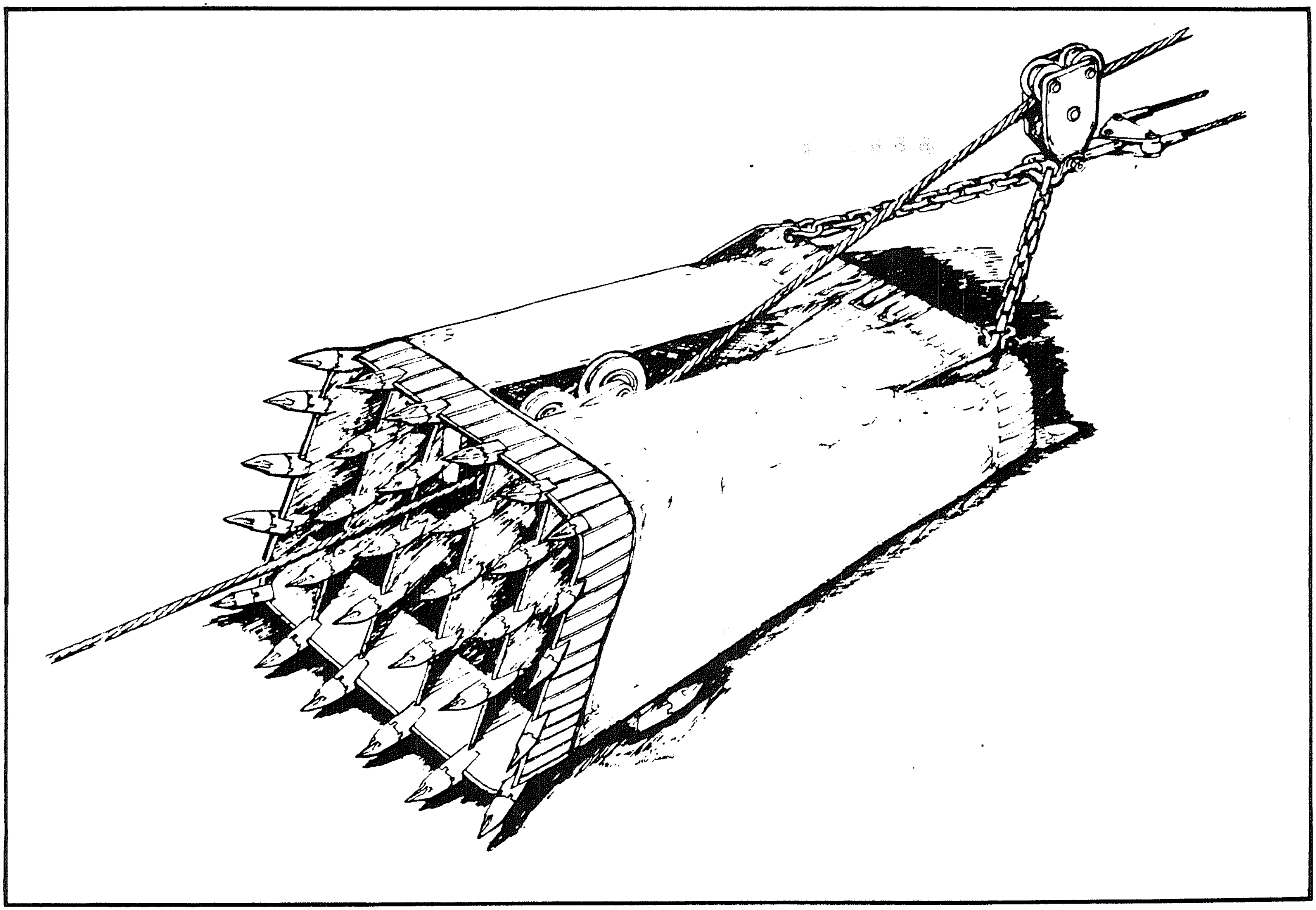


Figure 6 BUCKET (SHOWN RIPPING)

Broken coal need not be retained in the bucket for more than a few bucket lengths to keep the face area clean. In this approach, bucket attitude is maintained nearly horizontal as it approaches impact. Loose coal near the face is scooped up before impact. Additional coal broken loose during impact also falls into the bucket. This coal would be temporarily retained in the open projectile end of the bucket by the coal's natural friction angle. Later in the hoist cycle, as the bucket begins to pitch up, this coal would spill out onto the excavation floor, to be scooped up through the opposite end on some later cycle.

3.2.1.2 Ripping Shank Design

The underside of the bucket contains a number of ripping shanks, adjustable in depth from 6 to 24 inches. For a nominal lateral spacing of 20 inches, a total of five shanks would be employed. These would be arranged in a chevron pattern located about 1/3 of the bucket length back from the projectile face. The teeth are easily replaceable, and the shanks are fitted with easily replaceable wear guards.

The estimated total ripping load of the five shanks for a depth of 12 inches is 40 tons. Based on an instrumented test of a D9 ripper/dozer (conducted in coal at the demonstration mine), this value appears reasonable.

3.2.1.3 Projectile Process

Depending on seam thickness, between one-third (18-foot-thick seam) and one-eighth (50-foot-thick seam) of the total coal mined is extracted through the projectile process. The balance of coal produced is ripped.

During this process, the bucket accelerates as it descends along the track cable and the strategically placed front teeth impact into the coal face like projectiles. The average volume of coal broken loose by the teeth per impact determines the average advancement of the bucket along the track cable embedded in the coal seam. (For the productivity analysis, an average advancement of 2 inches per impact was assumed.)

Various outside studies of high-energy-impact rockbreaking revealed a relationship between materials (rocks) and the energy input per unit volume of rock fragmented. This is called the specific energy, and it is the generally accepted measure of efficiency in rock breaking: the lower the specific energy, the more efficient the rock-breaking process. According to an article in the August 1975 issue of CIM bulletin ("High-Energy Impact Rockbreaking," by I. Grantmire and I. Hawkes), the specific energy values for various rocks and breaking processes range between 1.0 and 1,000 ft-lb/in^3, with the majority of values under 100 ft-lb/in^3.

Based on the projectile tooth tests which were conducted at the contractor's facility, and measuring only the amount of coal removed from the imprint left by the pointed tooth, a specific energy value of 103 ft-lb/in^3 can be calculated for fracturing coal with a single projectile tooth.

A 20,000-pound bucket traveling at 30 feet per second has a kinetic energy of nearly 280,000 foot-pounds. If a very conservative estimate for specific energy is used (100 foot-pounds per cubic inch), 2,800 cubic inches of fractured coal would be produced during impact. For a frontal area of 48 square feet, the average bucket advancement would be 0.4 inch per impact.

However, the efficiencies achievable by multiple teeth are greater than what is suggested by the test of the single tooth. A significant amount of additional coal fracturing and breakout occurs between adjacent teeth working in unison. The efficiency of the projectile process is improved further by raising and lowering the bucket slightly between successive blows to vary the impact print on the coal face and enhance the fracturing process.

3.2.1.4 Ripping Process

Most of the coal produced is broken loose from the floor of the excavation by the ripping action of the bucket as described in the following. After the bucket has impacted into the coal face, the track cable is slackened to allow the bucket to rest on the excavation floor. Simultaneously, the hoist drum is started to haul the bucket. An array of short ripping shanks protruding from the bottom of the bucket then begin to rake the floor. Under bucket weight alone, the ripping teeth penetrate the coal until the bucket bottoms out. Under this equilibrium, the ripping operation is sustained until the bucket is fully loaded with loose coal ripped during a previous cycle. The track cable is then tensioned to lift the bucket clear of the floor and disengage the teeth.

Table 2 contains dimensional details of a number of ripper or scarifier attachments manufactured by the Caterpillar Tractor Corporation. In comparison, a D9H dozer at the demonstration mine is barely able to rip to a depth of 18 inches against the bedding plane using only two standard shanks. Caterpillar also has done a considerable amount of work in correlating rippability of various materials and their seismic velocities, and has determined that easily ripped formations are characterized by lower seismic velocities.

The following assumptions were verified by tests:

- The ripping shanks can penetrate unbroken coal in the same way as existing rippers and agricultural subsoilers penetrate other materials.

- The inclined bedding planes and ripping direction do not adversely affect ripping capability.

- The dragline vectors pulling upwards on the bucket do not restrict excavation to impractical limits.

- The dragline cable loads at various depths are reasonable.

- The tooth designs and optimum tooth clearance angles used in the industry are acceptable.

- The swath generated by single shanks cause adequate fracturing between the selected spacing of multiple shanks.

Table 2 CATERPILLAR RIPPER/SCARIFIER ATTACHMENTS

Vehicle	Track-type bulldozer							
	D3B	D4E	D5B	D6D	D7G	D8K	D9H	D10
Gross vehicle weight (tons)	7.2	9.7	12.9	15.7	22.1	35.3	46.7	95.2
Number of shanks	5	5	5	5	3	3	3	3
Overall width (outer shanks)	41	68	84.4	84.4	78	92	106	98
Shank spacing, center	14	17	21.1	21.1	39	46	53	49
Shank spacing, outer	14	17	21.1	21.1	--	--	--	--
Maximum penetration	10.25	16.6	20.6	22.5	29	29	38.5	44

Vehicle	Track-type loader						Wheeled motor grader		
	931B	941B	951C	955L	977L	983B	12G	14G	16G
Gross vehicle weight (tons)	8.3	13.2	14.2	17.8	24.8	40	15	20.3	27
Number of shanks	5	5	5	5	5	5	5	7	7
Overall width (outer shanks)	56	68	68	80.6	84.4	92	84	96	109.4
Shank spacing, center	14	17	17	17	18.2	23	21	18.6	19.7
Shank spacing, outer	14	17	17	18.3	24	23	21	14.7	17.5
Maximum penetration	10.25	16.6	15.6	14.3	20.1	20.4	18.2	15.8	17.8

All dimensions in inches.
From Caterpillar Performance Handbook, Tenth Edition, October 1979.

- The vertical load from the track cable is sufficient to disengage the ripping teeth.

- The bucket remains in an upright position during the ripping cycle.

3.2.2 Surface Machine

An anticipated production surface machine (Figures 4 and 5) would consist of the surface structure, track and hoist cable systems, and conveyor system. It may also include a separate storage hopper. The machine is approximately 40 feet high, 25 feet wide (parallel to the highwall), and 50 feet long. Its weight is estimated at 160 tons.

A self-propulsion capability should be included; however, skid mounts may be provided to facilitate towing instead of a self-propulsion capability.

3.2.2.1 Surface Structure

The anticipated surface structure of a production machine will be comprised of tubular support members connected together by socketed weldments at the node points. This allows the structure to be disassembled for highway transport. The nomenclature of these weldments is identified in Figure 4. The main support weldment (9) consists of a 5-foot-diameter tubular weldment that serves as a primary skid. It is reinforced with steel rings uniformly spaced along its length. Further reinforcement is provided by longitudinal webs positioned between rings at a 45-degree circumferential spacing. Provision is made for attaching a towing bridle to either conical end of the tubular skid.

The ground lock node weldment (10) includes the anchor attachment for the track cable hydraulic tension cylinder (4), the ground lock assembly and cylinder (2), and two eyes. In the retracted position, the base of the ground lock serves as a skid. The head sheave node weldment (11) supports the track cable chain tension sheave, the chain anchor point, and the large head sheaves for the hoist cables. The face pad node weldment (12) includes a 6-foot-diameter steel pad (3) which can pivot from a ball-and-socket attachment to make flat contact against the highwall and provide support for the cable loads during the mining operation.

The lower front section of the surface structure contains the dumping chute and feeder conveyor and the drum hoist. A protected area located on the upper deck contains the control station and provides the operator with a good vantage position. The upper half of the structure is armored with steel plate to withstand falling rock.

3.2.2.2 Preliminary Structural Analysis

A preliminary analysis was conducted for the anticipated production-size surface structure in order to verify the size selection and adequacy of the primary structural members. The critical stress areas are located in the back pipe and in the two bottom pipes. The critical load in the back pipe is a column-type loading of 200 tons from the track cable tensioning cylinder. The critical load in the two bottom pipes is a bending load resulting from the weight of the drum hoist.

Since the structure is relatively large and involves an operator, and since it was subjected to vibration-induced loading from the track cable and the hoist cable, the deflections of all the primary structural pipes and all the floorings are limited to 1/360 of the span lengths.

All plates, bars, and structural members of the surface structure are defined as ASTM A36 steel; all pipes are specified as ASTM A53, Grade B. The allowable yield stress for these materials is 36 ksi, and the ultimate stress is 60 ksi.

The allowable column-loading of the steel pipes is limited to the loads stipulated in the AISC specification for structural steel buildings. All the structural components are also sized to limit maximum stresses to one-third of the material yield stress and one-fifth of the material ultimate stress. The structural analysis was based on a design load of 100 tons on the track cable and 40 tons on the hoist cable.

The results of the analysis revealed that the back pipe and the two base pipes need to be 24 inches in diameter, with a minimum wall thickness of 1/2 inch. The 5-foot-diameter pipe located at the front needs a 1/2-inch wall thickness and (to transmit localized high surface loading from protruding rocks) requires an internal rib structure approximately 6 inches deep. The remaining pipes can be either 24 inches in diameter with a 3/8-inch wall thickness, or 18 inches in diameter with a 1/2-inch wall thickness. In general, equipment support plates need to be a minimum of 1/2 inch in thickness, with an incorporated rib structure for carrying localized heavy loading in the area of the drum hoist. These ribs should be 1/2 inch thick, 2 to 4 inches deep, and on 2-foot centers. The rib structure also serves to reduce deflections for the operators in the controls area.

The relatively large size of the basic structure will result in excessively high floor deflections under the repeated tensioning and loosening of the hoist cable. These deflections would be on the order of 0.7 to 0.8 inch for 40 tons of on-and-off loading of the cable. To reduce these deflections to an acceptable level for standing operators and maintenance personnel, it is necessary to provide structural members which can effectively carry the drum hoist floor-loading up to the structure supporting the large sheave. These members will serve an additional purpose by providing a shield for the hoist cable from the drum hoist to the sheave. The shield is needed, of course, to protect personnel from the whipping cable during the rapid loading and unloading of the bucket hoist cable.

3.2.2.3 Track Cable System

The track cable system includes the cable anchor and the tensioning mechanism. Normal working loads induced by a fully loaded bucket are limited to 20 percent of the track cable's rated breaking strength. To ensure that the track cable can be salvaged for reuse at the completion of each excavated hole, the anchor is designed to fail before the cable.

3.2.2.4 Track Cable Anchor

A number of different approaches were considered for the cable anchor. Cement grouting was selected because it is in common use in the mining and construction industries, and it is tolerant of moisture and water which can be expected to collect at the bottom of the pilot hole. The grouting mixture can be pumped to the bottom of the hole through expendable tubing. To determine strengths achievable in coal, it was necessary to conduct a test at the demonstration mine. This test also established that the cable is easily fed into a 22-degree downward dipping hole with the assistance of gravity.

Table 3 shows projected grout lengths necessary to achieve anchor strengths equal to 50 percent of breaking strength for different diameter ropes. The values shown are scaled up from the observed strength of 1-inch-diameter 6x19 IWRC wire rope by a ratio of rope diameter. This assumes that strength is proportional to grouted surface area. The test cables were untreated and the protective coating of grease was left undisturbed. Shorter grout lengths may be possible by degreasing the cable or by using a solid, serrated anchor bar. On the other hand, smooth locked-coil-type rope may require a longer grout length.

Table 3 ANCHOR GROUT LENGTH FOR VARIOUS CABLE DIAMETERS IN COAL

Wire rope diameter (inches)	Breaking strength (tons)	Grout length,* for 50 percent rope strength (feet)
1	58	20.0
1-1/2	127	29.4
2	218	37.8
2-1/2	335	46.5
3	495	57.2

* 2:1 water to cement ratio, 48-hour cure.

3.2.2.5 Track Cable Diameter and Excavation Depth

Maximum tension in the track cable occurs when a loaded bucket is at mid-span. Tension decreases for a given span length as deflection is allowed to increase. For this reason, penetration depths obtainable for any given cable diameter increase with seam height. Track cable tension also decreases as the seam pitch increases.

Table 4 shows the minimum seam height necessary for 200 feet of penetration and the maximum penetration depths obtainable in a 50-foot-thick seam for various rope diameters. The values shown assume a seam angle of 22 degrees, a design safety factor of 5:1 on cable breaking strength, a bucket height and gross weight of 6 feet and 16 tons, a 2-foot thickness of roof coal left in place, and 50 feet of unsupported track cable outside the excavation. The breaking strengths shown are for round or locked-coil tramway rope.

If connected directly to the tensioning cylinder, the track cable would require a large-diameter head sheave. (The recommended sheave-to-cable diameter ratio is 100:1 for locked-coil rope.)

Table 4 PENETRATION FOR VARIOUS SEAM THICKNESSES AND CABLE DIAMETERS

Track cable diameter (inches)	Breaking strength (tons)	Minimum seam height for 200 feet penetration (feet)	Maximum penetration in 50-foot-thick seam (feet)
1-1/2	127	45.2	232
2	218	30.1	411
2-1/2	335	22.6	607
3	495	18.2	818
3-1/2	640	16.0	970

Seam angle = 22 degrees, bucket weight = 16 tons, span = penetration + 50 feet, deflection = seam height - 8 feet.

3.2.3 Hoist Cable System

The hoist cable system employs dual hoist ropes wound on a modern mine-rated divided drum hoist unit. To minimize the effects of twisting, the ropes are of opposite lays. Drum and sheave diameters are a factor of 60 times the rope diameter. Dual hoist ropes permit a significant reduction in drum and sheave diameters from that necessary for a single rope of equal strength.

3.2.3.1 Cable Size

Table 5 shows the various cable speeds and hoist loads of a bucket cycle. The maximum load occurs during ripping, and this load determines rope diameter. The 40-ton ripping load shown is a best estimate for ripping to a depth of 1 foot, and appears reasonable, based on an instrumented test of a D9 ripper/dozer. For the assumed load and a 5:1 design factor, each rope requires a breaking strength of 100 tons. For a 6x37 IWRC class construction, this corresponds to a rope diameter of 1-1/2 inches.

TABLE 5 HOIST LOADS, SPEEDS, AND POWER LEVELS

Activity	Bucket weight (tons)	Average hoist load (tons)	Average bucket speed (feet per second)	Average power at bucket (horsepower)*
Lower empty bucket	10	3.75*	15	-204
Rip	--	40.0 (estimate)	1	145
Hoist full bucket	16	6.0*	10	218

* For 22-degree seam angle.

3.2.3.2 Hoist Unit

To provide good control over the wide ranges of load and speed shown in Table 5, an air-cooled DC shunt-wound drive motor was selected for the drum hoist. Static silicon-controlled rectifiers (thyristors) supply variable DC voltage to the motor from the mine's AC power system. This type of drive system has been found to offer high efficiency, lower overall costs, and excellent reliability. It also permits regenerative braking while the unbalanced bucket is lowered at a controlled rate of speed.

The divided drum is grooved to accept a single layer of wire rope. Multiple spring-set disc brake units operate directly on the drum and provide quick-action braking. Figure 7 shows a 90-inch-diameter divided drum with disc brakes. These are about the size envisioned for HAMS.

As the bucket begins its gravity-powered acceleration to impact speed, the drum is motor-accelerated to minimize the effect of drum inertia on rope drag. Near the point of impact, the drum is immediately braked to minimize the payout of excess rope.

The control system provides for three modes of operation: manual, semiautomatic, and full automatic. Bucket position is sensed by a digital encoder at the drum. The location of the face is continually updated in the control system's memory, and automatic operation with respect to this datum can be programmed.

The solid-state drive system is diagrammed in Figure 8. AC mine power enters the motor control unit which protects the system from input voltage or current anomalies. Power is then supplied through an isolation transformer to the SCR panel for conversion to DC motor power. The regulator module contains the solid-state circuitry for regulating the firing sequences and safety interlocks. The microprocessor-based programming and diagnostic module provides for automatically sequenced operation. Diagnostic programs are actuated each time the system is turned on. In the event of an automatic shutdown, the primary fault is announced by a flashing LED.

3.2.4 Conveyor System

The conveyor system consists of a chain conveyor section that feeds a pocketed-belt stacking conveyor. Both are rated at 200 tons per hour. The hydraulically driven chain conveyor is of a type that is standard in the coal mining industry. The speed of the conveyor is automatically controlled so that if the stacking conveyor is overloaded the chain conveyor will slow down to prevent spillage of coal from the stacking conveyor.

If the coal size is a problem, a breaker can be added on the chain conveyor section. Breakers are also standard in the coal industry and can reduce the feed size to a size suitable for the stacking conveyor.

The base of the stacking conveyor is pivoted from the surface structure. The discharge end is cable supported. A winch raises or lowers the conveyor. Maintenance can be performed in the lowered position so that a walkway is not required. The stacking conveyor is also slewable in azimuth so that a fan-shaped pile can be built or rock-contaminated coal can be discharged to a separate pile.

The pocketed-belt conveyor belting (shown in Figure 9) allows operation at angles up to vertical, though the normal operating angle is about 45 degrees.

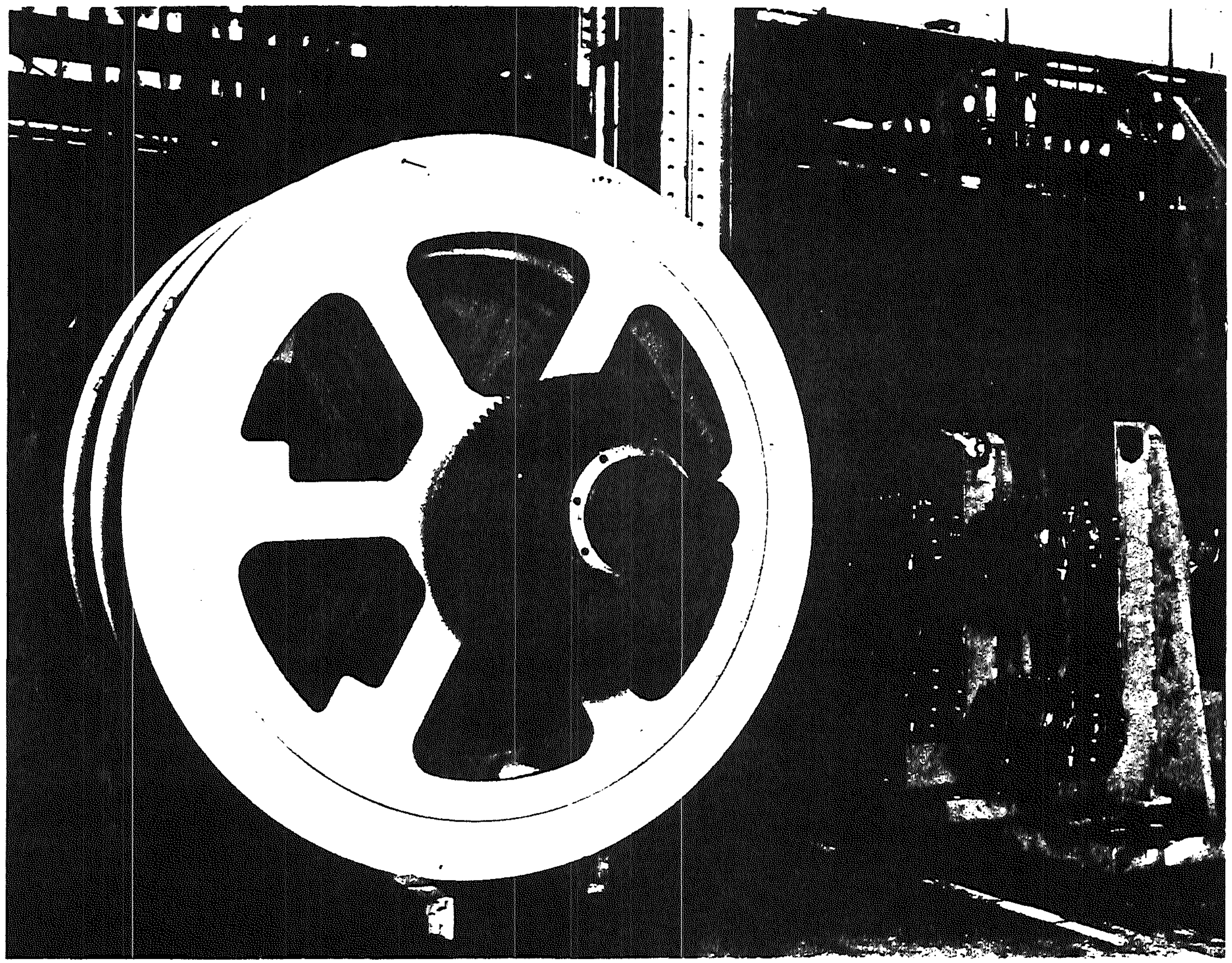


Figure 7   TYPICAL DRUM HOIST WITH DISC BRAKES

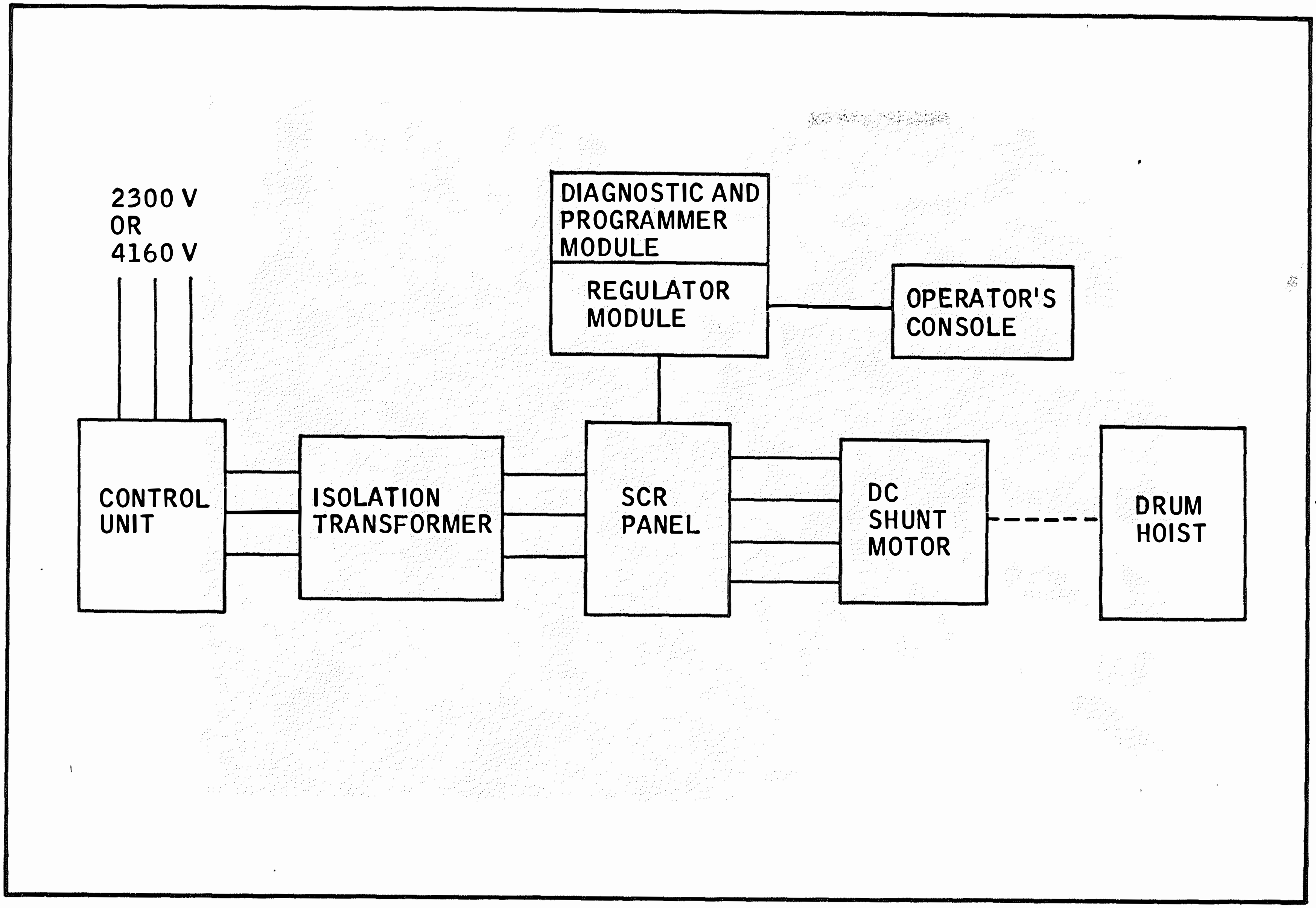


Figure 8 DRUM HOIST CONTROL SYSTEM

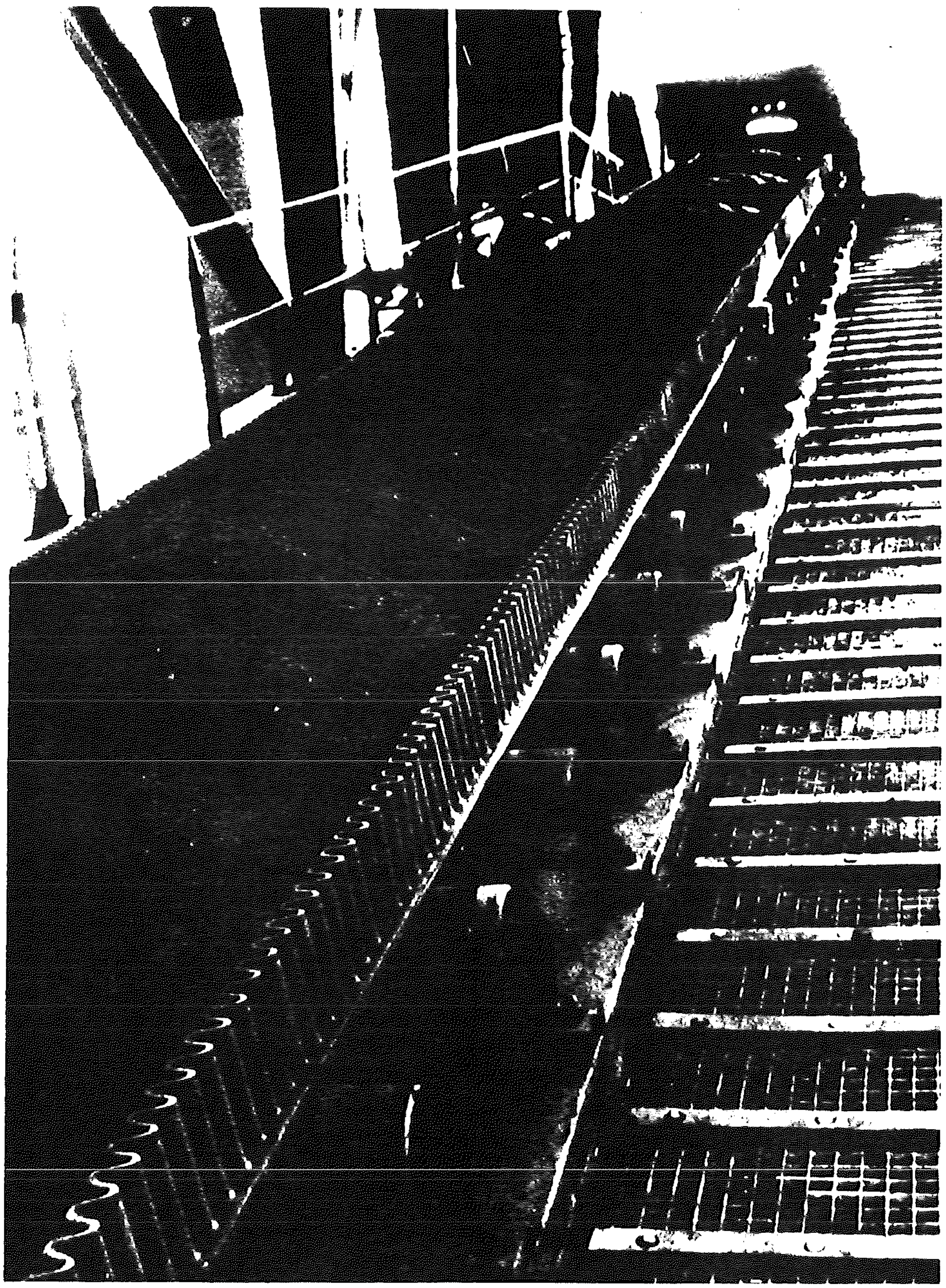


Figure 9 POCKETED-BELT CONVEYOR

3.2.5 Storage Hopper

An optional storage hopper can be provided with a 50- to 100-ton bin capacity to receive coal from the stacker and load it onto haulage trucks. The hopper would be skid mounted to permit positioning for optimum truck operation. The hopper would have a powered slide gate and controls accessible either from the truck cab or from the operator's station for quick loadout. The hopper would improve economy by speeding surface haulage, eliminating a front-end loader, and reducing the potential for product contamination.

3.2.6 Mining Plan

If a single coal seam is to be mined at the base of the final highwall, HAMS would be employed on a graded, level surface. The area would be prepared in a manner similar to a haulage roadway. Where multiple seams are to be mined, the various seams could be worked from the top down during development of the final highwall or from the bottom up during final pit filling. The most important factor to be considered is safety. The recommended approach for multiple seams is to mine from the bottom up. This avoids having to work below mined-out seams and weakened highwall.

The following subsections contain a discussion of the safety aspects of final highwall development, a description of the surface mine that agreed to cost-share the development and demonstration of HAMS, and a discussion of the proposed excavation geometry for this mine.

3.2.6.1 Final Highwall Development

The most hazardous location in an open-pit mining operation is usually near the highwall where HAMS would be employed. Highwall stability is the single most important factor regarding safety. Safety benches are normally created in the highwall in such a manner as to constrain rock falls and divert water from cascading further down the face.

Consideration should be given to the effects of time and weathering on local geological conditions. Blasting techniques can also greatly improve the integrity of the final highwall.

"Weather conditions often contribute to slope and highwall problems. Rainwater may seep into faults, joints, and other fissures, weakening the highwall by making the exposed surfaces in the cracks slippery. Cold temperatures create additional hazards when ground water freezes and expands in these areas. This expansion enlarges the gaps in the earth by exerting pressure against the walls of the fissure. The pressure is relieved when the rock on the exposed face of the highwall moves toward the pit, creating an even larger crack in the rock. The process is repeated when temperatures move above and below the freezing point in what is commonly called the freeze-thaw cycle, until the rock breaks free and falls..."[4]

4. "Mine Safety and Health," MSHA Safety News Periodical, September-October 1980.

Special blasting techniques can leave the highwall in the best possible condition. "These methods include line drilling, presplitting, and cushion shooting, and involve drilling a row of closely spaced parallel boreholes along the final excavation line. With line drilling, the boreholes contain no explosives but create a plane of weakness allowing the blasted material to break cleanly along what will become the pit wall. The line of drill holes serves the same function as a row of perforations on a sheet of postage stamps, but in three dimensions rather than two. The presplitting method involves a similar line of boreholes along the excavation line, but these holes are loaded with a light explosive charge and are fired just before the primary charges. The cushion blasting technique involves shooting the lightly loaded row of boreholes after the main hole pattern is blasted. These methods are often used in mountain highway construction..."[5]

3.2.6.2 Demonstration Mine

The demonstration mine selected, FMC's Skull Point Mine, is located near Kemmerer, Wyoming. The Skull Point Mine, which produces about 1,000,000 tons per year, is adjacent to and south of Kemmerer Coal Company's Elko-Sorenson Mine, which produces about 4,000,000 tons per year. These two mines are located within the HAMS Fork coal region and comprise its only active coal mines. This narrow region extends more than 300 miles north from the southwest corner of Wyoming, and falls within the overthrust belt, an area of intense activity for oil and gas exploration. Near an elevation of 7,000 feet in the eastern foothills of the Rocky Mountains, the weather can become quite cold in the winter. The seams mined are the Adaville formations which outcrop to the east and dip to the west. At the Skull Point Mine the dip angle is approxiately 22 degrees. The Adaville formations include some 13 identifiable seams which thicken, thin, split, and coalesce over very short distances. These formations and the South Haystack area 17 miles south of the Skull Point Mine contain the only strippable deposits in the region.[6]

The Skull Point Mine pit will ultimately expose some 7,200 feet of highwall to a depth of 600 feet. Some 17 million tons of strippable coal remain. Reclamation is planned to commence in 1986. Filling will begin at the north end of the pit and advance southward at approxiamtely 450 feet per year. The original contour will not be restored, but the general drainage pattern will be maintained and contouring will be similar to that of the surrounding area.

The Skull Point mine has four prominent coal seams. Of principal interest are the lower two seams, Number 1 lower and Number 1 upper, separated by a 20-foot parting. The lower seam presently averages 22 feet in thickness and the upper seam 47 feet. (These two seams converge to form a 100-foot-thick seam to the north on the Kemmerer Coal Company property.) The upper seam is slightly higher in quality since it has a lower ash and sulfur content. The upper two seams of note average 32 feet in composite thickness. Moisture content averages 22 percent, fixed carbon 39 percent, ash 5 percent, sulfur 1 percent, and the heating value 10,000 Btu per pound. In situ density is about 80 pounds per cubic foot.

5. MSHA Safety News Periodical, op. cit.
6. Keystone Coal Industry Manual.

The mine began producing in 1976. Mine management, staff, and union workers number about 100. Approximately 550,000 tons is shipped to FMC's Green River plant in Wyoming and used in the production of soda ash and phosphate chemicals. Another 200,000 tons is shipped annually to FMC's formcoke plant located near the mine. This is converted to 90,000 tons of coke and shipped to FMC's Pocatello plant in Idaho where it is used in the production of elemental phosphorous. The balance of production is sold outside. All transportation away from the mine is by rail.

Twelve and one-half million cubic yards of overburden have been stripped to date. The stripping ratio of overburden to coal is 5.2:1 (cubic yards per ton). The mine prepares the blast holes, which are contractor loaded. Overburden removal employs a 17-cubic-yard, 570-horsepower electric shovel, and five 120-ton trucks. Coal removal employs a 16-cubic-yard front-end loader and three 85-ton trucks. Overburden removal is conducted on a three-shift basis. Coal is mined on a single-shift basis. Coal is not cleaned but fed directly to a 4,300-ton-per-day (single-shift) tipple for sizing and loading into 100-ton rail cars. A coal sample is taken from each car and tested for Btu, ash, and sulfur content.

The highwall is maintained at an average angle of 55 degrees with 60-foot-wide benches spaced at 100-foot vertical intervals. Water seepage occurs all along the highwall and is collected in a sump at the bottom of the pit. This is pumped up to a 180-acre-foot settling pond area near the mine entrance. Current pumping rates are 150 gpm, but this is expected to increase to 400 to 600 gpm as the pit develops. In winter, seepage tends to freeze to the face of the highwall. Figures 10 and 11 show various views of the Skull Point operation.

The reserve base available to HAMS for a depth of 200 feet is estimated at 5.75 million tons in the highwall. This is based upon an average composite seam thickness of 100 feet for a distance of 7,200 feet. An additional 1.8 million tons are available in a 40- to 50-foot-thick outcropping seam to the south of the pit. This seam is estimated to outcrop for a distance of 5,000 feet.

Based on an average composite seam thickness of 300 feet for a length of 2 miles, the 200-foot-depth reserve base at the neighboring Elko-Sorenson mine is estimated at 25 million tons.

3.3 DEMONSTRATION PROTOTYPE EQUIPMENT

The selected equipment for the demonstration test was a subscale prototype of the slackline excavator with projectile bucket concept described in Subsection 3.2.

3.3.1 Projectile Bucket For Demonstration Testing

The size of the projectile bucket which was used for the demonstration tests was scaled down from the anticipated production-mining-sized bucket. The demonstration bucket was designed and fabricated at approximately 64 inches high, 80 inches wide and 12 feet long. It was formed from 3/4-inch-thick

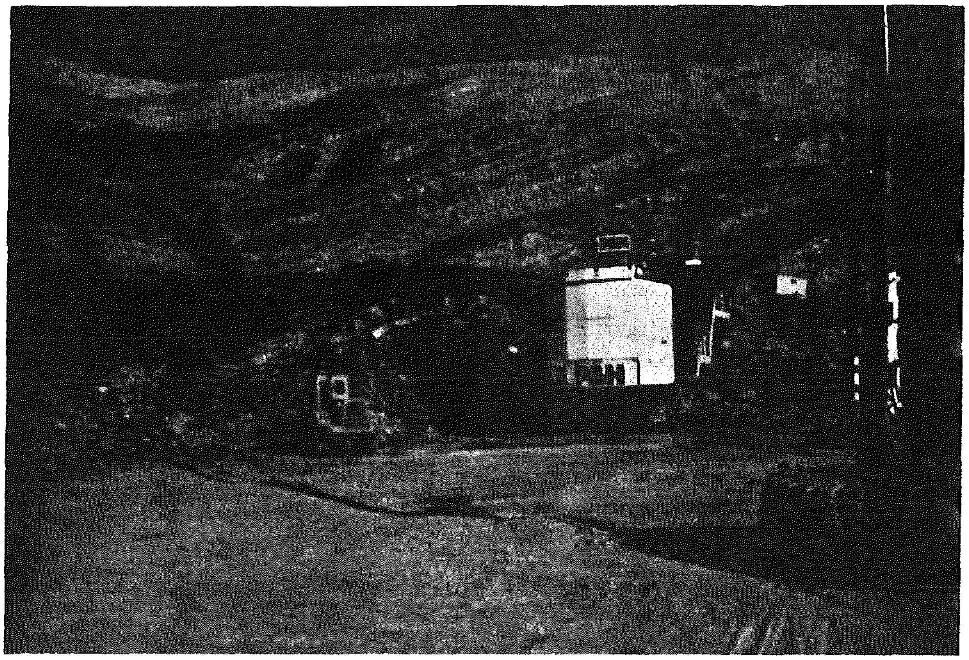


Figure 10 STRIPPING OVERBURDEN, LOOKING NORTHWEST FROM PIT FLOOR

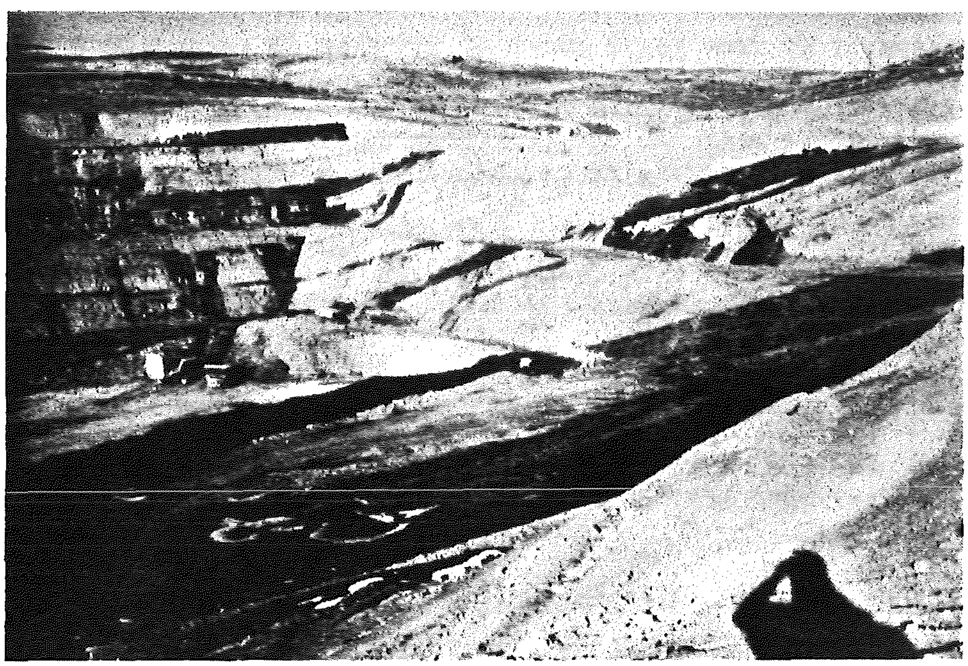


Figure 11 STRIPPING OVERBURDEN, LOOKING NORTHWEST FROM PIT EDGE

plating and had 1-1/2-inch-thick steel end framing. There were 33 conical-shaped teeth on the impact end of the bucket, and three adjustable ripper shanks with teeth that protruded through the bottom of the bucket. The hoist end of the bucket had six bottom shovel teeth and two side-clearance shovel teeth. The bucket weighed 20,000 pounds, including a 1,300-pound carriage assembly and a 2,000-pound hoist shackle-and-chain assembly. A picture of the bucket used for the demonstration tests is shown in Figure 12. Of all the concepts which were evaluated, the projectile bucket concept appeared to be the simplest to design, fabricate, and operate, and to have the greatest potential for coal recovery beyond the economically feasible limits of open-pit mining.

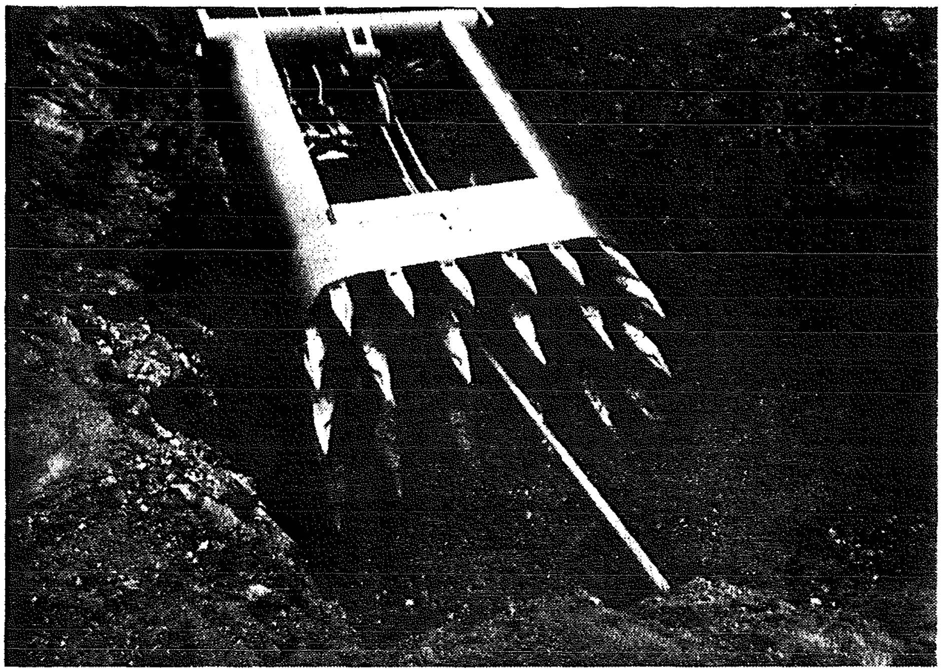


Figure 12 DEMONSTRATION TEST PROJECTILE BUCKET

3.3.1.1 Impact Mining Capability

It was anticipated from projectile progress calculations (Section 3.2.1.3) that a velocity of 30 feet per second would be required to obtain sufficient mining advancement per impact to make the projectile bucket concept feasible. To achieve this velocity requires a vertical drop of 22 feet, or 61.4 feet of track-cable travel at a 21-degree angle. The initial mining at each new setup did not give the full 30 feet-per-second velocity until the mining progressed enough to allow the 61-foot bucket travel down the track cable. However, repeated impacts at a slower velocity soon removed enough coal to give the longer track run, and the impact bucket concept proved feasible.

3.3.1.2 Ripping Capability

Three ripping shanks with conical teeth were installed with the capability of varying the depth that the shank protrudes from the bottom of the bucket. The six fixed shovel teeth at the hoist end of the bucket also acted as ripping teeth as the bucket was pulled along the floor of the excavation. The three ripping teeth were initially set for 12 inches of penetration and performed very well in ripping and loosening the coal as the bucket was hauled back along the mine floor. It was later determined, however, that the six fixed shovel teeth both loosened and removed the coal in a single operation, and the three adjustable ripping teeth and shanks, being superfluous, were removed from the bucket.

3.3.1.3 Dragline Loading Capability

As stated above, the six shovel teeth at the hoist end of the bucket worked very well in not only loosening the coal on the floor of the mine but also scooping the coal into the bucket as it was drawn along the floor of the mine. When the track cable was slackened, the bucket could be made to dig as deeply as required to completely fill the bucket with coal. In a production operation, there does not appear to be any problem which would hinder filling the bucket by this dragging method.

3.3.2 Projectile Tooth Tests

Tests were conducted with a single projectile tooth mounted on a carriage weighing 250 pounds. The carriage was released to travel 37 feet down a cable 22 degrees from horizontal and impacted into a coal or simulated coal target. About 5 inches of penetration was obtained during the initial blow to each block, and it was determined the tooth spacing could be wider than the anticipated 12 inches. Subsequent blows to the same impact point produced less penetration and more fines. To reduce the amount of fines and increase the amount of coal broken loose, a wider tooth spacing was found to be desirable. A discussion and data for the single-projectile-tooth tests are given in Appendix B.

3.3.3 Cable Anchor Tests

A number of methods for anchoring the track cable to the bottom of the pilot hole were considered. These included expandable wedge bolts, expandable rubber packers and various plastic, gypsum, and cement grouting systems. A cement grouting system, commonly used in mining and construction industries, was chosen as the method of securing the cable. Subscale tests were conducted to determine the required length of grout to give the cable holding strength to support the mining operation. It was determined from the subscale tests that a grout length of 20 feet would provide a nominal anchor strength equal to half the 100-ton breaking strength of the 1-1/2-inch wire rope or track cable. The nominal working load, applied by counterweight, was about 30 tons.

A complete discussion of the subscale cable anchor tests is given in Appendix C.

3.3.4 Surface Machine

To minimize the cost of the demonstration tests, the necessary surface equipment was leased from Bigge Crane & Rigging Co. The structure and hoist assembly shown in Figure 13 was much smaller than the anticipated production-mining surface equipment discussed in Subsection 3.2.

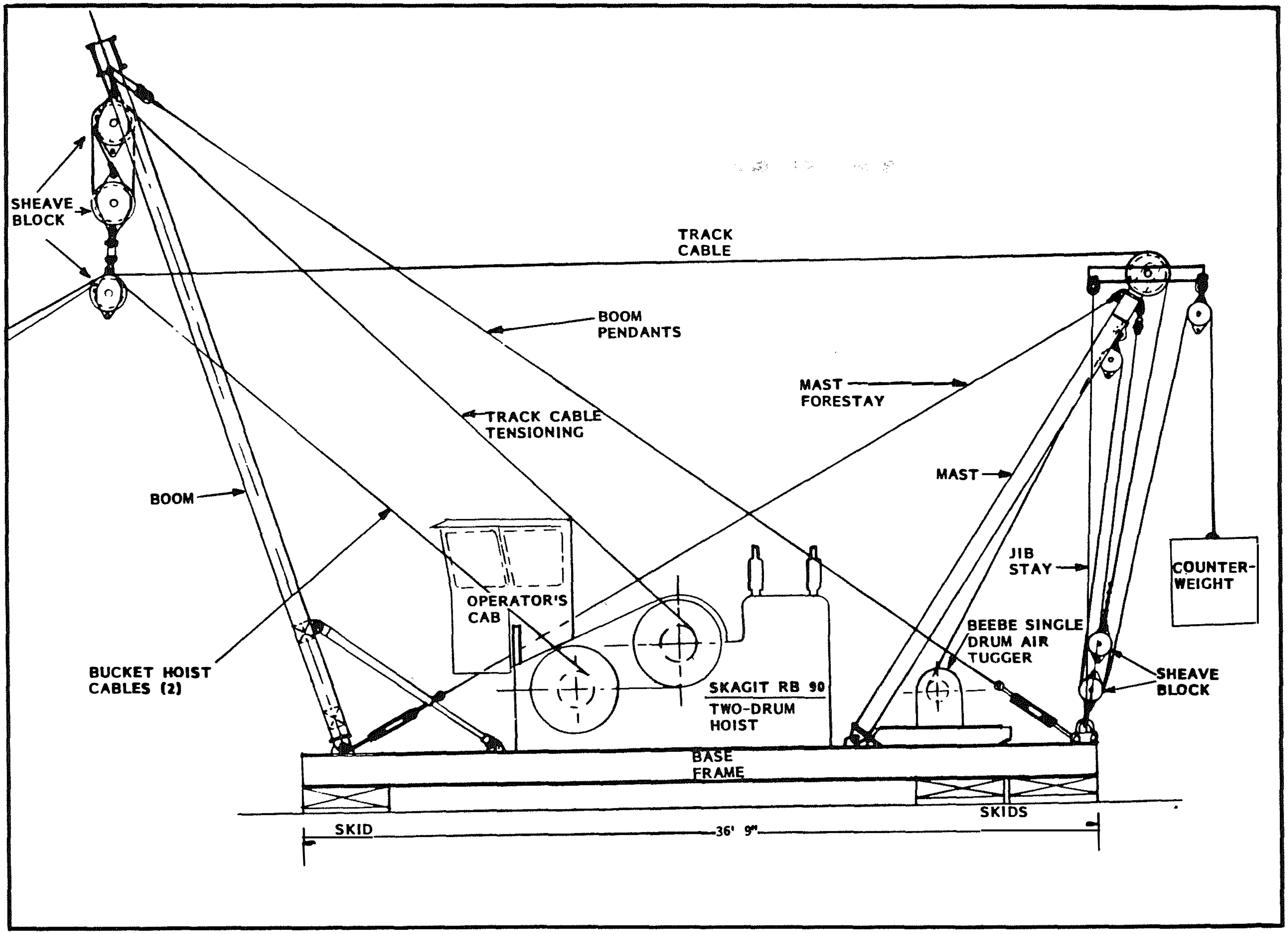


Figure 13 DEMONSTRATION TEST SURFACE MACHINE

The leased equipment included a Skagit RB90 diesel-powered, 3-drum hoist unit with capacities of 60 tons at 40 feet per minute and 5 tons at 330 feet per minute. The hoist unit included an enclosed operator's cab, which was a necessity for the cold weather encountered during the demonstration tests. A custom-designed and fabricated derrick structure and all the rigging (except the track cable) was included in the lease from Bigge Crane & Rigging Co.

3.3.5 Conveyor System

The anticipated conveyor system for production mining was discussed in Section 3.2. However, no conveyor system was included in the demonstration test mining setup because an adequate system did not warrant the additional cost. Instead, the coal was dumped in a pile at the top of the excavation and moved away with front-end-loading equipment.

3.3.6 Dumping Methods

During the demonstration test, two methods of dumping the coal out of the bucket were used. In the first method, coal was dumped out of the hoist end of the bucket by using a dumping horse. In the second method, coal was dumped out of the impact end of the bucket without the use of the dumping horse.

3.3.6.1 Dumping From the Hoist End

The coal was dumped out of the hoist end of the bucket by rocking it over the pivot pins of the dumping horse. Two problems became apparent with this method of dumping: there was not enough room under the dumping horse to dump a full bucket of coal; and engagement of the bucket with the pivot pins was awkward, frequently causing the horse to be knocked out of alignment, necessitating its being realigned. Figure 14 shows the bucket being emptied by using the dumping horse.

3.3.6.2 Dumping From the Impact End

Dumping out of the impact end of the bucket eliminates the need for the dumping horse and the time required to index the bucket on the pivot pins. It also allowed several buckets of coal to be dumped before the coal pile had to be moved with a front-end loader.

The bucket was pulled up the upper sheave block and held by the hoist cables while the track cable was relaxed. This tipped the bucket down and allowed the coal to run out of the impact end. Coal being dumped out of the impact end of the bucket is shown in Figures 15 and 16.

3.4 EQUIPMENT DEMONSTRATION

The equipment demonstration was divided into two test series. The first test series was set up in the bottom north end of the FMC Skull Point Mine's main production pit. When the demonstration test schedule was delayed and the mine's production operations schedule called for moving into the demonstration area, the HAMS equipment was moved out of the pit and set up in an unmined portion of the mine's main coal seam to the south of the main production pit.

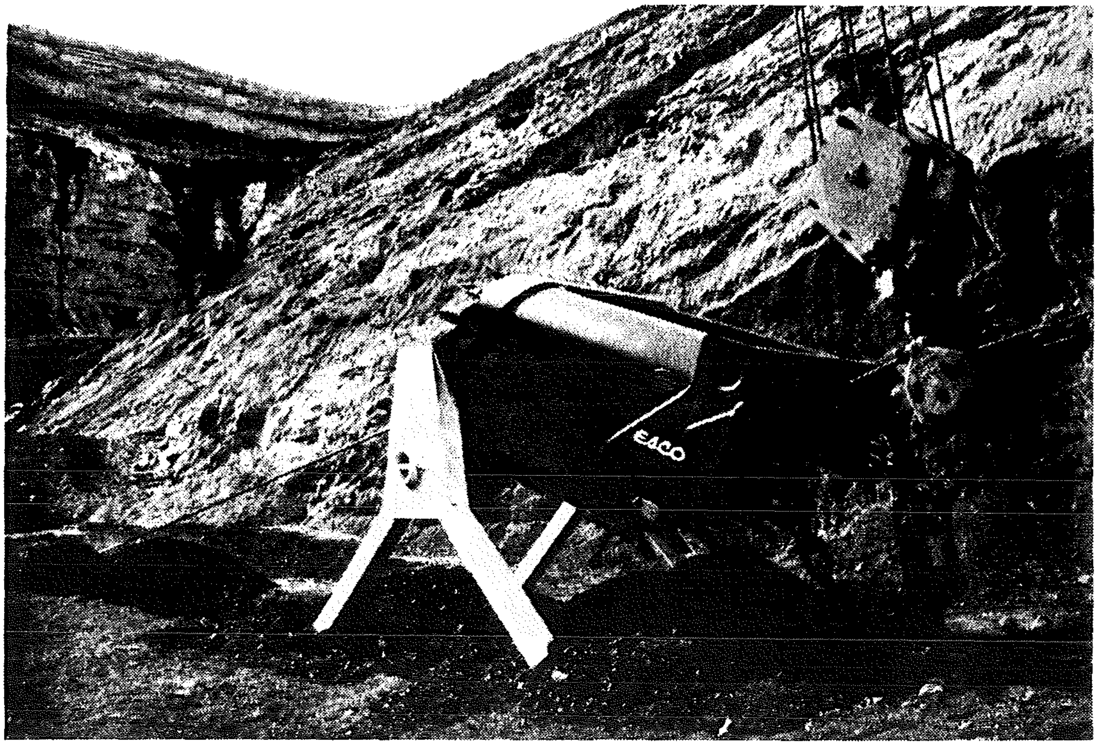


Figure 14 EMPTYING BUCKET USING DUMPING HORSE

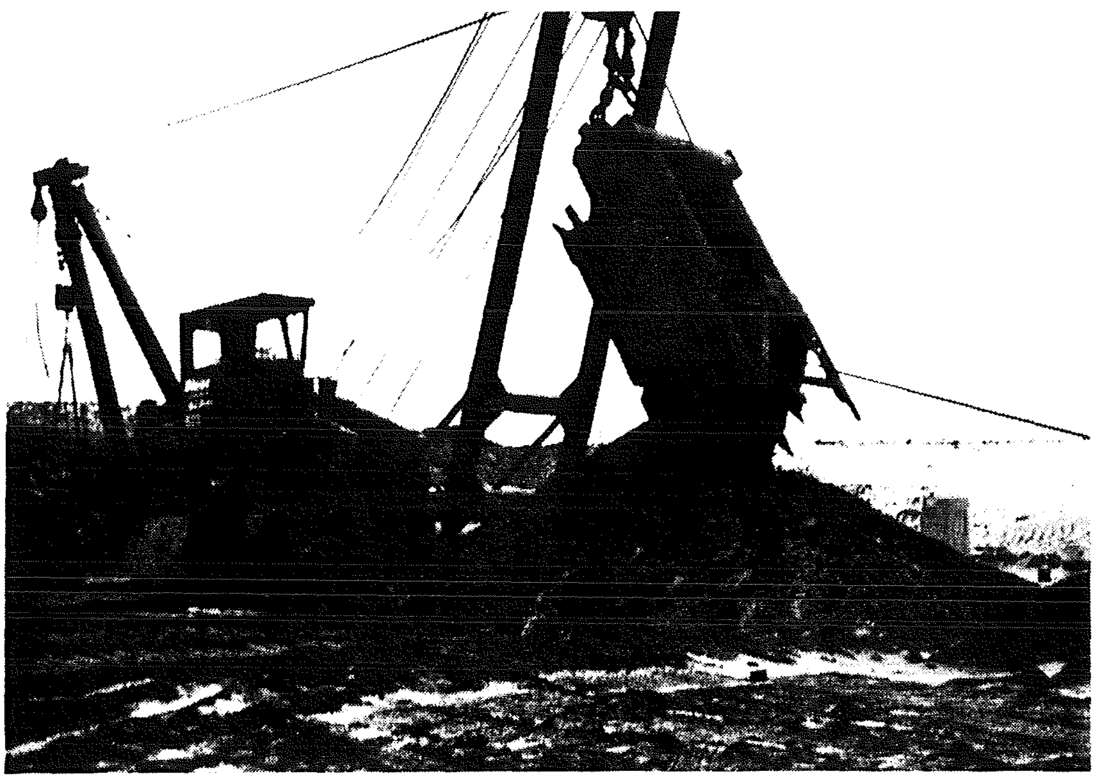


Figure 15 BUCKET HOISTED FOR EMPTYING FROM IMPACT END

Figure 16 **EMPTYING BUCKET FROM IMPACT END**

3.4.1 Initial Setup in Pit

Nine 4-inch-diameter holes were drilled into the main coal seam, known as No. One Upper, in the bottom of the pit. The holes were on a 21-degree angle to match the coal seam and were 5 feet from the top coal/sandstone interface. The holes varied from 115 to 365 feet deep, with the average about 200 feet. The 1-1/2-inch-diameter 6x37 class IWRC wire rope track cable was inserted into each hole and anchored in place by pumping metal/cement grout into the bottom 20 feet of each hole through a 1-inch-diameter plastic (PVC) tube which was then pulled out of the hole. Each hole had 135 feet of wire rope protruding from it to allow connecting and working distance to the hoist rig. The grout length of 20 feet provides a nominal anchor strength equal to half the 100-ton breaking strength of the wire rope. The nominal working load, applied by counterweight, was about 30 tons.

A 50-foot-wide roadway was constructed parallel to the hole centers to accommodate the hoist rig. A berm, or escarpment, was created on the side of the roadway nearest the drilled holes to react the horizontal pulling loads of the hoist rig.

The prefabricated hoist rig, bucket, and carriage were assembled and ready for free-running operation after 2 days' employment of a local crane and rigging company. Figure 17 shows the mining rig set up in the pit.

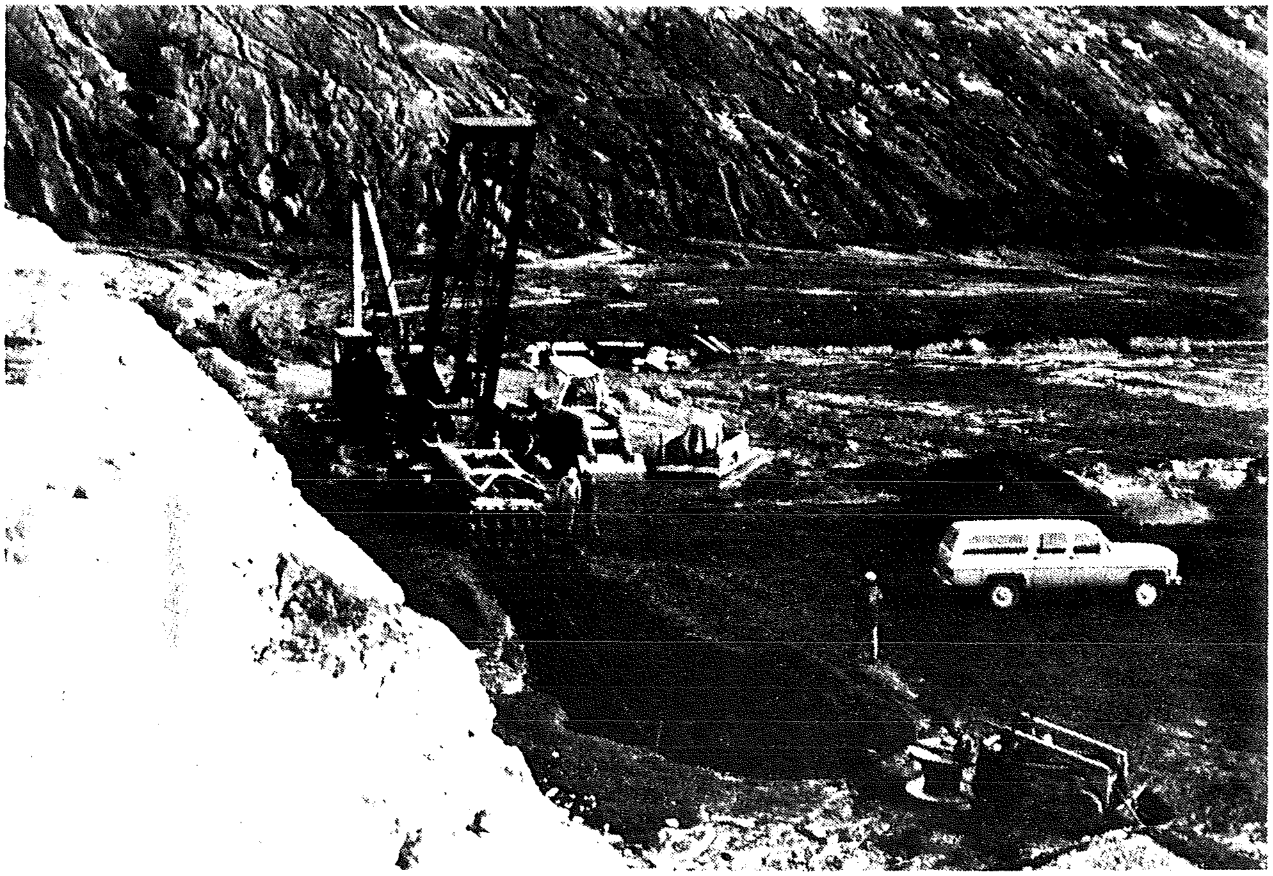


Figure 17 MINING RIG SET UP IN PIT

3.4.2 Feasibility Mining

During the first week of mining, the selected concept proved it had potential as a practical method of mining. An open trench about 7 feet wide, 30 feet long, and 6 feet deep was excavated during the week even though actual operating time was only a few hours each day. The bucket appeared to perform well in the ripping/loading mode and to break up large amounts of coal during impact. However, it became apparent that only a fraction of the coal broken loose during impact was being captured by the bucket and removed, and broken coal accumulating at the face would eventually be pulverized by successive bucket impacts.

3.4.3 Equipment Modification

From observation of the bucket behavior during many mining cycles and analysis of the impact process, a number of problems were recognized and several modifications to improve bucket performance were recommended. The modifications which could feasibly be made at the test site were the first to be effected.

3.4.3.1 Recognized Problems

The problems which were observed were as follows:

- When the bucket was released to free-roll down the track cable, it did not have the correct attitude or orientation to impact the coal correctly. The impact end of the bucket was too low, so the bucket was not level during impact.

- The centerline force of the impacting bucket was much higher than the bucket's lower teeth which first impact the coal, causing the bucket to pitch forward or try to somersault until restricted by the track cable.

- Because the impact end of the bucket was too low, the bucket would not scoop up the coal at the excavation face, but instead pushed the coal like a plow into the working coal face without scooping it into the bucket.

- Any coal which was picked up by the impact end of the bucket mostly fell out as the bucket was hauled back.

3.4.3.2 Recommended Modifications

Several modifications were considered which could improve bucket performance by partially eliminating the observed problems. These modifications were as follows:

- The bucket should be modified to lower the center of gravity and bring it more in line with the impact of the lower teeth to give a scooping action instead of a pitching action.

- A new bucket carriage should be designed and fabricated to allow the attitude or orientation of the bucket to be adjustable. This would allow various attitudes to be selected, and the bucket performance for the various attitudes could be evaluated.

- The center and/or top row of teeth should be extended to make them contact the coal face at the same time or before the bottom row of teeth. This would position the center of gravity of the bucket behind the teeth penetrating the coal.

- Several minor modifications should be tried to help keep the coal from sliding out of the bucket, i.e., baffles, grids, etc. If these modifications fail, a clamshell concept should be pursued.

Because some of the desired modifications would require complete bucket redesign and fabrication, partial modifications were initiated in an attempt to evaluate any improvement in bucket performance.

3.4.3.2.1 Tooth Arrangement

The middle row of six teeth were extended by about 14 inches by welding 4-inch-wide grader cutting blades over the top of the original teeth. The blades were pointed by 45-degree cuts on each side and braced to prevent bending. The pointed blades extended about even with the bottom teeth. The extended teeth contacted the coal shortly after the bottom teeth, and the bucket pitching was reduced for several impacts. The coal face eroded after repeated impacts and the bucket again pitched, but not as severely as before the blades were added.

The four top teeth were then extended by the same method so the teeth were even with the extended middle row and the bottom row of teeth. Again the extended teeth reduced the bucket pitching until the coal face eroded and the added-on teeth no longer contacted the coal face at the same time as the bottom row of teeth.

The reoccurrence of the bucket pitch appeared to be caused by the additional dislodged coal from the added-on teeth falling in front of the impact end of the bucket, which was not oriented correctly to scoop up the coal. Thus, even though changing the teeth would help bucket advancement for a few impacts, it became apparent that it was not a permanent improvement unless the bucket orientation was also corrected.

3.4.3.2.2 Bucket Balance and Attitude

An attempt to change the bucket attitude as it impacted the coal face was first made by raising and lowering the track cable. With the track cable slackened by the air tugger and lowered by the head sheave, it was hoped a sufficient scooping action could be obtained to prevent the bucket from pitching up when impacting the coal. The bucket performance was slightly better, but lowering of the hoist end of the bucket was still needed to give better scooping action.

An attempt was then made to change the balance of the bucket by adding weight to the hoist end of the bucket and removing some of the original teeth from the center area of the impact end of the bucket. The teeth which were removed were not making contact with the coal face upon impact and thus did not affect bucket advancement.

The weight added to the hoist end of the bucket was in the form of lengths of used railroad rails which were very brittle and difficult to weld. Despite this difficulty, as much as 1,500 pounds of additional weight were added. The weight did help balance the bucket so the approach attitude was flatter, and performance improved. However, the railroad rails were so brittle they would break off after only a few impacts and had to be replaced repeatedly. The additional material welded on the hoist end of the bucket also hindered the dragging operation of the bucket. After several attempts to balance the bucket by adding weight, it was determined that even though performance was improved, adding weight to the present bucket was impractical. Attention was turned to the carriage as a method of varying the bucket attitude, and the design of a new carriage which could be varied was initiated while testing was continued with the original fixed carriage.

3.4.3.2.3 Dumping Methods

The original dumping method was to set the bucket over the pivot pins of a dumping horse (see Figure 14) and relax the cables so the bucket would tip and dump out the hoist end. There was some difficulty positioning the recesses in the bucket over the pivot pins on the dumping horse, and frequently the dumping horse was knocked out of line and had to be repositioned. By removing the dumping horse from the dumping area and pulling the bucket to the top sheave blocks while relaxing the track cable with the air tugger, it was possible to dump the bucket contents out of the impact end (see Figures 15 and 16). Dumping the coal from a higher position allowed more coal to be dumped before it became necessary to clear the coal from the dumping area. This method also made it easier to clear the coal from the dumping area with a small front-end loader, as the dumping horse was no longer in the way to restrict loader movement.

3.4.3.2.4 Baffles and Clamshell

The bucket was originally designed with front and internal baffles to prevent the coal which was captured in the bucket from spilling out while the bucket was being hoisted out of the mine for dumping. Although these original baffles worked well in retaining the coal, the combined attitude of the bucket and angle of the front baffle prevented the coal from being scooped into the bucket as anticipated.

Because it was evident that the coal was being pushed ahead of the baffle instead of flowing over it as intended, the front baffles were removed to allow the coal to be scooped into the bucket. Removing the front baffles did help in loading coal into the impact end of the bucket; however, most of the coal then fell out as the bucket was being hoisted back from the coal face and out of the mine.

Several methods were then tried to minimize restriction of coal flow into the bucket and maximize retention of the coal until the bucket was out of the mine and in the dumping area. Small strips of expanded metal were welded on the floor of the bucket close to the impact end. These acted as riffle plates to reduce the coal sliding out of the bucket when it was hoisted out of the mine. The expanded metal did help hold the coal stable while the bucket was hoisted from the mine, but it also hindered dumping the coal at the dump area.

As the mine became deeper, the water in the bottom of the mine also became deeper, and the problem of holding the coal increased. Several types of baffles were experimented with in an attempt to hold both dry coal and wet coal in the bucket. Only the baffles which completely closed off compartments in the bucket kept the slurry of water and coal from running out of the bucket while it was being hoisted out of the mine.

It was determined that some type of gate or clamshell system was required to allow coal to be pushed into the bucket upon impact, and then to close and retain the coal and water slurry while the bucket was hoisted out of the mine. A clamshell concept was designed, fabricated, and adapted to the bucket (see Figures 18 and 19). A latching spring was designed to hold the clamshell open until the bucket impacted the coal face. Upon impact, the clamshell snapped free to close and capture the coal which was in the bucket.

This concept worked very well to scoop up the slurry of coal and water, but much of the water drained out of the captured slurry and ran back into the mine as the bucket was being pulled up. The clamshell also worked well as a dumping gate to release the coal. This was accomplished by pulling the clamshell locking lanyard when the bucket was in the dumping area.

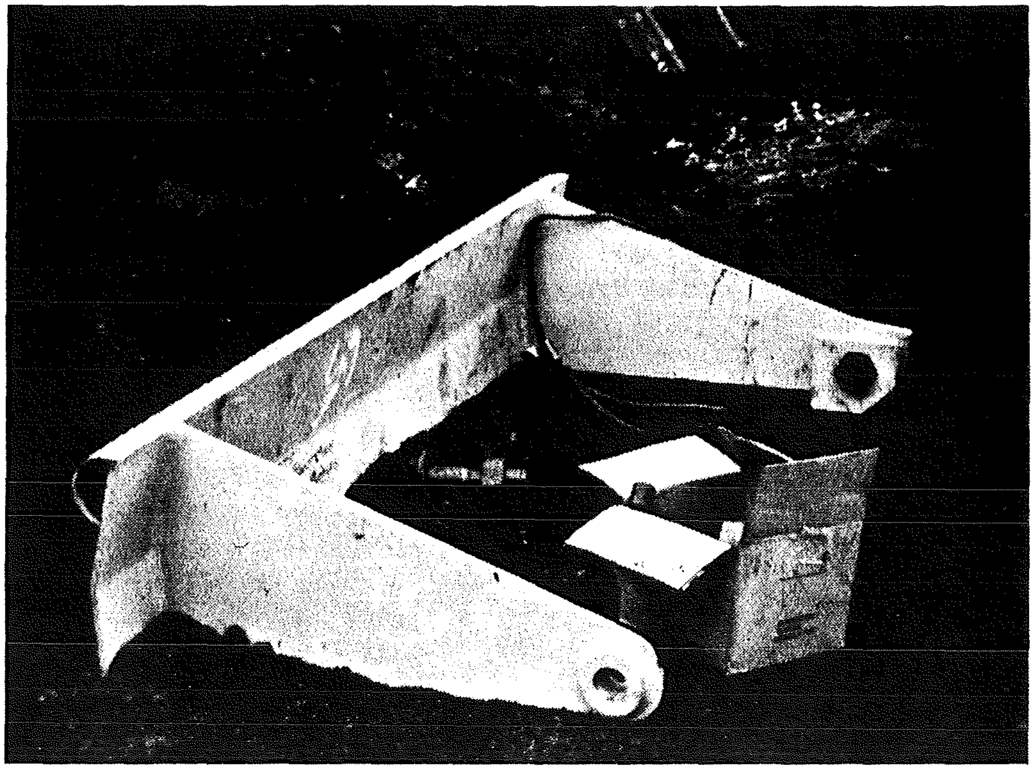


Figure 18 CLAMSHELL

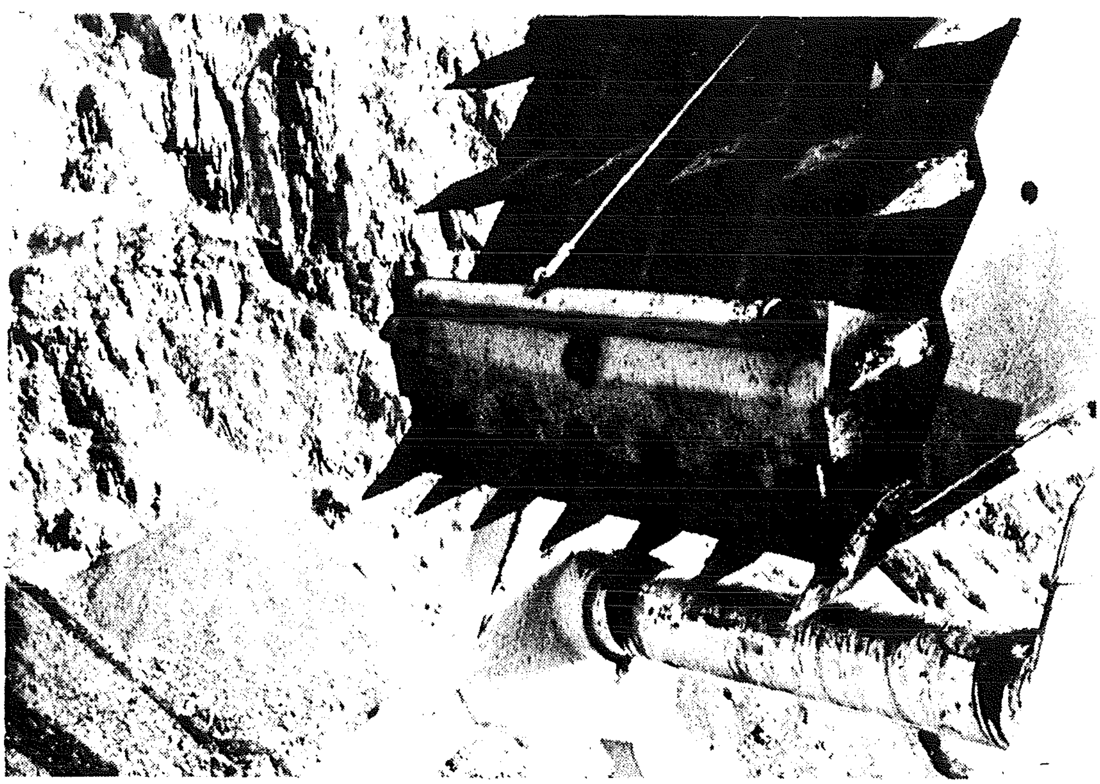


Figure 19 CLAMSHELL ON BUCKET

In the final analysis, because the water in the mine acted as a brake to the bucket impact advancement, it was concluded that the projectile bucket concept does not work well in flooded conditions.

3.4.3.2.5 Machine Anchors

The original production machine was designed to be positioned against the mine's pit highwall with a large support pad resting against the highwall to keep the machine stable during mining operations. When the decision was made to mine in a seam in the floor of the pit away from the highwall, a berm, or escarpment, about 2 feet high was created on the operating side of the prepared roadway to react to the horizontal pulling loads of the hoist rig. The machine's front skid was positioned against the berm.

Because the berm was softer on one side than the other, the pulling loads soon caused the machine to pivot slightly so the track cable leading to the back of the machine was no longer in line with the excavation. After realigning the machine several times, workers dug two holes measuring 4 feet by 6 feet by 6 feet deep at 45 degrees out from the back corners of the machine, and 4 yards of cement were poured into each hole to act as anchors for the machine. Wire rope, 1-1/2 inches in diameter, was imbedded in the cement to provide attachment points.

After the machine was attached to the anchors, correct alignment with the excavation was maintained.

3.4.4 Upper Seam Outcrop Setup

Due to accelerated production in the pit floor, the mine requested that the test rig be relocated to another site on their property. The new site was on an undeveloped area where a 50-foot-thick seam naturally outcrops at about 21 degrees. Approximately 15,000 cubic yards of topsoil, overburden, and weathered coal were removed by the mine in preparation to begin mining with the High Angle Mining System.

Four-inch-diameter track cable holes for two positions were drilled about 25 feet apart. The first was drilled to a depth of 200 feet down dip. About 2 feet beneath this, in the plane of the seam, a 12-inch-diameter "kerf" hole was drilled to a depth of 50 feet. The intent of the kerf hole was to provide a void for coal to break into. At the adjacent test position, the track cable hole was drilled to a depth of 105 feet. A 2-1/2-inch-diameter blast hole was drilled a few feet beneath this second track cable hole to a depth of 60 feet. Both track cables were inserted and grouted in place. The 60-foot blast hole was then loaded with explosive and blasted to determine the effect on projectile bucket mining of loosened coal around the track cable.

3.4.5 Adjustable Suspension Carriage

During the new site development, installation of track cables, and equipment move, a new suspension system, which permitted adjustment of the bucket's attitude, was designed and fabricated at the ESD Corporation facility in San Jose, CA. These components were shipped to the new test site and arrived in time to be retrofitted to the bucket prior to initiation of mining at the new site. The new adjustable carriage made a significant improvement in bucket performance. With the capability of changing the attitude of the bucket and observing the effects of each attitude change, an optimum or near-optimum bucket attitude was soon achieved. The new suspension carriage is shown in Figure 20.

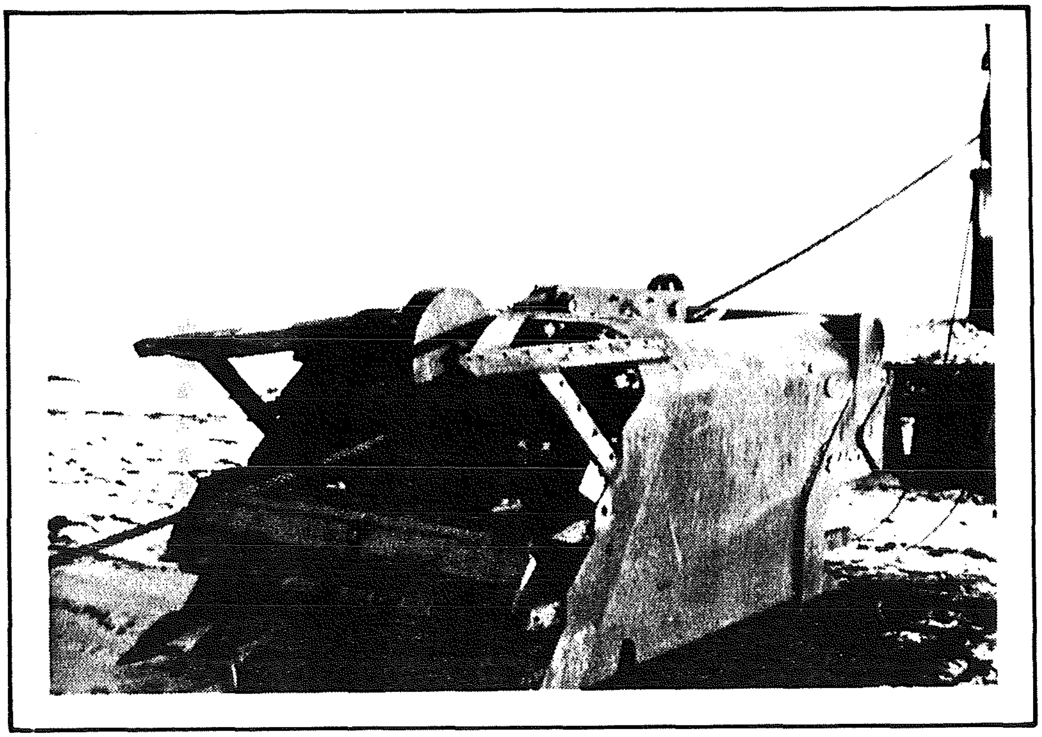


Figure 20 **ADJUSTABLE SUSPENSION CARRIAGE ON BUCKET**

3.4.6 Upper Seam Outcrop Mining

Although the upper coal in this area was very weathered, or decomposed, there was a gradual transition to solid coal as the excavation was made deeper. While mining was being conducted in the weathered or softer coal, an average of 5 or 6 inches advancement was achieved with each bucket impact. However, the bucket had not as yet progressed to mining underground, but rather was mining a trench. At this stage, the weathered or soft coal compacted into the bucket upon each impact and had to be dug out with a shovel during the dumping operation.

When the mining progressed until the bucket was underground, the composition of the coal improved and became solid. The dumping operation also improved as the composition of the coal improved.

At the start of the mining operation, at the point that the bucket first started to enter the ground, the track cable was located about 6 feet below the top of the 50-foot-thick coal seam. As mining progressed, the distance from the top of the coal seam increased, due partly to a slightly steeper angle of the track cable hole and partly from the longer bucket run allowing more deflection in the track cable to cause a lower bucket impact. During the final days of demonstration testing, the distance from the top of the bucket to the top of the coal seam, after impact at the bottom of the hole, was 12 feet. The impact of the bucket during mining jarred loose all the coal from the point of impact up to the sandstone layer over the coal seam. As a result, as the mining progressed deeper, more coal was removed by the bucket impact while the bucket advancement appeared to be less. Even though each bucket impact showed only 2 inches of advancement by actual measurement, the amount of coal removed from the face of the mine was equivalent to 6 inches of coal. The coal jarred loose by each impact fell into the mine at the coal face as the bucket was hoisted out. It was then scooped up during the next impact. The loose coal which fell at the face of the mine acted as a buffer and did not allow the bucket to penetrate as far as it would have if there had not been loose coal at the face of the mine.

As the mine progressed deeper, the coal just above the bucket seemed to try to form an arch and limit the amount of coal falling into the mine. However, after continued impacts, the coal would again jar loose and fall free from the point of impact up to the sandstone roof. It is anticipated that, if mining had continued, a lower roof would finally form and the advancement per impact would be increased. The configuration of the mine at the termination of the demonstration test is shown in Figure 21.

3.4.7 Causes of Delays in Progress

Progress of the mining demonstration was severely hampered by several surface rig breakdowns and problems developing from the freezing weather. It is estimated that less than one-fourth of the demonstration time was spent in actual mining.

3.4.7.1 Frequent Delays

The most common (and recurring) delays were as follows:

- Inability to get the surface rig equipment to start because of the cold.
- Ice in the air lines which would not allow the hoist brakes to release.
- Equipment moving out of line and requiring realignment before mining could continue.
- Dirt in the hydraulic check valves, requiring disassembly and cleaning.
- Inability to start support equipment in freezing conditions to remove coal from dumping area or to furnish compressed air for air arc or impact tools.
- Chains and cables breaking, twisting, or pulling loose and requiring replacement, straightening, or repair.
- Dirt in diesel fuel necessitating line and filter cleaning.
- The haul-back chain-link assembly becoming tangled and requiring a crowbar to untangle.
- Rain, snow, and ice making poor working conditions.

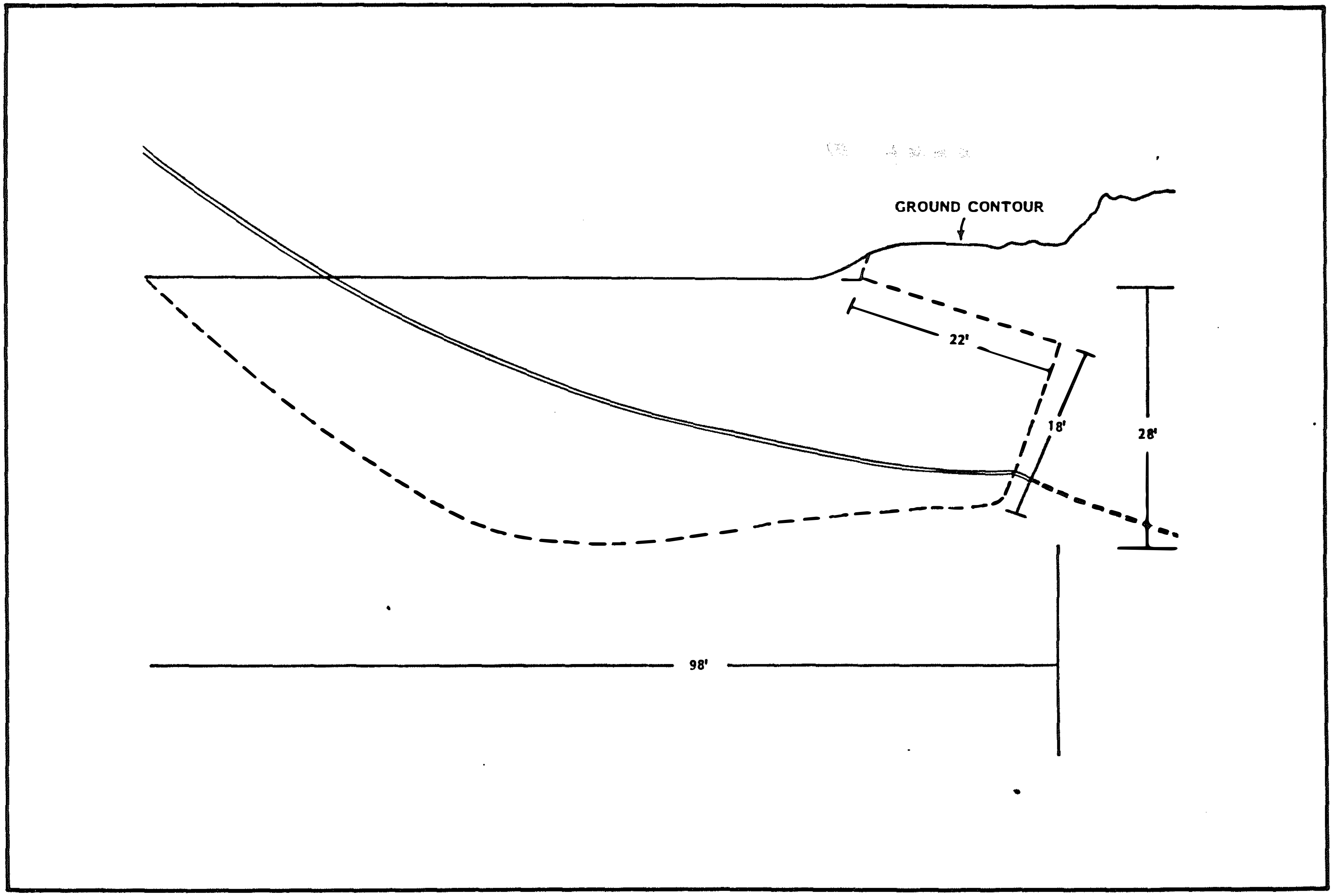


Figure 21 CONFIGURATION OF MINE AT TERMINATION OF DEMONSTRATION TEST

3.4.7.2 Major (Single-Occurrence) Delays

Two major surface rig breakdowns occurred, as follows:

- The main support frame of the surface rig bowed down in the middle and required 4 days to straighten and reinforce.

- The main mast support braces both buckled, allowing the mast to fall back over the surface rig. The mast and surface rig sustained extensive damage, and it was estimated that 10 to 14 days would be required to repair the damage.

Since the demonstration tests were scheduled to be terminated in two weeks, no attempt was made to repair the damaged surface rig, and preparations were initiated to close down the demonstration operations.

3.5 SLACKLINE EXCAVATOR CONCEPT WITH MOTORIZED BUCKET

Evaluation of the problems encountered while mining with the projectile bucket led to the conclusion that these problems could be eliminated by adapting a motorized mining head to the bucket. The concept for such a motorized bucket is described in the following subsections. A productivity analysis of the slackline excavator with motorized bucket is given in Appendix D.

3.5.1 Rationale for a Motorized Bucket

As discussed earlier, the presence of water in the face area of the development (hole) will seriously affect advancement of a projectile bucket by absorbing much of its energy. However, a motorized bucket using components of proven mining equipment can operate in or under water and advance the development at a much faster rate.

Thus, it seems practicable to use a motorized bucket to develop the initial hole and a conventional dragline bucket to excavate the remaining coal in the seam. Once the development of the first hole has been completed, the motorized bucket will move to an adjacent embedded track cable and proceed to develop another hole while the dragline bucket excavates coal at the preceding development.

Significant improvement in the advancement rate of the projectile bucket beyond 3 inches per blow is not expected, although some improvement may be possible. On the other hand, the motorized bucket can advance up to 12 inches or more, if adequate power is provided to shear coal to that depth. Furthermore, the advancement rate for the motorized bucket is relatively more constant since it can function under water almost equally as well as out of water.

Advances of this magnitude for a motorized bucket are predictable because cutter chains, which would be used on the bucket, have been used on several models of continuous miners over a period of many years. Their rate of advancement is well known. Although operating conditions in a high angle mining situation are somewhat different, they are not expected to present any difficult or unsolvable problems.

3.5.2 Design and Operation of Motorized Bucket

3.5.2.1 Description

The motorized bucket (see Figure 22, Sheets 1, 2, and 3) contains two 30-inch-wide cutter chains to shear coal from the face. Each chain is mounted on a bar which pivots on a common tubular drive shaft. Sprockets attached to the drive shaft drive the cutter chains. The tubular shaft is supported by the bucket sidewalls using bolt-on stub shafts and self-aligning spherical roller bearings. The drive shaft is driven by a 2:1 roller chain drive at each end. A 10.05:1 planetary gear reduction unit driven by a fixed-displacement hydraulic piston motor drives each roller chain drive. At the other end of each cutting chain bar, separate shafts are mounted on self-aligning take-up bearings. Each shaft has a sprocket to guide the cutter chain, drums to support the outer edges of the chain, and a drum with conical cutter bits on the outboard end. One shaft has a face-core breaker mounted on the inboard end. A 5-inch-diameter hydraulic cylinder raises both bars as the cutter chains shear coal from the face. Another hydraulic cylinder, by means of a device that grasps the track cable, forces the bucket into the coal face when the cutters are shearing coal. The bucket is suspended by and travels on two 24-inch sheaves similarly to the projectile bucket. Bridle attachment and bridle for attaching the hoist cable are the same as for the projectile bucket. Bucket rated-cubic-yard capacity is approximately the same. A double hose reel is located on the surface to reel in the fluid supply and return hoses as the bucket is raised from the hole. Fluid power is provided by a hydraulic power unit also located on the surface.

3.5.2.2 Components

Many of the components that "motorize" the bucket are commercially available. Rough-cut calculations to size some of the components selected are included in Appendix E. Some of the major components are as follows:

- Cutter chain - This is a 30-inch-wide chain with one or a pair of cutter bit blocks welded to each chain link. Carbide-tipped conical cutter bits are secured in each bit block. The cutter chain, called a "Ripperveyor,"[R] is available from Joy Manufacturing Co. and is used on their 10CM, 11CM, 12CM, 12HM, and 14CM models continuous miners. Sprockets for driving the cutter chain are available with hub configured to meet customer requirements.

- Planetary gear reduction - This is an American Lohman Corporation unit selected because it is more compact than other gear reduction types. A motor spacer box can be included for direct mounting of the drive motor.

- Drive motor - This is a fixed-displacement piston motor made by Denison Division of Abex Corporation. The motor is rated at 5,000-psi-maximum continuous-operating pressure and a flow rate of 45 gpm at 1,000 rpm. Maximum theoretical torque is 8,200 lb-in. at 5,000 psi.

- Bearings - The drive shaft is mounted on self-aligning spherical roller bearings manufactured by McGill Manufacturing, Inc., SKF Industries, Inc., or others. The cutter chain bars use plain cylindrical/flanged bronze bearings where the bars attach to the drive shaft for support.

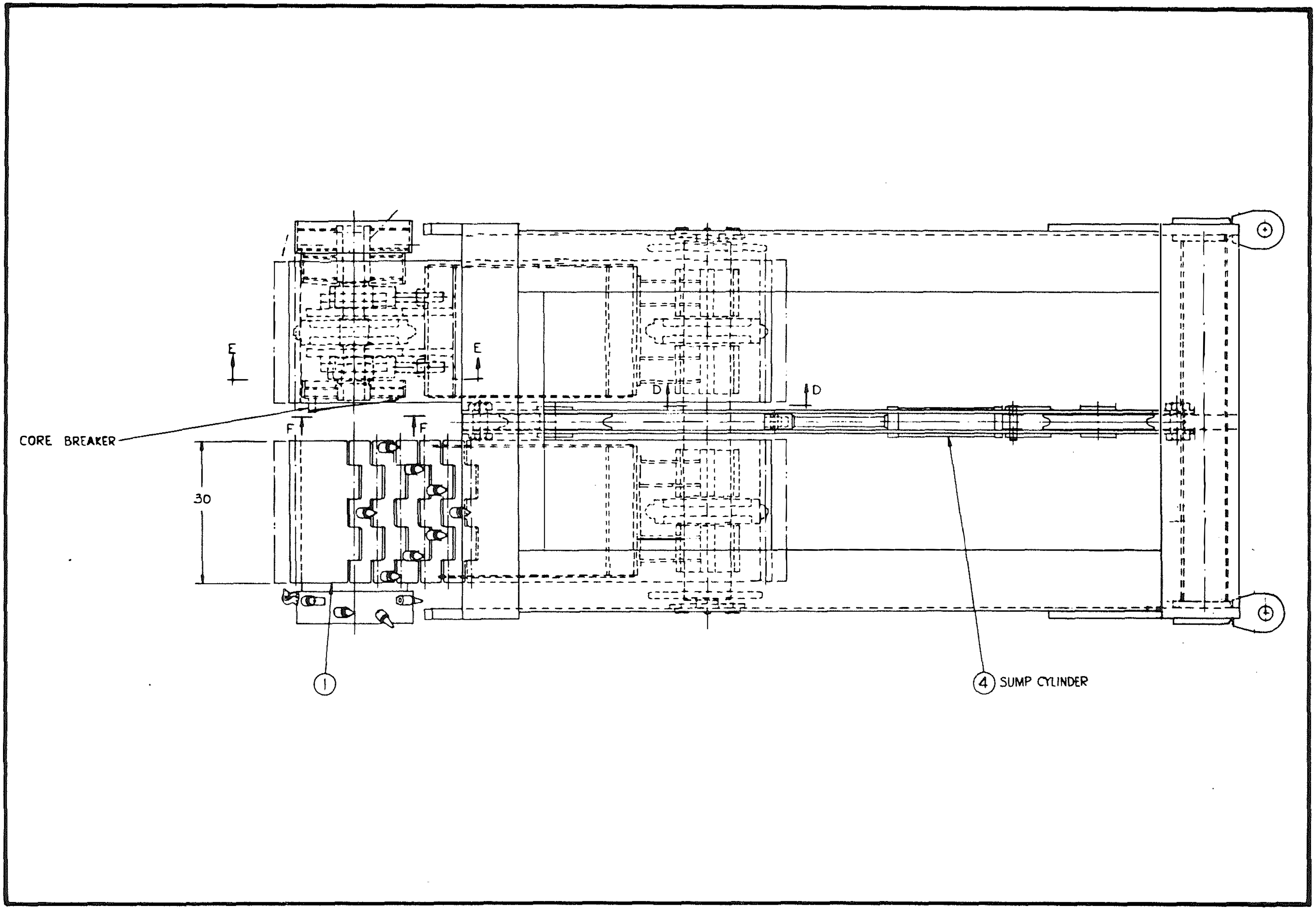


Figure 22 MOTORIZED BUCKET (Sheet 1 of 3)

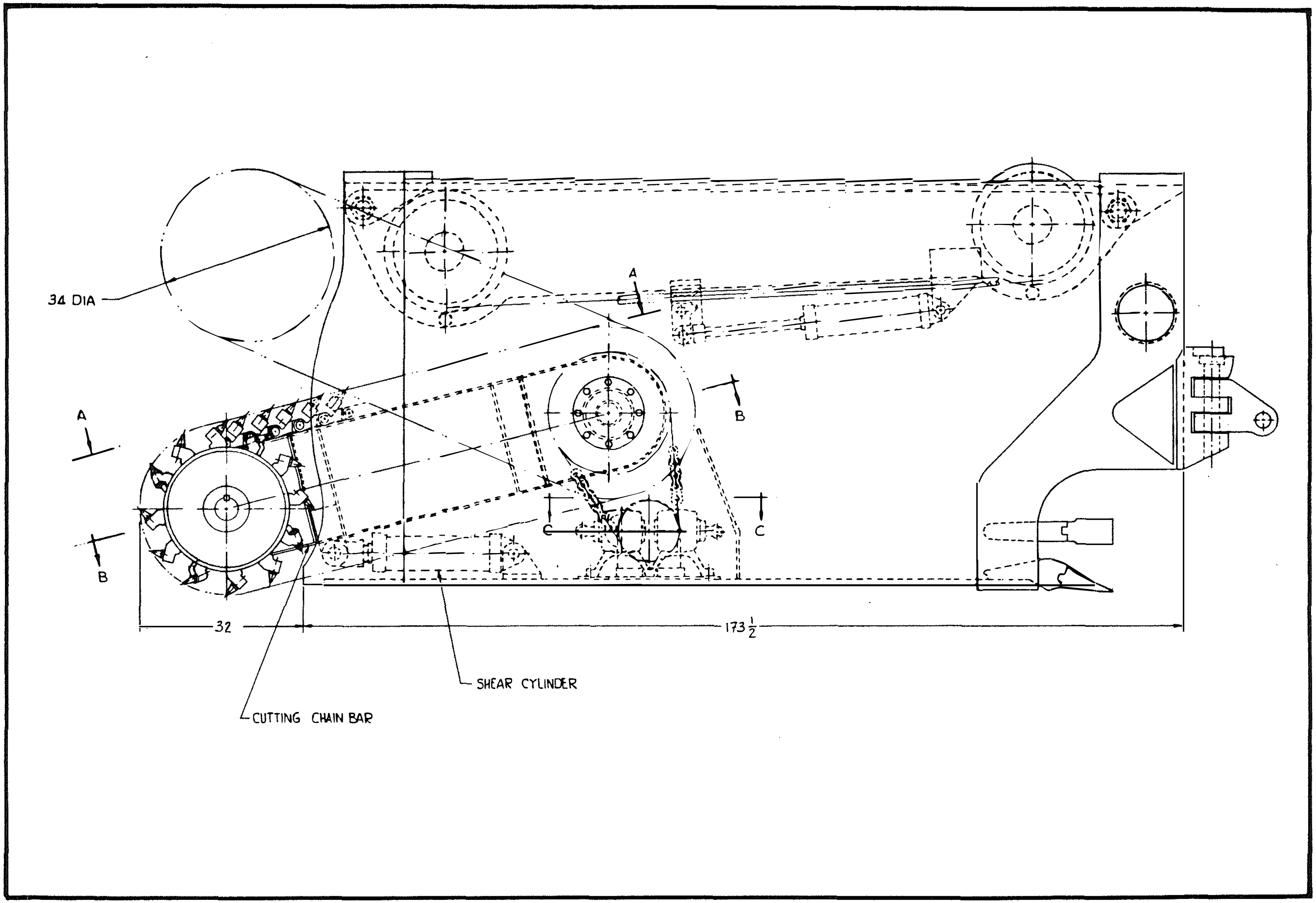


Figure 22 MOTORIZED BUCKET (Sheet 2 of 3)

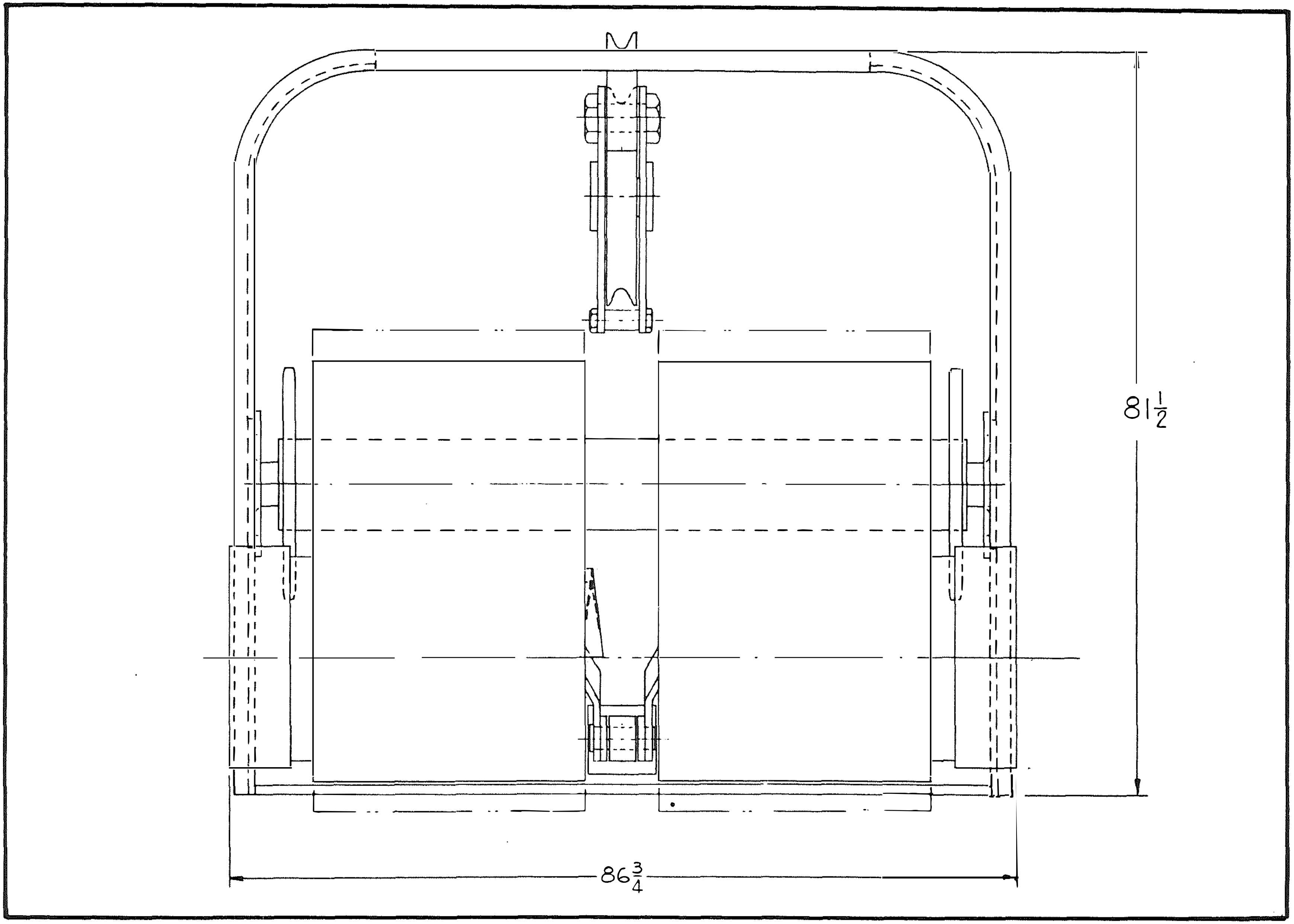


Figure 22 MOTORIZED BUCKET (Sheet 3 of 3)

These bearings are available on order from several manufacturers, including Magnolia Metal Corp., Marion Bronze Co., and Wisconsin Centrifugal Inc., to name but a few. The two driven shafts are mounted on self-aligning take-up bearings made by Link-Belt and others.

- Hydraulic cylinders - These are rated for 3,000-psi service or 5,000-psi non-shock load service. These cylinders are available from numerous manufacturers. Each hydraulic cylinder has an internal spring that causes the cylinder rod to extend or retract, depending upon cylinder application, when fluid pressure to the cylinder is released.

- Bucket sump mechanism - This device grasps the track cable when activated by a hydraulic cylinder. Once the cable has been firmly grasped, the hydraulic cylinder can exert force against the bucket to move it along the cable and against the face.

- Bucket - The projectile bucket used throughout the demonstration is too small to contain the items described above. Therefore, a new bucket is required for the motorized version. Since impact loading is greatly reduced for the motorized bucket, the basic box can be constructed of thinner material.

- Suspension - The bucket travels on two sheaves mounted between steel plates bolted to the bucket. The arrangement is essentially the same as that used on the projectile bucket.

- Hydraulic components - Various other hydraulic components will be required, some located on the bucket and others on the surface.

  - A pump and reservoir located on the surface supplies fluid under pressure (up to 5,000 psi) to the drive motors on the bucket.

  - Two large hoses (1-1/2-inch inside diameter) supply fluid to and return fluid from the drive motors. A hose reel located on the surface is needed to reel in each hose when the bucket is raised to the surface.

  - A scavenger pump located on the bucket may be required if backpressure on motor case drains exceed the manufacturer's recommendations. Otherwise, the case drain will dump into the large motor-return line.

  - Some of the fluid flow to each motor must be diverted to the hydraulic cylinders for raising the cutter chain bars and keeping the cutter chains forced into the coal face. Pressure-reducing valves and check valves may be required to reduce fluid pressure to the cylinders. Initially this arrangement would be tried without the use of flow direction control valves for each cylinder. A sequence valve is needed to cause the sump cylinder to stroke before the shear cylinder strokes. This sequence provides a sump cut before the shear cut begins.

3.5.2.3 Theory of Operation (Operating Scenario)

In the original concept for high angle mining, the coal seam is exposed at a highwall resulting from surface-mining operations. In this situation, starting the hole with the motorized bucket needs no prior preparation of the coal seam; in other situations it may be necessary to prepare a face for the cutting chains to start against. A typical operating scenario is given in the following paragraphs.

Daily operation begins by lowering the bucket down the track cable to the face. As the bucket is being lowered, the hose reel pays out both hoses. When the bucket has reached the face, hydraulic fluid under pressure is directed to the sump cylinder and the cutter chain drive motors. At this moment the cutter chains start operating and the sump cylinder in its first 4 to 6 inches of stroke causes a device to securely grip the track cable. The remaining 14 to 16 inches of stroke force the bucket and cutter chain to sump into the coal face. When the sump cylinder can stroke no farther, a sequence valve allows fluid to flow to the shear cylinder. The shear cylinder then forces the cutter chain to shear upward along the coal face by pushing on both cutter chain bars. As the cutter chain bits shear upward and into the coal, chunks of coal are broken off. These chunks drop onto the cutter chain and are carried to the rear of the bucket.

Once the cutter chain bar has ranged vertically full stroke, it needs to be returned to the starting point. This is accomplished by stopping the fluid flow to the bucket-mounted hydraulics. A spring in the shear hydraulic cylinder, along with the weight of the cutter chain and bar assembly, causes the assembly to return to the starting position. The cable-gripping device also releases and a spring in the sump cylinder retracts the cylinder rod, pulling the gripping device with it. If feasible, another shear pass up the coal face can now be repeated by directing fluid flow to the bucket-mounted hydraulics.

Periodically the bucket must be raised to the surface by the hoist rope. As the bucket is hoisted, the reels take up the two hoses. Dumping of coal is done in the same manner as used with the projectile bucket.

3.6 ECONOMIC EVALUATION

A cost analysis and comparison of the various augering systems and the slackline excavator concepts, both projectile bucket and motorized bucket, was conducted on a per-ton basis. The two slackline concepts were similar in total cost of capital investment and operating expenses (the motorized bucket being somewhat lower in cost) and also compared quite favorably with the auger systems. Additionally, of course, as established elsewhere in this report, only the slackline excavator concepts are capable of operating through the full range of seam pitch required of the High Angle Mining System. Details of the cost analysis and comparison are given in Appendix F.

Further, the slackline excavator concept is capable of producing coal at a cost lower than that of surface-mined coal. This may permit the final highwall to be pushed back beyond the normal economic stripping limit.

The cheaper cost of the coal produced by the High Angle Mining System would offset the more expensive costs of the surface-mined coal. Any additional coal that is economically recoverable by surface mining would add to the reserve base attributable to HAMS. An analysis of the slackline productivity for the projectile bucket is given in Appendix G.

4.0 CONCLUSIONS AND RECOMMENDATIONS

As a result of work performed under this contract, it can properly be concluded that the High Angle Mining System, as conceived and tested by ESD Corporation, is a feasible method for the mining of coal from seams varying widely in thickness and pitch. The system should be employed to extract coal lying beyond the present economic limit of a strip mine or open-pit highwall, thus allowing the capture of large coal reserves which are otherwise not economical or practical to surface mine.

It is further concluded that, while the projectile bucket can mine coal at a reasonable rate, the incorporation into the system of a motorized bucket head will increase production rates and make production more predictable. Guiding the motorized bucket by the preinstalled track cable will give positive control of mining direction and will ensure the desired spacing between excavations.

ESD recommends that, as funding becomes available, a full-scale High Angle Mining System be fabricated and incorporated into the normal production of a mine for a 1-year evaluation period. Such a program should include both the motorized bucket for driving the initial excavation to the bottom of the track cable and the dragline bucket to remove the coal in a slot shape to the bottom of the coal seam.

The system should also include a conveyor system and a storage hopper for rapid removal of the mined coal. This will allow a realistic evaluation free of delays caused by coal piling up and preventing normal bucket dumping.

Appendix A

ALTERNATE METHODS EXAMINED

APPENDIX A

ALTERNATE METHODS EXAMINED

A.1 DESCRIPTION OF EXISTING SYSTEMS

The existing systems which are discussed in this section are mining systems that are capable of at least limited down-seam mining, and which were examined for possible application to HAMS. This presently appears to include only augering and auger-like machines. A standard augering machine is shown in Figure A-1.

Augering machines are relatively low in cost and produce clean coal with a small proportion of fines. They have the highest productivity of any mining system in terms of tons mined per manshift.

Though augers are normally used in nearly horizontal applications, there is at least one known example of operation at around 30 degrees down-dip. Fully shrouded in a steel tube, augers can elevate coal at angles exceeding 45 degrees. The limits in a bore hole of parent coal are not known.

The RSV Thin Seam Miner, developed in the Netherlands, has been undergoing evaluation in the U.S. for about 3 years. It has a ranging drum-type cutterhead rotating on a horizontal axis. Coal is conveyed to the surface by auger-screw flights. These are shrouded and should be capable of conveying coal up a 45-degree incline. The angular limit for feeding between cutterhead and auger conveyor is not known. The RSV has two basic advantages over augering machines: the excavation height is variable, and the rotational speeds of the cutterhead and conveyor screw can be independently controlled for maximum efficiency.

A.1.1 Auger System

Augering is a common method of extracting coal from outcropping seams and from highwalls left at the economic stripping limit. It is essentially a drilling or boring process that uses a number of auger flight stages to achieve penetration depths of up to 250 feet. The diameter of single heads range from about 21 inches up to 84 inches (Figure A-2). Double- and triple-element cutterheads produce oblong holes and are popular for thinner seams.

The rotating barrel-type head with pilot bit and annular rings of cutting picks and breaker bars is the most popular. These combine good advancement rates and uniformly sized cuttings with very few fines.

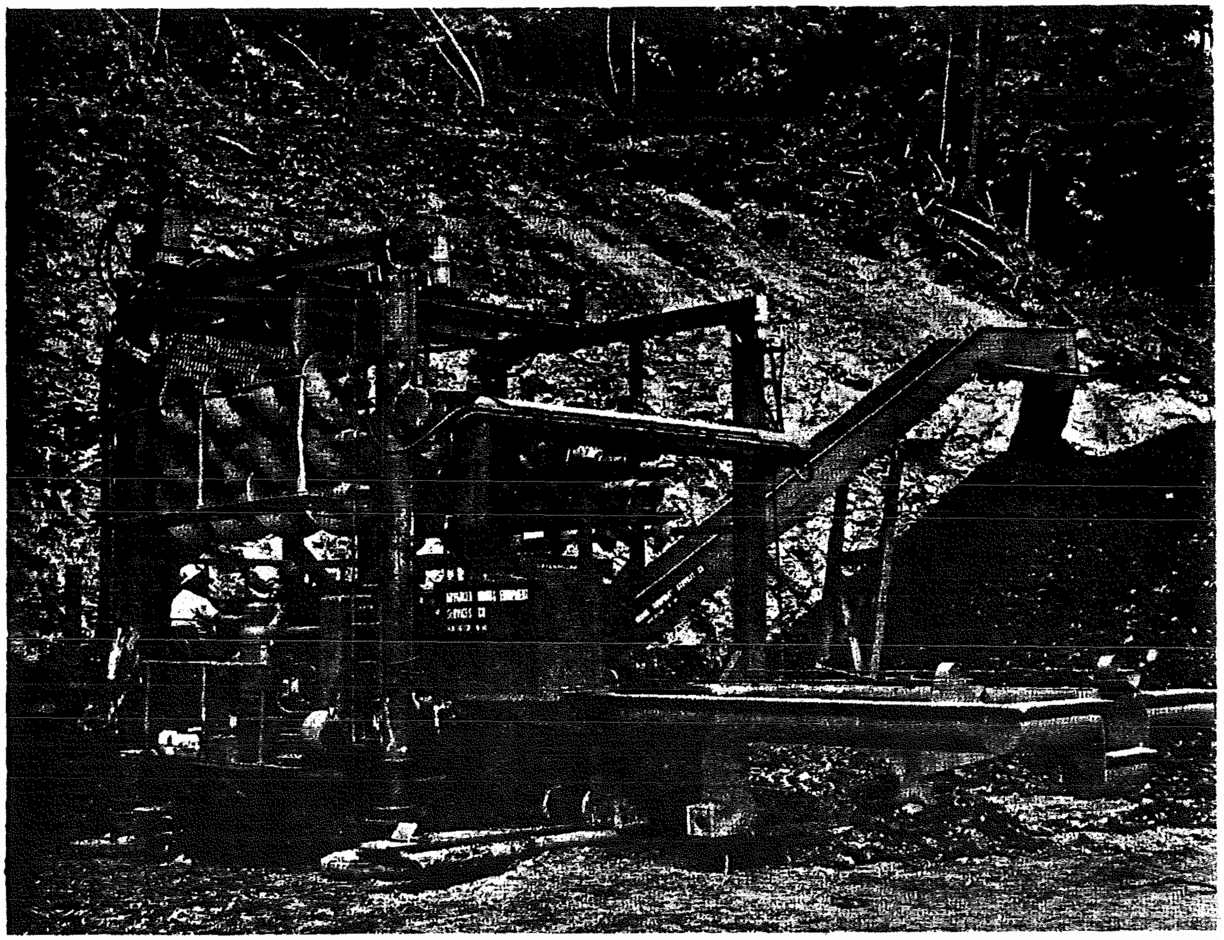


Figure A-1 STANDARD AUGER MACHINE

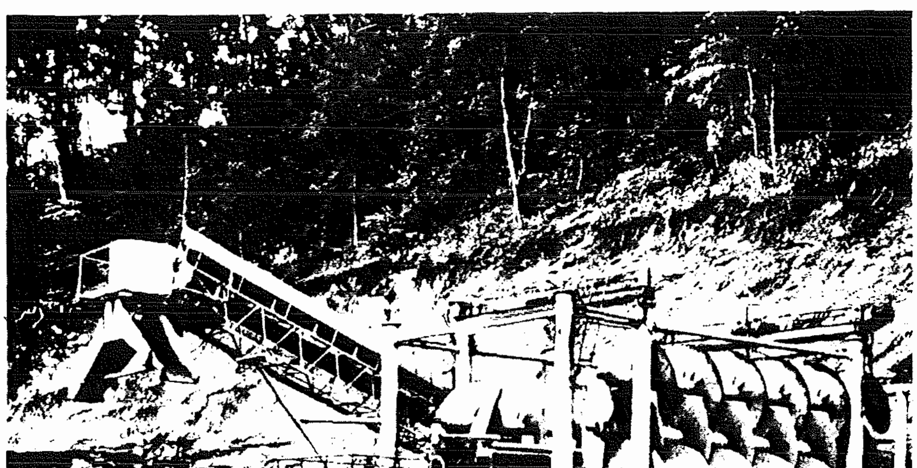


Figure A-2 JOY 84-INCH AUGER

The standard practice is to select an auger diameter approximately 12 inches smaller than seam thickness and enter the seam above center. This method accommodates some "droop" of the cutterhead due to gravity. The cutterhead can be steered upward by applying thrust, or allowed to "droop" by relaxing the thrust. This makes steering difficult, at best. Planned webs between holes do not normally exceed 8 inches, but may approach 12 inches or more for the largest diameters. Figure A-3 compares production rates attainable for various seam heights and cutterhead configurations. These are the highest obtainable of most (if not all) mining systems in terms of tons per manshift.

Recovery ratios of up to 80 percent have been achieved but are believed to average closer to 35 percent. This is due principally to auger diameters too small for the seam being worked, penetration depths less than maximum, and excessive web thickness between holes. Previous holes are often breached because of augerhead wandering.

The Coal-Tex Division of Vecellio & Grogan has developed an augerhead prototype that produces a 42-inch-square hole while advancing, which is widened to a rectangular hole as the string is withdrawn (see Figure A-4). The rectangular width is variable up to about 6 feet, 6 inches. Various other methods for backreaming blind pilot holes are possible and may offer some potential for overcoming the basic factors mentioned above that limit the recovery ratio.

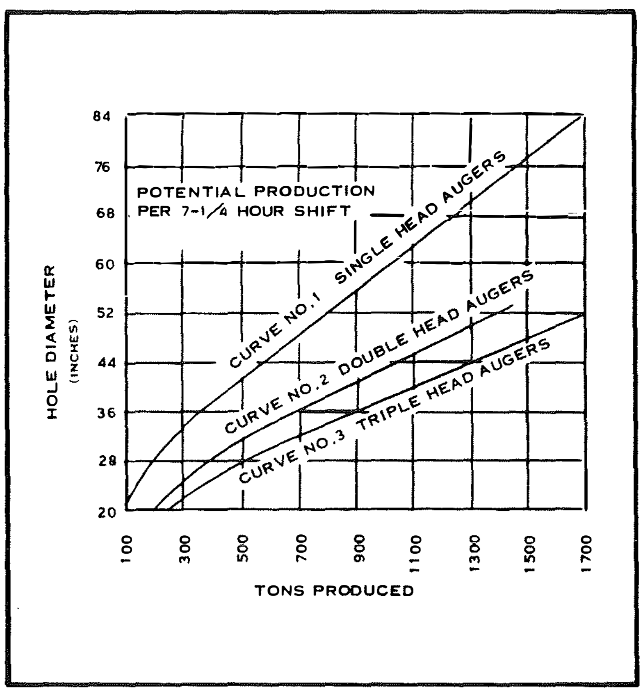


Figure A-3 PRODUCTION FROM AUGERS OF VARIOUS DIAMETERS AND DESIGNS

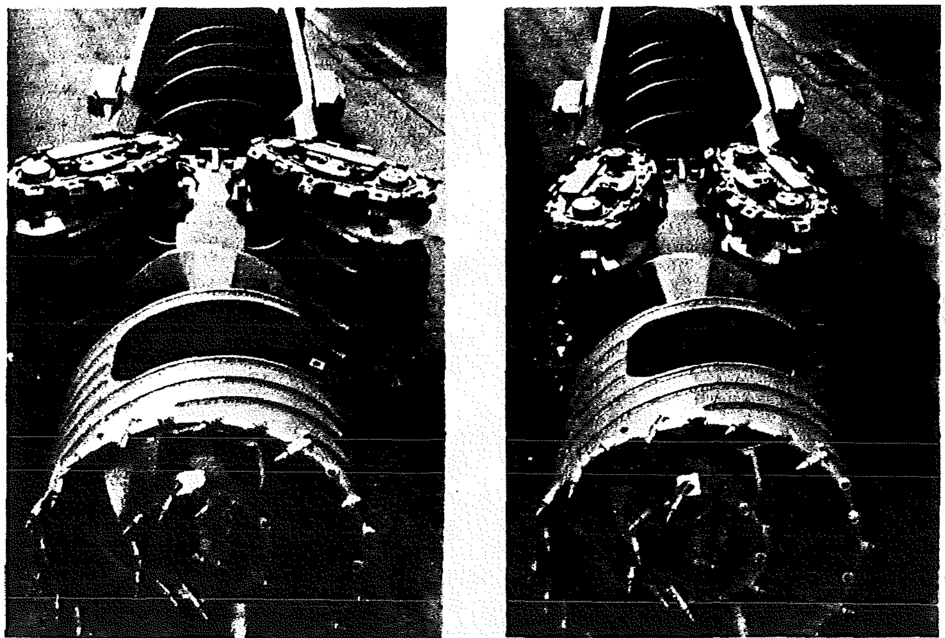


Figure A-4 RECTANGULAR AUGERHEAD

Normal down-dip angles rarely exceed 10 degrees. A 30-inch-diameter triple-head auger was experimented with at a 30-degree down-dip angle but was apparently unsuccessful. The problem encountered--staying within the seam--appears to be unrelated to the steep down-dip angle, except perhaps for steering. Auger screws are capable of conveying coal up a 45-degree incline, and higher, if rotated within a steel sleeve or casing. The angular limits in an unsleeved hole are not known. If tip clearances do not degenerate, performance should remain comparable.

A.1.2 RSV Thin Seam Mining System

The RSV Thin Seam Mining System is a relatively new development that combines a vertically ranging rotating drum-type cutterhead and a twin counter-rotating screw conveyor system. A head-position sensor and coal-thickness sensors feed information to the operator who is able to remotely steer the cutterhead.

The system is capable of cutting a 7.4-foot-wide hole from 24 to 60 inches high. Maximum cutterhead advancement of 220 feet is attained by adding conveyor modules, each of which is 20 feet long. A production capacity of up to six times that of augering is claimed.

The system is manufactured by Rhine-Schelde-Verolme (RSV) which is headquartered in the Netherlands. The Thin Seam Miner Leasing Corporation of Houston, Texas, a joint venture by Coal Systems, ICC Corporation, and the RSV subsidiary, Coalpart, plans to purchase 200 systems during the next 5 years. These would be leased or operated as joint ventures with mining companies. Since July 1979, a prototype unit has been operating near Buckhannon, West Virginia (see Figure A-5).

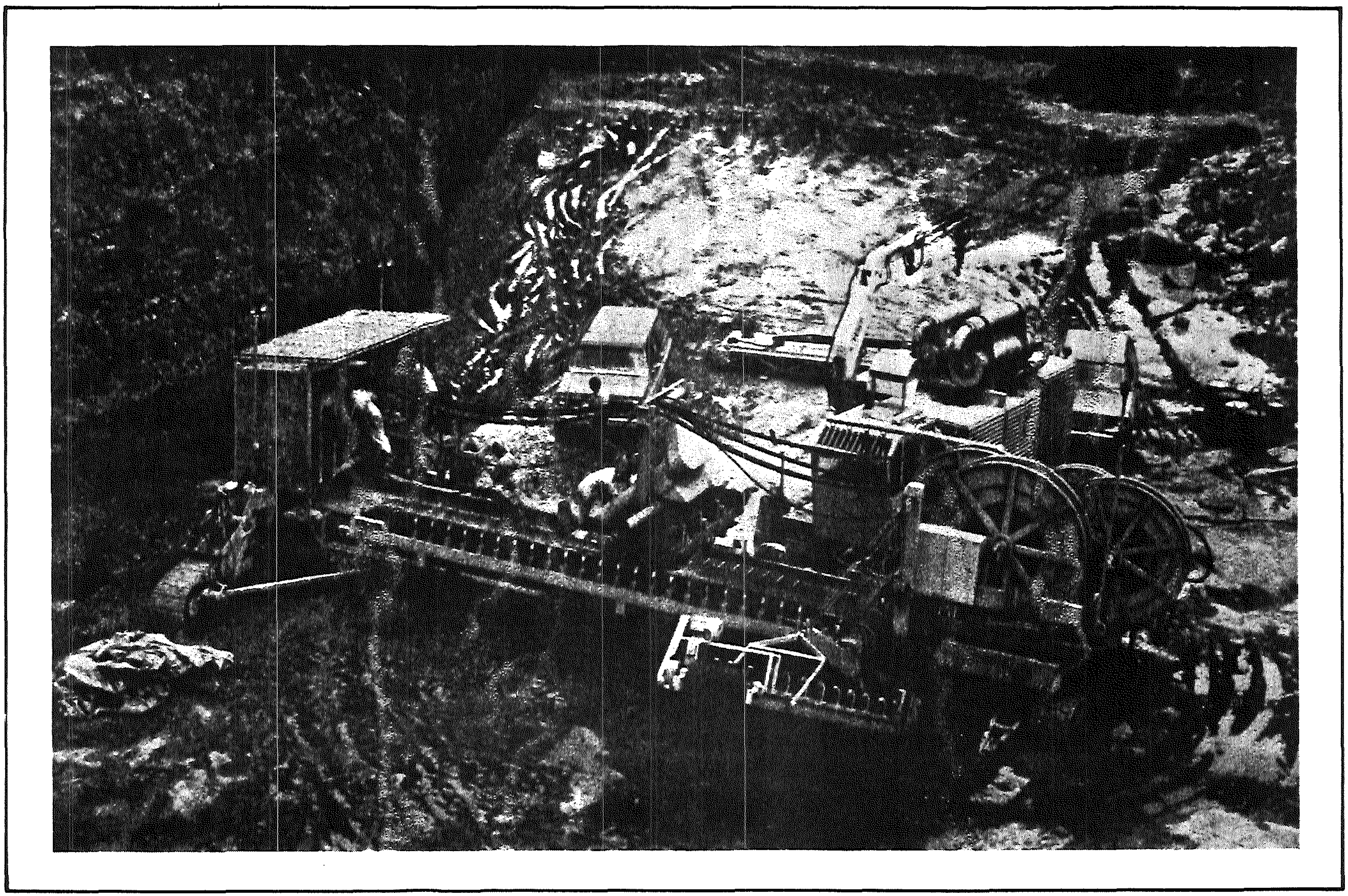


Figure A-5 RSV THIN SEAM MINER

The RSV system overcomes the basic problem of lack of seam-height versatility of auger systems. The shrouded auger conveyor system would also appear to be capable of down-dip operation to 45 degrees.

A.1.3 Eckenrode Auger Concept (Figure A-6)

This concept uses standard augering equipment to drill a round hole 42 inches in diameter and 200 feet deep. A special cutterhead removes coal from the roof in a multipass operation. The cutterhead consists of an inclined ripper-chain arrangement that is driven by the rotating auger string through a right-angle gearbox. The cutter chain is raised about 48 inches for each pass until a vertical slot about 42 inches wide and up to 16 feet high is achieved. Cuttings are removed by the standard auger string. This is a proprietary concept of an eastern augering contractor.

A.1.4 New Edna Miner Concept (Figure A-7)

This is another development of Coal-Tex. The machine produces a 5-foot-square hole while advancing which is enlarged to 8 feet wide by 12 to 16 feet high during retreat. Cuttings are conveyed by an extensible belt which limits penetration depth to 1,000 to 1,500 feet. The cutterhead array is similar to the rectangular auger except that in addition to the side wings, a third wing has been added to the top. This wing telescopes to a 7-foot width and is raised to the desired height. It is powered by a rotating drill pipe string which can also be used for ventilation air. The drill pipe string and belt conveyor are supported by brackets spaced about 12 feet apart. A short cone-shaped auger feeds coal to the belt.

A.2 CUTTING MECHANISMS

This section focuses on the various types of cutting mechanisms which were examined for potential for application to HAMS. Most of the machines discussed here are functionally limited in down-dip capability by the problem of feeding cuttings from the face, through the cutterhead, to the conveyor system, which may be located above the cutterhead.

A.2.1 Continuous Miners

Included in this category are drum, ripper, and boring types, as well as their hybrid variations. These are characteristically limited in operation to about 15-degrees down-dip by their gathering and feeding mechanisms. By improving these functions, operation to 45-degrees down-dip is conceivable for most. A surface winch would be necessary to control ascent and descent for pitches exceeding 20 degrees or so. Radio and pendant types of remote control are well developed.

The boring types of machines, such as those manufactured by Goodman, Joy, and National Mine Service (Figure A-8), produce a smooth and well-contoured hole (Figure A-9) which can enhance stability. They are capable of very high production rates depending on size. Twelve tons per minute is common.

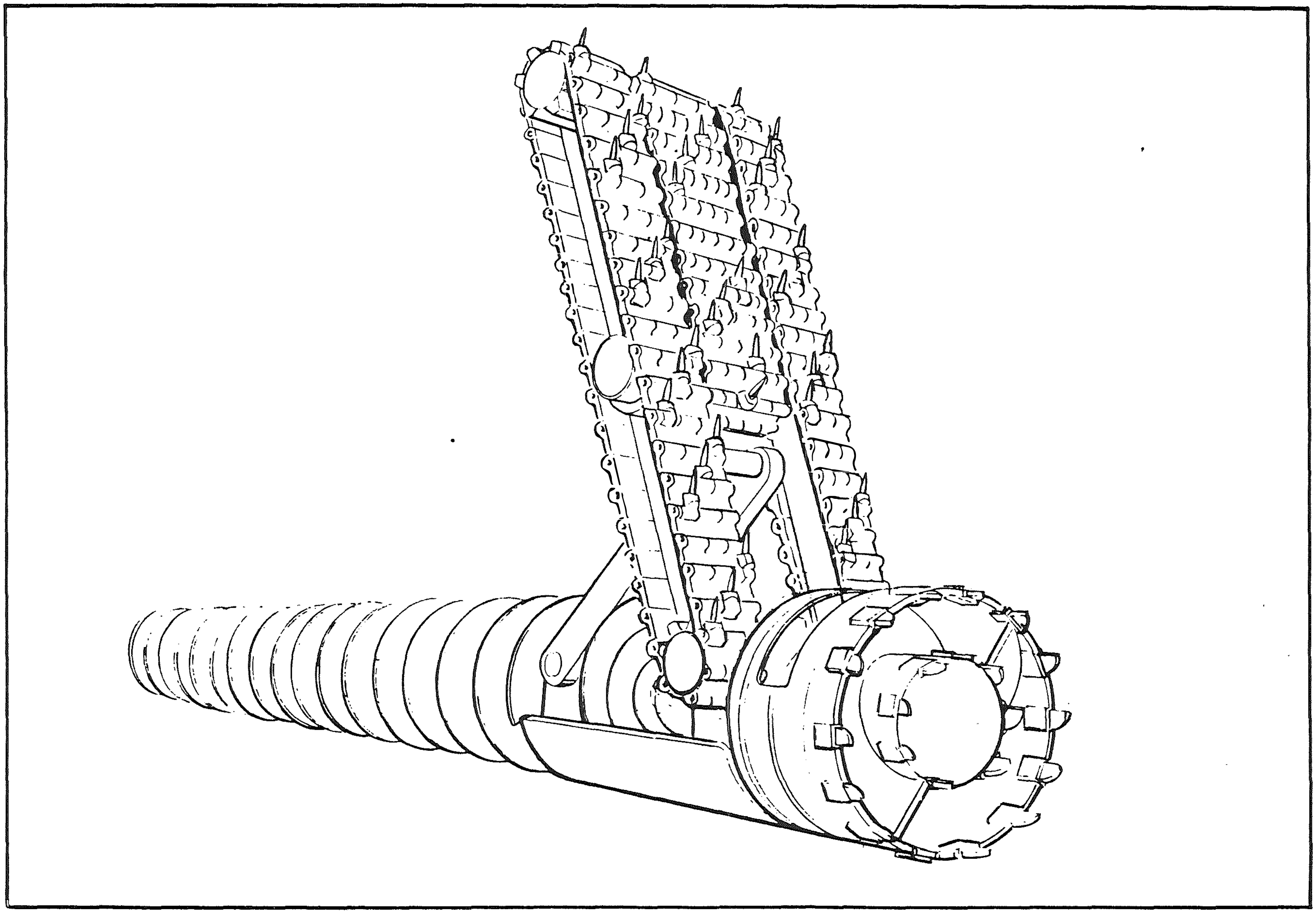


Figure A-6 ECKENROE AUGER CONCEPT

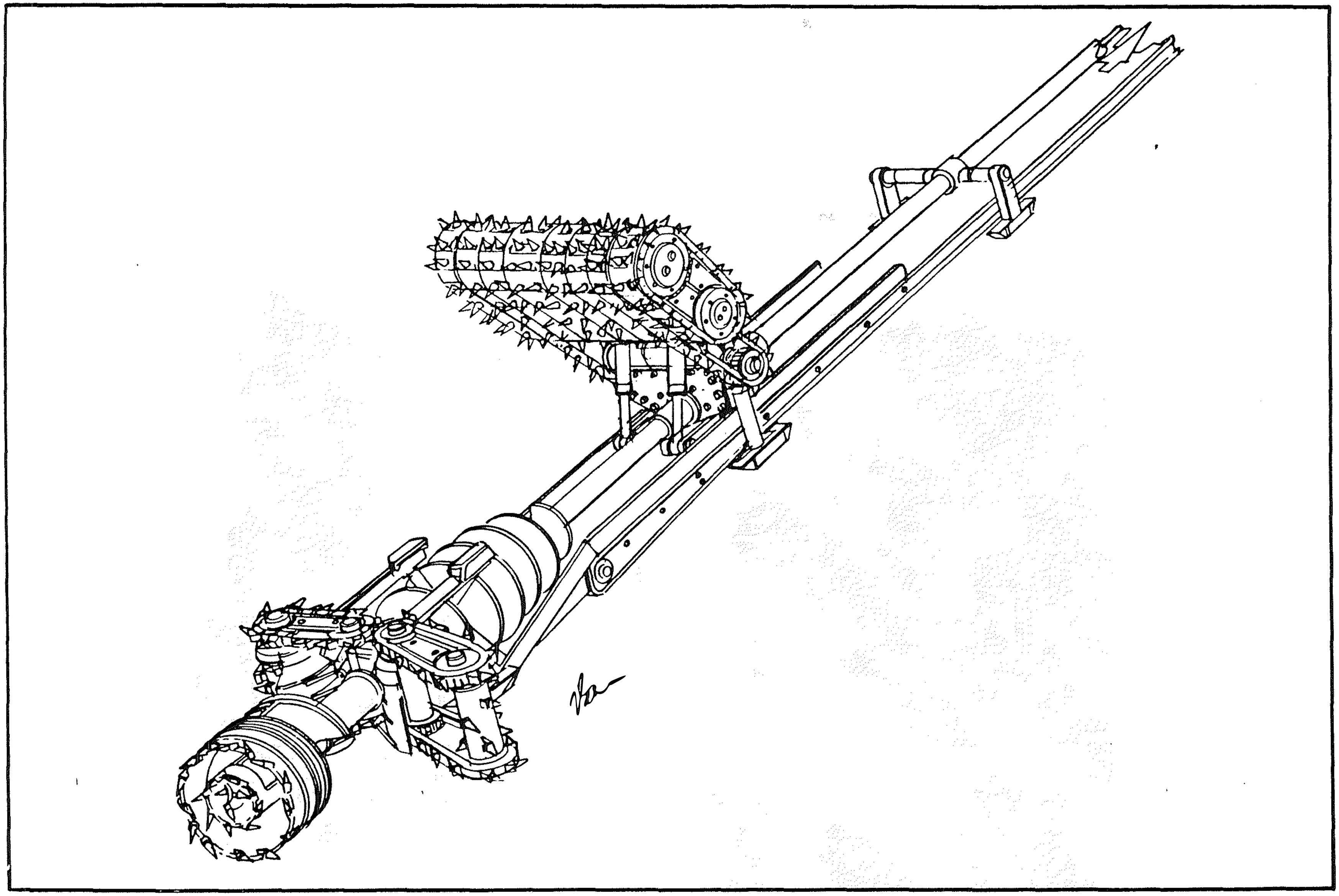


Figure A-7 NEW EDNA MINER CONCEPT

A version of the Alkirk Cycle Miner has been demonstrated in both up- and down-dip angles to 30 degrees. It is believed capable of operation to 45-degrees down-dip. The unique pilot bore locking mechanism enables it to pull itself upgrade.

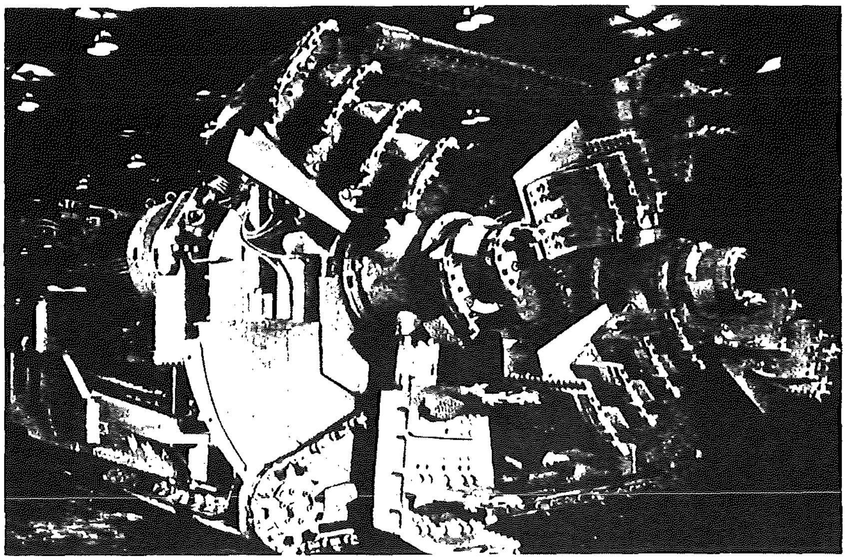


Figure A-8 JOY TWIN BORING MINER

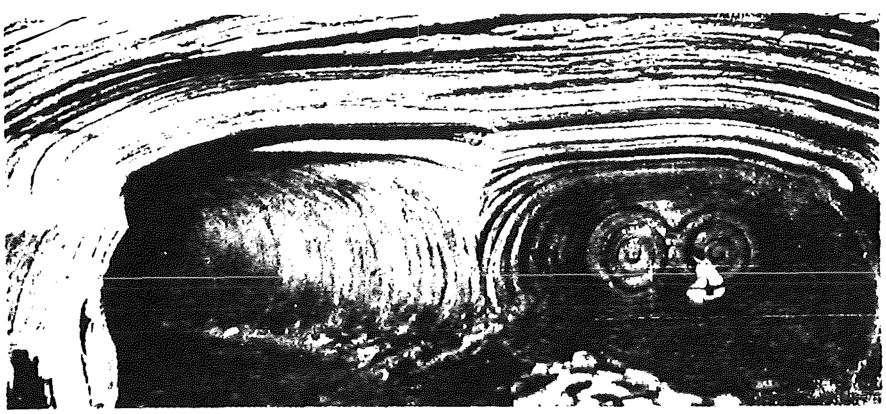


Figure A-9 ENTRY DRIVEN BY JOY TWIN BORER

Another version of the Alkirk miner (Figures A-10 and A-11), without the self-sumping feature, was experimented with at the Kemmerer Coal Company mine during the 1960's. It was operated at approximately 20-degrees down-dip, and initially had a skip-hoist-type of haulage system. The skip-hoist car, however, nearly filled the bore hole and virtually trapped the operator stationed at the miner. The skip hoist was replaced by an extensible flat belt (Joy). At the surface, coal was fed to a large hopper equipped with an off-conveyor for loading haulage trucks.

Any of the continuous miners could proceed cross-seam at a reduced angle in a manner similar to that demonstrated by a Joy 6CM ripper miner at the Kemmerer Coal Company mine (and described in the first task report under this contract). This reduced the pitch angle to less than 10 degrees in a seam lying at 18 degrees. Recovery ratios achieved were quite low, less than 25 percent. This is because the number of passes was limited to an excavated depth of 50 feet or so. The method, however, maintained good roof control by leaving a maximum amount of top coal around a small face opening.

A.2.2 Tunnel-Boring Machines

Both vertical and horizontal tunneling machines are available in versions suitable for hard (see Figure A-12) or soft formations. Those designed for soft formations ranging from cohesive and competent to such cohesionless material as sand and gravel require large bore thrust pads or "grippers" (from which the generic term "shield-type" boring machine derives). Hole diameters range from 40 inches to 40 feet.

Tunneling machines designed for vertical application are capable of boring 10 to 15 degrees from vertical. Some horizontal versions are capable of boring to as much as 35 degrees down-dip. These are the approximate limits for feeding cuttings through the machine. Vertical machines produce a cone-shaped face. Cuttings gather by gravity to the center of the cone, from where they are extracted. Horizontal machines convey the cuttings away from the floor, where they tend to accumulate, by using the rotary action of the cutterhead to elevate the cuttings to a conveyor.

Cuttings are subjected to regrinding. Larger amounts of fines are to be expected. In general, these types of tunneling machines are very high in cost and not very versatile. They are of more interest in civil construction than in mining.

A.2.3 NCB-DOSCO In-Seam Miner

The in-seam miner resulted from a joint development effort of the British National Coal Board (NCB) and DOSCO Overseas Engineering LTD, a division of the Hawker Siddeley Group, in the early 1970's. Figure A-13 shows the arrangement of the cutterhead, which consists of cutter picks and gathering buckets mounted on an endless chain. The chain is driven by a pair of high-torque, low-speed hydraulic motors. Coal cuttings are conveyed away from the face by a chain flight conveyor. Face thrust is developed by a pair of hydraulic cylinders, each pushing from the base of a hydraulic walking jack. These are extended and locked into the floor and roof of the excavated hole. Other control cylinders provide a full steering capability.

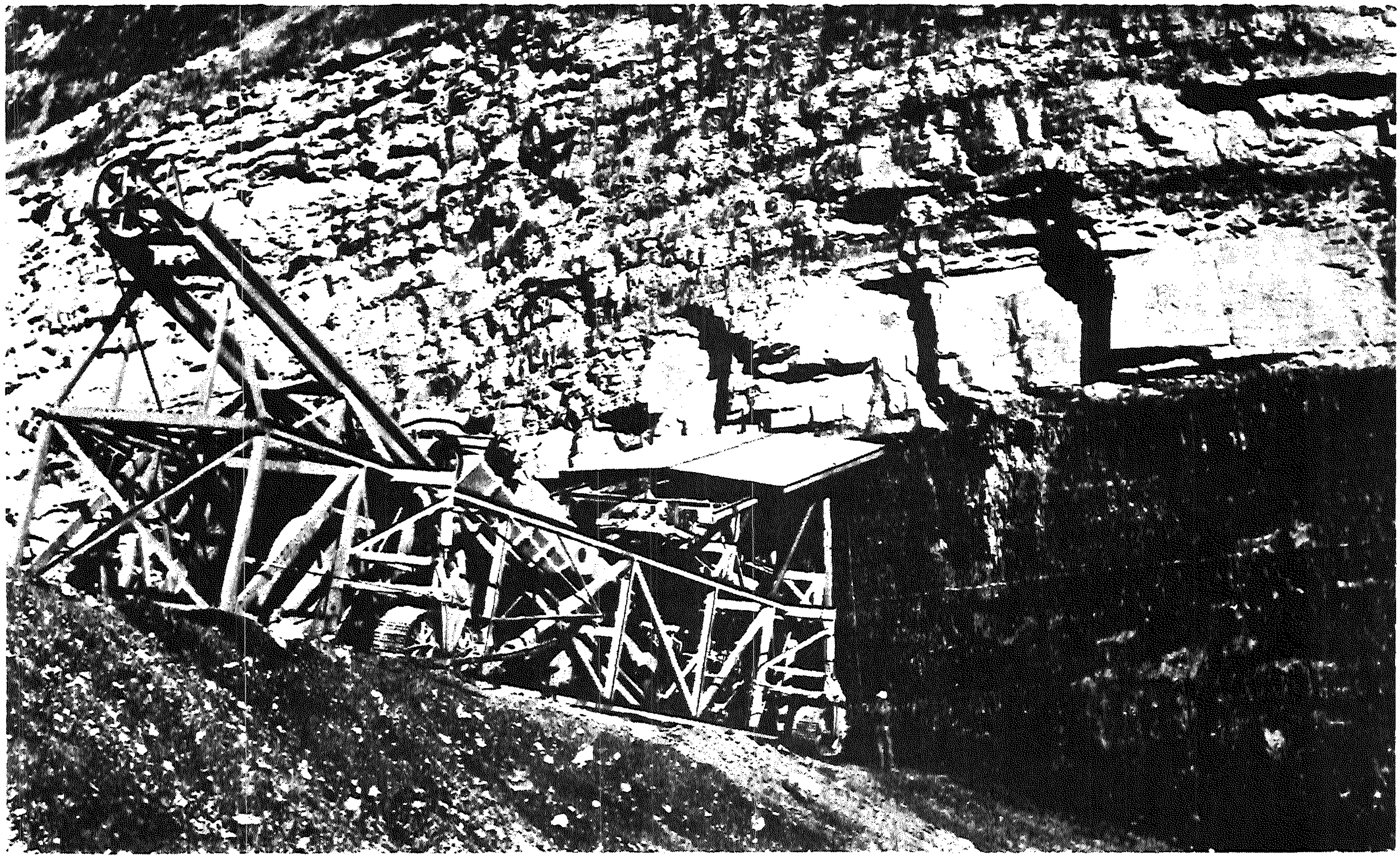


Figure A-10 ALKIRK MINER AT KEMMERER COAL COMPANY

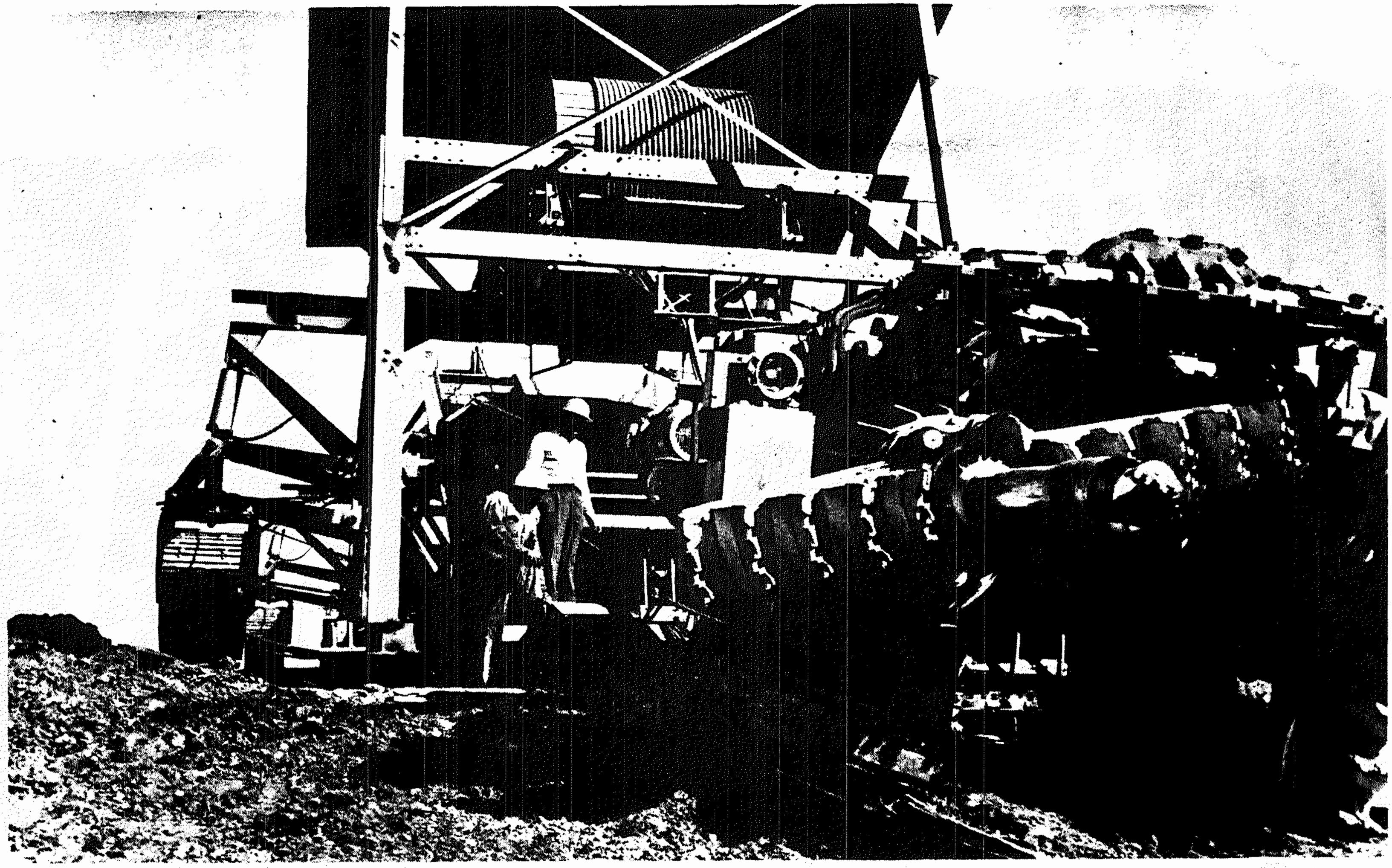


Figure A-11 ALKIRK MINER AT KEMMERER COAL COMPANY

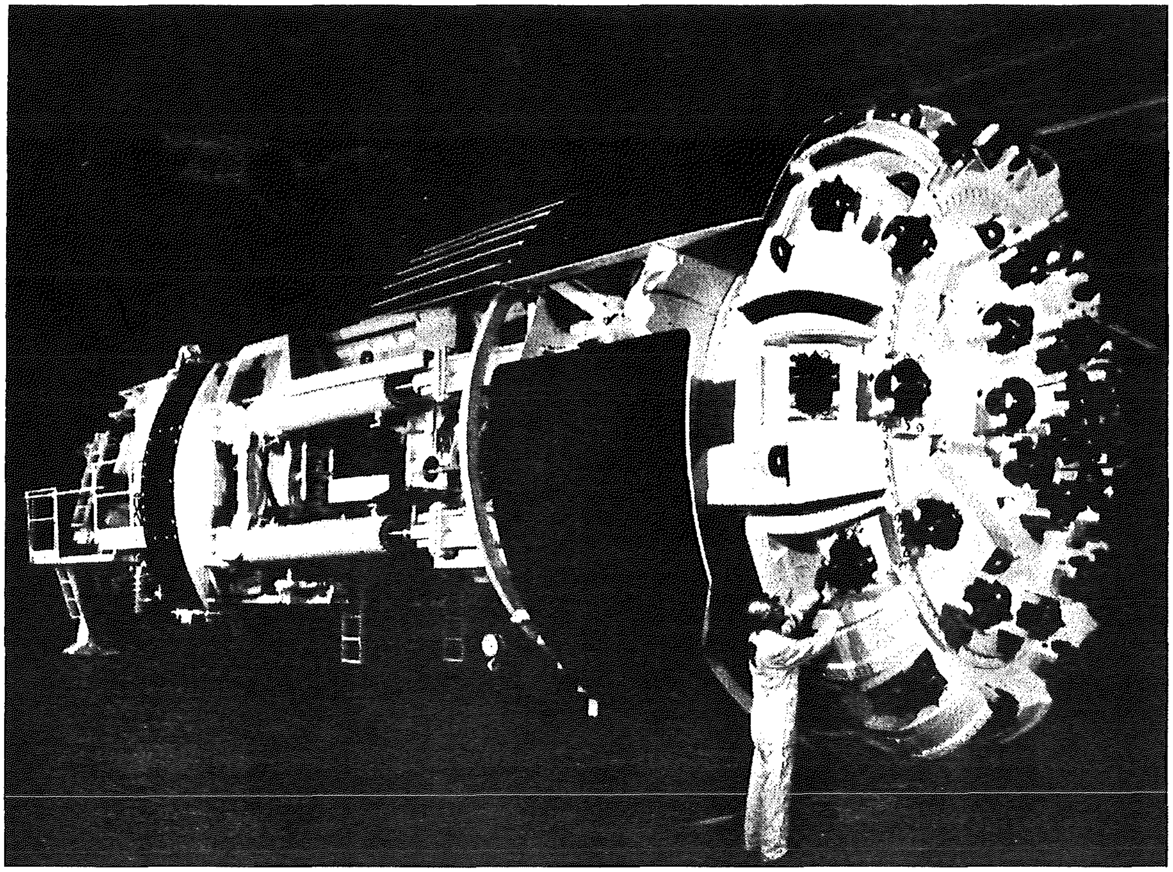


Figure A-12 TUNNEL BORING MACHINE, 18 FEET, 7 INCHES IN DIAMETER
(Courtesy of the Robbins Company)

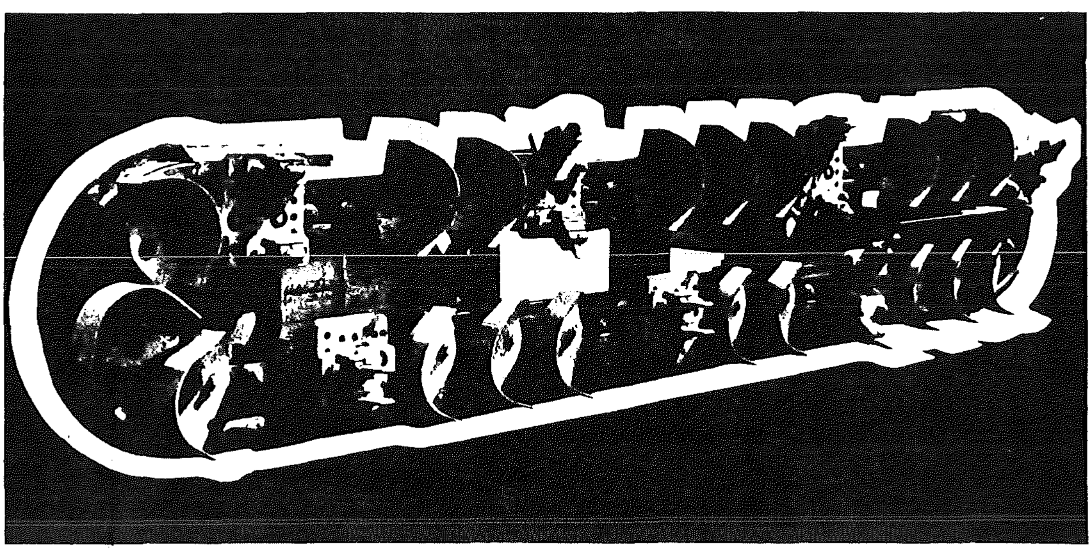


Figure A-13 IN-SEAM MINER (Front View Showing Pick and Bucket Arrangement)

The all-hydraulic version (shown in Figure A-14) is rated at 120 horsepower, and makes a well-contoured heading 11.5 feet wide and around 4 feet high, which is adjustable. An operator's control panel is integral with the machine. It is capable of advancing 2 inches per minute (or 5 feet per hour). This is equivalent to about 8-1/2 tons per hour. Its developer limits its down-dip capability to a gradient of 1 in 20, though this would seem conservative.

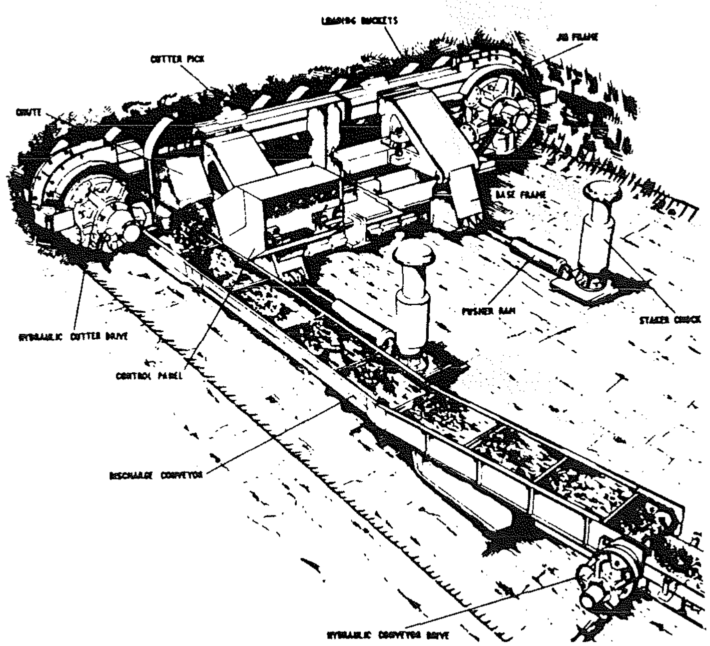


Figure A-14 IN-SEAM MINER

A.2.4 Boom Heading Machines

Boom heading machines, such as manufactured by DOSCO in Britain and Alpine Equipment Corporation (Figures A-15 and A-16) of State College, Pennsylvania, enjoy a wider popularity in Europe than in the U.S. Alpine originally represented an Austrian manufacturer but now designs and builds its own line of equipment. These machines are characterized by a relatively small rotating cutterhead mounted on the end of an articulated boom. The cutterheads are similar and contain an array of picks. The 36-inch-diameter cutterhead on the DOSCO machine rotates about the boom axis. The Alpine machine has two cutterheads which mount on either side of the boom and rotate about a common axis normal to the boom. The Alpine boom has an additional degree of freedom in that it is extensible.

The Alpine FT-6 machine demonstrated its high-precision capability during the refurbishment of the USBM's underground testing facility at Bruceton, Pennsylvania. Room heights were maintained within plus or minus 2 inches.

Boom heading machines can make well-contoured cross sections of just about any shape within the reach of the boom. The cross-sectional shape is easily varied as the machine advances. The major shortcoming is low rate of advancement. However, a second boom could be added, if necessary, to meet requirements. DOSCO had conceptually designed a twin boom miner in 1974 (Figure A-17).

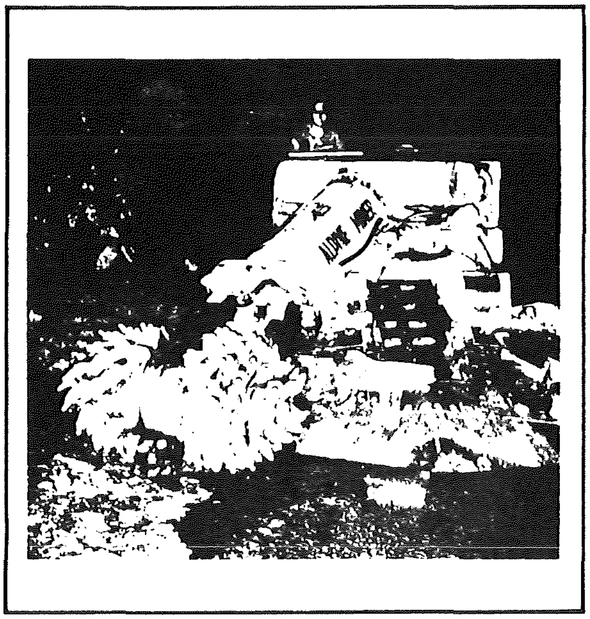


Figure A-15 ALPINE BOOM HEADING MACHINE

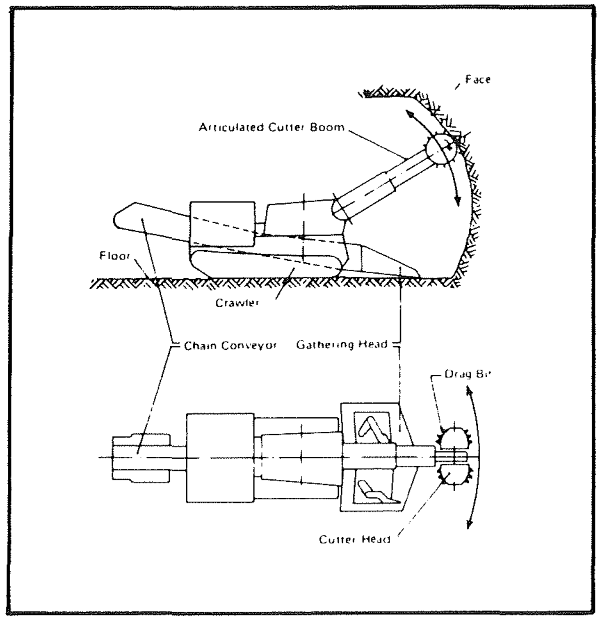


Figure A-16 CUTTER BOOM ARTICULATION

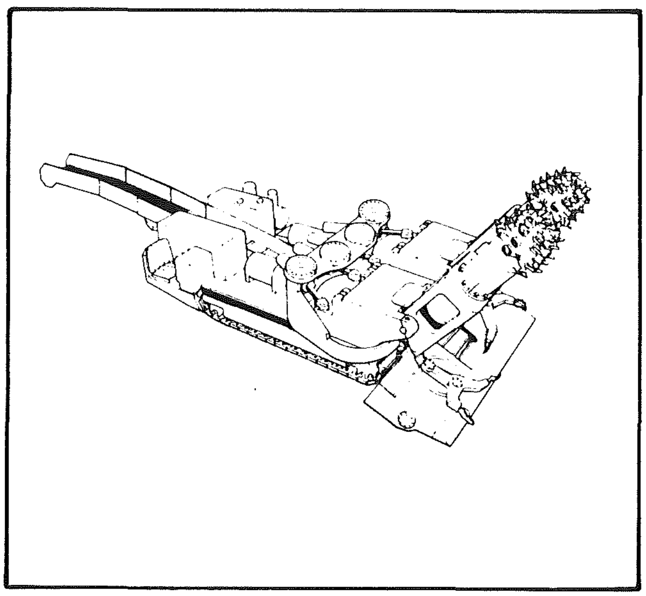


Figure A-17 ARTIST'S CONCEPT-- TWIN BOOM HEADING MACHINE

A.2.5 Dredge-Type Rotating Cutterheads

Dredges of the hydraulic type are designed for excavating materials lying underwater and transporting them in a slurry form to a discharge point, such as a hopper. Typical rotating cutterheads are shown in Figure A-18.

Suction dredges normally operate at depths of 30 to 90 feet but have been used at depths exceeding 200 feet. Cutter booms are normally deployed at 45 degrees down-dip but will cut and convey material at any angle. Output capacity depends on pump and cutterhead horsepower, digging depth, height of discharge, relative size and specific gravity of cuttings, line friction, and percentage of solids. Dredge output is usually calculated on a 10-percent volumetric ratio of solids to water.

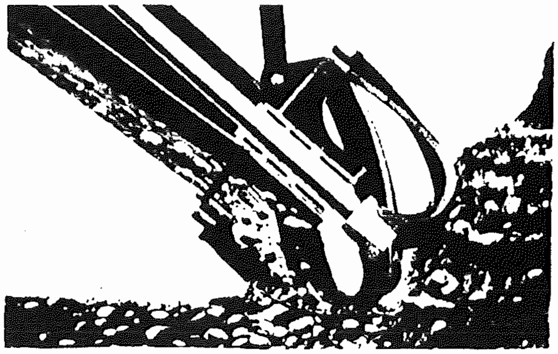
A 20-inch dredge is the standard general-purpose machine. It might typically have a 300-horsepower cutterhead, a 1,200-horsepower, 12,000-gallon-per-minute pump, and a rating of 350 cubic yards per hour of river bottom material. Based upon a mixture of sand, clay, and gravel weighing approximately 100 pounds per cubic feet, this is equivalent to 470 tons per hour.

Although extensively used in the mining of metals and other minerals, no example of coal mining by dredging was found. However, if the cutterhead can cut coal, gathering and conveying should not present a problem. At an in situ density of 80 pounds per cubic feet and a specific gravity of 1.3, coal should be conveyed almost as easily as sand or gravel and clay, which have a specific gravity of nearly 1.0 in water.

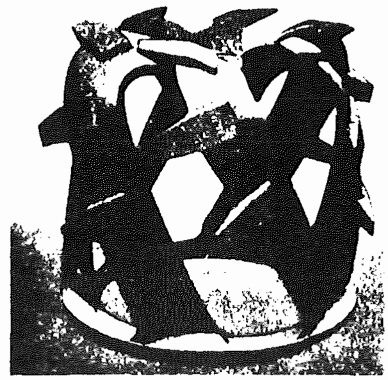
A.2.6 Slackline Excavator Concept with Projectile Bucket (Figure A-19)

This concept employs the slackline cableway technology pioneered by Colonel Sauerman some 70 years ago. A dragline-like bucket is suspended from a track cable on low-friction sheaves. The track cable is anchored to the bottom of a small pilot hole. The pilot hole is drilled into the coal seam parallel to and near the roof. The bucket excavates a vertical slot equal to the bucket width and a height approaching seam thickness.

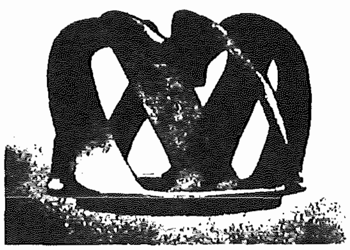
The bucket (Figure A-20) is released from the surface and travels down the track cable by gravity. At the bottom of the excavation the face is advanced through the mechanism of high-energy projectile impact. After the face is advanced about 6 inches through multiple bucket blows and trapping the coal broken loose, the track cable is slackened to lower the bucket to the floor of the excavation. The hoist cable is then hauled to rip additional coal. When full, the track cable is retensioned to lift the bucket clear. The bucket is then hoisted to the surface at a high rate of speed by the dragline. A modified dragline machine could provide the necessary surface equipment.



Artist's view of the cutter of a hydraulic pipe line dredge.



For special work, cutters may be provided with teeth of the so-called "shovel" type welded to the blades.



A spiral cutter with seven blades.

Figure A-18 DREDGE-TYPE ROTATING CUTTERHEADS (Courtesy of Ellicott Machine Corporation)

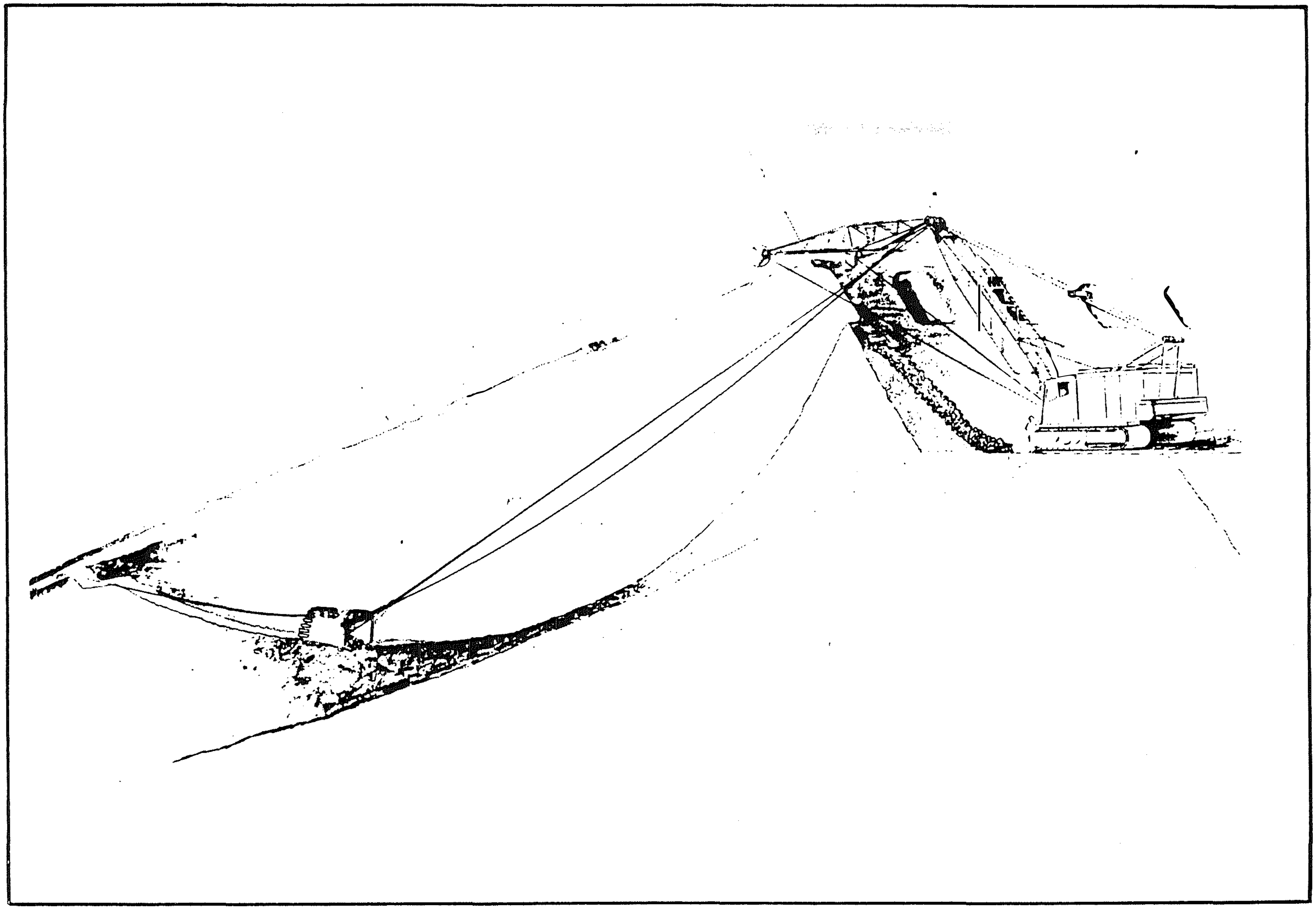


Figure A-19 SLACKLINE EXCAVATOR CONCEPT

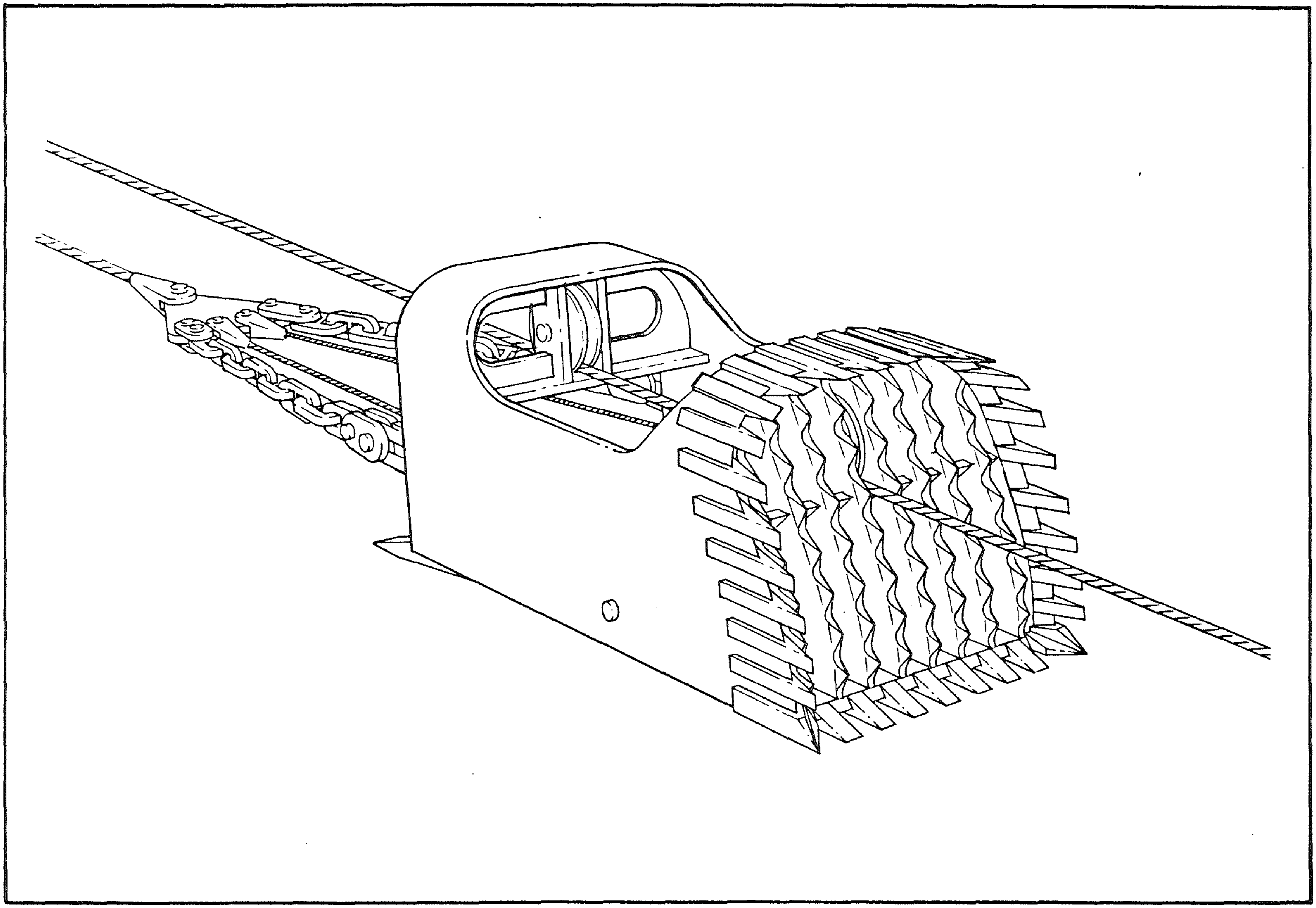


Figure A-20 PROJECTILE BUCKET CONCEPT

Appendix B

PROJECTILE TOOTH TEST

APPENDIX B

PROJECTILE TOOTH TEST

The slackline excavator concept developed for a High Angle Mining System employs a bucket that breaks coal loose and advances the face through projectile impact. A leading designer and manufacturer of buckets and teeth of all types recommended a sharply-pointed, conical design for the projectile teeth.

B.1 OBJECTIVE

The purpose of the limited testing discussed in this subsection was to ascertain full-scale tooth performance in real coal. This includes tooth advancement per blow, the geometrical shape of the cavity formed in the impact area, the degree of fracturing in the surrounding coal, and the effects of multiple blows to the same point.

B.2 SUMMARY

A projectile tooth was allowed to accelerate freely down a tensioned track cable and impact a coal target. The projectile carriage was weighted to 250 pounds, which simulates a 12-inch tooth spacing over an 8- by 8-foot projectile face of a 10-ton bucket.

The test target consisted of six blocks of coal arranged in a flat 2-by-3 matrix within a heavy steel framework. Furnished by the demonstration mine, each coal block was about 1 cubic foot. The voids between blocks and framework were filled with a gypsum plaster mixture.

Blows were approximately centered on each block within the matrix. About 5 inches of penetration was obtained during the initial blow to each block. As evidenced by the degree of face swelling, individual blocks appeared to be significantly, if not completely, fractured during the initial blow. It was hoped that the target would behave as a monolithic block of coal, but little fracturing (face swelling) across the boundaries of the impacted block into adjacent blocks was evident. Subsequent blows to the same impact point produced less penetration and more fines.

B.3 CONCLUSIONS AND RECOMMENDATIONS

Based on the degree of fracturing obtained within each coal block during the initial blow, a wider tooth spacing should be evaluated in future testing. A wider tooth spacing has the effect of increasing blow energy per tooth, all other factors remaining constant. For example, an 18-inch tooth spacing would increase the weight per tooth to about 500 pounds for the same 10-ton bucket with a face area of 64 square feet.

To reduce the amount of fines produced and increase the amount of coal broken loose, it appears highly desirable to vary the tooth-impact pattern for each subsequent blow. This may be achieved simply by varying track-cable tension during alternate blows to raise and lower the bucket.

B.4 DISCUSSION

B.4.1 Cable Rigging

Figure B-1 shows the test set-up. A mast consisting of a wide-flange steel beam was erected and guyed in place. A mast head pulley supported the 3/8-inch-diameter, 6x9 IWRC track cable. Both ends of the track cable were double-staked. A chain hoist provided the means for tensioning the track cable. Actual tension was measured with a mechanical load cell.

The projectile was raised with a light, 1/4-inch-diameter nylon rope and a manual hoisting winch (Figure B-2). Once raised, the projectile was released with a pull pin and lanyard.

B.4.2 Projectile

The projectile was suspended from the track cable by a pair of roller-bearing steel snatch blocks and two short lengths of chain. A universal tooth adapter was welded to each end of a rectangular steel tube. One end would mount any of a number of ESCO series 56 teeth, the other, any of a number of ESCO series 46 teeth. A saddle-type weight increased the combined weight of the projectile with teeth to 250 pounds.

B.4.3 Drop Height and Speed

By setting the potential energy of the raised projectile equal to the kinetic energy at the bottom, the ideal (frictionless) energy relationship is developed:

$$\text{Potential Energy} = Wh$$

$$\text{Kinetic Energy} = WV^2/2q$$

$$\text{or } V = \sqrt{2qh}$$

For h = 13.5 feet:

$$V = \sqrt{2 \times 32.2 \times 13.5} \qquad = 29.6 \text{ feet per second}$$

Impact speed was not directly measured. The duration of time between projectile release and impact provided an indirect indication of efficiency. From the ideal relationship:

$$d = 1/2\ at^2$$

For $d = h/\text{Sin } 0$ and $a = q \text{ Sin } 0$

$$t = \sqrt{2\ d/a} = \sqrt{2 \times h/g\ (\text{Sin } 0)^2}$$

For 0 = 22 degrees and h = 13.5 feet:

$$t = \sqrt{2 \times 13.5/32.2 \times .14} \qquad = 2.46 \text{ seconds}$$

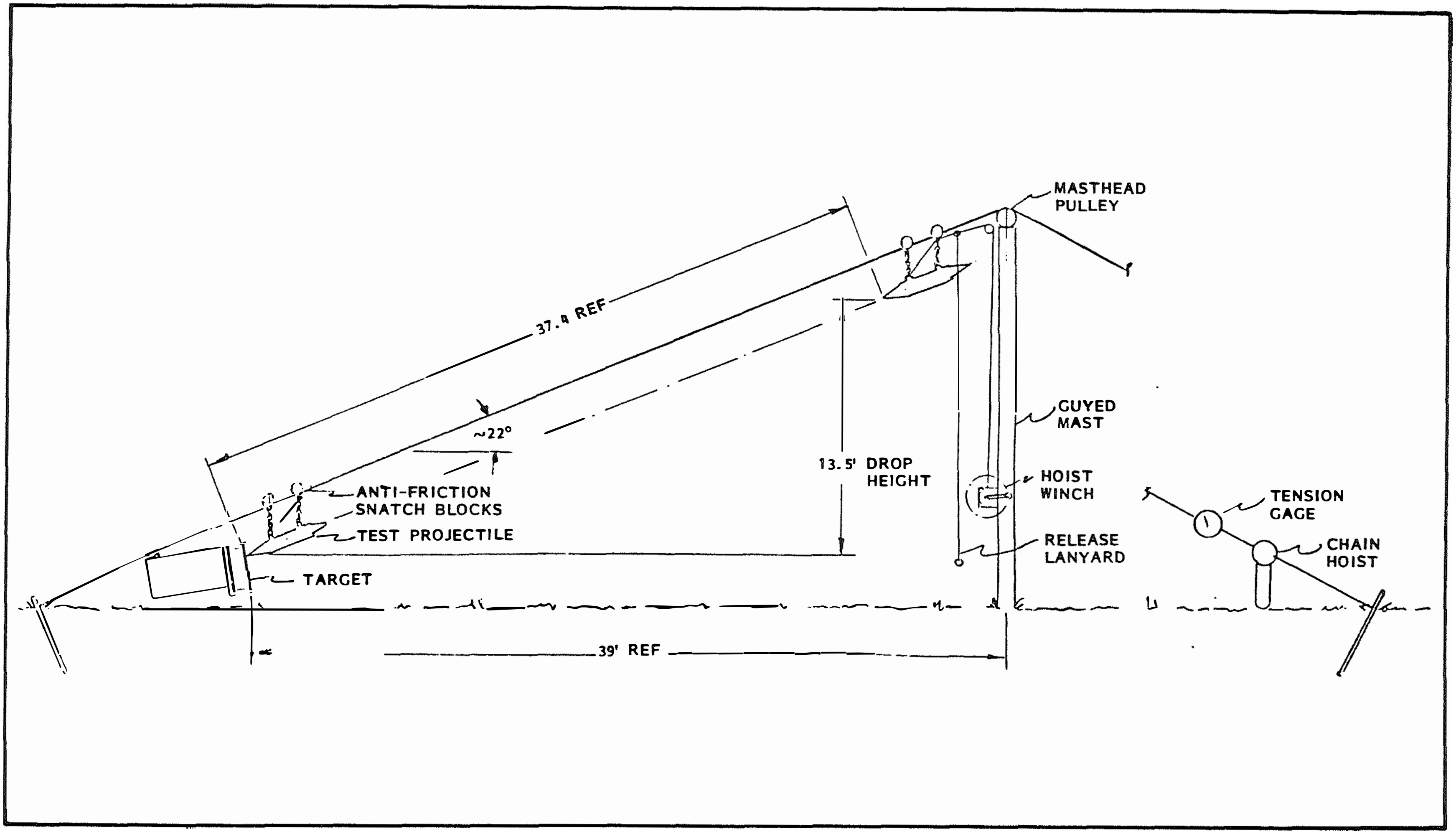


Figure B-1 PROJECTILE TEST ARRANGEMENT

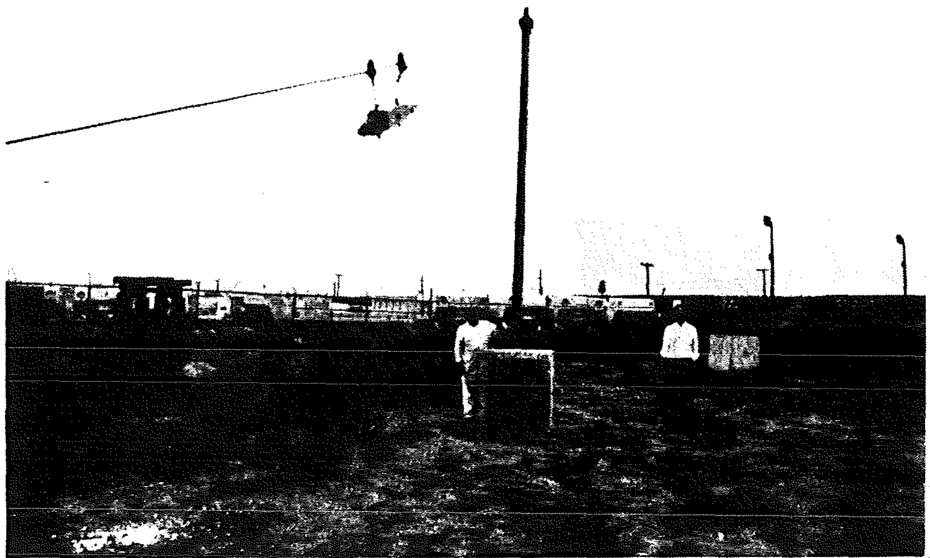


Figure B-2 HOISTING THE PROJECTILE

Observed acceleration times, as could best be determined by stop-watch, appeared to be between 2.3 and 2.5 seconds. Because of this and the near-frictionless arrangement, actual impact speeds were estimated to be very close to the ideal (frictionless) speed of nearly 30 feet per second.

B.4.4 Target

The target was made up of six individual coal blocks, each about 1 cubic foot in volume and 80 pounds in weight. Furnished by Skull Point Mine, each block was packaged in a heavy cardboard carton (triple-wall construction), lined with "rubberized hair," and filled with styrofoam "popcorn." Shipped by common carrier, they arrived in excellent condition. Upon receipt, the blocks were sprayed with clear lacquer to retard oxidation.

Figure B-3 shows the coal blocks arranged in the steel framework prior to grouting with standard quick-setting casting plaster. The framework was of standard channel, 12 inches thick by 20.7 pounds per foot, with an inside opening of 2 by 3 feet. The proper target face of each block was spray-painted red at the mine. These faces were arranged flush with the frame. Bedding planes were arranged parallel to the short side of the framework. Two-by four-inch fir framing was bolted to the backside of the framework. After grouting, the backside was faced with two layers of 3/4-inch-thick fir plywood.

The finished target block weighed about 800 pounds. To increase target mass, the target was chained to a 2- by 4- by 4-foot plaster block weighing approximately 2,400 pounds.

During projectile impact, the target would be pushed as far as 3/4 of an inch. This would indicate some loss of kinetic energy available to fracture coal, though probably less than 10 percent.

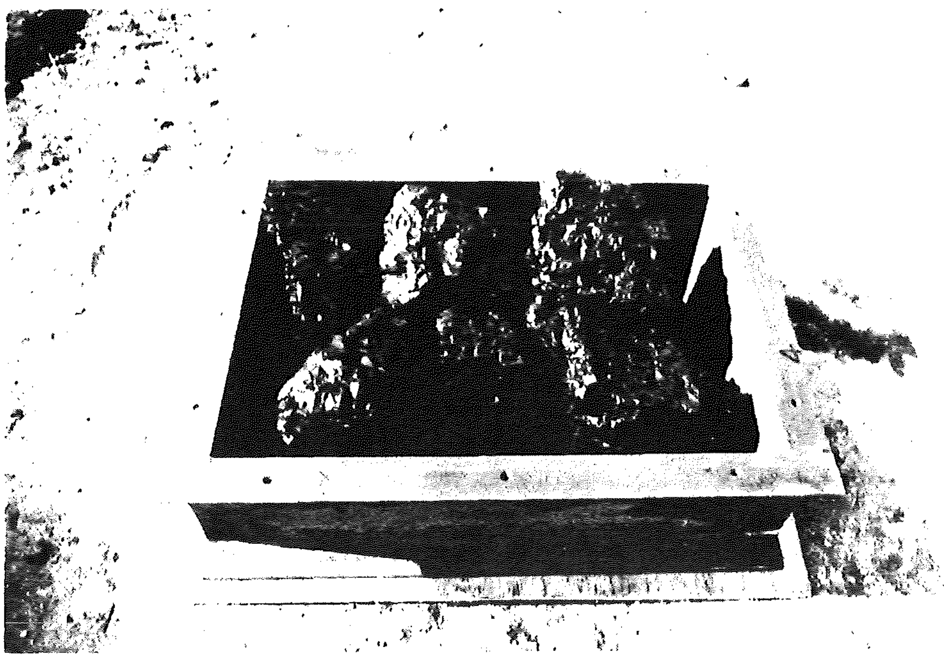


Figure B-3 **COAL BLOCKS IN FRAME PRIOR TO GROUTING**

B.4.5 Penetration

The face of the target was marked for six impact points spaced on 12-inch centers; each impact point was roughly centered on an individual coal block. By pulling the track cable from side to side, the impact point of the projectile could be walked across the face. By varying the track cable tension or adjusting the length of chain between the supporting snatch blocks and projectile, the impact point could be raised or lowered.

Figure B-4 shows the location of impact points and the blow sequence. Table B-1 shows the penetration depths obtained; the blow sequence number is indicated in parentheses. Areas A, B, and C were impacted with the 56 VIP tooth, area E was impacted with the 46 TVIP tooth. Areas D and F were left for future testing. Figures B-5 and B-6 show the styles of teeth available for both series.

It was hoped that the individual coal blocks grouted into place within the heavy steel frame would behave as a single large block of coal. The initial blow to each block would shatter it, as evidenced by surface swelling around the impact point. There was little evidence of fracture progressing through the grout interface to neighboring blocks. Figures B-7 through B-16 show the target's appearance at various points between projectile runs.

The single point 56 VIP tooth penetrated about 5 inches on the first blow. Material broken loose formed a large conical hole. The surface layer of grouting over the target face tended to prevent broken material from falling away. Subsequent blows tended to produce more fines and less penetration.

The 46 TVIP (forked) tooth exhibited good penetration but a smaller breakout volume. Material tended to break towards the open center of the tooth to produce a square hole of constant width.

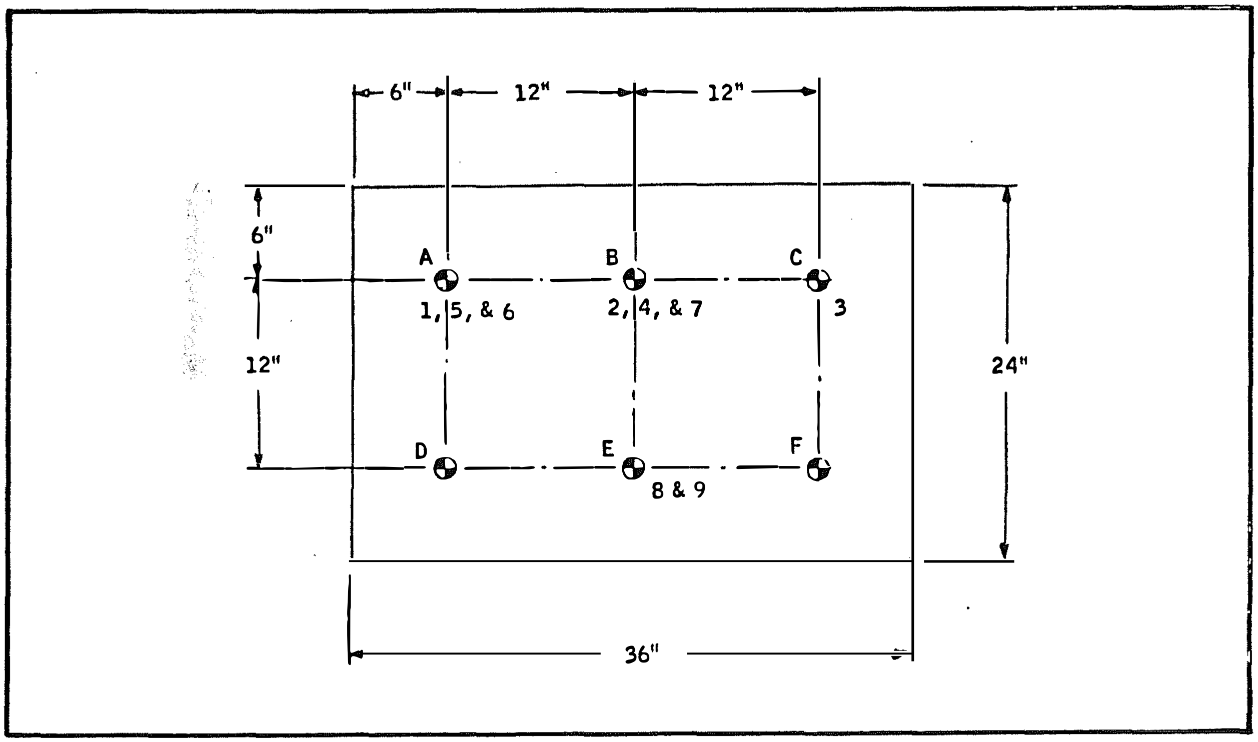
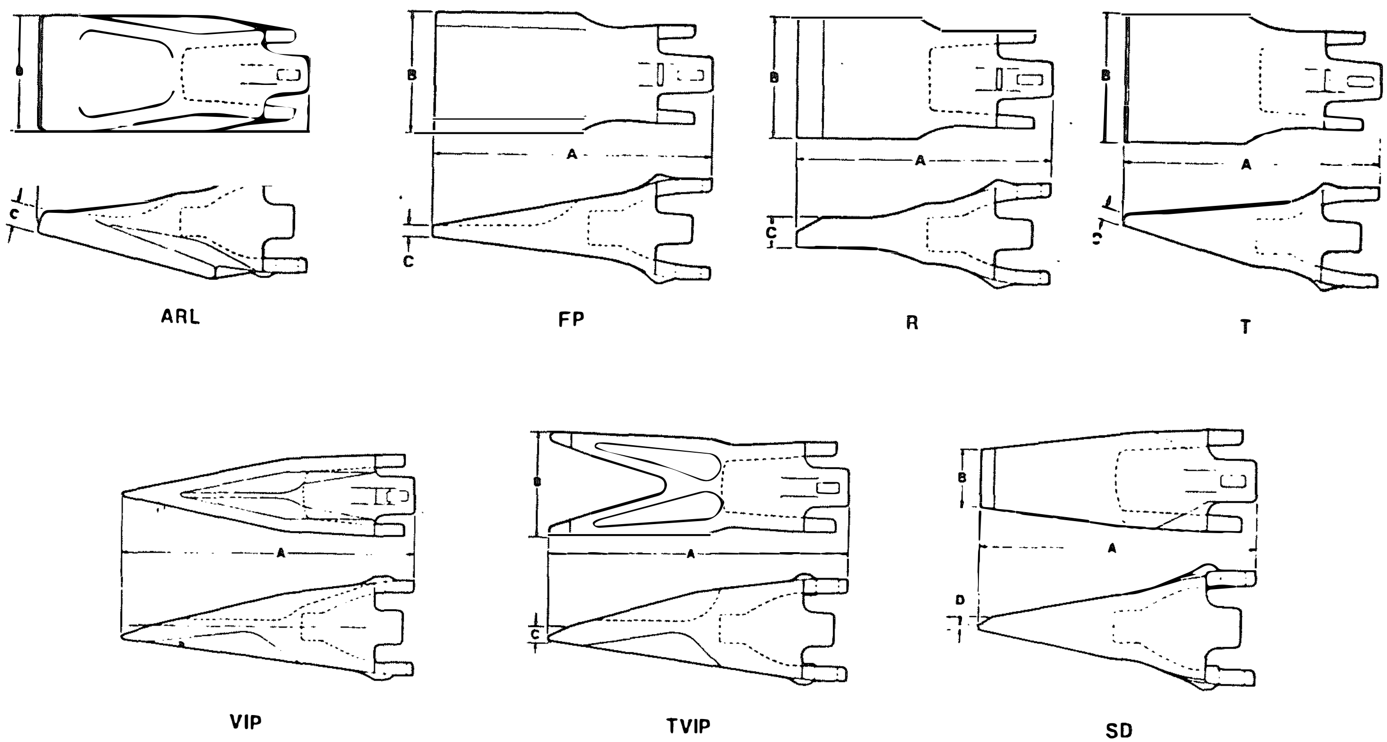


Figure B-4 LOCATION OF IMPACT POINTS (LETTERS) AND BLOW SEQUENCES (NUMBERS)

Table B-1 PENETRATION DEPTH IN INCHES

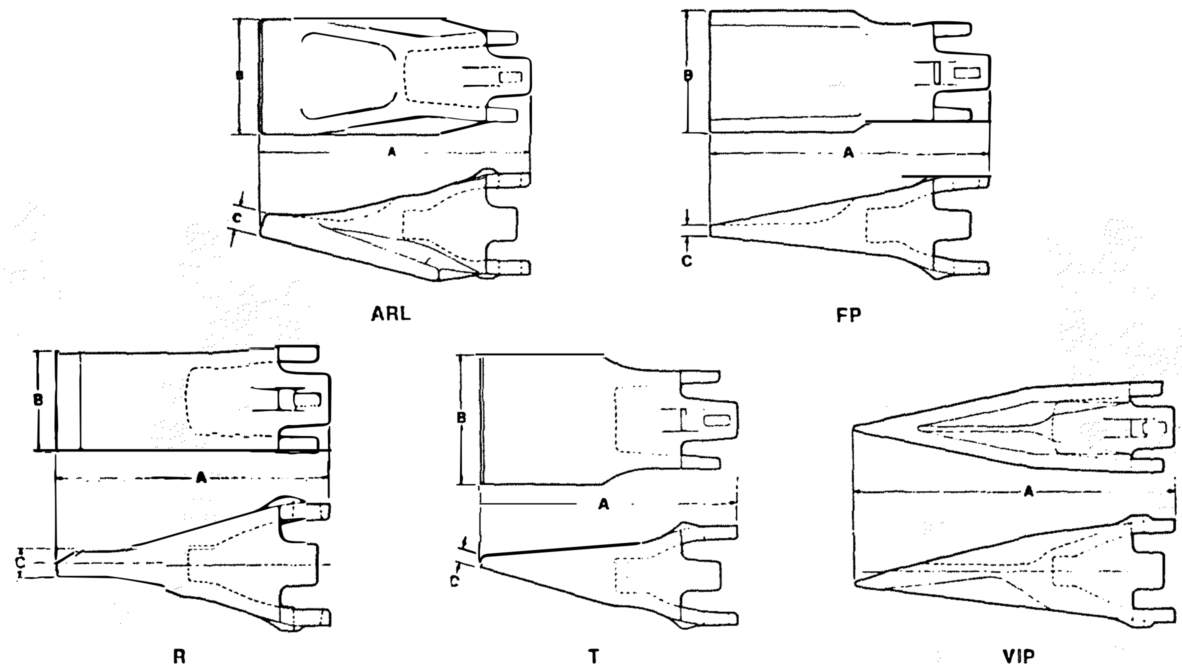
Tooth	56 VIP			46 TVIP
Area	A	B	C	E
1st blow	5 (1)	4 (2)	5 1/4 (3)	4 (8)
2nd blow	8 1/2 (5)	7 (4)	-	7 (9)
3rd blow	9 1/4 (6)	9 1/4 (7)	-	-



46 SERIES POINTS

PATTERN NUMBER	TYPE	A mm	A in.	B mm	B in.	C mm	C in.	WGT. kg.	WGT. lbs.	LOCKING SYSTEM PIN	LOCKING SYSTEM LOCK
46ARL	Centerline	295	11-5/8	127	5	26	1	11	25	46PN	46LK
46FP	Centerline	290	11-1/2	110	4-3/8	10	3/8	8	17	46PN	46LK
46R	Centerline	267	10-1/2	105	4-1/8	29	1-1/8	8	18	46PN	46LK
46SD	Centerline	277	10-7/8	57	2-1/4	14	9/16	7	16	46PN	46LK
46T	Non-Centerline	277	10-7/8	133	5-1/4	10	3/8	7	16	46PN	46LK
46TVIP	Non-Centerline	330	13	—	—	—	—	10	22	46PN	46LK
46VIP	Non-Centerline	330	13	—	—	—	—	8	18	46PN	46LK

Figure B-5 46 SERIES TOOTH STYLES



56 SERIES POINTS

PATTERN NUMBER	TYPE	A mm	A in.	B mm	B in.	C mm	C in.	WGT. kg.	WGT. lbs.	LOCKING SYSTEM PIN	LOCKING SYSTEM LOCK
56ARL	Centerline	368	14-1/2	159	6-1/4	29	1-1/8	21	47	56PN	56LK
56FP	Centerline	355	14	140	5-1/2	15	1/2	15	34	56PN	56LK
56R	Centerline	325	12-3/4	115	4-1/2	35	1-3/8	14	31	56PN	56LK
56T	Non-Centerline	330	13	135	5-3/8	15	1/2	15	33	56PN	56LK
56VIP	Non-Centerline	394	15-1/2	—	—	—	—	15	34	56PN	56LK

Figure B-6 56 SERIES TOOTH STYLES

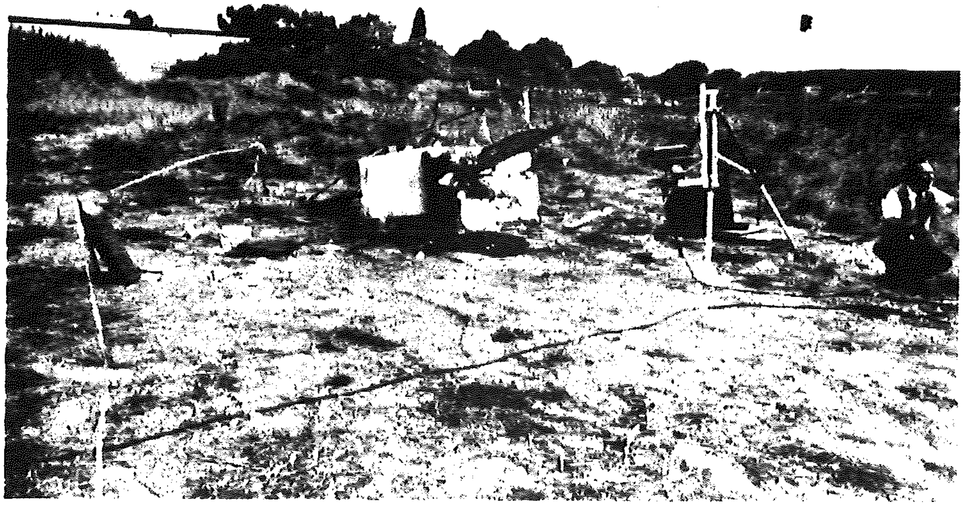


Figure B-7 FIRST IMPACT

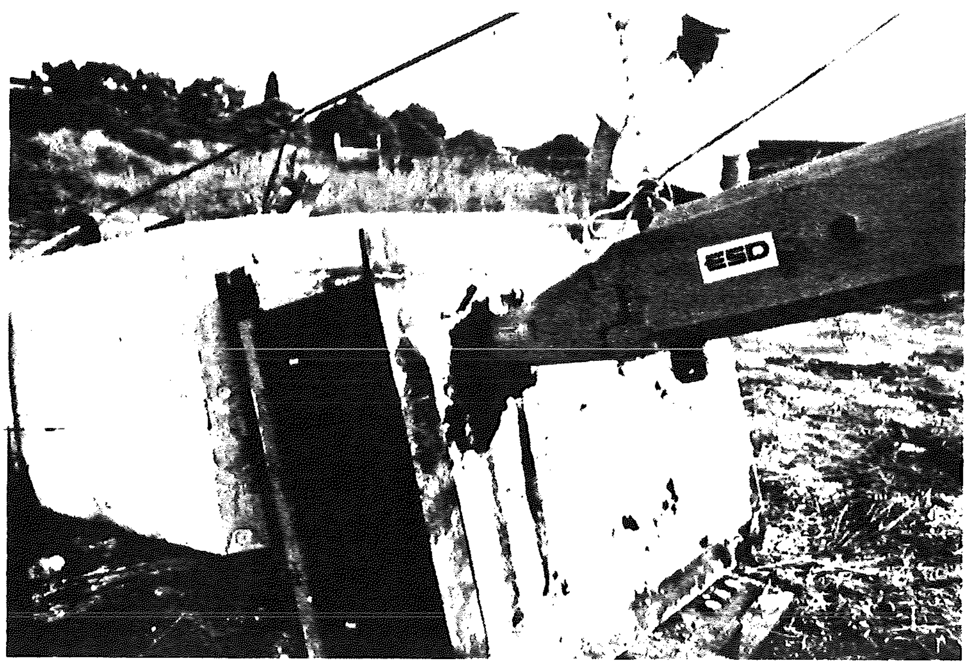


Figure B-8 FIRST IMPACT

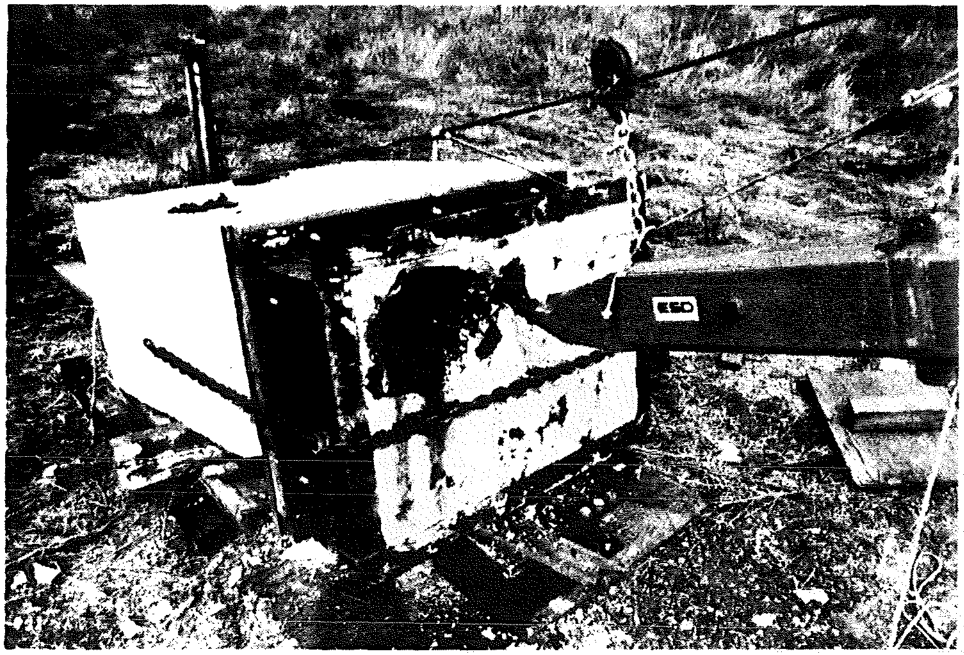


Figure B-9 SECOND IMPACT

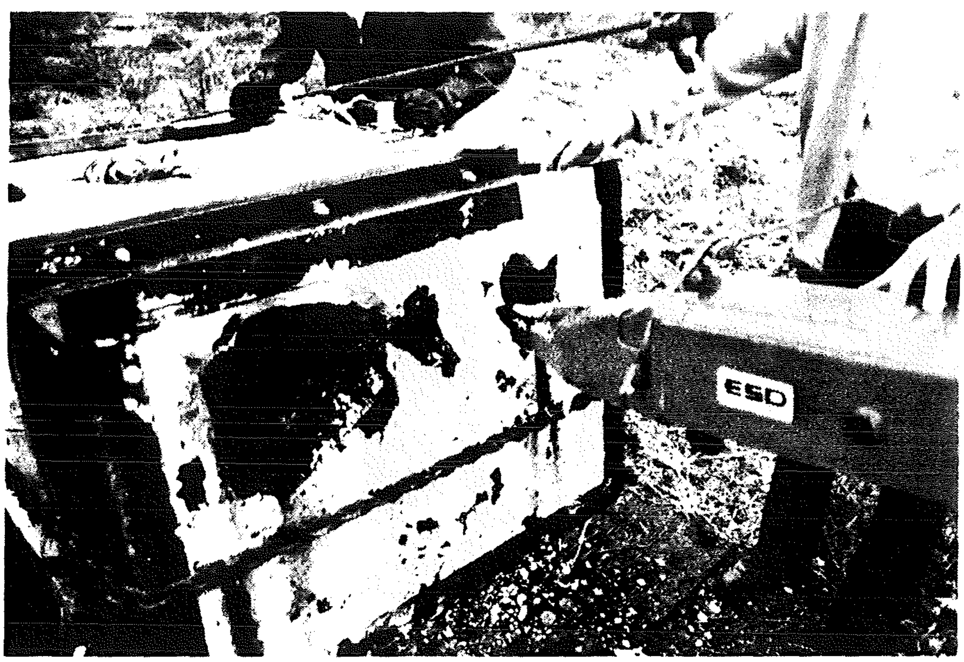


Figure B-10 THIRD IMPACT

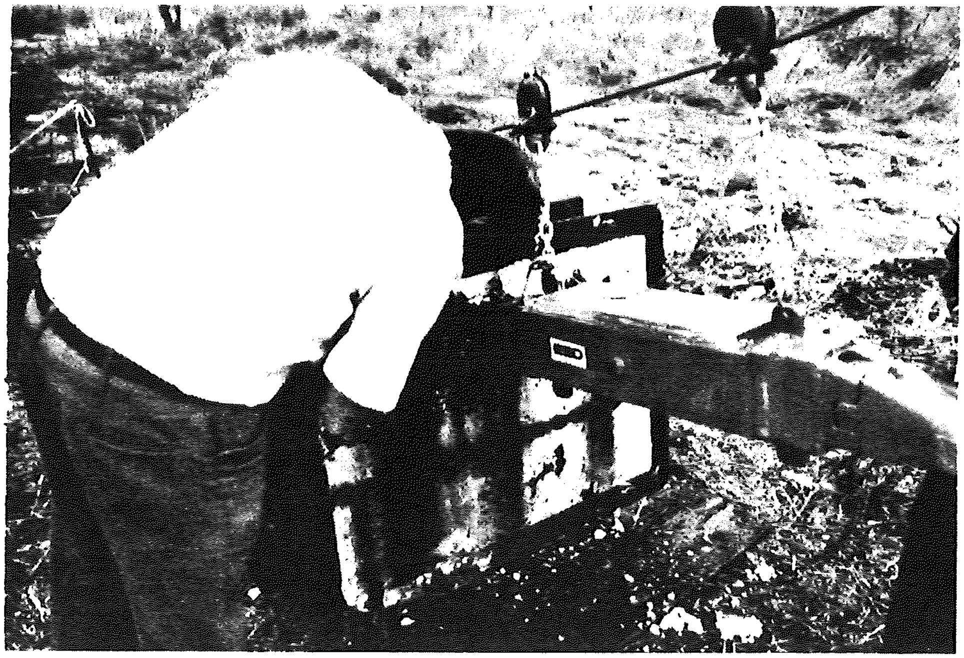


Figure B-11 FOURTH IMPACT

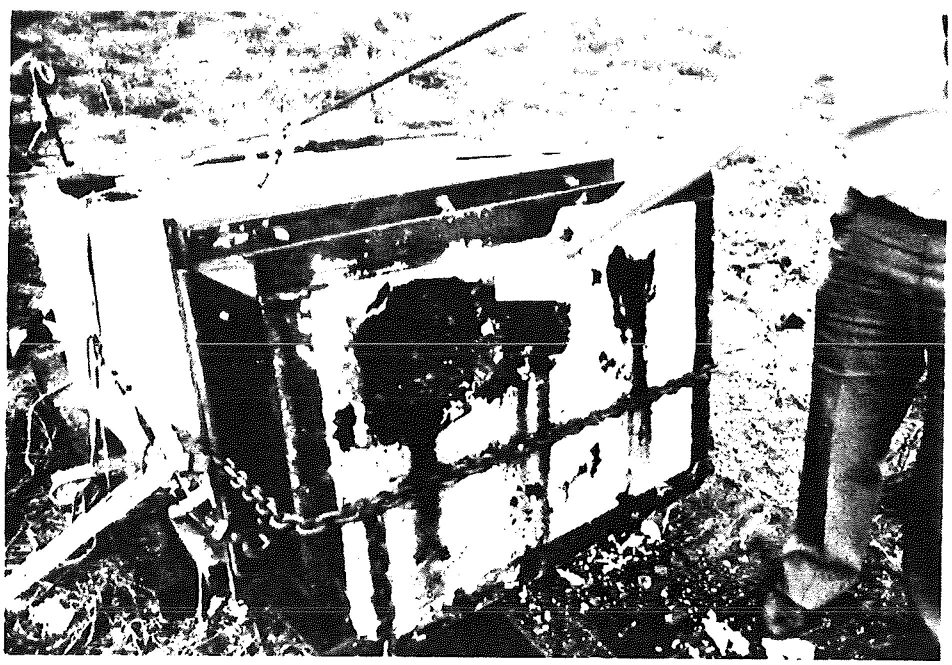


Figure B-12 FIFTH IMPACT

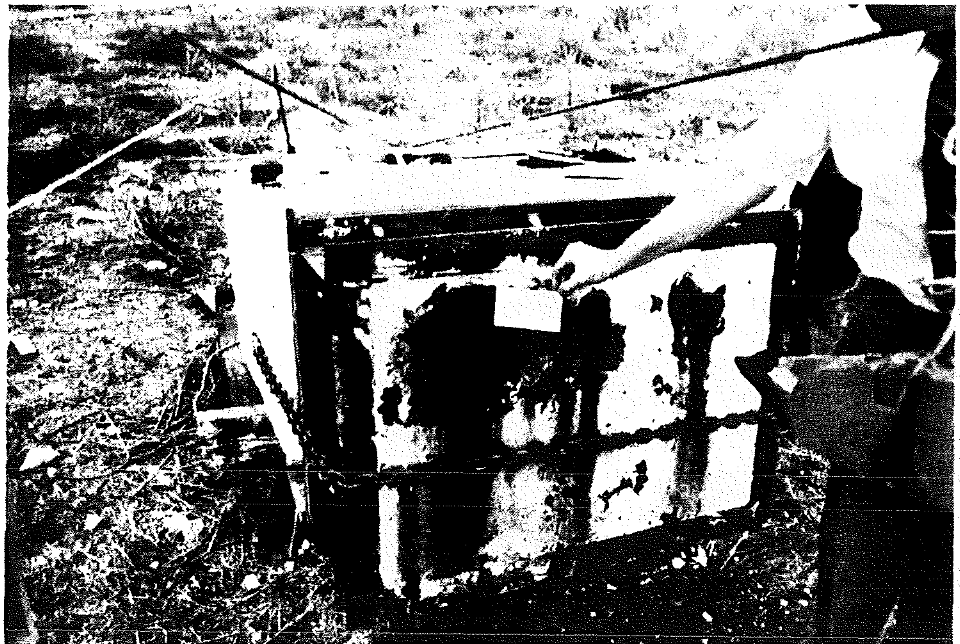


Figure B-13 SIXTH IMPACT

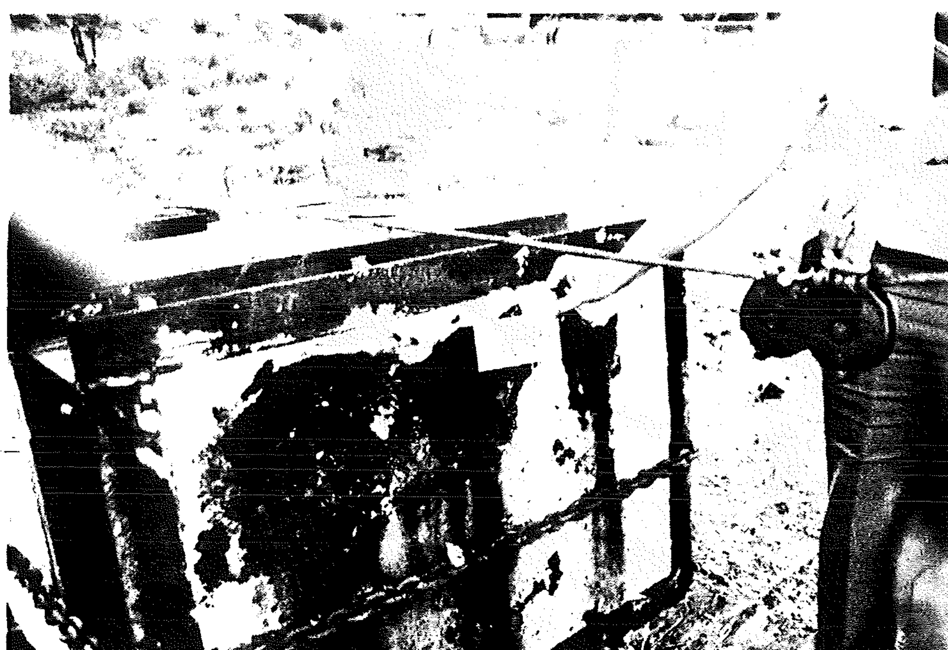


Figure B-14 SEVENTH IMPACT

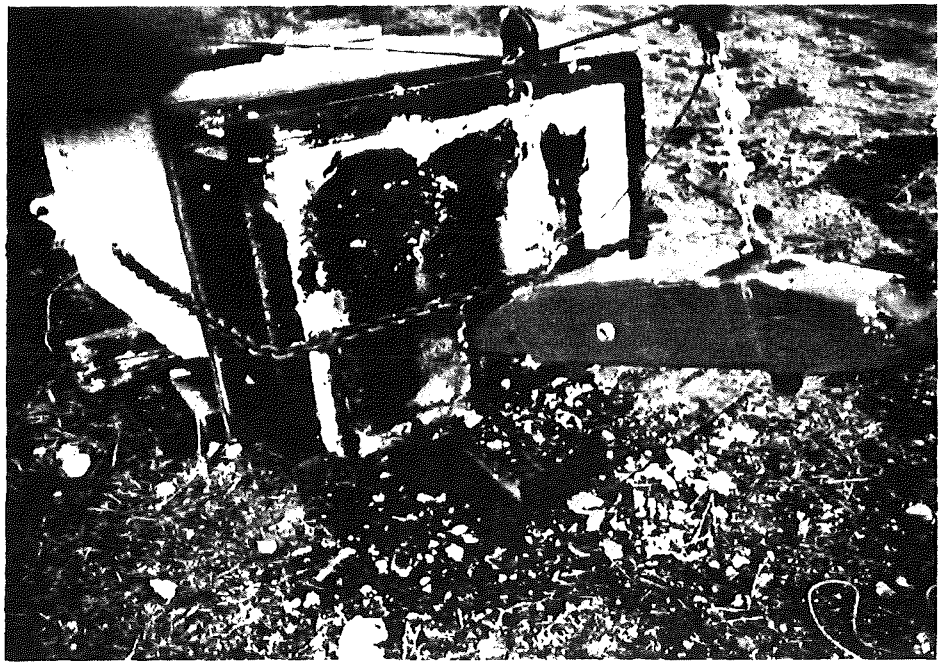


Figure B-15 EIGHTH IMPACT

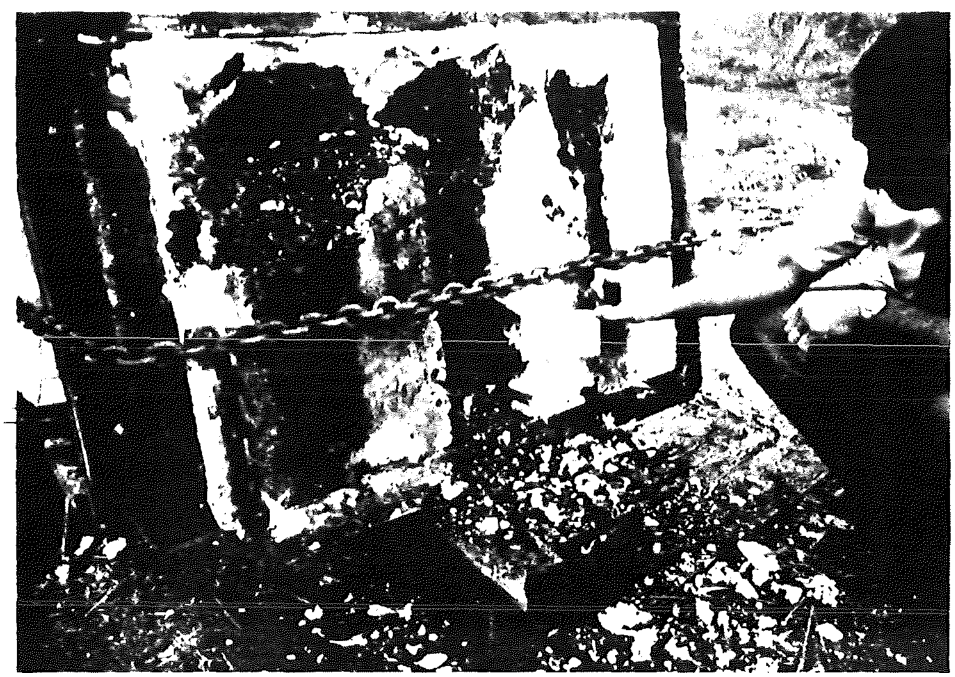


Figure B-16 EIGHTH IMPACT

Appendix C

CABLE ANCHOR TEST

APPENDIX C

CABLE ANCHOR TEST

C.1 INTRODUCTION

A number of methods for anchoring the track cable to the bottom of the pilot hole were investigated. These included expandable wedge bolts similar to those used for roof support in underground mining, expandable rubber packers similar to a thermos bottle plug, and various plastic, gypsum, and cement grouting systems.

Expansion bolts and rubber packers are not presently developed in a form useful for application as a cable anchor. Plastic grouting systems are not tolerant of water[1] which, in aquiferous formations, can be expected to collect at the bottom of the hole. Gypsum-based grouting systems will set and cure underwater but can eventually leach and weaken under continuous exposure to water.

Cement grouting systems, commonly used in the mining and construction industries, present the most attractive alternative. These offer high strength, fast curing times, and relatively low cost. The cement grouting mixture may be pumped into place from the surface through expendable tubing. Any water at the bottom of the hole would be displaced. No data, however, was found as to the strengths obtainable in coal.

C.2 OBJECTIVE

The principal objective, then, was to determine anchor strengths obtainable in coal as a function of grout length and cure time. Although higher strengths could possibly be achieved using specially serrated solid bars, it was decided to use plain wire rope in the "as received" condition, without attempting to remove the protective coating of lubricant.

A secondary objective was to evaluate the stability of the hole and any special problems in drilling the holes down-dip, inserting the cable into the hole, and pumping the grouting mixture to the bottom.

C.3 RESULTS

The test was conducted at the demonstration mine. Two-inch-diameter holes were drilled 17 feet deep into and parallel to a coal seam with a 22-degree downward dip. This was accomplished with a prototype portable, hand-held, air-powered drill developed by the USBM for anthracite probing. Plastic tubing was taped to the 1-inch-diameter, 6 by 19 IWRC wire rope prior to its

1. "Pumpable Rock Bolt," Engineering and Mining Journal, August 1973, Pages 76-79.

insertion. The wire rope and tubing was fed into the hole almost effortlessly, friction nearly balancing its weight. Various amounts of a quick-setting cement grouting mixture were pumped down each hole. After curing, the anchored cable was tensioned with a hollow hydraulic ram.

The results are shown in Figure C-1. The two lower curves indicate a linear relationship between strength and grout length. The strength per unit length appeared to more than double when cure time was increased from 16 to 48 hours.

Since the recommended volumetric mixing ratio of 3 parts cement to 1 part water was found to be difficult to work with (especially with the longer grouted lengths), a thinner 2:1 mixture was used. The strength achieved after a 48-hour cure was approximately 17 percent less than that of the recommended 3:1 mixing ratio.

Interestingly enough, none of the anchors failed catastrophically, but continued to yield at 60 to 70 percent of the maximum load achieved.

C.4 DISCUSSION

C.4.1 Hole Drilling

A lightweight, portable drill developed by Hamilton Engineering, Seattle, Washington, for the Bureau of Mines was used to drill the holes for the anchor (Figures C-2 and C-3). The all-pneumatic drill is capable of drilling 1-3/4-inch or 2-inch holes up to 80 feet deep. Holes at any angle are possible since the drill can flush the cuttings from the hole with either air or water. The weight of the drill is supported by two rubber anchors located in two holes to the sides of the main hole.

The anchors are expanded by tightening a steel bolt on each. A steel collet then expands, causing the rubber sleeve of the anchor to press against the inside of the hole. This pressure provides the necessary reaction which allows the rubber anchors to resist the high feed forces of the drill. Two- or 4-foot-long drill pipes are attached to an air motor and fed into the coal face by an air-operated ball screw mechanism.

The test site was located on a bench in front of an 18-foot coal seam at FMC's Skull Point Mine near Kemmerer, Wyoming. Two-inch-diameter holes were drilled into the coal parallel to the seam roof at an angle of approximately 22 degrees. The 17-foot-deep holes were drilled in an average of 49 minutes, including move and setup. The 2-inch hole for the rubber anchor was drilled first; in order to save setup time, only one rubber anchor was used. Though the developer of the drill recommends using a separate air drill to make this hole (for ease of handling), the same drill was used instead. An Ingersoll-Rand gasoline-powered compressor, rated at 150 cfm and delivering 100-psi air, was used to power all equipment.

The drill was first set on two 8- by 8-inch wooden timbers. The pilot hole was drilled about 2 feet deep parallel to the bench. This type of drilling was difficult because of the high feeding forces mentioned earlier. The forces were resisted manually by pushing against a protrusion on the rubber anchor assembly with a shovel acting as a lever. This arrangement was

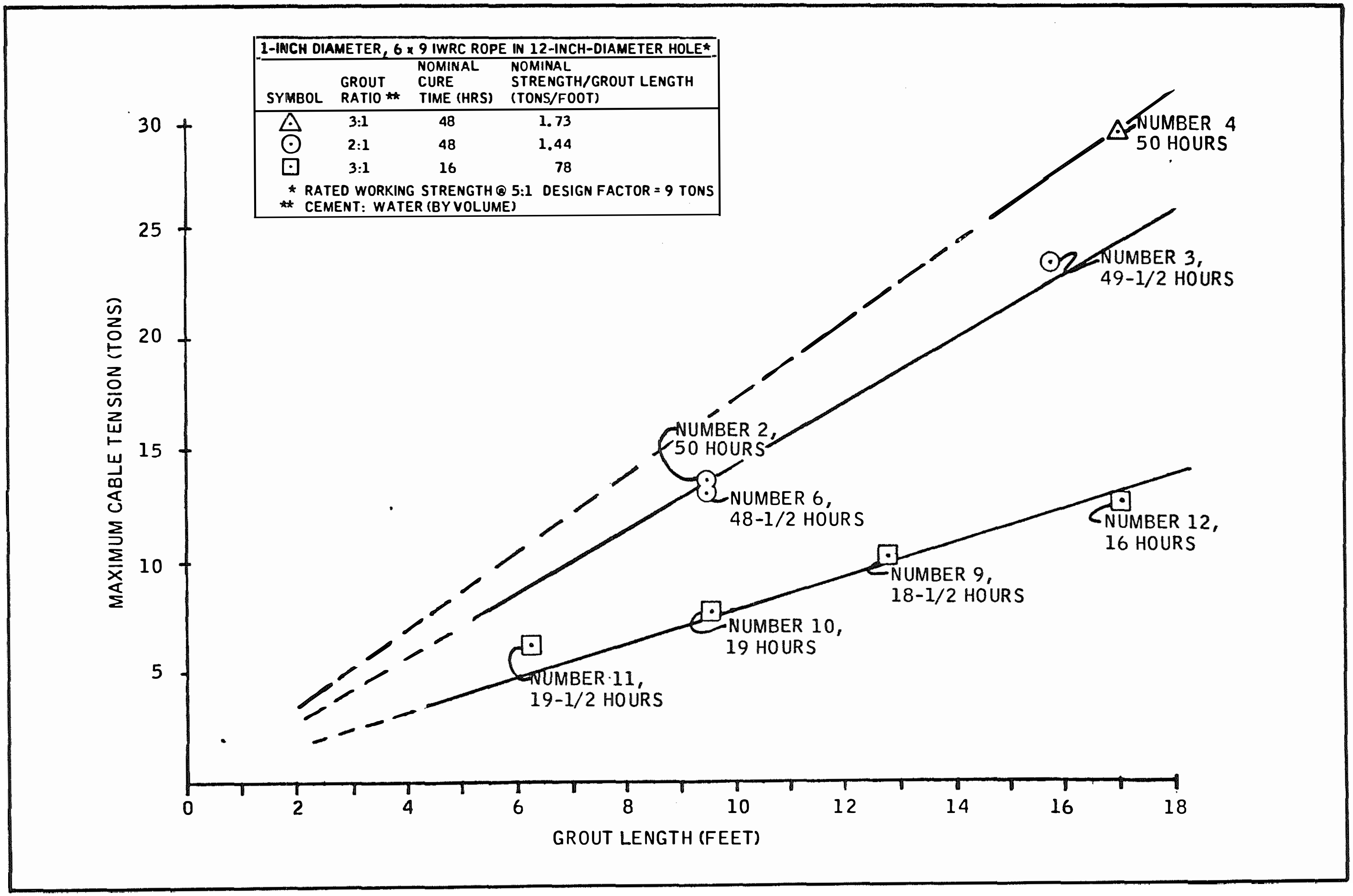


Figure C-1 ANCHOR STRENGTH VERSUS GROUT LENGTH

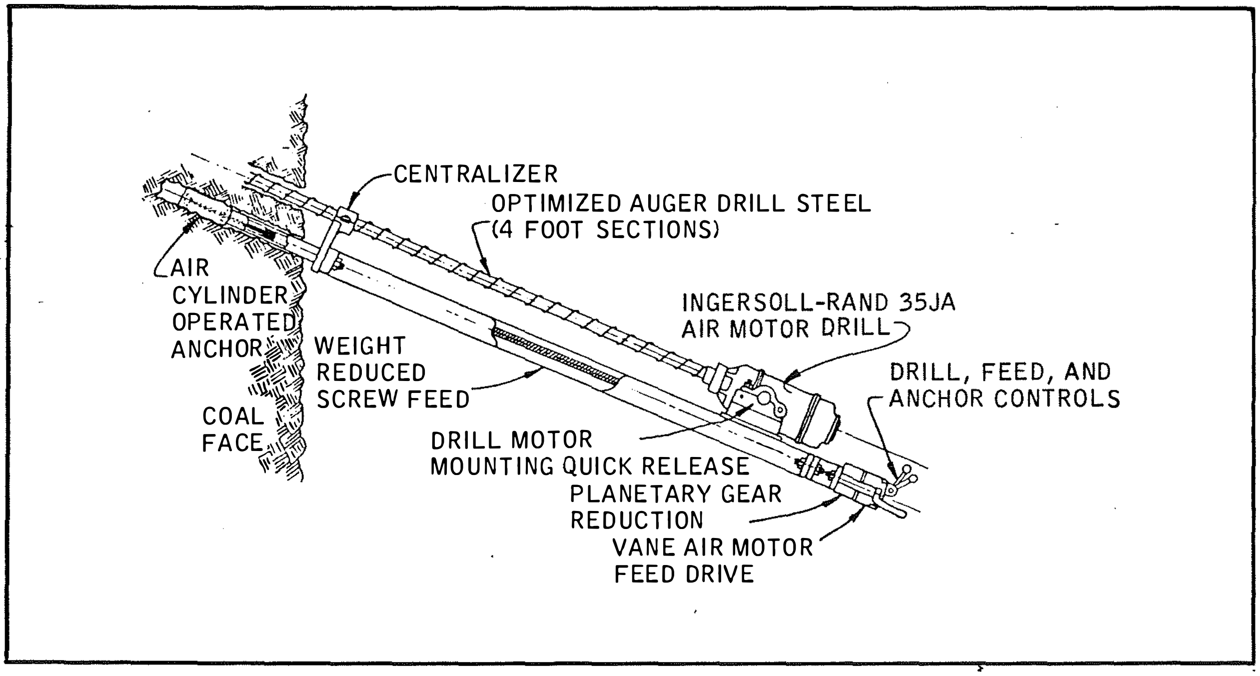


Figure C-2 LIGHTWEIGHT, PORTABLE DRILL

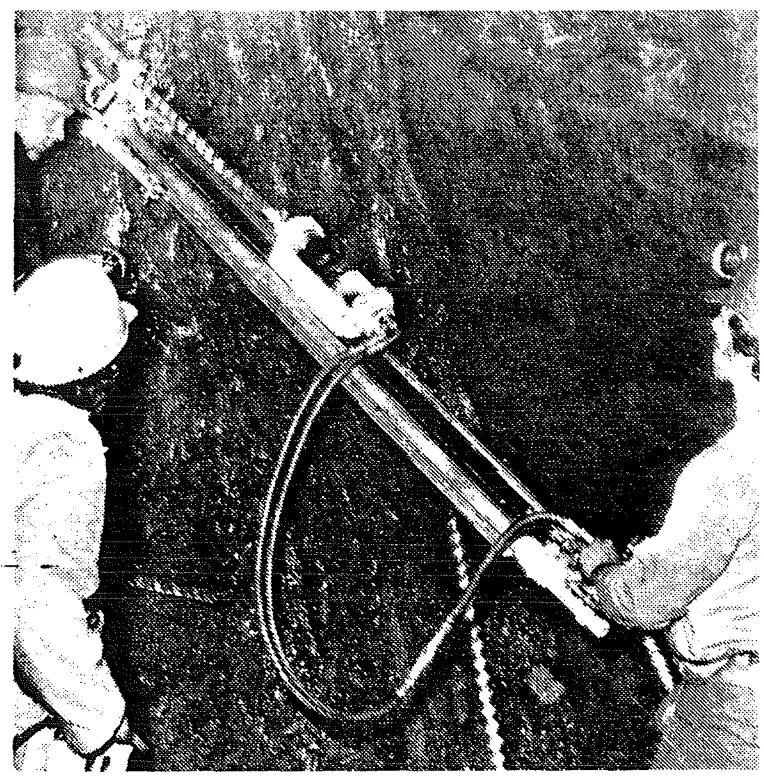


Figure C-3 OPERATION OF LIGHTWEIGHT PORTABLE DRILL UNDERGROUND satisfactory for the test, though the drilling of this hole tended to be slower than that of the main anchor hole. When drilling was complete, the rubber anchor was inserted into the hole and tightened.

Though the drill was designed to be self-supporting when both of the rubber anchors were used, only one anchor was used in the tests. In order to provide additional support, a large wooden spool was rolled underneath the controls end of the drill. The position of the spool was adjusted until the angle of the drill was about 22 degrees. The spherical pivot was tightened, and the drilling of the main test hole began. The coal was cut with a four-winged bit attached to the first drill pipe. Six of the 4-foot lengths of flightless drill pipe provided a hole 17 to 17-3/4 feet deep, depending on how deep the rubber anchor was inserted. Full air flush was used to remove the cuttings from the hole and to prevent clogging. A slow, continuous feed was possible for the majority of the time. Occasional plugging or binding of the drill string required stopping or reversing the drill until the condition was fixed. When the desired depth was reached, the drill string was uncoupled and manually withdrawn from the hole, which proved to be faster than using the drill's powered reverse. After removing the drill string, the anchor head assembly was disconnected, and the anchor was loosened and removed. The hole was measured by inserting a 1/2-inch-diameter ruled pipe. After recording the depth, the hole was plugged with a rag to prevent any material from entering.

C.4.2 Cable Insertion

After several holes were drilled, the cable was inserted into the hole in preparation for grouting. Twenty-five-foot lengths of 1-inch-diameter improved plow-steel wire rope with an independent wire rope core (IWRC) were used for the test. Polyethylene grout tubing (1/2-inch outer diameter by 3/8-inch inner diameter) was tied to the cable about 2 inches from the end. If more than a few hours had passed since drilling, the hole was remeasured and the depth was recorded. Far from causing difficulties, the manual insertion of the cable into the hole was accomplished quite easily. With gravity assistance, little manual effort was necessary.

In the hope of maintaining a large enough clearance to allow for measuring the depth of the grout, the tubing was taped to the cable in several places. Unfortunately, the tubing could not be prevented from resuming its coiled shape and spiraling around the cable. Depth measurement after cable insertion was not possible.

C.4.3 Grouting

A portland cement-based grout was used to anchor the wire rope at the bottom of the hole. The powdered material used was MagnaRok Weatherproof Expansion Cement (manufactured by Metalcrete Manufacturing Company, Cleveland, Ohio). MagnaRok possesses three characteristics which were important for this application:

- According to independent laboratory tests conducted by Pittsburgh Testing Laboratory (June 18, 1969), MagnaRok cures rapidly to high strengths. It reaches the maximum compressive strength of cement in 6 hours (3,000 psi) and 7,000 psi in 24 hours at a 4:1 cement-to-water ratio by volume. The maximum compressive strength of 10,200 psi (also at a 4:1 ratio) is obtained in 28 days, the curing time for standard portland cement. The quick cure prevents delays during actual operation, and the high strength reduces the amount of grout required.

- Because the coal seams at the demonstration mine contain water-bearing pockets, the grout's immunity to erosion by water was a very important consideration in selecting this material for the anchor.

- Since the grout also expands slightly, it has greater holding strength than a nonexpanding grout and fills any voids.

Metalcrete recommends mixing MagnaRok at a 3:1 cement-to-water ratio for pouring and a 4:1 ratio for troweling. Since our application required pumping the grout to the bottom of the hole, a 3:1 ratio was used for the first hole. Seventy-five pounds of cement (1-1/2 50-pound pails at 5 gallons per pail) were mixed with 2-1/2 gallons of water in a wheelbarrow. A shovel was used for stirring the mixture. The 4-1/2 gallons of grouting mixture produced turned out to be quite thick, though not as thick as the paste that the directions indicated it should be. The mixture was then poured into one of the empty 50-pound pails for pumping. A small hand pump (Model T6Z, Williams Form Engineering Corporation, Grand Rapids, Michigan) was inserted into the pail. A hose was attached with standard fittings to the pump and the grout tube.

The grout began to set in about 10 minutes and could not be pumped with the small hand pump available; however, a pneumatic piston pump was used to pump the grout before complete setting had occurred. Even so, the 3:1 mixture was judged to present enough of a pumping problem to warrant testing a 2:1 mixture in some of the holes. The remainder of the holes were grouted with the powered pump, since it was significantly easier to use than the hand pump. Clear water was run through the pump after each batch to ensure that the grout would not set.

The 50-pound grout pails held 5 gallons of water. Since less than 3 gallons of grout mixture would fill up the hole, a 5-quart container was also used when pumping the smaller quantities of grout. However, because of the rapid setting time, the amount of grout pumped was only measured to the nearest quart when the small container was used.

C.4.4 Cable Tensioning

After cure times ranging from 16 to 50 hours, the wire rope was ready for tension testing. Figure C-4 shows the 60-ton, hollow-bore hydraulic jack (Model T7Z, Williams Form Engineering Corporation) used to apply the test loads to the cable. The cable was threaded through the center of the ram and clamped on the outside with a Sauerman Tiger-Grip cable clamp (Figure C-5).

The Sauerman cable clamp consists of two wedges inside a shoe. Two of the inside walls of the shoe slant to match the slope of the wedges. The wedges are normally clamped to the cable with a U-bolt arrangement. When the load pulls on the eye of the shoe, the wedges are forced further down the sloped planes inside the shoe and exert an increased clamping force upon the cable.

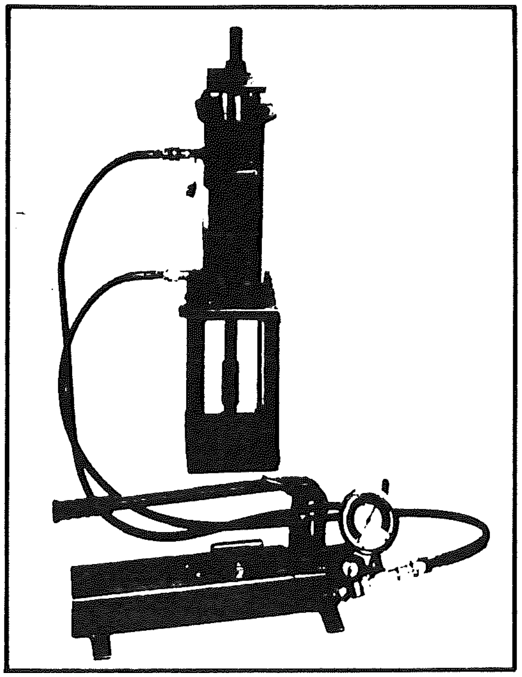


Figure C-4 HYDRAULIC TENSIONING JACK

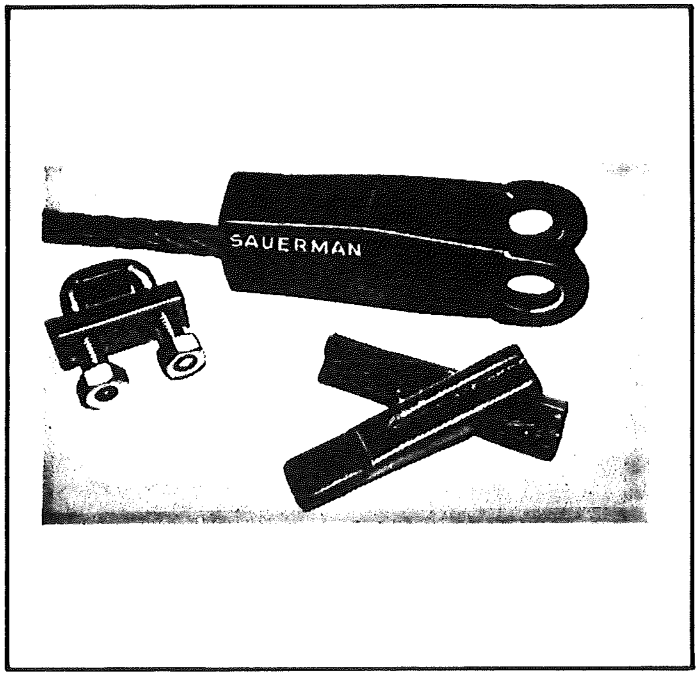


Figure C-5 CABLE CLAMP

It had been planned to transfer the cable load from the clamp shoe to the ram through an adapter in the form of a short length of rectangular steel tubing. This arrangement precluded the use of the U-bolt which was found essential to the operation of the cable clamp. The shoe was modified by removing the two ears and a second set of back-up wedges were clamped to the cable with a pair of U-bolts. This arrangement proved quite satisfactory, limiting slippage of the clamp assembly to 0.2 inch or less on seven of the eight tests. (The second test slipped 0.65 inch, perhaps because of insufficient tightening and inexperience with the clamping arrangement.)

C.4.5 Apparent Slip

Figures C-6 and C-7 show the apparent slip of the test anchors under load in terms of ram extension. The cables were unloaded at the end of each test to determine the net amount of apparent slip. Individual cables were also unloaded at selected points both before and after the onset of plastic yielding during the run. A considerable amount of elastic recovery is evident.

There are a number of sources of plastic slip: the grout interface between rope and coal, the crushing of coal at the base of the ram, and slippage of the cable clamp at the ram head. Most of the slippage appeared to occur within the grouting. Smaller amounts are attributable to cable clamp slip and coal crushing. Constructional stretch of the rope itself would be negligible, considering the relatively light loading and short lengths.

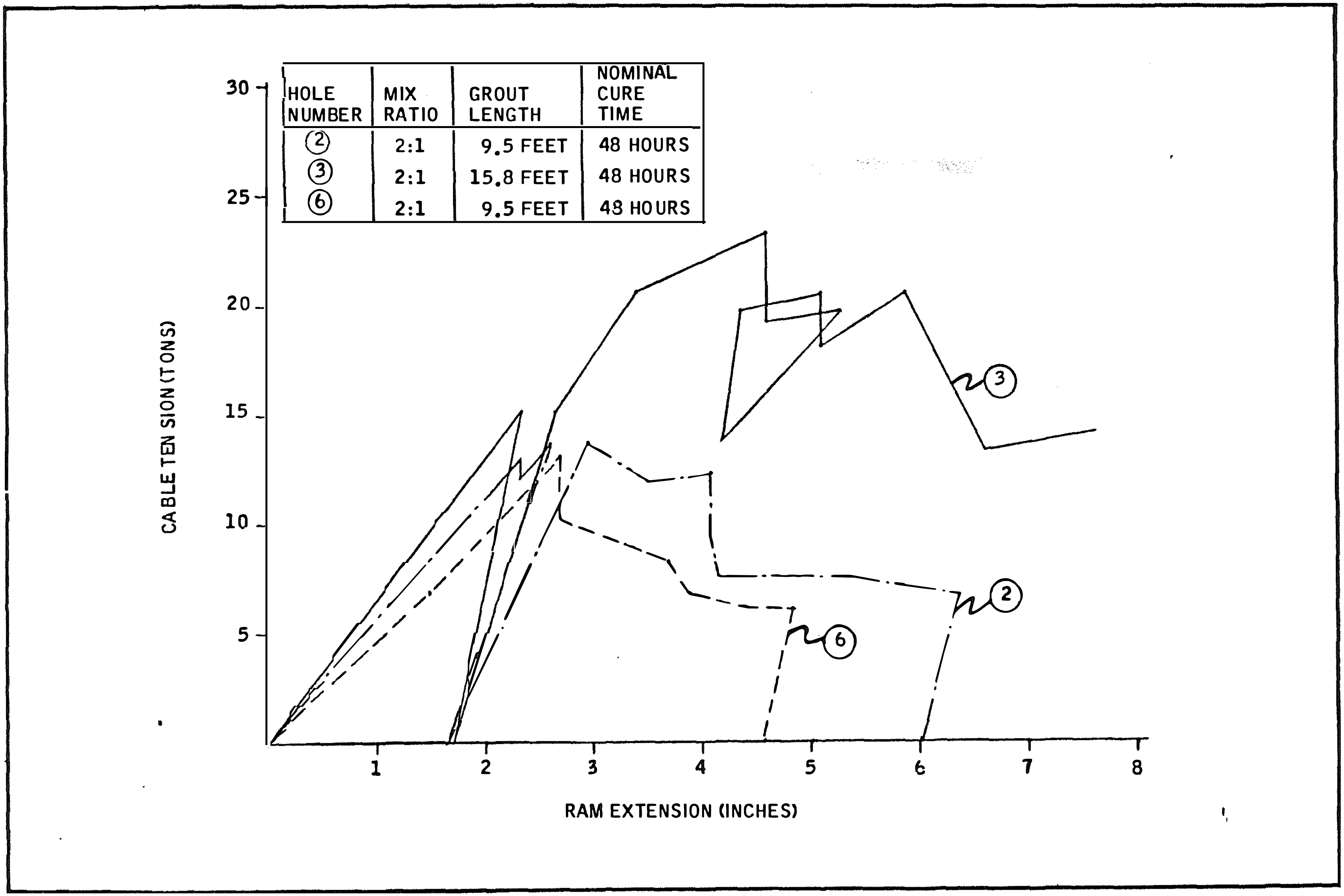
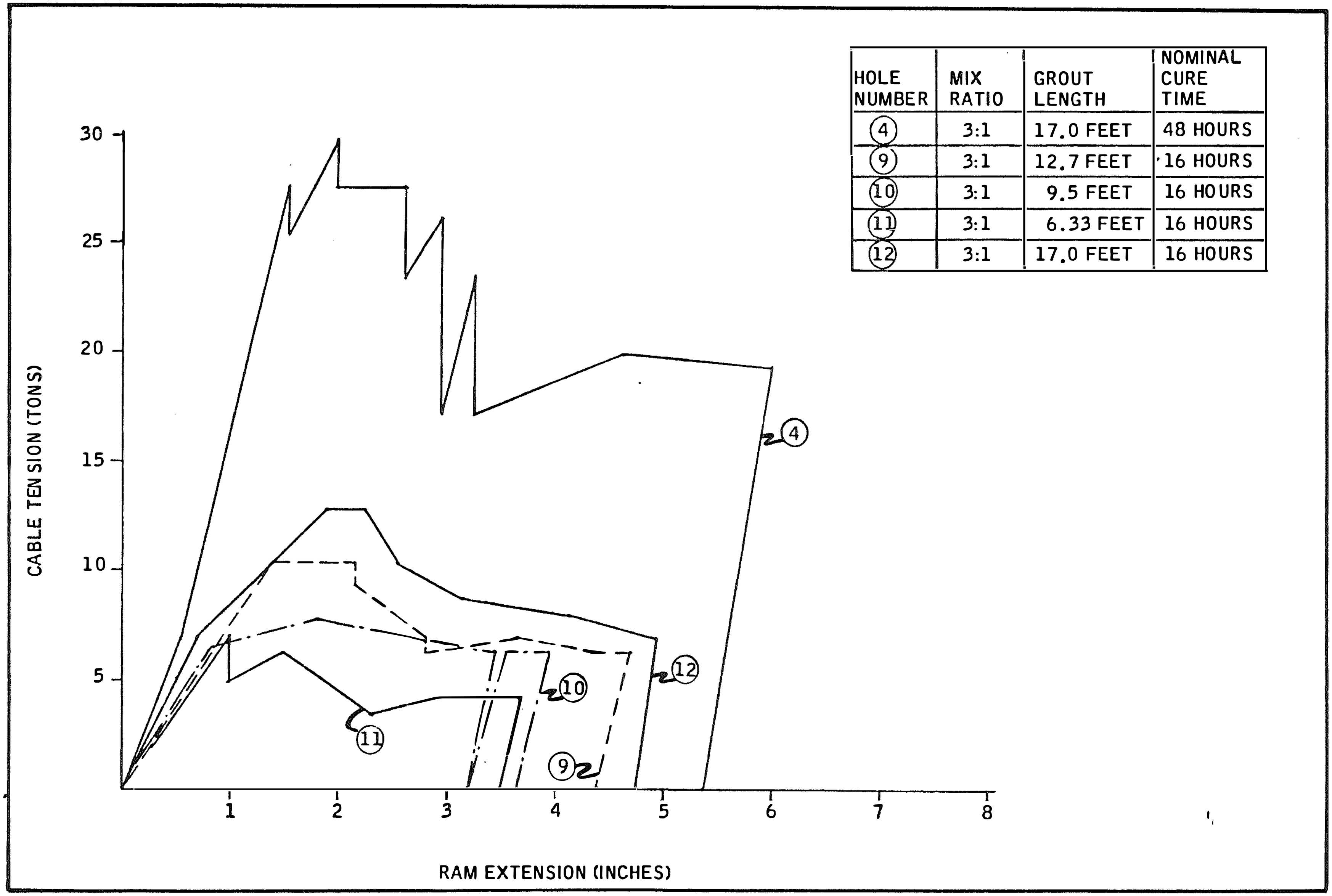


Figure C-6 APPARENT ANCHOR SLIP



HOLE NUMBER	MIX RATIO	GROUT LENGTH	NOMINAL CURE TIME
4	3:1	17.0 FEET	48 HOURS
9	3:1	12.7 FEET	16 HOURS
10	3:1	9.5 FEET	16 HOURS
11	3:1	6.33 FEET	16 HOURS
12	3:1	17.0 FEET	16 HOURS

Figure C-7 APPARENT ANCHOR SLIP

Appendix D

SLACKLINE PRODUCTIVITY
ANALYSIS, MOTORIZED BUCKET

APPENDIX D

SLACKLINE PRODUCTIVITY ANALYSIS, MOTORIZED BUCKET

D.1 SHIFT 1

During the first shift, the hole is mined 266 feet deep and the full seam thickness. In order to estimate productivity, the hole is divided into nine separate regions, 1A through 1I. Figure D-1 shows the regions graphically and Table D-1 summarizes the depth and productivity. Figure D-2 shows the Production Time Chart for the first shift.

D.1.1 Shift 1 - Productivity Development

Tables D-2 through D-10 detail the production time required for each region. After assuming a mining depth for the region, the table values were calculated as follows:

- BUCKET TRAVEL DISTANCE = $\frac{\text{End depth - start depth}}{2} + 16$ feet

  (The bucket dumps 16 feet outside the hole mouth.)

- NUMBER OF CYCLES

  -- Regions: 1A, 1B, 1D, and 1G: Number of cycles = $\frac{\text{Depth}}{\text{5.8 feet/cycle}}$

  -- Regions: 1C, 1E, 1F, 1H, and 1I: Number of cycles = $\frac{\text{Volume (tons)}}{\text{20 tons/cycle}}$

- GRAVITY RETURN

  -- Regions: 1A, 1B, 1D, and 1G:

  Distance = Average bucket travel distance

  -- Regions: 1C, 1E, 1F, 1H, and 1I:

  $$\text{Distance} = \frac{\text{End depth - Start depth + 20.8 ft}}{2} + 16 \text{ ft}$$

  $$\text{Time} = \frac{\text{Distance}}{\text{15 ft/sec}}$$

- DECELERATION = 12 seconds (including 10 seconds to approach face)

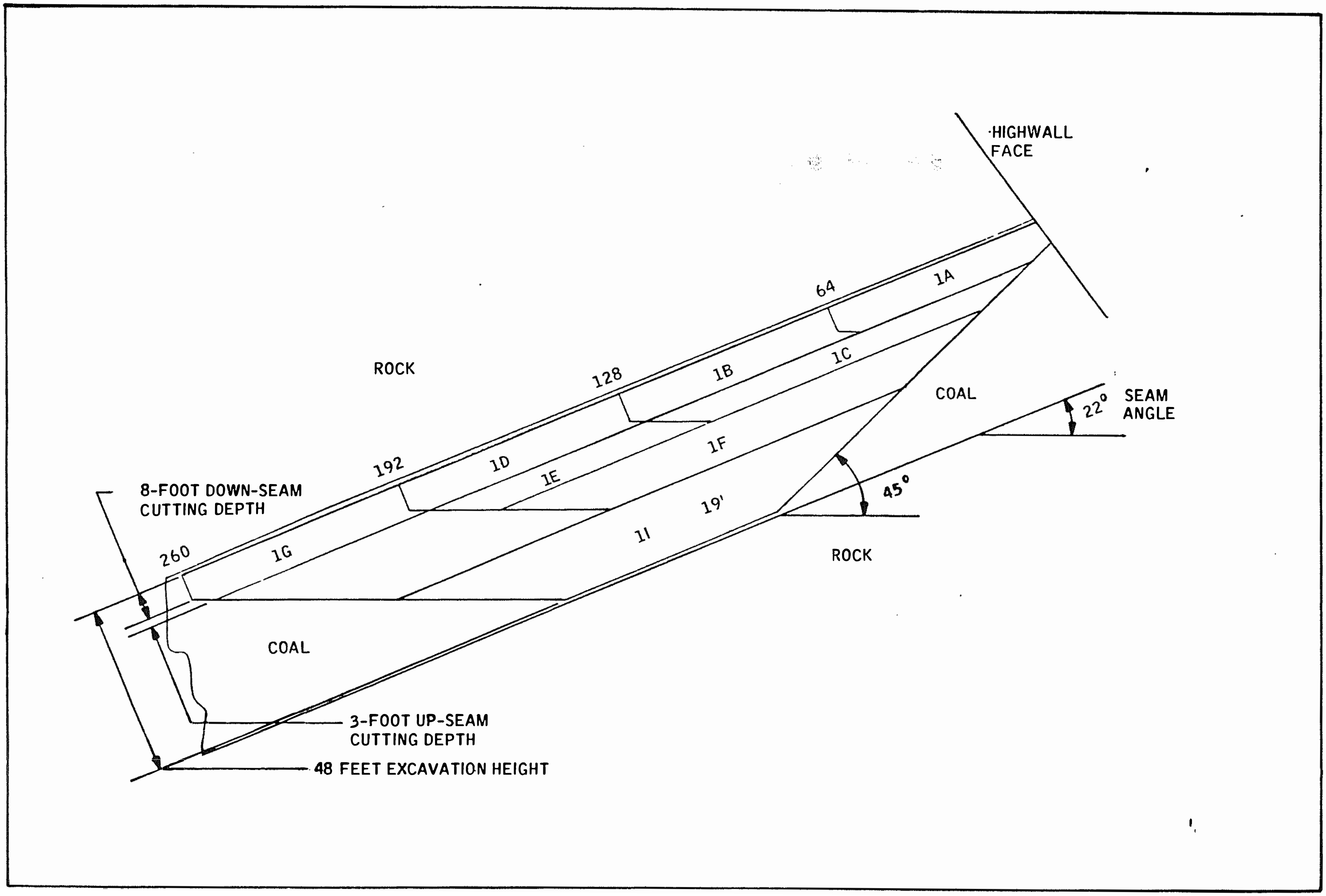


Figure D-1 MOTORIZED SLACKLINE EXCAVATOR, SHIFT 1 MINING PLAN (50-foot seam)

Table D-1 SLACKLINE WITH MOTORIZED BUCKET - PRODUCTION SUMMARY

Region	Shift 1									
	1A	1B	1C	1D	1E	1F	1G	1H	1I	Total
Average depth, feet	0-64	64-128	16-108	128-192	108-172	37-149	192-266	156-230	78-184	
Average height feet	0-11	0-11	11-17	0-11	11-17	17-29	0-11	17-29	29-48	
Tons Produced:										
Down-Seam Rip	161	161	0	161	0	0	190	0	0	673
Up-Seam Rip	59	59	180	59	120	420	70	420	620	2,007
Total	220	220	180	220	120	420	260	420	620	2,680
Cumulative Tonnage	220	440	620	840	960	1,380	1,640	2,060	2,680	
Cycles Required	11	11	9	11	6	21	13	21	31	134
Average Cycle Time, minutes	2.5	2.7	2.6	2.9	. 2.8	2.7	3.0	3.0	2.8	2.8
Total Production Time, minutes	28	30	23	33	17	56	40	62	86	
Cumulative Production Time minutes	28	58	81	114	131	187	227	289	375	
Production Rate, tons/minute	7.9	7.3	7.8	6.9	7.1	7.4	6.5	6.7	7.1	
Cumulative Production Rate, tons/minute	7.9	7.6	7.7	7.4	7.4	7.4	7.2	7.1	7.1	

ELEMENT NO.	DESCRIPTION	TIME (MIN) IND.	TIME (MIN) CUM.
1	TRANSPORT OPERATORS TO SITE	15	15
2	PRE-OP MAINT. CHECKS AND SERVICES	15	30
3	DIG REGION 1A 0-64 FEET, 11 FEET DEEP	28	58
4	DIG REGION 1B 64-128 FEET, 11 FEET DEEP	30	88
5	DIG REGION 1C 0-128 FEET, 17 FEET DEEP	23	111
6	REST PERIOD	10	121
7	DIG REGION 1D 128-192 FEET, 11 FEET DEEP	32	153
8	DIG REGION 1E 128-192 FEET, 17 FEET DEEP	17	170
9	DIG REGION 1F 128-192 FEET, 29 FEET DEEP	57	227
10	START REGION 1G 192-266 FEET, 11 FEET DEEP	13	240
11	LUNCH	(30)	
12	COMPLETE REGION 1G	27	267
13	DIG REGION 1H 192-266 FEET, 29 FEET DEEP	63	330
14	START REGION 1I 192-266 FEET, 48 FEET DEEP	30	360
15	REST PERIOD	10	370
16	COMPLETE REGION 1I	58	428
17	MISCELLANEOUS DELAYS	22	450
18	POST-OP MAINT. CHECKS AND SERVICES	15	465
19	TRANSPORT OPERATORS TO ENTRANCE	15	480

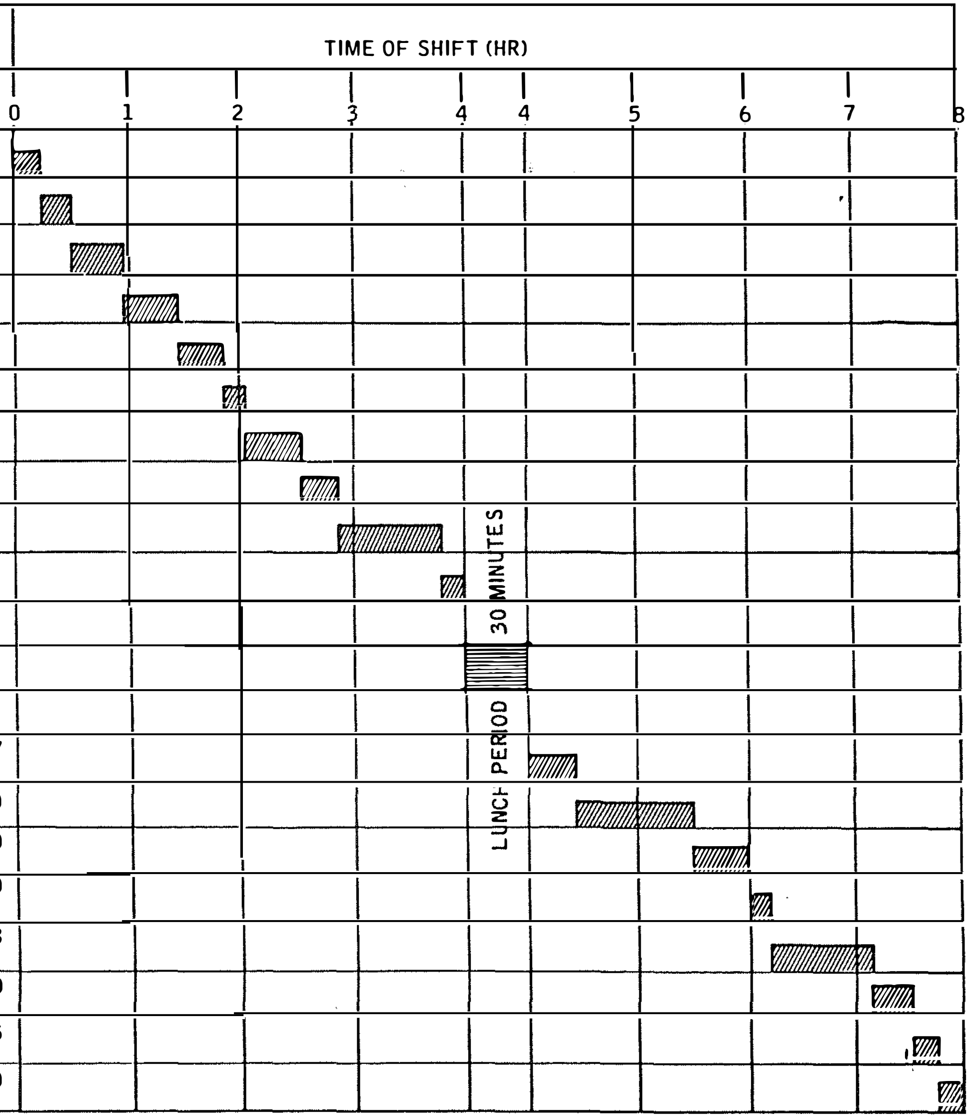


Figure D-2 SLACKLINE WITH MOTORIZED BUCKET PRODUCTION TIME CHART, FIRST-SHIFT OPERATION

Table D-2 SLACKLINE, MOTORIZED BUCKET (REGION 1A)

Hole Depth: Start 0 Feet
End 64 Feet   Length 64 Feet
Average Bucket Travel Distance: 48 Feet
(includes 28 feet beyond entrance)
Number of Cycles: 11
Bucket Size: 8'H x 8'W x 20'L   Capacity: 32 tons full, 20 tons working

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return: 48 Feet Rate = 15 feet/second (average)	3	
2	Deceleration: 2 Seconds Approach Face: 10 Seconds	12	
3	Down-seam Rip: 5.8 Feet Rate = 3.9 feet/minute (10 tons/ minute, 8'W x 8'H cut)	89	
4	Up-seam Rip: 5.8 Feet Rate = 10.4 feet/minute (10 tons/ minute, 8'W x 3'H cut)	34	
5	Hoist: 48 Feet Rate = 10 feet/second (average)	5	
6	Dump	8	
	Total Cycle Time (seconds)	151	
	Total Cycle Time (minutes)		2.5
7	Total Time-Minutes		27.6

Table D-3 SLACKLINE, MOTORIZED BUCKET (REGION 1B)

Hole Depth: Start 64 Feet
End 128 Feet   Length 64 Feet

Average Bucket Travel Distance: 112 Feet
(includes 28 feet beyond entrance)

Number of Cycles: 11

Bucket Size: 8'H x 8'W x 20'L   Capacity: 32 tons full, 20 tons working

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return: 112 Feet Rate = 15 feet/second (average)	8	
2	Deceleration: 2 Seconds Approach Face: 10 Seconds	12	
3	Down-seam Rip: 5.8 Feet Rate = 3.9 feet/minute (10 tons/ minute, 8'W x 8'H cut)	89	
4	Up-seam Rip: 5.8 Feet Rate = 10.4 feet/minute (10 tons/ minute, 8'W x 3'H cut)	34	
5	Hoist: 112 Feet Rate = 10 feet/second (average)	11	
6	Dump	8	
	Total Cycle Time (seconds)	162	
	Total Cycle Time (minutes)		2.7
7	Total Time-Minutes		29.6

Table D-4 SLACKLINE, MOTORIZED BUCKET (REGION 1C)

Hole Depth: Start 112 Feet

End 16 Feet   Length 96 Feet

Average Bucket Travel Distance: 80 Feet

(includes 28 feet beyond entrance)

Number of Cycles: 9

Bucket Size: 8'H x 8'W x 20'L   Capacity: 32 tons full, 20 tons working

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return: 91 Feet Rate = 15 feet/second (average)	6	
2	Deceleration: 2 Seconds Approach Face: 10 Seconds	12	
3	Down-seam Rip: --- Feet Rate = 3.9 feet/minute (10 tons/ minute, 8'W x 8'H cut)	0	
4	Up-seam Rip: 20.8 Feet Rate = 10.4 feet/minute (10 tons/ minute, 8'W x 3'H cut)	120	
5	Hoist: 70 Feet Rate = 10 feet/second (average)	7	
6	Dump	8	
	Total Cycle Time (seconds)	153	
	Total Cycle Time (minutes)		2.6
7	Total Time-Minutes		23.0

Table D-5 SLACKLINE, MOTORIZED BUCKET (REGION 1D)

Hole Depth: Start 128 Feet
End 192 Feet Length 64 Feet

Average Bucket Travel Distance: 188 Feet
(includes 28 feet beyond entrance)

Number of Cycles: 11

Bucket Size: 8'H x 8'W x 20'L Capacity: 32 tons full, 20 tons working

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return: 188 Feet Rate = 15 feet/second (average)	12	
2	Deceleration: 2 Seconds Approach Face: 10 Seconds	12	
3	Down-seam Rip: 5.8 Feet Rate = 3.9 feet/minute (10 tons/ minute, 8'W x 8'H cut)	89	
4	Up-seam Rip: 5.8 Feet Rate = 10.4 feet/minute (10 tons/ minute, 8'W x 3'H cut)	34	
5	Hoist: 188 Feet Rate = 10 feet/second (average)	19	
6	Dump	12	
	Total Cycle Time (seconds)	178	
	Total Cycle Time (minutes)		3.0
7	Total Time-Minutes		32.6

Table D-6 SLACKLINE, MOTORIZED BUCKET (REGION 1E)

Hole Depth: Start 172 Feet

End 108 Feet   Length 64 Feet

Average Bucket Travel Distance: 168 Feet

(includes 28 feet beyond entrance)

Number of Cycles: 6

Bucket Size: 8'H x 8'W x 20'L   Capacity: 32 tons full, 20 tons working

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return: 179 Feet Rate = 15 feet/second (average)	12	
2	Deceleration: 2 Seconds Approach Face: 10 Seconds	12	
3	Down-seam Rip: 0 Feet Rate = 3.9 feet/minute (10 tons/ minute, 8'W x 8'H cut)	0	
4	Up-seam Rip: 20.8 Feet Rate = 10.4 feet/minute (10 tons/ minute, 8'W x 3'H cut)	120	
5	Hoist: 158 Feet Rate = 10 feet/second (average)	16	
6	Dump	8	
	Total Cycle Time (seconds)	168	
	Total Cycle Time (minutes)		2.8
7	Total Time-Minutes		16.8

Table D-7 SLACKLINE, MOTORIZED BUCKET (REGION 1F)

Hole Depth: Start 149 Feet

End 37 Feet   Length 112 Feet

Average Bucket Travel Distance: 121 Feet

(includes 28 feet beyond entrance)

Number of Cycles: 21

Bucket Size: 8'H x 8'W x 20'L   Capacity: 32 tons full, 20 tons working

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return: 127 Feet Rate = 15 feet/second (average)	9	
2	Deceleration: 2 Seconds Approach Face: 10 Seconds	12	
3	Down-seam Rip: 0 Feet Rate = 3.9 feet/minute (10 tons/ minute, 8'W x 8'H cut)	0	
4	Up-seam Rip: 20.8 Feet Rate = 10.4 feet/minute (10 tons/ minute, 8'W x 3'H cut)	120	
5	Hoist: 106 Feet Rate = 10 feet/second (average)	11	
6	Dump	8	
	Total Cycle Time (seconds)	160	
	Total Cycle Time (minutes)		2.7
7	Total Time-Minutes		56.0

Table D-8 SLACKLINE, MOTORIZED BUCKET (REGION 1G)

Hole Depth: Start 192 Feet

End 266 Feet   Length 74 Feet

Average Bucket Travel Distance: 257 Feet

(includes 28 feet beyond entrance)

Number of Cycles: 13

Bucket Size: 8'H x 8'W x 20'L   Capacity: 32 tons full, 20 tons working

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return: 254 Feet Rate = 15 feet/second (average)	17	
2	Deceleration: 2 Seconds Approach Face: 10 Seconds	12	
3	Down-seam Rip: 5.8 Feet Rate = 3.9 feet/minute (10 tons/minute, 8'W x 8'H cut)	89	
4	Up-seam Rip: 5.8 Feet Rate = 10.4 feet/minute (10 tons/minute, 8'W x 3'H cut)	34	
5	Hoist: 233 Feet Rate = 10 feet/second (average)	23	
6	Dump	8	
	Total Cycle Time (seconds)	183	
	Total Cycle Time (minutes)		3.1
7	Total Time-Minutes		39.7

Table D-9 SLACKLINE, MOTORIZED BUCKET (REGION 1H)

Hole Depth: Start 230 Feet
End 156 Feet Length 74 Feet
Average Bucket Travel Distance: 221 Feet
(includes 28 feet beyond entrance)
Number of Cycles: 21
Bucket Size: 8'H x 8'W x 20'L Capacity: 32 tons full, 20 tons working

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return: 232 Feet Rate = 15 feet/second (average)	16	
2	Deceleration: 2 Seconds Approach Face: 10 Seconds	12	
3	Down-seam Rip: 0 Feet Rate = 3.9 feet/minute (10 tons/ minute, 8'W x 8'H cut)	0	
4	Up-seam Rip: 20.8 Feet Rate = 10.4 feet/minute (10 tons/ minute, 8'W x 3'H cut)	120	
5	Hoist: 211 Feet Rate = 10 feet/second (average)	21	
6	Dump	8	
	Total Cycle Time (seconds)	177	
	Total Cycle Time (minutes)		3.0
7	Total Time-Minutes		62.0

Table D-10 SLACKLINE, MOTORIZED BUCKET (REGION 1I)

Hole Depth: Start 184 Feet

End 78 Feet   Length 106 Feet

Average Bucket Travel Distance: 159 Feet

(includes 28 feet beyond entrance)

Number of Cycles: 31

Bucket Size: 8'H x 8'W x 20'L   Capacity: 32 tons full, 20 tons working

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return: 170 Feet Rate = 15 feet/second (average)	11	
2	Deceleration: 2 Seconds Approach Face: 10 Seconds	12	
3	Down-seam Rip: 0 Feet Rate = 3.9 feet/minute (10 tons/ minute, 8'W x 8'H cut)	0	
4	Up-seam Rip: 20.8 Feet Rate = 10.4 feet/minute (10 tons/ minute, 8'W x 3'H cut)	120	
5	Hoist: 149 Feet Rate = 10 feet/second (average)	15	
6	Dump	8	
	Total Cycle Time (seconds)	166	
	Total Cycle Time (minutes)		2.8
7	Total Time-Minutes		85.8

- RIP

  -- 10 tons/minute is the assumed ripping rate in both directions.

  -- Down-seam cutting area = 8'W x 3'H

  -- Up-seam cutting area = 8'W x 3'H

  -- Regions: 1A, 1B, 1D, and 1G:

  Down-seam ripping distance = Up-seam ripping distance

$$\frac{x \text{ ft}}{3.9 \text{ ft/min}} + \frac{x \text{ ft}}{10.4 \text{ ft/min}} = \frac{20 \text{ tons}}{10 \text{ tons/min}} = \text{Thus, } x = 5.8 \text{ ft}$$

  -- Regions: 1C, 1E, 1F, 1H, and 1I:

$$\text{Up-seam rip} = \frac{20 \text{ min}}{10.4 \text{ ft/min}} = 20.8 \text{ ft.}$$

- HOIST

  -- Regions: 1A, 1B, 1D, and 1G:

  Distance = Average bucket travel distance

  -- Regions: 1C, 1E, 1F, 1H, and 1I:

  Distance = Gravity return distance - 20.8 ft

$$\text{Time} = \frac{\text{Distance}}{15 \text{ ft/sec}}$$

- DUMP = 8 seconds (assumed)

D.2 PRODUCTIVITY FROM SHIFT 2 TO SHIFT 8

D.2.1 Equation Development for Shift Advance

The equation for describing the depth the excavation advances each shift is given by:

$$\text{Amount mined } \left(\frac{\text{Tons}}{\text{Shift}}\right) = \text{Rate } \left(\frac{\text{Tons}}{\text{Min}}\right) \times \text{Time } \left(\frac{\text{Min}}{\text{Shift}}\right) \quad (1)$$

- PRODUCTION RATE

  Production rate is a function of depth. It will decrease as the hole becomes deeper, since more time per cycle will be spent traveling.

$$\text{Rate } \left(\frac{\text{Tons}}{\text{Min}}\right) = \frac{\text{Cycle Rate } \left(\frac{\text{Tons}}{\text{Cycle}}\right)}{\text{Cycle Time } \left(\frac{\text{Min}}{\text{Cycle}}\right)} \quad (2)$$

-- Cycle Rate

Since a full bucket holds 20 tons, the cycle rate is 20 tons/cycle.

-- Cycle Time

Cycle Time = A + B + C + D + E + F

where A = Gravity return time = $\frac{\bar{x}\ (ft)}{15\ ft/sec}$

($\bar{x}$ = Average Depth)

B = Deceleration and approach face time - 12 seconds

C = Down-seam rip time (1.3 ft @ 3.9 ft/min)

D = Up-seam rip time (17.4 ft @ 10.4 ft/min)

E = Hoist time = $\frac{\bar{x} - 16.1\ (ft)}{10\ ft/sec}$ + 1 second deceleration

(Average depth reduced by the 17.4 - 1.3 = 16.1 of ripping)

F = Dump time = 8 seconds

$$\text{Cycle Time} = \frac{\bar{x}}{15} + 12 + \frac{1.3 \times 60}{3.9} + \frac{17.4 \times 60}{10.4} + \frac{\bar{x} - 16.1}{10} + 8$$

$$= 1.67\ \bar{x} + 138.39\ \text{(secs)}$$

$$= 0.00278\ \bar{x} + 2.3065\ \text{(mins)}$$

$$= C_1\bar{x} + C_2 \qquad (3)$$

Substituting equation (3) and equation (5) from Appendix E into equation (1):

$$15.36\ (x_2 - x_1)\ \text{tons/shift} = \frac{2}{C_1\ x + C_2}\ \text{tons/min}\ \left(\frac{\text{mins}}{380\ \text{shift}}\right) \qquad (4)$$

From Figure F-1, $\bar{x} = \frac{x_2 + x_1}{2}$. Since the bucket dumps when the front is

16 feet from the highwall face, $\bar{x} = \frac{x_2 + x_1}{2} + 16$. Substituting into

equation (4):

$$15.36\ (x_2 - x_1) = \frac{7,600}{C_1\ (\frac{x_1 + x_2}{2} + 16) + C_2}$$

$$15.36\ (x_2 - x_1) = \frac{15,200}{C_1(x_1 + x_2 + 32) + 2C_2}$$

Multiplying both sides by the right side denominator and dividing by 15.36

$$(x_2 - x_1)\ \ [C_1(x_1 + x_2 + 32) + 2C_2] = 990$$

$$C_1x_1x_2 + C_1x_2^2 + 32C_1\ x_2 + 2C_2\ x_2 - C_1x_1^2 - C_1x_1x_2 - 32C_1x_1 - 2C_2x_1 = 990$$

$$C_1x_2^2 + x_2\ (32C_1 + 2C_2) - C_1x_1^2 - 32C_1x_1 - 2C_2x_1 - 990 = 0 \qquad (5)$$

Since x_1 is known from the previous shift (x_1 for shift Y = x_2 for shift X)

This is simply a quadratic equation:

$$a\ x_2^2 + b\ x_2 + c = 0$$

where: $a = C_1$

$b = 32C_1 + 2C_2$

$c = -(C_1\ x_1^2 + 32C_1\ x_1 + 2C_2\ x_1 + 990)$

The roots are: $X_2 = \frac{-b \pm \sqrt{b^2 - 4ac}}{2a}$

Now, since $C_1 = 0.00278$ and $C_2 = 2.3065$

$a = 0.00278$

$b = 5.503$

Equation (5) can then be solved on a calculator using a program from Texas Instruments for solving a quadratic equation. A short section to calculate c for a given x_1 (the previous shift's x_2) is added.

D.2.2 Other Equations

- AVERAGE BUCKET TRAVEL $= \frac{x_1 + x_2}{2} + 16$

x_1 = final depth at end of previous shift

x_2 = final depth at end of present shift

- NUMBER OF BUCKET CYCLES

$$\frac{\text{No. of Cycles}}{\text{Foot of Advance}} = \frac{15.36 \text{ tons/foot}}{20 \text{ tons/cycle}} = 0.768 \frac{\text{cycles}}{\text{foot}}$$

$$\text{Number of bucket cycles} = 0.768 \frac{\text{cycles}}{\cancel{\text{foot}}} \text{ X } (x_2 - x_1) \frac{\text{feet}}{\text{shift}}$$

- AVERAGE CYCLE TIME $= \dfrac{380 \text{ mins/shift}}{\dfrac{\text{number of bucket cycles}}{\text{shift}}}$

- PRODUCTION TIME

-- Rip = Number of bucket cycles X $\frac{2 \text{ minutes}}{\text{cycle}}$

-- Travel and Dump = 380 minutes - Rip Time

- TONS MINED

-- $\frac{\text{total tons}}{\text{shift}} = (x_2 - x_1) \text{ feet X } 15.36 \frac{\text{tons}}{\text{foot}}$

-- Tons Ripped Down-seam

The down-seam rip zone is 8 feet of the 48-foot excavation. So:

$$\text{Tons produced by down-seam ripping} = \frac{8}{48} \text{ x total tons}$$
$$= \frac{1}{6} \text{ x total tons}$$

-- Tons Ripped Up-seam

Total tons - tons ripped down-seam

- AVERAGE TONS PER HOUR EXCLUDING MOVE $= \dfrac{\dfrac{\text{total tons}}{\text{shift}}}{\dfrac{8 \text{ hours}}{\text{shift}}}$

- CUMULATIVE AVERAGE **TONS** PER **HOUR** EXCLUDING **MOVE** =

$$\frac{\text{cumulative tons}}{\text{8 hours x number of shifts}}$$

- CUMULATIVE AVERAGE **TONS** PER **HOUR INCLUDING MOVE** =

$$\frac{\text{cumulative tons}}{\text{8 hours x number of shifts + 2-1/2 hrs}}$$

Appendix E

MOTORIZED BUCKET CALCULATIONS

ESD Corporation
Engineered Systems Development
A NESCO COMPANY

CALCULATION SHEET

Client	BUREAU OF MINES
Date	3/16/83
Project Title	HIGH ANGLE MINING SYSTEM
Subject	MOTORIZED BUCKET
Project No.	3 0 0 2 7 0 1 0 0
Sheet	1 of ____ Sheets
Calculated By	L. JACK
Checked By	

THE 30" CUTTING CHAIN IS USED ON THE 12HM CONTINUOUS MINER. THE MINER DRUM WIDTH IS 10 FEET 10 INCHES (130") AND DRUM MOTOR HORSEPOWER IS 350 (2×175 HP). THUS HORSEPOWER REQUIRED BY THE CUTTING CHAIN IS

$$\frac{30}{130} \times \frac{350}{.8} = 101 \text{ HP}$$

A SIX INCH WIDE CLEARANCE CUTTER ON THE OUTSIDE END OF EACH CUTTER CHAIN DRIVEN SHAFT COULD REQUIRE ANOTHER 25 HP (ESTIMATED).

$$T_D = \frac{125(63025)}{62} = 127\ 066 \text{ LB-IN} \quad \text{CUTTER CHAIN TORQUE}$$

ASSUME DRIVE MOTOR SPEED IS 1000 RPM

SPEED REDUCTION IS $n_1/n_2 = 1000/62 = 16.13$ RPM

ASSUME ROLLER CHAIN REDUCTION IS 2 TO 1
THEREFORE THE GEAR REDUCTION IS

$$16.13/2 = 8.06$$

GEAR REDUCER OUTPUT TORQUE IS

$$127{,}066/2 = 63\ 533 \text{ LB-IN (OR } 5294 \text{ LB-FT)}$$

$$\frac{5294}{.737} = 7183 \text{ NM}$$

$$T_N = T_{erf} \times f = 7{,}183 \times 2.00 = 14{,}366 \text{ NM}$$

NOMINAL TORQUE RATING FOR AMERICAN LOHMAN PLANETARY GEAR REDUCTION MODEL GPV 90-2111 IS 16,500 N·M, RATIOS AVAILABLE ARE 7.85 OR 10.05

THERMAL REQUIREMENT

$$n_T = \frac{T_N}{T_{erf}} \times n_{4h} \times f_\omega$$

$$= \frac{16500}{7183} \times 14 \times 1.0$$

$$= 1034.2 \text{ RPM}$$

$n_T < n_2$: SINCE 1034.2 IS NOT LESS THAN 62 RPM NO ADDITIONAL COOLING IS REQUIRED

GEAR REDUCTION INPUT TORQUE IS

$$T_M = \frac{127\,066}{2(10.05)} = 6\,322 \text{ LB-IN}$$

$$\frac{6322\,(100)}{5000} = 124.6 \text{ LB-IN}/100 \text{ PSI @ } 5000 \text{ PSI}$$

DENISON MODEL M46F-05P-103 PISTON PUMP IS RATED AT 164 LB-IN/100 PSI @ 5000 PSI THEORETICAL. (ACTUAL = 164 x .93 = 152.5)

OR $\frac{6322(100)}{.93(164)} = 4145$ PSI REQ'D PRESSURE

VOLUME REQ'D = 4.46 GAL/100 RPM x 10 HUNDRED RPM x 2 MOTORS = 89.2 GPM

ESD Corporation
Engineered Systems Development
A NESCO COMPANY

CALCULATION SHEET

Project No. 3 0 0 2 7 0 1 0 0

Client BUREAU OF MINES Date 3/22/83

Sheet 3 of ____ Sheets

Project Title HIGH ANGLE MINING SYSTEM

Calculated By L. JACK

Subject MOTORIZED BUCKET

Checked By ____________

HOSE DIAMETER SHOULD BE 1 1/2" MINIMUM FOR THIS FLOW RATE.

ROLLER CHAIN TENSION

$$\frac{127\,066(2)}{23.917} = 10\,625.6 \text{ LBS}$$

NO. 200 ROLLER CHAIN IS RATED AT 95 000 LBS AVERAGE STRENGTH

$$F.S. = 95000/10\,625 = 8.9:1$$

THE ACTUAL SPEED REDUCTION IS

$1000/2 \times 10.05 = 49.75$ RPM (INSTEAD OF 62 AS ASSUMED)

Appendix F

AUGER AND SLACKLINE COST ANALYSIS AND COMPARISON

APPENDIX F

AUGER AND SLACKLINE COST ANALYSIS AND COMPARISON

The cost analysis for each of the six concepts is broken into two categories, peculiar and common. Peculiar costs include the capital, operating, and maintenance costs to extract and deposit coal on the working bench or pit floor and are different for each HAMS concept. Common costs are related to surface haulage, processing, and loading the coal for shipment and do not vary on a per-ton basis for the different concepts. Included in the common costs are severance, ad valorem, and local taxes as well as royalty and reclamation costs.

The estimated costs for each of the six concepts are summarized and compared on a per-ton-of-coal-produced basis in Table F-1. It is noteworthy that the common costs are much greater than the peculiar costs and tend to level or equalize the total cost per ton of the various concepts. All costs are adjusted to 1980 dollars.

Capital investments in mine utilities or general facilities, such as pit drainage, sewage treatment, power distribution, buildings, roads, communications and rail spur are not included in the cost analysis, since these are supported by the surface operation.

F.1 CAPITAL COSTS

The calculation of the average annual investment cost for each of the concepts is based upon the true effective cost of the investment. For this analysis, it is the investment cost after taxes and takes into account the current 10-percent investment tax credit and the allowed depreciation charges (provided a company has a net profit and pays taxes each year). The 10-percent investment tax credit is allowed the first year and in effect reduces the investment principal by 10 percent.

The annual depreciation charges in all these concepts are calculated by the standard straight-line method with a common 11-year life and zero salvage value. These depreciation charges are tax deductible and generate a positive cash flow stream each year equal to the tax rate times the depreciation charge.

The method used for determining the average annual investment cost takes into account the present worth of these future depreciation deductions. A federal tax rate of 48 percent on profits is used, and a loan term of 11 years is assumed. The capital costs for the augering concepts are based on the current selling price of a 42-inch-diameter standard augering machine ($500,000). The capital costs for the slackline excavator concepts are based on the selling price of a 12-cubic-yard dragline machine ($3,000,000).

Table F-1 COST SUMMARY AND COMPARISON

	Standard Auger	Rectangular Auger	Eckenrode Miner	New Edna Miner	Slackline Excavator, Projectile Bucket	Slackline Excavator, Motorized Bucket
PECULIAR COSTS (per ton)						
Mining machine						
Capital*	$0.46	$0.56	$0.48	$0.16	$ 1.72	$0.92
Operating	1.32	1.16	1.12	0.33	0.50	0.32
Maintenance	0.28	0.39	0.39	0.39	0.33	0.35
Pilot hole drilling					0.11	0.11
Benching	0.07	0.06	0.02	0.00	0.01	0.01
Total peculiar costs	$2.13	$2.19	$2.11	$0.88	$ 2.67	$1.71
COMMON COSTS (per ton)						
Haulage**	1.40	1.40	1.40	1.40	1.40	1.40
Tipple	1.00	1.00	1.00	1.00	1.00	1.00
Reclamation	0.35	0.35	0.35	0.35	0.35	0.35
Black lung fee	0.25	0.25	0.25	0.25	0.25	0.25
Severance and ad valorem taxes	1.93	1.93	1.93	1.93	1.93	1.93
Royalties	1.93	1.93	1.93	1.93	1.93	1.93
Miscellaneous local taxes	0.02	0.02	0.02	0.02	0.02	0.02
Total common costs	$7.58	$7.58	$7.58	$7.58	$7.58	$7.58
NET COST	$9.71	$9.77	$9.69	$8.46	$10.25	$9.29

* Annual capital costs are offset by Federal tax benefits.
** Includes capital, operating, and maintenance costs.

F.2 OPERATING COSTS

Operating costs include direct labor, overhead supervision, and fuel. Direct labor costs are assumed to be 10 percent above the 1980 IUOE and UMWA wage scale which is common for nonunion western coal mines. Direct labor benefits, which include holidays, vacation, and sick leave, are estimated at 66 percent of direct labor. An additional 35 percent of direct labor is added to cover the costs of supervision and office staff. Fuel consumption is estimated separately for each concept.

F.3 MAINTENANCE COSTS

The maintenance costs include consumables, such as cutter bits, bucket teeth, oil, grease, filters, and cable, as well as labor. The maintenance costs for the auger concepts are based on actual costs for a standard auger. Maintenance costs for the slackline excavator concepts are based on published data for dragline machines adjusted to 1980 dollars.

F.4 PILOT HOLE DRILLING COSTS

The pilot hole drilling costs estimated for the slackline excavator concepts are assumed to be drilled by a contractor using a rotary or downhole motor rig. The estimate also assumes a 100-foot-per-hour average penetration rate.

F.5 BENCHING COSTS

It is assumed that working benches are created on loose fill during the normal reclamation process. Seams are mined from the bottom to top of the final highwall. The costs shown assume a bulldozer can level a 60-foot-wide bench at the rate of 10,000 square feet per hour.

F.6 COMMON COSTS

F.6.1 Haulage

The haulage cost is assumed constant on a per ton basis for each concept. It is based on a 10-cubic-yard front end loader loading two each 50-ton trucks and includes capital, operating, and maintenance costs derived in a manner similar to that used for the peculiar costs for the various HAMS concepts.

Each 50-ton truck is filled in 8 minutes and hauls an average of 100 tons per hour a distance of 2-1/2 miles to the tipple. The round-trip cycle is completed in 30 minutes.

F.6.2 Processing

Coal is sized and sprayed with a dust control solution as it is processed through the tipple. It is loaded into rail cars by an automated control system. The dollar-per-ton estimate assumes that no cleaning, drying, or other additonal processing is involved.

F.6.3 Other Common Costs

Federal Reclamation and Black Lung contributions are \$0.35 and \$0.25 per ton, respectively. The state severance and ad valorem tax rate of 17 percent reflects Wyoming coal. This rate is applied to the value of the coal at its first point of storage, which is related to the selling price. A value of \$15.47 per ton is used. Royalties are estimated at an additional 12-1/2 percent of this value. Miscellaneous local taxes are estimated at \$0.02 per ton.

Appendix G

SLACKLINE PRODUCTIVITY ANALYSIS, PROJECTILE BUCKET

APPENDIX G

SLACKLINE PRODUCTIVITY ANALYSIS, PROJECTILE BUCKET

G.1 SHIFT 1

During the first shift, the hole is mined 175 feet deep and almost the full seam thickness. In order to estimate productivity, the hole is divided into six separate regions, 1A through 1F. Figure G-1 shows the regions graphically and Table G-1 summarizes the depth and productivity. Figure G-2 shows the Production Time Chart for the first shift.

G.1.1 Shift 1 - Productivity Development

Tables G-2 through G-7 detail the production time required for each region. After assuming a mining depth for the region, the table values were calculated as follows:

- BUCKET TRAVEL DISTANCE = $\frac{\text{End depth - start depth}}{2}$ + 16 feet

  (The bucket dumps 16 feet outside the hole mouth.)

- NUMBER OF CYCLES = $\frac{\text{Depth}}{\text{Region}}$, Ft x $\frac{\text{cycle}}{\text{1/2 foot}}$

  (3 impact blows per cycle mine 6 inches deep.)

- GRAVITY RETURN

  -- Distance = Bucket travel distance

  -- Time = $\frac{\text{Distance}}{\text{15 feet/second}}$ + 1 second deceleration

- IMPACT BLOWS

  -- 8-1/2 seconds per blow

  Raise bucket 40 feet at 10 feet per second plus 1 second deceleration = 5 seconds

  Release bucket, accelerate and impact = 2-1/2 seconds

  Pause = 1 second

  -- Two extra blows were allowed for region 1A since the bucket will not reach the full impact velocity of 30 feet per second.

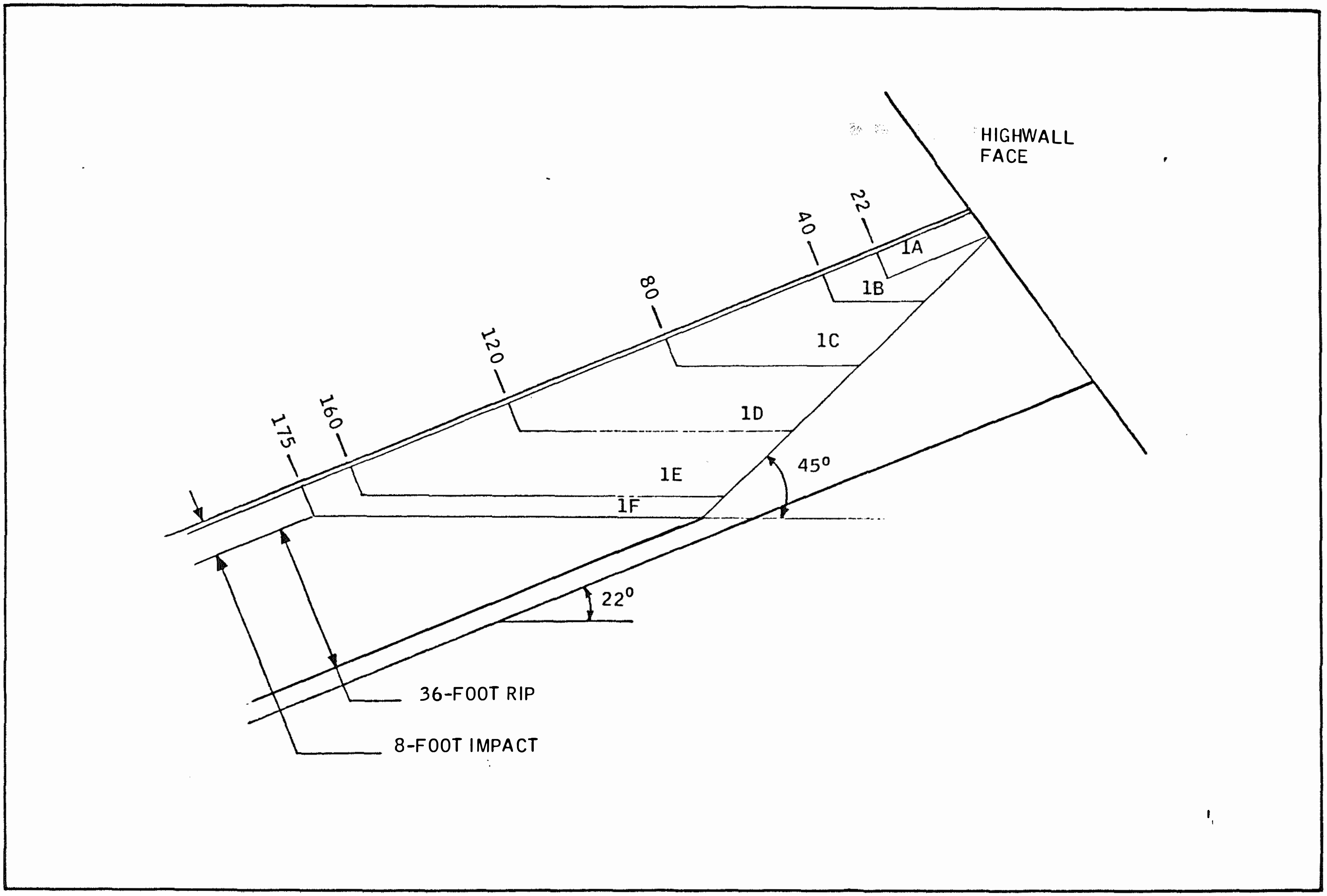


Figure G-1 PROJECTILE SLACKLINE EXCAVATOR, SHIFT 1 MINING PLAN (50-foot seam)

Table G-1 SLACKLINE WITH PROJECTILE BUCKET - PRODUCTION SUMMARY

Region	Shift 1						
	IA	IB	IC	ID	IE	IF	Total
Average depth, feet	0-24	24-40	40-80	80-120	120-160	160-175	
Average height, feet	0-8	0-16	0-25	0-33	0-41	0-44	
Tons Produced							
Impacted	61	41	102	102	102	38	446
Ripped	-	51	127	255	370	180	983
Total	61	92	229	357	472	218	1,429
Cumulative	61	153	382	739	1,211	1,429	
Cycles Required	48	32	80	80	80	36	356
Average Cycle Time, minutes	0.8	0.8	0.9	1.1	1.3	1.4	1.06
Total Production Time, minutes	40	23	68	87	106	51	
Cumulative Production Time, minutes	40	63	131	218	324	376	
Average Production Rate, (tons/minute)	1.5	3.5	3.3	4.1	4.5	4.3	
Cumulative Production Rate (tons/minute)	1.5	2.4	2.9	3.4	3.7	3.8	

ELEMENT NO.	DESCRIPTION	TIME (MIN) IND.	TIME (MIN) CUM
1	TRANSPORT OPERATORS TO SITE	15	15
2	PRE-OP MAINT. CHECKS AND SERVICES	15	30
3	DIG REGION 1A = 0 - 24 11 FEET DEEP	40	70
4	DIG REGION 1B = 24 - 40 11 FEET DEEP	26	96
5	DIG REGION 1C = 40 - 80 17 FEET DEEP	70	166
6	REST PERIOD	10	176
7	DIG REGION 1D = 80-110 11 FEET DEEP (START)	64	240
8	LUNCH PERIOD	(30)	
9	DIG REGION 1D = 110 - 120 (COMPLETE)	23	263
10	DIG REGION 1E = 120 - 160	106	369
11	REST PERIOD	10	379
12	DIG REGION 1F = 160 - 175	51	430
13	MISCELLANEOUS DELAYS	20	450
14	POST - 9P MAINT. CHECKS AND SERVICES	15	465
15	TRANSPORT OPERATORS TO ENTRANCE	15	480

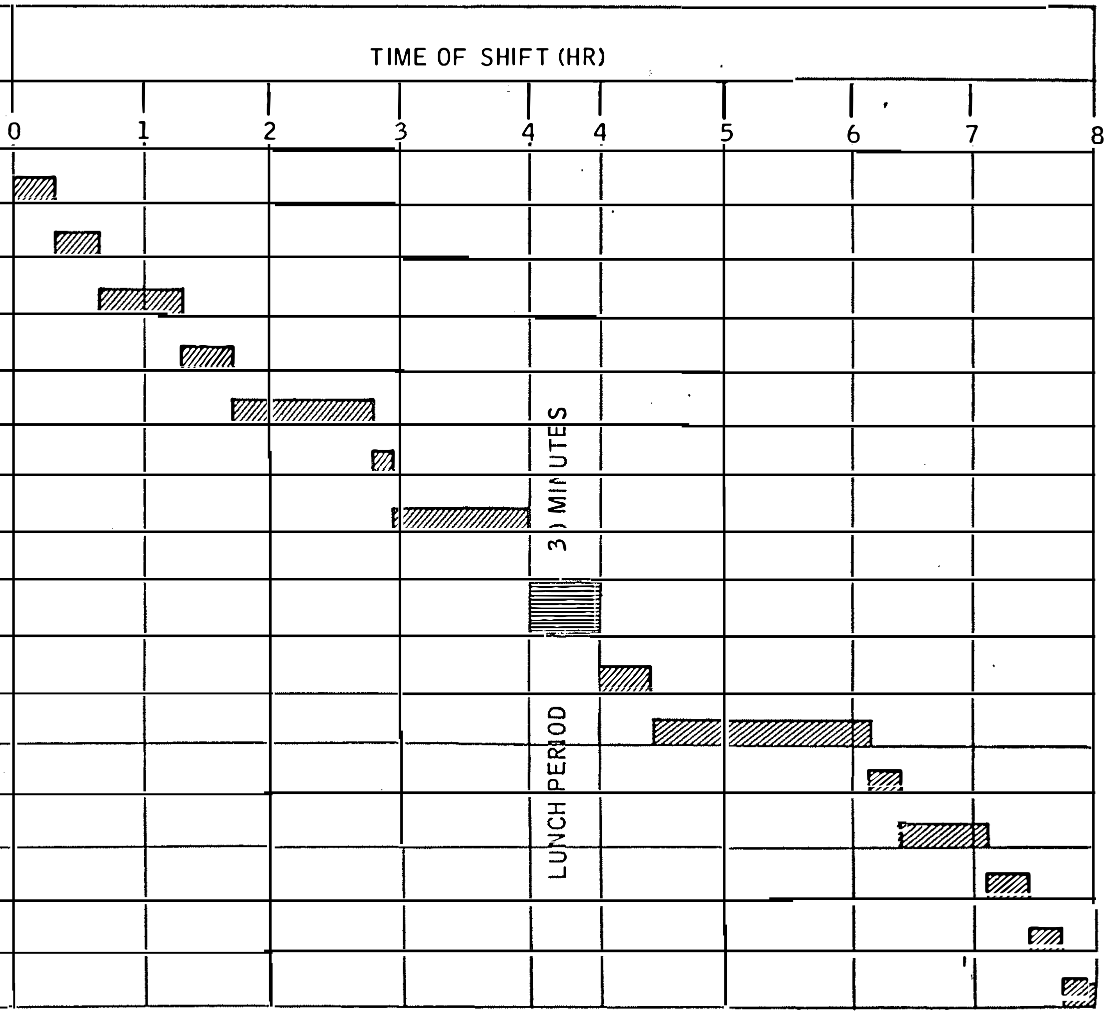


Figure G-2 SLACKLINE WITH PROJECTILE BUCKET PRODUCTION TIME CHART, FIRST-SHIFT OPERATION

Table G-2 SLACKLINE, PROJECTILE BUCKET (REGION 1A)

Hole Depth: Start 0 Feet, Length 24 Feet,
End 24 Feet

Bucket Travel Distance: (12 + 16) = 28
Average, including 16 feet beyond hole entrance

Number of Cycles: 48

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return 28 Feet Rate = 15 feet/second	3	
2	Impact Blows		
	Blow No. 2 & No. 3	17	
	Blow No. 4 & No. 5	17	
3	Rip 0 Feet Rate = 1 foot/second	0	
4	Hoist 28 Feet Rate = 10 feet/second	4	
5	Dump	8	
	Total Cycle Time (seconds)	49	
	Total Cycle Time (minutes)		0.82
	Number of Cycles 48 24 Feet @ 2 cycles/foot		
6	Total Time-Minutes		40

Table G-3 SLACKLINE, PROJECTILE BUCKET (REGION 1B)

Hole Depth: Start 24 Feet, Length 16 Feet,
End 40 Feet

Bucket Travel Distance: 32 + 16 = 48
Average, including 16 feet beyond hole entrance

Number of Cycles: 32

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return 48 Feet Rate = 15 feet/second	4	
2	Impact Blows Blow No. 2 & No. 3	17	
3	Rip 9 Feet Rate = 1 foot/second	9	
4	Hoist 39 Feet Rate = 10 feet/second	5	
5	Dump	8	
	Total Cycle Time (seconds)	43	
	Total Cycle Time (minutes)		0.72
	Number of Cycles 32 16 Feet @ 2 cycles/foot		
6	Total Time-Minutes		23.0

Table G-4 SLACKLINE, PROJECTILE BUCKET (REGION 1C)

Hole Depth: Start 40 Feet, Length 40 Feet,
End 80 Feet

Bucket Travel Distance: 60 + 16 = 76
Average, including 16 feet beyond hole entrance

Number of Cycles: 80

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return 76 Feet Rate = 15 feet/second	6	
2	Impact Blows Blow No. 2 & No. 3	17	
3	Rip 13 Feet Rate = 1 foot/second	13	
4	Hoist 63 Feet Rate = 10 feet/second	7	
5	Dump	8	
	Total Cycle Time (seconds)	51	
	Total Cycle Time (minutes)		0.85
	Number of Cycles = 80 40 Feet @ 2 cycles/foot		
6	Total Time-Minutes		68.0

Table G-5 SLACKLINE, PROJECTILE BUCKET (REGION 1D)

Hole Depth: Start 80 Feet, Length 40 Feet,
End 120 Feet

Bucket Travel Distance: 116 - 100 + 116
Average, including 16 feet beyond hole entrance

Number of Cycles: 80

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return 116 Feet Rate = 15 feet/second	9	
2	Impact Blows Blow No. 2 & No. 3	17	
3	Rip 20 Feet Rate = 1 foot/second	20	
4	Hoist 96 Feet Rate = 10 feet/second	11	
5	Dump	8	
	Total Cycle Time (seconds)	65	
	Total Cycle Time (minutes)		1.08
	Number of Cycles 80 40 Feet @ 2 cycles/foot		
6	Total Time-Minutes		87.0

Table G-6 SLACKLINE, PROJECTILE BUCKET (REGION 1E)

Hole Depth: Start 120 Feet, Length 40 Feet,
End 160 Feet

Bucket Travel Distance: 156 = (140 + 16)
Average, including 16 feet beyond hole entrance

Number of Cycles: 80

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return 156 Feet Rate = 15 feet/second	12	
2	Impact Blows Blow No. 2 & No. 3	17	
3	Rip 28 Feet Rate = 1 foot/second	29	
4	Hoist 128 Feet Rate = 10 feet/second	14	
5	Dump	8	
	Total Cycle Time (seconds)	80	
	Total Cycle Time (minutes)		1.33
	Number of Cycles 80 40 Feet @ 2 cycles/foot		
6	Total Time-Minutes		107.0

Table G-7 SLACKLINE, PROJECTILE BUCKET (REGION 1F)

Hole Depth: Start 160 Feet, Length 15 Feet,
End 175 Feet

Bucket Travel Distance: (167 + 16) = 183
Average, including 16 feet beyond hole entrance

Number of Cycles: 36

CYCLE ELEMENT

Number	Description	Time (seconds)	Time (minutes)
1	Gravity Return 183 Feet Rate = 15 feet/second	13	
2	Impact Blows Blow No. 2 & No. 3	17	
3	Rip 28 Feet Rate = 1 foot/second	30	
4	Hoist 155 Feet Rate = 10 feet/second	16	
5	Dump	8	
	Total Cycle Time (seconds)	84	
	Total Cycle Time (minutes)		1.4
	Number of Cycles 36 18 Feet @ 2 cycles/foot		
6	Total Time-Minutes		51.0

- RIP

  -- Depth = 6 inches

  -- $$\frac{\text{Feet ripped}}{\text{Cycle}} = \frac{\text{Volume of coal ripped per region}}{\frac{\text{(Number of cycles)}}{\text{Region}} \text{ X (Area ripped = 1/2 foot x 8 feet)}}$$

  NOTE: Region 1A is too shallow for ripping

- HOIST

  -- Distance = Bucket travel distance - ripping distance

  -- $$\text{Time} = \frac{\text{Distance}}{\text{10 feet/second}} + \text{1 second deceleration}$$

- DUMP = 8 seconds (assumed)

G.2 PRODUCTIVITY FROM SHIFT 2 TO SHIFT 16

G.2.1 Equation Development For Shift Advance

The equation for describing the depth the excavation advances each shift is given by:

$$\text{Amount mined } \frac{\text{(tons)}}{\text{(shift)}} = \text{Rate } \frac{\text{(tons)}}{\text{(min)}} \text{ x Time } \frac{\text{(min)}}{\text{(shift)}} \qquad (1)$$

- PRODUCTION RATE

  Production rate is a function of depth. It will decrease as the hole becomes deeper since more time per cycle will be spent traveling.

$$\text{Rate } \frac{\text{(tons)}}{\text{(min)}} = \frac{\text{Cycle Rate}}{\text{Cycle Time}} \frac{\left(\frac{\text{tons}}{\text{cycle}}\right)}{\left(\frac{\text{mins}}{\text{cycle}}\right)} \qquad (2)$$

  -- Cycle Rate

  Since a full bucket holds 6 tons, the cycle rate is 6 tons/cycle

  -- Cycle Time

  Cycle Time = A + B + C + D + E

  where A = Gravity return time = $\frac{\bar{x} \text{ (ft)}}{\text{15 ft/sec}}$ + 1 sec deceleration
  ($\bar{x}$ = Average Depth)

  B = Impact time = 17 secs (2 blows)

$C =$ Ripping time = 32 secs (32 ft @ 1 ft/sec)

$$D = \text{Hoist time} = \frac{\bar{x} - 32\ (\text{ft})}{10\ \text{ft/sec}} + 1 \text{ second deceleration}$$

(Average depth reduced by the 32 ft. of ripping)

$E =$ Dump time = 8 secs.

$$\text{Cycle Time} = \left(\frac{\bar{x}}{15} + 1\right) + 17 + 32 + \left(\frac{\bar{x} - 32}{10} + 1\right) = +8$$

$$= \frac{\bar{x}}{15} + \frac{\bar{x}}{10} + 55.8$$

$$= 0.167\ \bar{x} + 55.8 \text{ secs/cycle}$$

$$= 0.00278\ \bar{x} + .93 \text{ mins/cycle}$$

$$= C_1\ \bar{x} + C_2 \qquad (3)$$

Substituting equation (3) into (2):

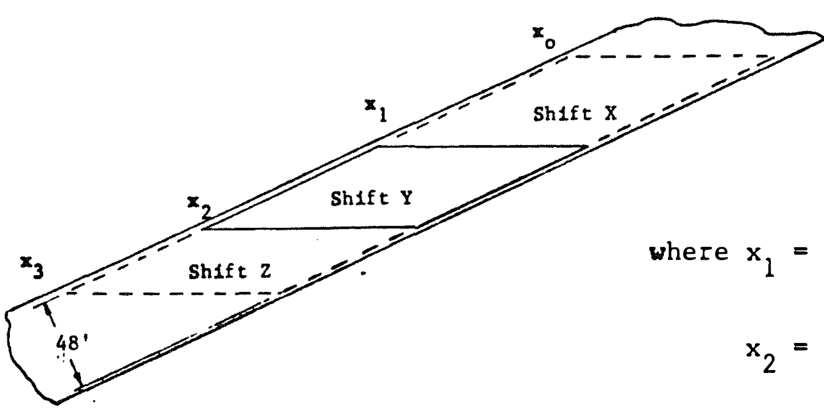
$$\text{Rate } \frac{(\text{tons})}{(\text{min})} = \frac{6 \text{ tons/cycle}}{C_1\ \bar{x} + C_2 \text{ mins/cycle}} \qquad (4)$$

- TIME

There are 380 productive minutes per shift.

- AMOUNT MINED PER SHIFT

After shift 1, the area of coal mined is represented in Figures G-3 and G-4.

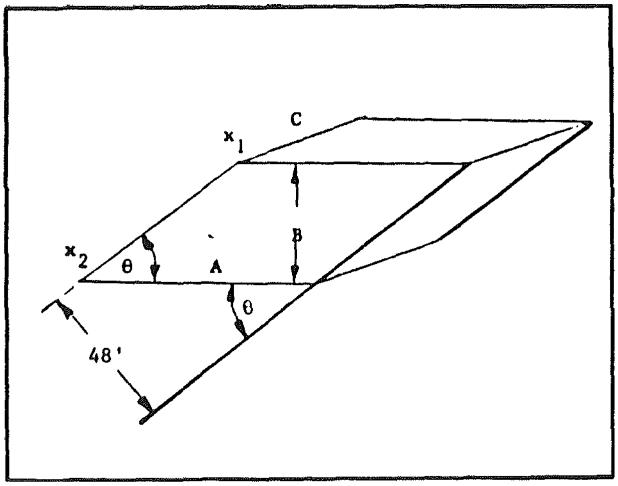


where x_1 = depth of hole at end of Shift X

x_2 = depth at end of Shift Y

x_3 = depth at end of Shift Z

Figure G-3 AREA OF COAL MINED



Volume mined = A x B x C

$A = 48/\sin\theta$

$B = (x_2 - x_1)\sin\theta$

$C = 8$

$$\text{Volume} = (8)\left(\frac{48}{\cancel{\sin\theta}}\right)(x_2 - x_1)\sin\theta$$

$$= (8)(48)(x_2 - x_1)\ \text{ft}^3/\text{shift}$$

$$= (8)(48)\left(\frac{80}{2000}\right)(x_2 - x_1)\ \frac{\text{tons}}{\text{shift}}$$

$$\text{Volume} = 15.36\,(x_2 - x_1)\ \frac{\text{tons}}{\text{shift}} \qquad (5)$$

Figure G-4 **MINED AREA**

Substituting equations (4) and (5) into equation (1):

$$\begin{array}{c} \text{tons/shift} \quad \text{tons/shift} \end{array}$$

$$15.36\,(x_2 - x_1) = \frac{6 \qquad\qquad \text{mins}}{C_1\,\bar{x} + C_2\ (380\ \text{shift})} \qquad (6)$$

From Figure G-3, $\bar{x} = \frac{x_2 + x_1}{2}$. Since the bucket dumps when the front is 16 feet from the highwall face, $\bar{x} = \frac{x_2 + x_1}{2} + 16$.

Substituting into equation (6):

$$15.36\,(x_2 - x_1) = \frac{2{,}280}{C_1\left(\frac{x_1 + x_2}{2} + 16\right) + C_2}$$

$$15.36\,(x_2 - x_1) = \frac{4{,}560}{C_1\,(x_1 + x_2 + 32) + 2C_2}$$

Multiplying both sides by the right side denominator and dividing by 15.36

$$(x_2 - x_1)\,[\,C_1\,(x_1 + x_2 + 32) + 2C_2\,] = 297$$

$$C_1x_1x_2 + C_1x_2^2 + 32C_1\,x_2 + 2C_2\,x_2 - C_1x_1^2 - C_1x_1x_2 - 32C_1x_1$$
$$- 2C_2x_1 = 297$$

$$C_1x_2^2 + x_2\,(32C_1 + 2C_2) - C_1x_1^2 - 32C_1x_1 - 2C_2x_1 - 297 = 0 \quad (7)$$

Since x_1 is known from the previous shift (x_1 for shift Y = x_2 for shift X). This is simply a quadratic equation:

$$a\,x_2^2 + b\,x_2 + c = 0$$

where: $a = C_1$

$b = 32C_1 + 2C_2$

$c = -(C_1\,x_1^2 + 32C_1\,x_1 + 2C_2\,x_1 + 297)$

The roots are : $x_2 = \dfrac{-b \pm b^2 - 4ac}{2a}$

Now, since C_1 = 0.00278 and C_2 = 0.93

$a = 0.00278$

$b = 1.949$

Equation (7) can then be solved on a calculator using a program from Texas Instruments for solving a quadratic equation. A short section to calculate c for a given x_1 (the previous shift's x_2) is added.

G.2.2 Other Equations

- AVERAGE BUCKET TRAVEL = $\dfrac{x_1 + x_2}{2} + 16$

  x_1 = final depth at end of previous shift

  x_2 = final depth at end of present shift

- NUMBER OF BUCKET CYCLES

  $$\frac{\text{No. of Cycles}}{\text{Foot of Advance}} = \frac{15.36 \text{ tons/foot}}{6 \text{ tons/cycle}} = 2.56\ \frac{\text{cycles}}{\text{foot}}$$

  Number of bucket cycles = 2.56 $\dfrac{\text{cycles}}{\text{foot}}$ X ($x_2 - x_1$) $\dfrac{\text{feet}}{\text{shift}}$

- AVERAGE CYCLE TIME = $\dfrac{380 \text{ mins/shift}}{\dfrac{\text{number of bucket cycles}}{\text{shift}}}$

- PRODUCTION TIME

  -- Impact & Rip = Number of bucket cycles X $\dfrac{17 \text{ seconds} + 32 \text{ seconds}}{60 \text{ seconds/minute}}$

  -- Travel & Dump = 380 minutes - Impact & Rip Time

- TONS MINED

-- $\frac{\text{total tons}}{\text{shift}} = (x_2 - x_1)$ feet X 15.36 $\frac{\text{tons}}{\text{foot}}$

-- Tons Impacted

The impact zone is 8 feet of the 48-foot excavation. So:

$$\text{Tons produced by impacting} = \frac{8}{48} \text{ X total tons}$$

$$= \frac{1}{6} \text{ X total tons}$$

-- Tons Ripped = Total tons - tons impacted

- AVERAGE TONS PER HOUR EXCLUDING MOVE = $\dfrac{\frac{\text{total tons}}{\text{shift}}}{\text{8 hours}}$

- CUMULATIVE AVERAGE TONS PER HOUR EXCLUDING MOVE =

$$\frac{\text{cumulative tons}}{\text{8 hours x number of shifts}}$$

- CUMULATIVE AVERAGE TONS PER HOUR INCLUDING MOVE =

$$\frac{\text{cumulative tons}}{\text{8 hours x number of shifts + 2-1/2 hrs}}$$