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Demonstration of Tunnel Boring Machine For Coal Mine Development

**Final Technical Report
Eastern Associated Coal Corp.**

October 1979

**Contract No. - U.S. D.O.E. ET-74-C-01-9068
(Formerly U.S.B.M. S0144075)**

U. S. DEPARTMENT OF ENERGY

Assistant Secretary for Energy Technology

**Division of Fossil Fuel Extraction
Mining Research and Development**

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FINAL TECHNICAL REPORT
AS OF
31 October 1979

This report represents work on a program that was originated by the Interior Department's Bureau of Mines and was transferred to the Department of Energy on October 1, 1977.

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Pittsburgh, Pa.

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TABLE OF CONTENTS

	<u>Page</u>
Abstract.....	1
Executive Summary.....	2
Project Summary.....	2
Conclusions.....	5
Acknowledgements.....	6
1. Objectives and History.....	7
2. Geologic Report.....	11
2.1 Rock Quality and Property Determination.....	14
3. System Description.....	15
3.1 TBM System.....	15
3.2 Materials Handling.....	16
3.3 Fragmentation.....	16
3.4 Roof and Ground Support.....	16
3.5 Shotcrete Liner.....	17
3.6 Ventilation.....	18
3.7 Illumination.....	18
3.8 Degasification.....	18
4. Operational Problems.....	20
4.1 General History.....	20
4.2 Ventilation.....	22
4.2.1 Volume and Pressure Survey.....	22
4.2.2 Results.....	24
4.3 Muck Handling and Haulage.....	28
4.4 Transformer.....	30
4.4.1 Requirements.....	30
4.4.2 Design, Testing and Approval.....	31

	<u>Page</u>
4.5 Degasification.....	33
4.5.1 Equipment.....	33
4.5.2 Procedure.....	33
4.5.3 Problems and Recommendations.....	34
4.5.4 Additional Degasification Activity.....	36
4.6 Conveyors.....	39
4.7 Cutters.....	41
4.7.1 General.....	41
4.7.2 Design.....	42
4.7.3 Performance.....	43
4.8 Shotcrete System.....	45
4.8.1 General.....	45
4.8.2 Materials Handling.....	46
4.8.3 Pumping.....	47
4.8.4 Emplacement.....	47
4.9 Guidance - Grade and Alignment.....	50
4.10 Labor Agreement.....	52
4.11 Flexicore.....	53
5. Demobilization.....	54
5.1 Procedure.....	54
5.2 Problems.....	55
5.3 Improvements.....	56

TABLES

		<u>Page</u>
I	RQD Log.....	57
II	Rock Type Distribution.....	58
III	Support Measures for Selected Support Category.....	59
IV	Compressive Strength & Hardness Test - Hole No. 1...	60
V	Brazilian & Hardness Test - Hole No. 1.....	63
VI	Compressive Strength & Hardness Test - Hole No. 2...	65
VII	Brazilian & Hardness Test - Hole No. 2.....	66
VIII	Compressive Strength & Hardness Test - Hole No. 3...	67
IX	Brazilian & Hardness Test - Hole No. 3.....	68
X	Compressive Strength Test - Hole No. 4.....	69
XI	Brazilian Test - Hole No. 4.....	70
XII	Compressive Strength & Hardness Test - Hole No. 5...	71
XIII	Brazilian & Hardness Test - Hole No. 5.....	74
XIV	Compressive Strength Test - Hole No. 6.....	78
XV	Brazilian Test - Hole No. 6.....	79
XVI	Chemical Composition Limits of Cutter Blade Material.....	80
XVII	Typical Mechanical Properties of Cutter Blade Material.....	81
XVIII	Equal Rating of Cutter Performance.....	82
XIX	Hour Meter Readings.....	87
XX	Tracking Distance and Cutter RPM.....	88
XXI	Monthly Production.....	90
XXII	Summary of Tunnel Activities by Month - 1977.....	93
XXIII	Summary of Tunnel Activities by Month - 1978.....	94
XXIV	Summary of Tunnel Activities by Month - 1979.....	95

FIGURES

		<u>Page</u>
1	Structure Map.....	96
2	Generalized Columnar Section in Tunnel Area.....	97
3	Entry and Tunnel Locations in Strata.....	98
4	Coal Cleat Orientations in Pittsburgh Seam.....	99
5	Degas Hole Orientations with Respect to Cleat System.....	100
6A	Basic Machine.....	101
6B	Basic Machine.....	102
7	Machine Cutting Head.....	103
8A	Ramp for California Traveler.....	104
8B	Arrangement Hopper and California Switch.....	105
9	Typical Stress-Strain Curve for Shaley Limestone....	106
10	Typical Stress-Strain Curve for Coaly Sandy Shale...	107
11	Typical Stress-Strain Curve for Dolomitic Conglomeratic Shale.....	108
12	Typical Stress-Strain Curve for (A) Coaly Limey Shale and (B) Limey Shale.....	109
13	Typical Stress-Strain Curve for Coal.....	110
14	Standardized Schematic of Wellhead Plumbing.....	111
15	Location of Degas Holes in Shaft and Tunnel.....	112
16	Location of Holes in Shaft.....	113
17	Sketch of Cutter Head Showing Cutter Position.....	114
18	15-1/2 Inch Rolling Disc Cutter.....	115
19	Graph of Penetration Rate.....	116
20	Graph of Monthly Progress.....	117
21	Placement of Shotcrete Liner.....	118
22	Roof Control Plan for TBM Tunnel.....	119

	<u>Page</u>
Appendix A - Rock Quality, Methods and Results.....	120
Appendix B - Financial Report	
Government Charges by Subcontractors.....	127
Government Charges for Additional Work.....	131
EACC Direct Contribution.....	135
Explanation of EACC Direct Contribution.....	139
Bibliography.....	140

ABSTRACT

A concept of mine development using a single entry bored with a tunnel driving machine was designed and completed. The effort was undertaken in hopes that the single entry could be rapidly advanced through the strata along the coal seam and increase access to coal reserves at a much faster rate than present mining systems.

The concept used was proven tunneling technology and it was not unreasonable to expect advance rates of 100 feet per day. The hardness of material to be mined was less than normally encountered in tunneling.

The project was completed and an 18-foot diameter tunnel was driven about 6,000 feet through the ultra-gassy Pittsburgh coal bed.

The achieved rate of advance fell short of the anticipated rate. The delays to boring were many and varied, most resulting from the complex nature of the project and original design or use of most components. Boring started on January 14, 1976 and was completed on August 25, 1979; 617 days were available for boring.

The project proved that a single entry could safely be mined through a gassy coal bed. The system incorporated seam degasification techniques, a unique system of rail haulage using battery-powered rail locomotives, a conveyor system to continuously discharge materials as the machine bored ahead, a concrete permanent divider to make the equivalent of two openings in the tunnel as it advanced and a shotcreting system that allowed a concrete liner to be applied. The system was in compliance with existing state and Federal mining laws as interpreted to apply to the unique tunneling project.

EXECUTIVE SUMMARY

Project Summary

One of the most pressing problems in bringing a new mining property to design capacity is mine development. Thousands of feet of multiple main entries are necessary before panel development and bleeder connections are mined. Especially in a gassy mine, the necessary development work is time consuming and delays achievement of full capacity with full utilization of plant and equipment. In fact, some operations never achieve a comfortable ratio between entry development sections and pillar mining sections.

Main entry development is expensive and slow due to the additional construction that is necessary, such as mainline track, several rows of ventilation stoppings, overcasts and graded floor or roof taken to maintain track grade or height.

Main entry maintenance after development is completed is a continual problem that increases with the age of the entries. Roof conditions causing the need for additional support and falls of roof are the usual result of increased age. Entry restrictions caused by such falls or bottom heaving which limit the mine ventilating capacity is not an uncommon condition in multiple access entries.

Longwall mining now offers advantages over conventional or continuous mining machinery to the extent that coal reserves are mined by the longwall method where possible. Sectional development often is a production bottleneck because multi-entry requirements due to federal regulations retard panel advance, and longwall machinery can finish a panel before another is made ready.

The delay in mine expansion caused by mine development was never more vivid than at the Federal No. 2 mine. This mine was started in 1968 in the Pittsburgh coal bed, and it was just beginning to take shape when the requirements of the 1969 Federal Coal Mine Health and Safety Act came into effect. The new regulations and the uncertainties of the time caused additional problems in mine layout and development.

As a result of its Federal No. 2 experiences, Eastern Associated Coal Corp. was very conscious of a need for more rapid mine development techniques. The U. S. Bureau of Mines also was well aware of such needs, and they understood that mine development would limit rapid expansion of national coal production.

The tunnel project was an attempt to transfer tunnel driving technology to the coal mining industry

to provide a rapid, low maintenance method of entry development. The success of tunneling in hard rock is well known, and the round tunnel is one of the most stable openings that can be made underground. It was felt that an eight-to ten-foot diameter tunnel would make ideal longwall retreat entries.

Several system modifications had to be made in order for the normal tunnel system to comply with existing state and federal laws. The first requirement faced was the need for a permissible machine. Only one such machine had ever been approved by the U. S. Bureau of Mines as permissible. This machine bored a 17-foot diameter hole. It was found to be available for the project. The owner of the machine was subcontracted to supervise the project.

The logical mine location was where mining would not be done for several years. Future mining in the selected area would dictate the need for a shaft about 6,000 feet from existing mine entries. The shaft was sunk through 778 feet of overburden to the coal bed and the tunnel was projected to that shaft. This distance of about 6,000 feet was considered a fair place to test the tunnel system.

The system problems caused by regulations were many and varied. The final design came after many conferences with representatives of MSHA, then MESA, the W. Va. Department of Mines, the U.M.W.A., the U.S.B.M. and various consulting engineers.

The requirement that two separate and distinct passageways must be available was a major design problem. The solution was pre-formed concrete slabs installed horizontally in the tunnel to form two separate openings. The concrete slabs had sufficient structural strength and fire rating to be acceptable for that duty. Doors with ladders were installed at necessary intervals to provide escape routes. A temporary steel divider was used between the end of the concrete portion and the tunneling machine. This divider was attached to the machine and moved as the machine advanced. Overlap between the concrete and steel divider provided the flexible air seal.

The requirement for a fresh air intake escapeway precluded the use of a conveyor belt or trolley haulage in the intake. The conveyor belt also could not be placed in the main tunnel return as the haulage system. This led to the design of a rail-mounted battery-powered haulage system.

The question concerning the location in the tunnel at which permissible equipment requirements begin caused difficulty. There was no set location determined, however MESA's position that all electric devices inby the permanent

divider would be permissible was accepted. The conveyor belt on the upper and return side of the movable temporary steel partition terminated just outby that point.

The power distribution system caused considerable design problems and delay. The tunnel machine was powered by 480 volts; however, this voltage was too low for the transmitted distance, so a high voltage (7,200) transformer was necessary. None was approved permissible, nor was there any design criteria or approval procedure for such a device. EACC eventually was able to obtain an experimental permit to use a 7,200 volt explosion-proof transformer designed for and tested in an explosive atmosphere.

The roof support requirements were met by installing two roof bolters 12 feet outby the cutters. Roofbolting was done concurrently with boring. A canopy was installed on the machine.

The ventilation requirements were set with regard to the methane liberation expected. The temporary divider was installed and machine construction modified to allow the air current to sweep the face. Dust control procedures required water sprays on the machine head.

A shotcrete tunnel liner about 3 inches thick was applied as the tunnel was advanced. This liner was intended to eliminate weathering of the strata adjacent to the coal bed and reduce methane emissions into the tunnel outby the face.

A degasification system was employed to drill holes at an angle on each side of the tunnel to a distance of several hundred feet ahead of the tunnel. Gas from these holes was collected in a pipeline laid in the return on top the concrete partition. The pipeline terminated at the borehole into the mine and the gas collected was exhausted to the atmosphere on the surface.

The total system was composed of equipment, concepts and procedures new to coal mining. In summary, the following units trailed the modified tunnel machine -- all were mounted on track: a degasification drill unit for drilling horizontal holes in the coal bed; a shotcrete machine which included hose, a pump unit and batch (mixing) car complete with conveyors to move sand, cement and aggregate to the mixing car; a "california" switch arrangement which consists of two parallel tracks accessed via a ramp carried about 3 feet above the main track by a carriage device. The conveyor belt system installed on the temporary divider discharged material to either track through a chute arrangement. The primary and secondary belts overlapped, allowing up to 250 feet of advance before the california switch arrangement was moved up.

Various problems with the system components limited the rate of advance of the tunnel. Down time on any component usually caused delays to the tunnel boring machine. Probably the overriding conclusion was that too many things had to work all the time.

The problems from natural conditions did not materially affect the success of the project. High methane liberation caused some problems at various times, and an extremely gassy area was encountered 125 feet from the shaft. The gas liberation was so high that face concentrations could not be kept below 1.0% within 8 inches of the coal or roof surfaces. The configuration and size of the machine made prompt dilution of this methane difficult, although the return air as it left the machine never contained more than 0.6% methane. This high methane content was eventually captured in drill holes and piped to the surface up the shaft, and at a location in the main return outby all machinery. Also, an unsuccessful effort was made to seal off the gas flows with resin injection. This last methane delay lasted about two months.

Roof problems were of no major consequence; falls of bolted roof did not occur. However, some rib sloughing near the machine and a rib roll occurred once near the california switch causing the concrete partition to sag on that side.

The tunnel project was a unique method of driving a single entry in a coal mine. Other methods of single-entry advance that have been tried in this country have nothing in common with this project, nor can this tunnel project be compared with conventional tunneling.

Conclusions

It was concluded that single-entry mining with tunnel boring machines is technically feasible and it can be done while complying with coal mine health and safety laws and regulations.

Lessons learned from this project can be used to design more suitable system components that would increase the reliability of the system -- but the economic comparison at this time favors the more conventional methods of mine development.

Another mine where methane liberation is less of a problem would eliminate the need for degasification and reduce the critical problems of methane liberation. The use of shotcreting could be changed considerably at such a location. It could be eliminated altogether or done at periodic intervals outby the "face" area. The tunneling could be reduced to boring, roof bolting, installing partitions and materials handling.

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1. OBJECTIVES AND HISTORY

Excavating development entries with a Tunnel Boring Machine (TBM) was an idea initiated by Eastern Associated Coal Corp. (EACC). An unsolicited proposal was submitted to the U. S. Bureau of Mines (USBM) by EACC in 1971 for the demonstration. Its purpose was to determine the economics and technical capabilities of a TBM system when applied to coal mine development. The TBM system is not a rapid production method and this project should not be viewed as an attempt to use it as such; it should be viewed as an alternate means of opening new mining areas for production. This type of rapid development should result in a greater number of mining units being placed in operation in existing mines as well as future mines.

A preliminary study of the economic and technical feasibility of using a TBM system for coal development was conducted under a USBM contract by Cobbs Engineering, Tulsa, Oklahoma, in the latter part of 1973. A search to find a machine for the demonstration led to McGuire Corporation of Ft. Smith, Arkansas, which was operating the only permissible machine in existence.

McGuire had driven three slopes with the machine for new coal mines, one for the Wabash Mine in Southern Illinois, and two for the Dilworth Mine in Greene County, Pennsylvania. In June of 1974, a cost-sharing contract between EACC and the USBM was signed to demonstrate "The Practical and Economic Possibilities and Application of Rapid Excavation Tunnel Machine and Techniques to Coal Mine Entry Development". McGuire Corporation was engaged as a subcontractor to EACC to operate their machine for the demonstration.

Up to this time the methods of entry development have utilized the conventional or continuous miner for forming entries and pillars. Though these systems are coal producing ones, their rates of advance under currently applicable regulations and constraints are very slow. The number and location of working sections which may be placed in operation at one time is limited by the rate at which access is gained to virgin coal. The concept of using the TBM for driving development headings, mains and sub-mains, would permit the outlining of large tracts of coal within the mining property early in the life of the mine. This would not only allow flexibility as to the number of sections working at any time but, also in the selection of which coal areas are to be mined to insure good product control. In gaseous mines, this procedure would aid in the degasification of the coal prior to mining.

The TBM system was designed around a modified Calweld Boring Machine built by Calweld Manufacturing Company, a division of Smith International, Inc., Santa Fe Springs, California. The machine had been approved originally by the

USBM and later by the Mining Enforcement and Safety Administration (MESA) as being electrically permissible. Since the demonstration would be taking place in an operating coal mine, the system would come under the jurisdiction of both Chapter I, Title 30, of the Code of Federal Regulations (30 CFR) as interpreted and administered by District 3, MESA, and the Mining Laws of West Virginia enforced under the direction of the State Department of Mines. Because of its permissibility, the Calweld machine was selected as the nucleus of the system to speed the project to the demonstration phase. Though modifications to the machine would be necessary, time would be saved by averting the extensive testing and approval procedure that a new machine would require.

Modifications to the machine were made to meet three goals: the first, to comply with the requisites of State and Federal standards; second, to make the tunneling system compatible with the mine operation; and third, to incorporate as many new and advanced health and safety concepts as was practical. It had been decided that as few experimental permits as possible would be sought, thus placing the emphasis on a system which could operate under existing coal mine standards.

Modifications were necessary to meet such standards as dust and methane control. The original Calweld machine controlled dust by sealing it in the face with a solid bulkhead located just behind the rotating head. Due to the hazards created by the presence of methane, sealing the coal face for dust control was not possible. The solid bulkhead was replaced with an expanded metal screen and the head was fitted with water spray nozzles for suppressing dust, similar to those used on continuous miners. The expanded metal screen allowed the ventilating air to sweep the methane, liberated by boring, from the face and dilute it to a safe concentration. By Federal guidelines, mining must cease if the methane content exceeds one percentum by volume and electrical power must be removed from the TBM proper. To quickly detect these methane levels, the machine was equipped with an automatic methane sensing system as required by Federal standards.

To make the Calweld borer compatible with the Federal No. 2 mine system, the machine had to be altered to enlarge the cutting diameter from 17 feet to 18 feet. This required that the cutter head be completely rebuilt and that the hydraulic frame jacks be extended. Two factors affected the choice of the 18 foot diameter tunnel; ventilation requirements and track haulage clearances. The history of methane liberation in the area where the TBM would be placed indicated that large volumes of air would be necessary for boring. To keep ventilating pressures to a minimum the cross sectional area of the tunnel would have to be relatively large. A complicating factor was the haulage system in Federal No. 2 Mine. Ten-ton mine cars are used as a primary means of transport to a centrally located skip hoist. It was necessary to use these same 10-ton mine cars in the tunnel to make the TBM system compatible with mine haulage. A track-

mounted California switch would solve the problem of switching in the single entry and could serve as a loading dock where mine cars could be filled. However, restrictions in the open cross-sectional area of the tunnel caused by the mine cars placed side-by-side on the switch demanded an even larger tunnel size as did the clearance between cars and tunnel walls. The 17-foot bore of the original size Calweld machine would not allow adequate room for the movement of men and ventilating air past the cars. The addition of 27.5 square feet of cross-sectional area provided the space necessary for these purposes.

Changes to the original machine were also made in several areas for improved health and safety. When used to bore slopes, the Calweld borer had no roof bolting machine included as an integral part of the system. Bolting and other support activities had been accomplished using jack leg pneumatic drills on a platform between the primary and secondary units. Temporary support forward of this platform, in the form of a hydraulically activated metal shield set against the top 120 degrees of the tunnel, protected the TBM operator. The type of ground expected to be encountered prompted the addition of two rotary percussion roof drills to the primary unit. These were placed as close to the face as practical, one on each side of the machine. The drill on the right side occupied the area formerly reserved for the operator's station. A passive support canopy was placed over these areas to protect the drill operators from spalling rock. The TBM operator's station was moved to the forward right side of the secondary unit to place him in an area which allowed greater freedom and control of the machine. All of the working areas were lighted to a level in excess of 0.06 foot lamberts lumens to provide a safer working environment. These examples of machine modifications demonstrated the primary thrust of the TBM project; the adaptation of an excavation method, proven in civil works worldwide, to a new environment with its associated hazards and limitations.

Modifications were completed at McGuire's Fort Smith, Arkansas, shop in March of 1975. The machine was then disassembled and shipped to the mine. Concurrently with the machine modification phase, the underground site to be used for the assembly of the machine was being constructed by EACC. The chamber, later termed stall, was 25 feet high, 20 feet wide, 50 feet long and was equipped with a 10-ton bridge crane. The first pieces of the machine were transported underground on June 2, 1975. The machine was transported in units small enough to be moved through the mine's main haulage entries. The basic machine and power unit were assembled by the end of 1975. The support equipment and the California switch trailing the TBM required a full 750-foot section of completed tunnel for installation. Boring of this initial 750-foot section with the modified system began January 14, 1976. Boring with the entire system was not started until January 19, 1977.

The length of the development period reflected the numerous problems which had to be dealt with in order to make the entire system legally workable. Obtaining an experimental permit for the 7,200-volt face transformer was probably the longest delay and most difficult of these problems. The project was transferred to the newly created Department of Energy in October of 1977 and was completed under its direction.

2. GEOLOGIC REPORT

A prerequisite to any tunneling endeavor is a complete and detailed geological study of its proposed path. Such a study yields valuable information which enables the engineer to prepare against hazards likely to be encountered through the course of the tunneling project. Success or failure could hinge on knowing what to expect and being prepared to handle conditions.

The tunneling project planned at Federal #2 mine (Pittsburgh coal seam) in western Monongalia County, West Virginia, (N3937.5 - W8015.0) was fortunate to be boring through horizontal sedimentary layers of proven regularity. Conditions likely to be found are well defined because of the extensive mining operations taking place in the immediate area. The following description is based on information gathered from these mining operations as well as materials published by the West Virginia Geological Survey.

Eastern Coal's Federal #2 mine is located in the northern coal fields of West Virginia in Clay and Battelle Districts of Monongalia County (see Geologic Maps). This area is on the Appalachian Plateau and features low relief and general northeast-southwest trending of its geological structures. These structures consist of gently rolling anticlines and synclines. As shown in Figure 1, the Belle Vernon Anticline and the Whitely Syncline come together and die out within the mining property near the location of the tunnel stall where boring was planned to begin. The Waynesburg syncline runs in a general north-northeast - south-southwest direction along the western edge of the mining property. This succession of very gentle folds, in which the beds of rock are so nearly horizontal that they appear level, make this area ideal for horizontal boring operations. This is well demonstrated by the core drilling data obtained at 1,000' intervals taken along the tunnel path. It shows a ten foot rise in seam elevations for the 6,000' distance with small rolls in between. The over burden thickness ranges from 650' to 1,300'. This relief was formed by the erosion of the plateau.

Figure 2 shows a columnar section of the overlying strata. Most of the surface strata down to an elevation of approximately 750' is composed of the Dunkard Group of the Permian system (refer to Geologic Maps). This group is made up primarily of sandstones and sandy shales which are extremely variable. The sandstone may change to a shale within a horizontal distance of only a few yards. Included in this group are thin layers of impure limestones and calcareous shales with at least a few thin layers of coal with associated fireclay and red and green shales. The most important sediment in the Dunkard Group however, is sand.

Underlying the Dunkard Group is the Monongahela Group which is part of the Pennsylvanian System. Its thickness is fairly regular ranging from 350' to 400', with a dominance of limestone and shales. There are also sandstones and several coal beds. The coal beds are much thicker and purer than those found in the Dunkard Group. The Pittsburgh coal marks the bottom of the Monongahela Group. It is in this coal bed that the boring was planned to take place.

Conditions in association with the Pittsburgh cyclothem were well known in the general area of the proposed tunnel site because of the mines completely surrounding it: Blacksville #1 (Consol) to the North; Osage and Arkwright (Consol) to the East; Federal #1 (Eastern Coal) to the South; and of course Federal #2's present mining operations west of the tunnel stall. The seam thickness averages 8' in the area of the proposed tunnel as shown by core drillings. This is consistent with averages found in these mines. The immediate roof rock is an interstratified coal, bone, and shale just over 7' thick in the tunnel area. As presently mined, this roof material must be protected from air and moisture by leaving 6 to 8 inches of roof coal in place (referred to as head coal) (see Figure 3a). This procedure prevents air slaking, a process which occurs progressively upward as the surface shale absorbs moisture, loses its strength and falls. The air slaking process is rapid in this area because of the thin layers of coal intermixed with the shale. An advantage of the tunneling operation is the inherent protection against this problem as a result of the circular opening. Figure 3b shows that the immediate roof is removed almost to the well consolidated layers of sandstone and shale above, resin bolted, then sealed against moisture by the shotcrete tunnel lining. The interstratified immediate roof was expected to actually prove an advantage to boring because of the ease with which it might be broken.

There were other problems brought about by strata characteristics. The coal seam is sometimes interrupted with clay veins which cut across the regular bedding. These anomalies cause hazardous roof conditions, generally resulting in falls before support can be applied. Underlying the coal is a fireclay which has high strength if left undisturbed in situ, but which is also subject to a loss of strength after exposure. It is very common for the bottom to crack and heave up to 18 inches within a few months after mining.

In the north-south mining direction (N 10° E) air slaking and heaving are much more severe than in the main development direction, (S 80° E). This anisotropy of the strata in the horizontal plane is related to the prevalent joint or cleat system present in the coal bed. The systematic (dominant) joint set is referred to as face cleats while the truncated joint, i.e., joints which are approximately normal to and which extend between but do not cross the face cleats, are called butt cleats (Nickelsen and Haugh, 1967). Cleat systems are thought to be formed by some tectonic stresses evidenced in the structure since the face cleats are nearly always perpendicular to the axial trends of the local folds (McCulloch, et.al., 1975). At Federal #2 mine the face cleats strike N77° W and the butt cleats strike N 17° E, 4° from perpendicular (see Figure 4). This is consistent in theory with northeast-southwest trending of the local structures. The mine layout was planned according to the cleat directions with the main development headings nearly parallel to the face cleat, to take advantage of the natural fractures in the coal.

The directional roof and floor effects (air slaking and heaving) are therefore related to the cleat system, but are probably not a result of these fractures; more likely, they are another reaction to the residual tectonic stresses which produced the cleats and the anticlines and synclines in the area. The tunnel path was projected to take advantage of this residual tectonic stress pattern by paralleling the primary stress direction. This alignment is believed to reduce horizontal pressures on the walls of the tunnel due to tectonics.

The cleat system was also important in relation to the direction of the degasification drilling. Research conducted by the Pittsburgh Mining Research Group has demonstrated that horizontal degasification holes which parallel the butt cleats are the most productive. The permeability of the coal is 2.5 to 10 times as great along the face cleat direction (McCulloch, et. al., 1975), therefore, holes which intersect the face cleats will effect methane liberation from a larger area. This directional permeability is consistent with the distinguishing traits of the face cleats, i.e., more pronounced and continuous.

For this project the tunneling direction was ideal for minimum methane liberation from the ribs, however, the degasification holes might prove to be somewhat less effective than anticipated because of the small angles at which they intersected the face cleats (see Figure 5). Problems might result from the gas liberation since the Pittsburgh seam in this area is known for its high methane content, as much as 200 ft³/ton (Kim, 1975).

Conclusions

In general the tunneling site at Federal #2 mine was geologically ideal for boring operations through sedimentary, coal-associated strata. The tunnel was properly oriented with respect to the apparent tectonic forces and the ground control methods to be employed were expected to provide a safe and sound working environment. Degasification could prove to be a problem and various drilling orientations were to be tried to determine an optimal direction.

¹Jenkins, F. M., Report on Geological Survey of Tunnel Boring Machine Project Site, December, 1975.

2.1 ROCK QUALITY AND PROPERTY DETERMINATION

In addition to the geologic report a "Rock Quality and Property Determination" was performed by the Penfel Company, of Morgantown, West Virginia. This internal report to the Bureau of Mines was intended for the purpose of comparing roof support requirements, as assessed by the Q Quality determination, with the proposed roof control plan for the TBM project. The Q Quality determinations were performed on NX size (2-1/8 inch diameter) roof cores provided by Eastern Associated Coal Corp. The cores were taken at 1,000-foot intervals along the centerline of the tunnel with the final core (No. 6) being extracted from the center of the proposed shaft.

The support requirements of each support category determined by the Q Quality method were compared with the roof control plan proposed for the project and found to be adequate for all areas except one. When boring advanced through the anticipated problem area the roof control plan was superceded by installing twice the number of bolts. As boring continued into more stable conditions as indicated by the report, the standard bolting pattern was reimplemented. Throughout the duration of the project not a single roof control failure occurred.

3. SYSTEM DESCRIPTION

3.1 TBM SYSTEM

The basic machine consists of two units (see Figure 6A&B), the primary unit and the secondary unit. The primary unit contains the cutting and loading functions and associated hydraulics for rotating and thrusting the cutting wheel, gripping the unit in the tunnel, and resetting after a cutting stroke is completed. Two bolting stations and a short 27 foot primary belt are also located on this unit. The secondary unit houses the transformer and breaker box, the main electric motors and hydraulic pumps, hydraulic fluid storage tanks and the operators station. The belt transfer from the small primary belt to the secondary belt is located on this structure.

Immediately outby the secondary unit, a 50 foot skid mounted trailing unit (tertiary unit) is used to aid the following support functions: (1) the laying of track, (2) shotcreting, and (3) divider support beam installation. The trailing unit is equipped with a monorail and air powered cranes to facilitate the movement and placement of materials.

Temporary steel dividers, which slide along steel beams anchored to the rib, extend for a distance of 320 feet behind the trailing unit platform to the permanent divider. These temporary dividers advance with the boring machine and provide support structure for the secondary and tertiary belts. The space under the first 150 feet of the temporary dividers is reserved for the degasification drilling equipment; the remainder is taken up by the shotcreting equipment and the California switch. This rail car switching unit consists of a double track resting on a carriage with trucks riding on the main track (see Figure 8A&B.) This switch unit is 390 feet long with a 40 foot ramp from the main track up to the carriage track. Supply cars as well as 10-ton mine cars are switched in and out with battery powered locomotives.

3.2 MATERIALS HANDLING

There are six pick-up buckets, incorporated into the cutting wheel, which collect and remove material excavated from the face. The pick-up buckets carry the fragmented material from the bottom of the tunnel invert to a point directly overhead where the material free-falls into a hopper mounted immediately behind the cutting wheel. This hopper feeds the 36-inch primary conveyor which in turn feeds into another hopper arrangement, for the secondary conveyor belt, that allows for the movement necessary to stroke forward some 27 inches.

The secondary conveyor is mounted to the temporary divider panels and moves forward in 27 inch increments as the boring machine is reset after boring a full stroke. The outby portion of the secondary conveyor overlaps the tertiary conveyor and results in a telescopic action when the temporary divider is pulled forward with boring machine reset.

The tertiary conveyor is fixed to the dump station at its outby extremity and rests on roller brackets at its inby extremity to allow movement of the temporary divider.

The automatic loading unit consists of a hydraulically operated loading chute and car spotter which provides for uninterrupted loading of several 10-ton mine cars on the California switch. One mine car has the capacity for approximately 8 inches of boring advance.

3.3 FRAGMENTATION

Thirty-seven rolling disc cutters and one centrally-mounted multiple cutter are arranged on the cutting wheel to effect circular orbits spaced approximately 3 inches apart. Three of the disc cutters gage the 18 foot diameter tunnel; in addition, spring loaded scraper blades on the perimeter of the cutting wheel help to smooth the tunnel walls. (Figure 7).

A water spray system designed for dust abatement was added to the TBM for this project. Approximately 50 gallons per minute of water is carried through the shaft to the cutting wheel where 35 spray nozzles are placed so as to apply a spray at or near each cutter.

3.4 ROOF AND GROUND SUPPORT

Two bolting machines, one located on each side of the primary unit 12 feet from the face, provide for the installation of the initial roof support. Six five-foot resin grouted bolts are placed on five-foot centers spanning the arch of the roof. (Figure 22) The drilling method is a hydraulically operated rotary/percussion drill equipped with a through-the-steel vacuum dust collection system. Each drill has two dust collectors, each with a one-half cubic foot capacity. The four collectors are located over the inby end of the secondary conveyor and have drop bottoms for ease of cleaning.

The finished drill holes may vary in diameter from 1 inch to 1-1/8 inch to 1-3/8 inch drilled to a depth of just over 5 feet. The 5-foot rebar rod with a forged head varies correspondingly from 3/4 inch to 7/8 inch to 1 inch. A quick setting epoxy resin is used to grout the rod in the hole. The resin is packaged in cartridge form and is broken and mixed by spinning the rod in the hole. A standard bearing plate is set directly against the roof to maintain maximum clearance.

The above described bolting plan represents the minimum amount of bolts installed in the approved roof control plan for the TBM project. Where roof conditions dictate, extra bolts are installed as necessary.

In addition to the roof bolting operation the tunnel is lined with approximately 3 inches of shotcrete. This lining aids stabilization by protecting the roof and ribs from the mine atmosphere.

3.5 SHOTCRETE LINER

The shotcrete system in its final form consists of the following components: (1) bulk material cars, (2) materials handling conveyor, (3) batching car, (4) mixing augers, (5) pump unit, (6) emplacement hose and attendant air line.

The bulk material cars were to be loaded from stockpiles on the surface and transported with other supplies to the California switch in the tunnel. The shotcrete materials-handling belt is located between the left set of rails on the inby end of the California switch. As shotcrete materials are needed, the bottom dump material cars are positioned over the belt and the various materials, i.e., course aggregate, sand and cement, are transferred, one at a time, to the three compartment batch car.

From the batch car these materials are fed simultaneously, at a predetermined rate, onto a single conveyor belt that dumps into the mixing hopper. The mixing hopper feeds into first, the dry mixing screw conveyor and then, while water is introduced, to the wet screw conveyor. The discharge from the wet mixing screw goes into the pug mill of the pumping unit.

The positive-displacement pump feeds into a flexible hose which, in turn, feeds a 3 inch rigid pipe called a slick line. The slick line is used as far as conditions permit before terminating with the 2-1/2 inch application hose. The end of the application hose is reduced in diameter by a pneumatically supplied nozzle for directional control of the concrete. At this point, the concrete is fragmented by expanding air and sprayed onto the exposed roof, rib and floor of the tunnel. After successive passes, enough build-up is attained for a competent concrete liner. (See Figure 21).

3.6 VENTILATION

The tunnel face was ventilated by air coursed through the tunnel's upper and lower compartments. The intake air was brought in the lower portion, swept the face, and exhausted through the upper compartment of the tunnel. There was a minimum requirement of 50,000 cubic feet of air per minute measured at a control point 140 feet from the face.

Temporary dividers extended from behind the trailing unit to just inby the loading point where permanent flexicore dividers were placed on beams to the tunnel mouth. The temporary steel dividers supported by beams bracketed longitudinally along the tunnel ribs. Thus a complete separation existed from the overcast at the mouth of the tunnel to the face. The temporary divider was moved up as the machine advanced and served as support for the secondary and tertiary conveyors. Flexicore deck sections were installed as the temporary divider advanced. These deck sections consisted of prestressed concrete beams cast with two pre-tensioned cables. Each deck is 16-1/2 feet long, 2 feet wide and 6 inches thick. Doors through the divider were provided every 80 feet with ladders to the lower invert. Every third door was made larger and equipped with swing down stairs for access between upper and lower compartments. Provisions were made in the ventilation system at the mouth of the tunnel to reverse the direction of air flow in the upper and lower compartment during an emergency.

3.7 ILLUMINATION

Fifty permissible 30 watt fluorescent lamps were placed on the primary and secondary units to provide an average intensity of 0.06 foot lambert lumens at all work areas. These lamps were operated by 110 volt AC power furnished from a 480 volt/110 volt 25 KVA transformer. Both the transformer and control panel were located on the secondary unit.

3.8 DEGASIFICATION

Degasification was incorporated into the TBM project as a supplemental method of controlling the methane expected to be liberated in the virgin boring area at rates commensurate with the anticipated rapid advance rate. Historically virgin areas of the Pittsburgh seam have been known to liberate as much as 200 cubic feet of methane per ton of coal produced.

In particular, the objectives were to collect and drain gas in advance of boring and to prevent gas build-up behind the liner that, through leaks, could compromise the quality of the intake air ventilating the face.

The system utilized wells drilled horizontally at the center of the coal seam. The wells were aligned 10 to 15 degrees from the longitudinal axis of the tunnel at staggered locations on the right and left ribs. (See Figure 15).

A specially designed drill was incorporated into the TBM system to facilitate the drilling of the degas wells. The drill has a hydraulic chuck and operated with a 5 foot stroke.

Gas produced from these wells was channeled into a 6 inch pipeline laid in the upper compartment of the tunnel. At the tunnel overcast the 6 inch line was connected to an 8 inch line that extended through the mine approximately 3,000 feet to a 24 inch vertical borehole through which the gas was transported to the surface where it was released to the atmosphere.

Each well was equipped with an automatic shutoff valve activated by a methane detection system. The detection system consisted of a series of eight sensors placed in strategic locations to constantly monitor the atmosphere surrounding the pipeline. In the event these sensors detect over one per cent methane in the mine atmosphere the automatic valves shut-in the wells at the wellhead. This precludes a buildup of gas from the line should a leak develop. Other safeguards included a system scheme which closed the valves in the event the collection system was damaged by a roof fall. A one half inch PVC air line was strapped to the top of both the 6 and 8 inch pipelines. In the event of a roof fall this line is broken, releasing the compressed air supply to the automatic valves, causing the wellheads to be automatically shut-in.

4. OPERATIONAL PROBLEMS

4.1 GENERAL HISTORY

The first stage of tunnel boring presented several unique operational problems. These persisted for the first 725 feet of tunnel excavation and were problems due primarily to two conditions. The first condition resulted from the lack of space for the installation of the complete back-up system of automated loading which required considerable tunnel footage. Secondly, the initial boring had to be accomplished using 480 volt power until an approved 7200 volt face transformer was available.

The tunnel boring machine was assembled in a stall area especially excavated for this purpose. The stall was 25 feet high, 20 feet wide, 50 feet long and was equipped with a 10-ton bridge crane to facilitate assembly of the boring machine.

Special steel side supports were fabricated and placed under the side jack housings in order to prevent the machine from rolling due to torque reaction before it was socketed into the tunnel. Also special steel pads were made and placed under the reset jack's feet due to the flat floor of the assembly stall not conforming to the circular invert of the 18 foot tunnel cross-section.

The steel side supports did not prove adequate to carry the machine thrust reaction. The front belly jack was then securely tied down to effect suitable anchorage, but without success. It soon became evident that this arrangement would not suffice as the belly jack was being wrenched from the machine. This plan of attack was abandoned in favor of another method of anchoring the machine.

Steel plates with eyes were secured to the floor with epoxy resin rock bolts near the front of the machine. Near the rear, steel eyes were welded to the machine frame. These sets of eyes were tied together with steel cables and hydraulic cylinders.

The cables were tensioned by means of the hydraulic cylinders and boring proceeded by pulling the machine into the face. Use of this method continued until enough advancement into the tunnel had been made for the side gripper jacks to engage the rings of the tunnel at which time it was no longer required.

During this socketing-in phase, the machine power and skid units traveled through the overcast constructed at the end of the stall. This overcast could only be partially completed to allow for the passage of the equipment. During this passage, ventilation had to be controlled with the use of temporary and movable stoppings and brattice cloth.

Some difficulty was encountered with the entry of the trailing units into the tunnel cross-section. This was overcome by hand excavation to let the units skid into the tunnel.

After the last trailing unit had cleared, the overcast could be completed. A lower deck was installed and doors hung. By opening or closing the doors and ports, ventilation could be reversed as the need would arise for operations such as welding or cutting on the cutter head. This would allow intake in the upper compartment of the tunnel and return air in the lower compartment. Being an emergency measure, the doors were maintained in a locked posture to prevent unauthorized movement.

The steel divider panels were installed on the lower deck of the overcast and abutted to the rear of the skid unit. A tow bar was fastened to the gripper jacks by wire rope and hydraulic cylinders. Wire rope ran from the tow bar back to the divider panels in order to pull them along. The tow bar was fastened to the skid unit by welded beams. Due to this rigid connection, the skid unit was damaged. This condition was corrected by replacing the beam connections with turnbuckles and pinned connections. This arrangement allowed for rotation of the components due to irregularities in the invert and solved the problem.

Mucking during the first stage of boring was performed by using a primary and a temporary secondary conveyor. This presented a problem as there was only a single track in the tunnel entry before the special switching arrangement was installed. As a result, the train of muck cars, when loaded, had to be taken from the tunnel before another string of empty cars could be brought in. Outby of the loading point, any work being performed had to cease and all equipment being used for that work had to be removed to allow passage of the train. During these tramming cycles, boring was stopped because of the interruption of the muck loading operation.

When sufficient footage for the back-up system had been bored, the temporary secondary conveyor was removed. It was replaced with a permanent secondary conveyor through which a tertiary conveyor telescoped as boring proceeded. The loading point was located outby the major construction areas.

A loading and passing arrangement called a California Switch was installed in January of 1977. The California Switch consists of double tracks mounted on rail trucks which ride on the single track placed in the tunnel invert as boring advances. Access to the California Switch, which is elevated approximately 3 feet to take advantage of the width near the tunnel centerline, was gained by the use of a 40 foot ramp that had a permanent connection to the switch assembly and a floating connection to the tunnel rail. The double track allowed a string of empties to be loaded while a train of loaded cars were being switched out. With this set-up boring was continuous.

4.2 VENTILATION

Late in 1976 the John T. Boyd Company of Pittsburgh, Pennsylvania was commissioned by Eastern Associated Coal Corp., at the request of USBM, to investigate the technical problems and the feasibility of the overall ventilation system for the TBM project at the Federal No. 2 mine. The investigation included a detailed ventilation survey of a portion of the tunnel, evaluation of problem areas, recommendations for practical solutions to ventilation problems and projection of ventilation parameters for the full 6,000 feet of the tunnel ventilation system.

The study included recommendations and designs to maintain adequate ventilation at the face to control liberated methane and dust at the roof bolting stations. These recommendations were based on detailed surveys and inspection of the operations and considered the mine's ventilation system, as well as requirements and constraints of the TBM and state the federal regulations.

4.2.1 VOLUME AND PRESSURE SURVEY

The tunnel ventilation system consisted of a single split of air taken from 1 East Mains ventilation near the mouth of the tunnel. The only parallel splits within this circuit were in the shop and compressor station areas. A relatively small quantity of airflow was regulated directly to returns through these areas. Main intake air passed through the stall area and under the overcast just outby station 0+00, the beginning of the tunnel. The tunnel was divided by a horizontal platform 6 feet from the top and 11.5 feet above the floor. The lower portion with a cross sectional area of 155 sq. ft. provided the intake air course. Air was directed the length of the tunnel to the face. Its course was periodically obstructed by various pieces of operating and support equipment, including the California switch which might have rock and supply cars on one or both tracks, shotcrete equipment and the degasification drill. The designated control station was located beyond this equipment at the outby end of the trailing unit. At this point 50,000 cfm intake air was required by MSHA.

At the face, air passed through the lower half of the TBM and up around the jacks and borer to the upper portion of the TBM. Return air was directed through the upper section of the tunnel. Flow was obstructed immediately outby the face portion mainly by the structure of the extensible secondary conveyor. Down stream from the loading station the return air course was effectively clear and unobstructed to the overcast. Planned methane drainage piping and related installations were not expected to cause significant obstruction. At the overcast, the air flow split to return air courses on either side of the stall area, with flows through the regulator in each split to the main returns.

In order to measure basic ventilation parameters, several types of standard ventilation survey procedures were used. These included volume measurements by standard anemometer traverse and pressure surveys using precision altimeters, magnahelix gages and a micro-manometer.

Volume survey data was derived from standard vane-anemometer traverses at selected stations through the circuit where velocities were less than 2,000 ft. per minute. Each measurement was based on averages of three or more successive traverses. These repetitive measurements yielded data which varied less than 2% from the average at each station. The cross sectional area at each station was obtained from actual measurement of the openings, supplemented by measurement of TBM cross section in the project plans.

Pressure surveys using precision altimeters were conducted in accordance with procedures discussed in U.S. Bureau of Mines Information Circular 7809, and MESA Information Report 1050. Four surveys were completed between December, 1976 and December, 1977. For each survey a pair of Wallace and Tiernan and a pair of Paulin altimeters were used.

The precision altimeter survey technique for determination of relative air pressure in a ventilation system is sensitive to many variables for which measurement and corrections are routinely made. The following table presents the magnitude of pressure difference resulting from some changes in standard conditions.

<u>Change of</u>	<u>Yields Pressure Difference</u>
One foot in elevation	.015 in. water gage
10% increase in intake velocity (50 fpm)	.003 in. water gage
10% increase in return velocity (100 fpm)	.013 in. water gage

The initial surveys of the TBM indicated that the actual pressure differences along important segments were near or less than the accuracy of the instruments. In addition, periodic pressure changes of a greater magnitude were encountered. These changes were ascribed to "local disturbances".

Local disturbances included all causes of erratic pressures and volumes within the TBM system that were not measured by the base altimeter station and which yielded data inconsistent with the pressure gradient and airflow of the system.

Trains moving in the open intake were estimated to have a minor effect on the system ventilation. Trains have a cross-sectional area of 20% of the intake area and a maximum adverse exposure of 12 minutes per trip at completion of the tunnel. Measurement across the shotcrete machine in the open tunnel indicated the pressure drop would be very small.

These and other unidentified sources singly and in combination could radically effect the steady state flow of the system for short periods of time. The pressure traverses were performed on shifts when the TBM was not in operation. Nonetheless, on occasion, the presence of a disturbance could be identified during a pressure traverse, and at these times the survey was temporarily halted until the disturbance had been corrected. Evaluation of the pressure data indicates that some disturbances were undetected at the time of the survey.

For all precision altimeter surveys, spot checks and special measurements were made using sensitive magnehelix gages of several ranges or a micromanometer. These instruments were calibrated and cross checked prior to the survey.

The final survey was conducted during the UMWA strike in 1978 when tunnel activities included shotcreting of intake and returns immediately outby the California switch; there were also inspection tours by firebosses, surveying personnel, management and visitors. Local disturbances were observed on the altimeter survey.

The final altimeter survey was followed by a pressure traverse using a micro-manometer and 600 ft. of 1/4 inch diameter polyethelene tubing. The procedure used was to extend the tubing and measure the static pressure drop along the air course between ends of the tubing.

4.2.2 RESULTS

The following table indicates the general decline in ventilation efficiency over the period December 8, 1976 through December 15, 1977:

Date of Survey	cfm		Efficiency %
	Station 0+00	Control Station	
12/8/76	77,000	45,000	58.4
12/21/76	78,000	46,500	59.6
3/30/77	84,000	47,500	56.5
4/1/77	92,000	51,500	56.0
6/21/77	86,000	44,000	51.2
7/6/77	79,000	43,000	54.4
12/15/77	84,000	41,000	48.8

A volume survey of December 15, 1977 showed ventilation quantities at selected locations to be as follows:

Location	Station	Quantity cfm
Main Intake	A	98,500
Tunnel Mouth	0+00	84,000
Beginning California Switch	17+04	73,000
Outby Loading Station	19+84	71,000
End Permanent Divider	22+36	61,500
Control Station	25+84	41,000

A pressure survey made at the same period of time indicated the following pressure losses at selected locations:

<u>Location</u>	<u>Station</u>	<u>Pressure Drop Intake to Return (in. W. G.)</u>
Tunnel Mouth	0+00	0.810
Beginning California Switch	17+04	0.400
Loading Station	19+84	0.310
End Permanent Divider	22+36	0.200
Control Station	25+84	0.060

Pressure gradients for major section of the tunnel were established as follows:

<u>Segment</u>	<u>Gradient (in. W.G./100 ln. ft.)</u>
Open Intake	.0038
Open Return	.022
Intake over California Switch	.0088
Return over Conveyor	.037

Frictional pressure loss constant (K) and resistance factor (R) were calculated based on final survey data. The K & R factors for major tunnel segments are listed as:

	<u>K</u>	<u>R/1000 ft.</u>
Open Intake	20.9×10^{-10}	.054
Open Return	16.4×10^{-10}	.311
Intake across the California Switch	75.4×10^{-10}	.195
Return across the Conveyor	60.4×10^{-10}	1.223

Leakage sources and losses were identified for various segments of the tunnel as follows:

<u>Location</u>	<u>Major Sources</u>	<u>Quantity (cfm)</u>	<u>Equiv.Area (sq. ft.)</u>
Overcast Outby 0+00	Cracks, door seals at air lock	7,000	3.1
Permanent divider	Stairway doors, scuttles, cracks by I-Beams	650/100 ft.	.33/100 ft.
Loading chute	Leakage around chute	2,000	1.4
Gantry and temporary divider	Box beam, above and behind I-Beam	20,500	23.7

Ventilation conditions were projected for the entire 6,000 feet of tunnel for three cases:

Case A. 50,000 cfm to control station with conditions as found on December 15, 1977 excepting cars on California switch with loading chute open.

Case B. Same as Case A except leaks between the loading station and control station were reduced by 36% and in the permanent divider by 50%.

Case C. Same as Case B except loading chute was partially closed; leakage was reduced in the permanent divider by 70%, in the chute by 57% and between the chute and control station by 64%.

The following parameters were calculated for three cases:

	<u>Case A</u>	<u>Case B</u>	<u>Case C</u>
Volume at 0+00	142,000 cfm	97,000 cfm	73,300 cfm
Pressure at 0+00	2.998 in.W.G.	1.868 in. W.G.	1.264 in.W.G.
Efficiency	35.3%	51.3%	68.2%

Case B was considered achievable and was within the capacity of the existing mine ventilation system. A volume of 114,400 cfm could be delivered to the TBM area in the mains at Station A and a pressure difference of 2.118 was available between Station A and the tunnel returns.

Having decided on the major parameters for the three cases presented, the pressure/quantity survey data were projected to completion of the tunnel. The following is a summary of that calculation:

	<u>12/15/77</u>	<u>Case A</u>	<u>Case B</u>	<u>Case C</u>
Tunnel Length (ft.)	2,677	6,000	6,000	6,000
Volume to control station (cfm)	41,000	50,000	50,000	50,000
Volume to 0+00 (cfm)	84,000	142,000	97,400	73,300
Pressure across 0+00 (in.W.G.)	.790	2.998	1.868	1.264
Leakage equiv. area (sq. ft.)	38.2	51.4	31.6	17.3
Efficiency @ 0+00	48.8	35.2	51.3	68.2

The Boyd Company recommended and Eastern accepted the parameters set forth in Case B as being reasonable and achievable. The following data summarizes the connection between Station 0+00 and the main intake (Station A) 200 feet outby the overcast:³

³Boyd, J. T. Company, Ventilation Analysis, Report 974, August, 1978.

Case B		
Total Required		
<u>Location</u>	<u>Pressure</u> <u>(in. Water Gage)</u>	<u>Volume</u> <u>(cfm)</u>
Across Intake and Return		
Control Station	0.060	50,000
0+00	1.868	97,400
Overcast		7,000
Shop		3,000
Compressor Station		7,000
Shock & Friction	0.250	
Station A - Total	2.118	114,400

Upon completion of the tunnel the parameters set forth in Case B were found to be a very accurate projection of ventilating conditions. The total pressure drop from intake to return at Station A was 2.28 inches at time of intersect with the shaft. The measured quantity at the face control point was in excess of 60,000 cfm. A diligent leakage control effort was responsible for maintaining this quantity of air. Leaks in the permanent divider were minimized by frequent patching from the overcast to the inby end of the permanent divider.

During the initial period of downtime attributed to the excess gas in the face area, extra personnel were available to correct leaks in the temporary divider by replacing seals, shotcreting and removing unnecessary obstructions. The gantry which had always been a major source of air leaks was completely covered with thin plastic sheeting on the pressure (intake) side to provide the optimum ventilation seal.

4.3 MUCK HANDLING AND HAULAGE

The vast quantities of muck generated by the tunnel boring operation did not fit into the dumping and cleaning cycle for normal run-of-mine production. Since no attempt would be made to recover the coal from the tunnel product, it was treated and disposed of as mine rock would be. The implications of this were that the large quantities of muck coming from the project had to be stored in mine cars until time was available to dump rock at the hoist shaft. In times of high production surges at both the tunnel and from the coal producing sections, this would result in mine car shortages due to standing loads. In addition, problems were encountered with excessively wet muck sticking in the mine cars and spilling from the surface refuse belt.

As the muck was cut from the face, it was scooped into the buckets on the wheel by plows. The buckets discharged onto the primary conveyor. The conveyors in turn loaded mine cars which were transported to a rotary dump at the mine production shaft. Once on the surface the muck was directed from the skips around the preparation plant to the rock holding bin. From here the muck was transported directly to the refuse area where it was disposed of by layering and compacting with earth fill to prevent spontaneous combustion. No attempt was made to recover the coal due to the high percentage of reject material and the anticipated high rate of production.

Spillage under the machine had to be hand mucked into the pick-up buckets. This was slow and laborious when muck had to be carried from the rear to the front of the machine. A small conveyor was acquired for this purpose. The muck was then tossed on this belt and dumped into the buckets in the rotating head.

Rail haulage of muck presented several difficulties. Due to the coal mining laws restricting the installation of trolley wire in fresh air escapeways and the tunnel being a single entry with the lower compartment serving as both the escapeway and haulage way, battery locomotives and transportation became a vital part of the project.

In the early stages the project was furnished standard mine locomotives equipped with shop-built battery cars. While these proved sufficient, they also created many problems and much downtime due to maintenance and charging procedures.

A small battery locomotive was acquired from one of Eastern's other mines for use on the project. This locomotive served the project well except for its size and weight which comprised the reliability necessary for this particular element of the operation.

The effect of the Federal Regulations severely restrained the haulage operation in the tunnel. Battery

locomotives were prohibited from traveling inby the permanent divider. This resulted in using air tuggers to place and spot cars and supplies in the early development of the tunnel. The far-reaching effect of the regulations was a complete design change in the location and operation of the loading station. In the original concept the loading station was to be located in the outby end of the temporary divider and move forward with that unit. The California Switch would also be towed by the skid unit resulting in intermittent forward movement of the entire loading system.

Having made this design change the later mode of operation was a stationary loading station outby the temporary divider making it necessary to idle the boring operation to move the California Switch forward in 250 foot increments.

To restore reliability to the haulage element of the operation, Eastern purchased two combination battery/trolley locomotives for exclusive use on the project. These locomotives were designed to operate in tandem to meet the service requirement of tunnel haulage. A charging station was set up in the general area of the tunnel for convenience and necessity of changing batteries.

Many problems followed as the new locomotives were put into full service. Initially, a problem with the battery chargers resulted in a series of burned contactor tips due to improperly charged batteries and subsequent low voltage. This was traced to improper wiring of the chargers when received from the manufacturer. Following this problem was an episode of traction motor failures so severe that a temporary halt was called to the project. By appealing to the higher management of the manufacturer of the locomotives, the proper involvement was finally attained and the problem was traced to improper safeguards within the wiring network of the units. A factory-authorized rewire of the units was completed and satisfactory service prevailed through the duration of the project.

4.4 TRANSFORMER

4.4.1 REQUIREMENTS

The original 4160 VAC, 750 KVA Crittenden permissible transformer could not be used at Federal No. 2 mine because of its incompatibility with the 7200 VAC underground power system. Normally the transformer could simply be changed from 4,160 to 7,200 volts with the reconnection of the primary phase leads. This could not be done due to an imposed regulation (18.45(d)(5) and (6) CFR Title 30) which the Mine Safety and Health Administration (MSHA) enforced.

Regulation 18.47(d)(5) implies that 4,160 volts is the maximum voltage permitted to permissible face equipment and 18.47(d)(6) reserves MSHA the right to require additional safeguards for high voltage equipment or modify the requirements to recognize improved technology. This was a first attempt to utilize 7,200 volts to supply power through a portable trailing cable to a permissible piece of mining equipment. Many delays occurred while trying to determine what additional safeguards were required, the technology available to comply and finally designing and testing these designs.

Due to these numerous delays in obtaining an approved 7,200 volt transformer and system design, a temporary approval was initiated to use 480 volts directly through a 500 MCM trailing cable. This permitted 725 feet of boring to be accomplished which allowed time and room to install the final machine components while pursuing the 7,200 volt system design.

4.4.2 DESIGN, TESTING AND APPROVAL

During the initial boring of the 725 feet necessary for the California switch installation, a 480 volt trailing cable was acceptable for power supply. Due to the projected distance of nearly 6,000 feet, the voltage drop resultant from transmission at 480 volts would have made the size of the trailing cable prohibitive. For this reason, it was decided to seek modification of the approved 480 volt system to allow a 7,200/480 volt face transformer.

The root problem of using a 7,200 volt transformer for the TBM was its intended location in an area where explosion-proof equipment is required.

Concurrent with the initial boring effort, design and construction of an acceptable enclosure and switch gear was progressing. Eastern's electrical engineers, working in concert with the USBM, MESA and the McGuire Corporation, designed and built an enclosure for subsequent testing to withstand at least four phase-to-phase faults at the maximum available fault current of 3,500 amperes. The duration of the current was initially specified at eight cycles or twice the breaker clearing time, whichever was greater.

The enclosure was tested for the following end results:

1. No visible discharge of flame or sparks from the enclosure.
2. No rupture of any part of the enclosure.
3. No permanent distortion of the enclosure exceeding 0.040 inch per foot.
4. No surface temperature exceeding 302°F.
5. No ignition of a surrounding explosive methane gas mixture.

In addition to the testing of the enclosure, the following requirements were imposed for electrical compliance:

1. Two independent circuit breakers in series shall provide the overcurrent protection.
2. One of the breakers shall be located on the surface.
3. The overcurrent devices of both circuit breakers shall be set at 50% of the available phase-to-phase short circuit current. At this setting the breakers shall clear the fault within one-half the test time stipulated above.

4. The transformer shall be provided with thermal overload protection. Such protection shall be set to remove the primary power before the insulation temperature rating is exceeded.
5. In addition to the testing required in CFR 75.800, the following tests shall be performed:
 - a. Upon initial installation, all protective devices shall be electrically tested and adjusted.
 - b. The protective devices of the surface circuit breaker shall be tested by means of primary injection upon installation and at periods not to exceed three months thereafter.

Testing was arranged by the USBM at the Westinghouse High Power Laboratory in Pittsburgh. The first test revealed that, under the specified fault conditions, the enclosure was not of sufficient strength. The maximum energy transfer under the test conditions was such that the physical dimensions of an enclosure strong enough to properly contain the fault condition was prohibitive.

Eastern's electrical engineer proposed to the controlling agencies the use of current limiting fuses to reduce the magnitude of the energy transfer and to reduce the test time duration to 3 cycles. In concert with this design change, the case was strengthened externally by the addition of structural beams wrapped around the case. Internally, a low density silicon rubber foam was used to provide expansion area in the top of the enclosure. The transformer and enclosure were again tested by the Westinghouse facility and the results were satisfactory to all concerned.

By this time the boring of the initial 725 feet of tunnel had been completed and installation of the California switch was nearing completion. Installation of the approved transformer occurred concurrently with the completion of the California switch installation.

Boring with the total system commenced in January of 1977 and progressed for approximately four months before a ground fault in the transformer occurred. Removal and inspection of the transformer revealed that the insulation of a lead had been damaged presumably during installation. Also, during the subsequent rebuild of the transformer, a faulty heat sensor was replaced with an improved design. Boring to completion, some 4,800 feet, was uninterrupted by transformer problems.

4.5 DEGASIFICATION

4.5.1 EQUIPMENT

A drill and auxiliary equipment were selected that would best suit the requirements for drilling in the confines of the tunnel. To further tailor the equipment to the job, other apparatus was designed and fabricated on site. Included in this was a track-mounted water car that enabled recirculation of the drilling water and collection of the drill cuttings.

The drilling equipment consists of the following major items: (1) track-mounted drill and power unit, (2) track-mounted water car with integral water pump and water/gas separator, (3) hydraulic wire line hoist, (4) 100 joints of BQ drill rod in 10 foot lengths and (5) stuffing box.

4.5.2 PROCEDURE

A predetermined plan called for drilling 500 foot wells on alternating sides of the tunnel at 250 foot intervals. This plan would result in having a completed well on line in advance of boring at all times.

The 6-inch pipeline in the upper compartment of the tunnel was advanced in 240-foot increments as boring advanced. Due to space limitations beside the conveyors, the pipeline could only be advanced to the dump station. From this point inby to the well site, 2-inch line was suspended from the rib to make the connection from the 6-inch line to the well complete. The 2-inch line was suspended because of the forward movement of the temporary divider panels as the boring machine was reset.

After the pipeline extension was complete, the collaring guide was positioned, using resin-grouted rods, on the rib at the well site which had previously been located and protected from shotcreting. The drill was then moved right or left off its platform of the rail trucks and positioned to drill through the collaring guide and anchored securely in place.

A 3-inch pilot hole was drilled to a depth of 20 feet and subsequently reamed to a diameter of 6 inches. When the reaming operation was complete, the collaring guide was removed and the coal around the collar was chipped away in such a way as to recess the wellhead for protection from damage that could occur from exposure to rail traffic at a later date. A 4-inch black steel pipe with a 4-inch gate valve was inserted in the reamed hole and grouted in place as a means of controlling gas flow during and after drilling.

Drilling was started with a 20-foot section of NW drill rod with a front and rear stabilizer. The NW rod was slightly larger in diameter and more resistant to flexing than the BQ rod

planned to be used from this point on. The stabilized section was designed to combat the natural tendency of the hole to drift.

After the 20-foot stabilizer section was inserted in the hole, the stuffing box connections were made. This device effected a nearly airtight connection around the drill rod and collected the gas, water and cuttings to be piped to the water car. The gas, water and cuttings entered the water car through a gas/water separator box that allowed the heavier water and cuttings to drop out the bottom of the box while the lighter gas escaped through an outlet in the top of the box that connected to the gas discharge line.

From this point until completion, the well was surveyed with a downhole instrument at 50-foot intervals to make certain that no conflict existed with the well direction and the direction of the tunnel. When the desired completion depth was obtained the drill string was removed and 1-1/2 inch PVC rigid pipe was inserted to the end of the well for future grouting purposes. Past experience in mining through open holes of this type had dictated that provisions be made for grouting before blockage occurs in the well that would preclude inserting the grout pipe completely to the back of the hole.

Upon completion of this operation, the standardized wellhead plumbing was installed (see Figure 14) and the well was put in service. At this time the drill and associated equipment was readied for moving to a new location. The water car had a capacity to hold the cuttings from two 500 feet wells before cleaning was necessary. If cleaning were necessary, it would be done at this time in the cycle, otherwise the cycle would begin again with the extension of the pipeline.

4.5.3 PROBLEMS AND RECOMMENDATIONS

Problems with the degasification drilling effort will be addressed here in the order they appeared in the drilling cycle.

Extending the 6-inch pipeline was a problem that intensified as the tunnel advanced in distance. Provisions were not made in the temporary divider to accommodate the moving of 6-inch pipe from the lower to the upper compartment. Ideally, an access door would have been provided in the temporary divider for this purpose. Pipe could then have been transported by rail to the temporary divider and taken through the access door to the area where it would be used. As it was, the only access for the pipe to the upper compartment was an opening in the overcast at the mouth of the tunnel which resulted in transporting one joint at a time manually using a pipe dolly designed for that purpose. As the tunnel advanced, this segment of the cycle became so time-consuming that an extra crew had to be added just to transport pipe. On several occasions boring was delayed by the degas operation for this reason.

Inserting the 4-inch casing in the 6-inch reamed portion of the well was a recurring problem. Deflection of the 3-inch pilot hole and subsequent deflection of the reamer resulted in a slightly curved hole. The rigidity of the 4-inch pipe was such that manual insertion was impossible at times. The casing would have to be removed, cut off and reinserted, causing an unnecessary delay. This problem was overcome by using a stabilized bit assembly for drilling the pilot hole, after which a reamer was substituted for the stabilizer behind the pilot bit. The pilot bit would then follow the open hole and lead the reamer in a straight line. After this procedure was adopted no more problems were encountered in inserting the casing.

Directional control of the drill string was a major problem. To effect methane drainage from a target area in advance of boring, it was necessary to control the direction of the well. The NW stabilizer section used was successful in controlling the horizontal deflection but, due to its weight, caused the bit to deflect into the rock underlying the coal seam. Due to this lack of vertical control the NW stabilizer was abandoned.

After the abandonment of the stabilizer and before personnel were trained in the use of the down-hole surveying instrument, a well was drilled on the left side of the tunnel to a depth of 940 feet. On July 12, 1978, this well was intersected by the boring machine at station 32+42. The boring operation was halted by a MSHA control order until the hole was properly plugged with cement grout on August 1, 1978. The surveying procedure, which was optional at that time, was incorporated into the degas drilling permit as a mandated requirement.

In a subsequent drilling effort directed by the USBM, Pittsburgh Mining Safety Research Center (PMSRC), a configuration of 3 stabilizers, separated by 2-foot sections of drill rod, successfully controlled both horizontal and vertical deflection.

Methane collection and control presented a problem when high flow rates were encountered. The gas/water separator would pressurize to such an extent that the excess gas would enter the water car and exit through the clean-out panels in the top of the car. These panels were not designed to be air tight but could easily be redesigned to solve this particular problem. This problem presented itself on the last well drilled and due to time constraints was the reason for the abandonment of the well at a depth of only 90 feet.

Initially, water traps were not included in the standardized wellhead plumbing because the wells were not producing water. Later, as the tunnel advanced into a wet area, the wells also became wet and water traps were included in the plumbing scheme.

However, before the water traps were available for installation, the wet wells were flowing water into the pipeline causing a pressure situation to develop. This was not a major concern on completed wells but it caused methane control problems on wells that were being drilled. When this situation surfaced, drilling was stopped and the pipeline water was blown out with compressed air. With proper maintenance of the water traps, this problem did not recur.

Undulation of the coal seam in proximity to the tunnel gradient was a problem that resulted in abandoning the last two projected wells. From station 48+00 to completion, the coal was not exposed in the lower compartment of the tunnel. It was necessary to start the last two wells in the rock sub-adjacent to the coal and drill at an angle up to the coal. Once the coal contact was made, it was necessary to stop the upward trajectory of the hole and maintain contact in the coal. Vertical control of the drill string was limited and time consuming but none-the-less possible. In this particular circumstance, the available drilling time was governed by boring advance rates, i.e., the amount of time consumed in boring 240 feet. If such time constraints were not imposed, more experimental techniques could be tried and developed.

The experience gained in degasification drilling through the duration of this project had demonstrated that successful completion of a well depended on starting the well in the coal seam and controlling the direction of the well. If the coal seam was not exposed, the preparatory work involved in collaring the well had to be done in rock, rather than in coal. The degas drill is designed to drill in coal formations as are the drill rods and bit assemblies. Drilling the pilot hole and subsequent reaming poses an extreme hardship on the drill and results in excessive breakage of drilling equipment. Recessing the wellhead in rock rather than coal also becomes a time-consuming operation. If the well can be collared in the coal seam, the direction controlled and the completed well put in service immediately upon completion, degas drilling can effect a very favorable reduction of methane liberation.

4.5.4 ADDITIONAL DEGASIFICATION ACTIVITY

On May 31, 1979, a clay vein was intersected by the boring machine. Historically unstable roof conditions and high methane concentrations are associated with clay veins in this area of the Pittsburgh seam. After boring some 20 feet past the intersect with the clay vein, methane concentrations detected by the TBM sensor system were such that the face power supply was automatically interrupted. Encountered concentrations were found in the 5 to 10 percent range. On June 4, 1979, a withdrawal order was issued by MSHA on the face area of the tunnel for excess methane concentrations.

At this time it was assumed that the methane was being liberated by the super-adjacent "rider coal" seam. Ventilation controls were intensified by the addition of 2 auxiliary fans positioned at the skid unit with tubes extending through the boring machine to the face. In the upper compartment face area, "air movers" operating from the compressed air supply were directed at localized hot spots. On June 7, 1979, the withdrawal order was lifted and boring resumed. An advance of 3 feet was attained with intermittent episodes of gassing off before EACC management voluntarily decided to stop the boring operation as a safety precaution.

After much experimentation with different configurations of ventilation devices, the face concentrations still could not be reduced to acceptable levels. Again, on June 13, 1979, MSHA issued a withdrawal order on the tunnel face area. Management, in concert with the enforcement agencies and the USBM, decided to try two approaches to reduce the methane liberation in the face area.

In the face area, it was decided to inject the roof with a resin in an effort to seal off the cracks transmitting the methane. Short holes were drilled into target areas to accommodate mechanical packers. The injection equipment was connected to the packers and quicksetting resin grout was pumped into the numerous cracks in the roof. The resin, designed for roof control, did not effectively seal off the gas. In addition, test holes were drilled outby the face at various locations to pinpoint the location of the gas body. From these test holes it was determined that the gas was, in fact, in advance of the face and, in all likelihood, had been confined to that area by the impermeability of the clay vein.

Concurrently with the injection effort, a USBM, PMSRC-directed degasification drilling program was initiated at Shriver shaft ahead of the tunnel. Three holes were projected to be drilled from the shaft toward the tunnel. The first attempt was made in the rider coal above the main seam because of the earlier theories developed by first-hand experience in the tunnel. Surprisingly, this well was a "dry hole" and was terminated at a depth of 77 feet.

The drilling equipment was repositioned and the projected well on the right side of the tunnel in the main seam was drilled to a depth of 200 feet. The flow from this well was measured and found to be in the range of 30 cfm, which is considered normal. The equipment was again repositioned and the remaining well was drilled on the left side of the tunnel in the main seam. The flow from the left well was measured and found to be in the 18 cfm range which is also considered normal due to its nearly parallel path to the face cleats of the coal formation (see Figure 16).

While face liberation had dropped significantly, concentrations in the vicinity of the boring machine were still not within acceptable levels. At this time, the drilling equipment was moved back into the tunnel to continue the degas drilling. Holes were drilled to the right and left in the main seam at locations just outby the secondary unit. When the flows were measured it was evident that the main seam at this location had effectively been degassed by the tunnel itself.

Excessive liberation was continuing from cracks in the tunnel roof. The next attempt was to intercept these cracks with a drill hole. The first hole was directed in the roof, at a very slight angle to the longitudinal axis of the tunnel, for a distance of 65 feet with a yield of less than 1 cfm. The drill was repositioned at a greater angle to complete a hole higher in the stratigraphic column and success was finally attained. The first hole was drilled to a depth of 27 feet, followed by a second hole to a depth of 20 feet, when liberation in the face area subsided almost totally. Flows of 70 cfm were measured from the two holes collectively (see Figure 15).

A plan for collecting and diffusing the gas through a manifold and 3-inch pipeline system to a point outby in the return was submitted to MSHA for approval. Upon approval, the plan was quickly implemented and on August 2, 1979, the withdrawal order was lifted and boring resumed.

As further assurance against a methane problem, the drilling equipment was moved back to Shriver shaft for drilling another hole directly into the now identified producing horizon. After preparatory work was completed, a hole was drilled in the rider coal seam that occurs stratigraphically above the rider coal. Problems developed with the drill string and the hole was abandoned at a depth of 35 feet. However, at this point the yield from the well was 96 cfm with a shut-in pressure of 18 psi. Subsequent flow measurements have shown no significant decrease in flow from this particular well.

Boring from this point to completion was accomplished with no interruptions whatsoever due to methane liberation. Once the additional degas drilling was completed, no appreciable concentration of methane could be found in the face area. Without the additional degas effort, boring of the final 125 feet would not have been possible.

4.6 CONVEYORS

Numerous conveyors were required for the complete tunnel boring system. The muck handling conveyors included the primary, secondary, tertiary and loading point belts. A belt was installed under the California switch to receive shotcrete ingredients from the material cars. Where it emerged from the inby end of the California switch, it rose to an incline and discharged onto a short transverse belt in order to charge the appropriate bin in the batch car. The batch car had compartments for fine aggregate, coarse aggregate and cement. A horizontal belt under the car dumped onto an incline belt which fed the shotcrete mixing auger. The auger was also inclined to mix the materials, add water and convey the mix to charge the shotcrete pumping machine.

Several problems were experienced with the muck conveyors. Boring started with a temporary secondary conveyor which had to be replaced with permanent secondary and telescopic tertiary. When the limit of the telescopic travel was reached, the tertiary conveyor had to be manually reset. This involved moving the loading point hopper and conveyor and drawing the tertiary forward into the secondary.

Oversize pieces of muck would hang up on the conveyors and hoppers. A cutter was torn from the machine head and was conveyed through three conveyor systems and out of the mine. Its ultimate destination was the preparation plant where it caused considerable damage to the shaft in the crusher. As a result, the plant was down for several days for repairs. The condition was remedied by installing a grizzly bar screen at the head of the primary belt to prevent large objects from entering the conveyors.

Other maintenance items included belt repairs, bearing replacements and transmission/speed reducer change-outs. Initially, shaft breakage caused numerous delays but was overcome by changing to larger diameter shafts for drive pulleys.

Belt slippage was a problem corrected by installing rubber lagging on the drive pulleys. Winged, self-cleaning tail pulleys were replaced with rubber lagged ones. Muck spillage required considerable hand mucking from under the conveyors, especially when the muck was exceptionally wet.

The shotcrete material belt under the California switch caused problems at the transition curve from horizontal to incline. It tended to lift off the idlers and whip. Correction of this condition required the use of hold down training idlers.

The first shotcrete batch car, mixing auger and shotcrete machine presented such insurmountable problems that the entire system was replaced with redesigned equipment. The second batch car was initially designed for the horizontal belt to charge directly into the mixing auger. This proved to be very unsatisfactory as lack of headroom caused charging to be erratic. The solution to this situation included adding an incline belt to the batch car. The incline belt discharged into a hopper between the incline and the auger, allowing a surge capacity and choke feeding. The auger required cleaning after each use. As the auger wore out due to abrasion, it was replaced with one having hard-facing applied to the flights.

4.7 CUTTERS

4.7.1 GENERAL

The Calweld Tunnel Boring Machine Model No. E-14170-001, Serial No. TM-64, was originally constructed as a 17' - 0" diameter machine. The requirement of this project dictated the use of an 18' - 0" diameter machine.

Consequently, the design of the machine was modified and the gripper system and cutter head was enlarged to accommodate an 18' - 0" diameter tunnel. This work was carried out in the offices and shops of McGuire Corporation at Fort Smith, Arkansas.

With the machine thrust being approximately 4,600 pounds per diameter inch, a 3-inch cutting kerf spacing was selected. This resulted in dressing the machine head with 36 disc cutters; eight of these being gage cutters, and one conical center cutter. (see Figure 7). The load per cutter is approximately 27,000 pounds.

A canvass of various cutter manufacturing concerns was made for the purpose of determining suitability and availability of cutters on the market. After careful consideration and evaluation of the data supplied by the manufacturers, the decision was made to utilize Kennametal KDS-15 steel disc cutters with a 15-1/2 inch detachable cutting blade supplied by Subterranean Tools, Incorporated, Denver, Colorado, a division of Kennametal Incorporated, Mining Tool Group, of Bedford, Pennsylvania.

As Kennametal did not manufacture a conical center cutter, Subterranean Tools supplied a Reed QK-1, 12-inch conical cutter manufactured by Reed Mining Tools, Inc., Houston, Texas, a member of the Mining Equipment Group of Baker International Corporation.

The cutters were mounted in forged Kennametal KRS15 saddles welded to the cutter head spokes. The exception to this was the center conical cutter which was bolted to a mounting plate on the machine head.

All disc cutters are located in numbered positions (see Figure 17). The conical cutter covers the center 12 inches of the bore. Position No. 1 disc cutter tracks a 15-inch radius. This radius is increased by 3-inch increments for position No. 2 through position No. 28 which is at a radius of 96 inches. Position No. 29 through position No. 36 are gage cutters mounted at various angles with respect to the machine head.

GAGE CUTTER MOUNTINGS

Position (No)	Radius (inches)	Cutter Angle (degrees)
29	96-3/4	15
30	99	25
31	101-5/8	38
32	103-7/8	48
33	105-3/4	55
34	107-1/4	67
35	108	67
36	108	67

4.7.2 DESIGN

The basic design of the disc cutters consists of an assembly of several various components (see Figure 18). The cutting edge is in the form of an annulus of triangular cross-section. It is press-fitted on the hub against a shoulder and secured with a ring in a groove on the hub. Tapered roller bearings are fitted into the hollow hub and on the cutter shaft. Locking nuts hold these bearings in position. Mechanical seals are held by retainers on each end, outboard of the bearings. A lubrication fitting is installed in one end of the shaft and connected to the bearing cavity by internal passages in the shaft.

The feature of the removable cutting blade enabled many of the disc cutters to be salvaged when the cutting edge failed or wore out by merely installing a new blade on the old hub. Another advantage of this feature was that various materials could be tried for blade manufacture without a great deal of lost time or expense.

The Kennametal KDS - 15 cutters were supplied with a cutting blade of A.I.S.I. No. 4340 alloy steel, heat treated to a hardness of 52 - 54 on the Rockwell "C" scale. A timken HH221410 bearing cup was enclosed within the hub. Mounted on the shaft was a Timken HH221442 bearing cone. Seal assemblies were Type 27, Code B02 as manufactured by Crane Packing Company, Morton Grove, Illinois.

Initially, as cutter replacements became necessary, the salvageable cutters were returned to Kennametal for rebuilding. Later, in order to expedite and control the supply of rebuilt butters, a quantity of 4340 blade material, bearings and seals were acquired by McGuire Corporation and rebuilding operations were performed at Fort Smith, Arkansas.

In a search for a more economical cutter blade, it was decided to change to cast steel rather than to flame-cut the part from rolled steel plate with its resulting drip material being wasted. The blades were cast of 8630 alloy steel by Green Country Casting Corporation, Muskogee, Oklahoma.

Due to a large backlog of other orders at Green County Castings, delivery of the 8630 blades became long and difficult. Production was switched back to the manufacture of blades from 4340 hot rolled steel plate. When the supply of the Crane Packing Company seals were exhausted, it was impossible to obtain additional ones, as Crane had ceased production of this particular type. An alteration to the hub dimension was made to accommodate a Caterpillar seal No. SK1078-G-9-76 from the Caterpillar Tractor Company, Peoria, Illinois.

A final change in cutter blade material was made to test another alloy. The material utilized in this change was Firmex Orange hot rolled plate, a trade name of International Alloy Steel Company, a division of Curtis-Noll Corporation, Cleveland, Ohio. All blades manufactured of 4340, 8630 and Firmex Orange were heat treated to 53 - 54 Rockwell "C" hardness. Table XVI lists the chemical composition limits of cutter blade material while Table XVII lists the typical mechanical properties for each blade alloy.

4.7.3 PERFORMANCE

Several modes of cutter failure were observed. Blades failed by normal wear, mushrooming of the edge, spalling, breaking in a radial direction, loosening from the hub due to failure of the retaining ring, and by developing flats. When bearing failures occurred, flats were almost certain to develop. Entire cutters were lost from the saddles and saddles could be torn off the head. On one occasion, a cutter, torn from the head, traveled through the muck handling system into the mine dumping system and into the crusher at the preparation plant where it caused severe damage and lost time to the mine.

The original Kennametal blades were heat treated to a depth of only about 1/2 inch. Once the blade was worn down to that point, the remainder would wear very rapidly. Another problem noted in the process of changing cutters was that some of the original Kennametal cutters were opposite handed on the shaft, which resulted in a momentary problem. In this instance when the shaft is interchanged, two adverse conditions result: 1) the blade tends to push on the snap ring instead of the shoulder on the opposite side, and 2) the threads on both keeper nuts roll in the wrong direction creating the tendency to unwind. Due to the nature of the pressed fit employed in assembly, the cutters were shipped to the Fort Smith shop to swap the shaft ends. This operation was necessary for approximately 17 cutters.

The blade of the cutter is machined to have an interference type fit, i.e. negative measurement. Initially, an interference of 15 thousandths was tried and found to cause cracks in the blade during assembly. Through trial and error, the interference of 8 thousandths was used with success.

Several failures of the cutter saddles were experienced. The side thrust reaction from the cutters resulted in a bending moment about the base of the saddle at the attachment to the machine head. The mode of failure was breakage of the welds, thereby tearing the saddle from the head. Instances of only half the saddle being torn off as well as the entire saddle occurred.

The original Kennametal saddles consisted of weldments comprised of steel forgings and plates. Complete forgings intended for smaller cutters had been cut in half and rewelded into a wider and deeper configuration to accommodate the larger cutters.

The standards of these forgings were welded to base plates. The base plates in turn were welded to the machine head. For gage cutters, part of the base plate and standards were machined off and steel plate wedges inserted between the head and saddle to produce the various gage angles.

These weldments proved to have weakness in the welded joints. As the failures occurred, they were replaced with saddles flame-cut and machined from a single piece of solid steel plate.

The solution arrived at to reduce failures consisted of welding 1/2" x 6" x 6" triangular steel gusset plates to each side of the standards (4 per saddle). This resulted in a greater moment-resisting capacity thereby greatly reducing failures.

The equal rating of cutter performance is given in Table XVIII. In rating cutters for equal performance, consideration must be given to the penetration rate and strata bored during the life of the cutter. Figure 19 graphs the average penetration rate in feet per hour over the life of the project from the hour meter readings listed in Table XIX. Other variables to be taken into consideration when comparing cutter performance are the tracking distance and cutter RPM as shown, by position, in Table XX.

The overall cost per lineal foot of tunnel for cutter manufacture and rebuild falls in the \$40 - \$50 per foot range. This cost would be reduced significantly by using the final design of the cutter evolved from the project exclusively and by following a regular schedule of inspections and replacements.

Monthly production rates, grouped by years, are shown in Table XXI. A graph of monthly progress is shown on Figure 20. The various elements comprising the total of tunnel activities are shown as a percentage of the available production hours per shift in Tables XXII, XXIII and XXIV.

4.8 SHOTCRETE SYSTEM

4.8.1 GENERAL

An Italian-made, Aliva 600, shotcrete machine was purchased from International Ground Support of Denver, Colorado. This machine was a dry type and was totally air operated. The materials were mixed in the auger and charged into the machine hopper. Water was mixed at the nozzle.

A second air compressor had to be acquired to meet the volume of air required by this equipment. In addition, an aftercooler was added to prevent freezing in the air motors.

A tremendous amount of downtime was experienced with this machine. Mechanical and batching difficulties were encountered. Rebound was excessive and required extensive cleanup. Leakage of material from the rotating wear plate could not be controlled; this generated vast quantities of dust and pelted personnel with particles of aggregate. Due to these problems this system was finally abandoned in favor of a wet mix system.

4.8.2 MATERIALS HANDLING

Numerous difficulties were experienced in the equipment and procedures for handling the materials used for shotcreting. As the project progressed, these difficulties were overcome when an efficient material handling system was developed.

A batch car manufactured by International Ground Support (IGS), Incorporated of Denver, Colorado was utilized. It discharged into a mixing auger also from IGS which fed the shotcrete machine (pump).

Mine cars loaded with shotcrete materials (sand, cement and coarse aggregate) were spotted in the stall area. Here a clamshell, working from the overhead crane, unloaded the materials into tote boxes for transporting and dumping into the batch car at the face area.

Problems were experienced due to insufficient compressed air volume for driving the air motors on the batch car and auger. Also, the air motors were freezing and stalling due to excessive moisture content. Another compressor was added and an aftercooler installed to solve both problems.

The batch car and auger were troublesome and would not feed and mix satisfactorily. Breakdowns were frequent and bearings failed often. This system was abandoned and a new one designed that would address the shortcomings of the initial system.

The new equipment was built by McGuire Corporation at its Fort Smith, Arkansas shop. It included hopper-bottom material cars, material conveyor and a new batch car. A new 3,000 series RPM mixing auger manufactured by B and B Fabrication and Machining, Inc. of Stanton, California was purchased.

The material handling belt was installed under the left hand siding of the California switch. Hopper cars were spotted over the belt and the shotcrete materials dumped. It was conveyed and elevated to a point in by the California switch where it loaded a transfer belt, which in turn charged the appropriate bin of the batch car.

A horizontal belt on the batch car directly fed the mixing auger. This arrangement resulted in difficult and erratic feeding, creating a non-uniform mixing of ingredients. This problem was solved by the addition of an inclined conveyor between the batch car and auger. The inclined conveyor filled a surge hopper which, in turn, choke-fed the auger, giving a uniform and consistent mix of the shotcrete material.

4.8.3 PUMPING

With the installation of the redesigned materials handling system, the shotcreting problem shifted to other segments of the system. The most outstanding and labor-intensive problem manifested itself in the shotcrete pumping equipment. With the changeover from the dry system to the improved wet system, two identical positive displacement pumping units were purchased.

At short distances, i.e., 200 feet and under, pumping the wet mixed material presented no unusual problems except for an occasional plugged line. However, working within the confines of the 250-foot boring cycle presented pumping distances of up to 350 feet by the end of each boring cycle. The pumping unit downtime and maintenance requirement varied in direct proportion of the pumping distance.

Maintenance and repair of the pumping unit became so intensive that the spare unit had to be cannibalized in order to maintain the operating unit in working order. It was very difficult to replace parts fast enough to bring the spare unit back to a complete condition.

The anticipated boring rates of 100 feet per day would never have been realized due to the inability of this particular support activity to keep pace. One of the complicating factors causing the long pumping distances was the presence of the degas drilling equipment in by the shotcreting equipment. Incorporated in the shotcrete system design is the ability to charge the batch car at the end of the California switch and use the self-tramming function to move the batch car and pumping unit closer to the application area, thus minimizing the pumping distance. Had the immediate in by area been unobstructed by the drilling equipment, shotcreting would possibly have been within the capacity of the pumping equipment.

4.8.4 EMPLACEMENT

The last step in the shotcreting operation is the process of emplacing the wet mixed material on the roof, rib and floor of the tunnel. This is accomplished by piping the mixed material, first through a flexible 3-inch hose connected to the shotcrete pump. This flexible connection is necessary due to the tramming action used to charge the various bins in the batch car. Since the batch car, mixing augers and pump are interconnected, the charging operation results in moving the entire shotcrete train.

From the flexible 3-inch diameter hose, the mixed material is pumped through sections of 3-inch diameter rigid pipe called "slick line" that are coupled together with snap lock victaulic

couplings. At the termination of the slick line, a 50-foot section of flexible emplacement hose is used. Initially, a 2-inch diameter hose was used for ease of application but was later changed in favor of a 2-1/2 inch diameter hose as an effort to reduce the load on the already overtaxed pumping system. Although this change made the task of applying the material a bit more strenuous, a reduction in the pumping machine downtime could be readily discerned.

The transition from the 3-inch slick line to the 2-1/2 inch emplacement hose is accomplished by using a reducer cone. The other end of the emplacement hose is fitted with a pneumatically supplied nozzle. Concrete pumped to the nozzle is fragmented by the expanding air and sprayed onto the exposed surfaces of the tunnel.

At one time, additive was introduced to the mix at the pump to accelerate the setting time, but its use was abandoned due to the caustic nature of the accelerated mixture. Chemical burns were frequently experienced by the shotcreting personnel and due to concern for their safety the accelerator was omitted from the mix.

The design specifications for the shotcrete liner called for a nominal 3-inch thickness. Maintaining the uniformity of the thickness was a major problem. A build-up of plus 3 inches was not uncommon in the lower one-third of the tunnel cross section due to the effects of rebound and gravity. This was of special concern when the gage of the tunnel was reduced sufficiently to prevent the forward movement of the California switch assembly. When this occurred, the tight spots had to be eliminated by using compressed air chippers which added to the non-boring time associated with moving the California switch.

More commonly though, the liner thickness was not of sufficient depth to form a competent concrete structure. It was necessary to make successive passes in order to attain enough build-up of material for the nominal thickness. Due to the excessive downtime experienced with the pumping system, it was not always possible to make successive passes without delaying the boring operation. On many occasions, boring was delayed because the shotcreting operation was behind.

Shotcreting played an important role in the ventilation of the tunnel in that the shotcrete was used to make the air seal between the longitudinal beams and the rib. When the shotcrete operation was behind, usually ventilation quantity was not adequate to proceed with boring. Since the longitudinal beams were installed in advance of shotcreting, the area behind the beams in the upper compartment was a difficult spot to cover with the concrete application. On numerous occasions, these areas received only a flash coating that resulted in air slaking and spalling at a later date, making it necessary to reapply the shotcrete with a smaller hand-fed machine.

Rebound of the coarse aggregate in the mix was a major clean-up problem in the upper compartment. The conveyor structure could be covered for protection from the shotcrete rebound, but the build-up on the top of the temporary divider was such that frequently compressed air chippers had to be used to break up the material for loading on the belts. Needless to say, this was a labor intensive task that required personnel usually assigned more productive chores.

Shotcrete played an important part in the secondary support of the longitudinal beams that ultimately support the permanent divider. The primary support for these beams comes from 3 foot resin-grouted rods used to attach the brackets that support the beams. The supplemental and primary load-carrying strength comes from the shotcrete application around and under the brackets.

In one area of the tunnel, station 48+00 to station 58+00, water was encountered seeping from the north rib. In this wet area, shotcrete application was very difficult due to the eroding effect of the water. As a consequence, the build-up of shotcrete around the brackets was less than adequate. At station 53+50, the short resin rods failed allowing the rib to roll off taking with it the brackets supporting two longitudinal beams. This failure caused 20 slabs to fall out of position on the left side, resulting in a delay of more than one week to the project. Once the problem was identified, the entire wet area was rebolted with 8-foot resin grouted rods installed directly through the beams on 10 foot centers and at closer intervals in selected problem areas.

4.9 GUIDANCE - GRADE AND ALIGNMENT

Guidance for the tunnel-boring machine was provided by a TCL laser grade instrument from AGL Corporation of Jacksonville, Arkansas. Machine roll was controlled by observation of a water tube level fastened to the bull gear case.

A laser line location was selected in the lower right-hand quadrant of the tunnel bore. This area was chosen in order to be easily visible to the mole operator. Care was exercised to avoid interference with equipment, walkways and the California switch.

At least two targets were set at intervals along the rib. The initial set up required the use of conventional surveying techniques. The laser was mounted and adjusted to shoot through the holes in the targets. Two more targets were mounted on the mole, one forward and one rearward. The laser beam passed through a hole in the rear target striking a bull's eye on the front target.

Periodically, as the tunnel progressed, the light intensity fell off due to distance and the laser had to be moved forward. This was accomplished by mounting a third rib target between the second rib target and the machine. When this target was adjusted so that the light penetrated its hole, the laser could then be moved forward of the first target and the first target removed. This leap frog procedure required no additional surveying.

Frequently, the thickness of the shotcrete liner interfered with the laser beam and had to be chipped away for clearance. Occasionally, personnel or moving equipment interfered with the beam, but the intermittent interruption had no adverse effects. Dancing of the beam due to refraction from stratification of differing air temperatures was minimal.

Due to operator negligence, the boring machine was allowed to dip below grade at approximately station 30+00. To mitigate this circumstance and allow the rigid trailing equipment to be advanced through this area, the track elevation was raised as was the permanent divider. As a consequence, the area in the upper compartment was reduced to the point that sufficient clearance for the conveyor systems was not available. In pulling the equipment through this swag, the conveyors and fire detection/suppression equipment sustained damage that ultimately delayed the project. The problem repeated itself again but to a lesser degree, causing no damage or delays.

The tunnel gradient was predicated on known elevations at the stall area and at the Shriver shaft bottom. However, when the tunnel had advanced to station 56+40, a 13-foot error was discovered in the Shriver shaft bottom elevation. The shaft bottom was actually 13 feet higher than it was originally thought to be. At this late

date, it was impossible to make the severe grade correction necessary to make a flat connection. A drift control mirror capable of deflecting the laser beam was secured and the maximum acceptable grade correction of plus 3 percent was made, resulting in a final connection error of minus 6 feet. The final centerline intersection of tunnel and shaft occurred six feet off center to the left. At this point in time, it is not known if the error was in the initial survey or in the tunnel itself.

4.11 FLEXICORE

When the required number of steel divider panels were installed to provide the necessary space for the construction activities such as shotcreting and degas drilling, a gantry was erected for the installation of the concrete deck sections termed flexicore. The purpose of the gantry was to maintain the distinct separation between intake and return while the permanent divider sections were being installed. This gantry was fastened to and pulled by the steel panels; it rolled on wheels on top of the previously installed flexicore panels. As boring proceeded, cracks appeared in some of the flexicore panels directly under the wheel loads. Several flexicore slabs failed completely and others had to be removed and replaced.

After the slab failures, the integrity of the slabs came under fire by the work force. The adequacy of the slab strength was questioned to such a degree that field tests were demonstrated by the manufacturer to representatives of the work force to reestablish confidence in the slabs.

Frequently, slabs were encountered with hairline longitudinal cracks caused by swelling of the casting tubes during curing. The work force consensus was that these fractures compromised the load carrying capacity of the slab and were responsible for the failures. Selected slabs with longitudinal cracks were loaded to 7,000 pounds without failure. Similar slabs without cracks were loaded to the same degree without failure and to the satisfaction of everyone involved.

The problem of excessive concentrated loading was due to there being only four wheels on either side of the gantry and the rigid manner in which it was fastened. The four wheels were replaced by an equalizing truck assembly of 32 wheels on either side, thereby distributing the load over 3 slabs rather than only one slab. It also allowed all wheels to bear load regardless of any irregularities encountered which was impossible with the initial design.

Man escapeways in the divider slabs were required at 80-foot intervals. A shorter slab was required at these locations. The short slab was supported on one end in the normal manner on the wall beam. The other end rested on a steel frame suspended from each adjacent slab. This required a temporary support for the short slab until the frame could be suspended from the next inby slab. Cribbing was used for the first escapeway but proved to be too time consuming. Thereafter, hydraulic roof supports were used resulting in a fast, safe and efficient operation.

5. DEMOBILIZATION

5.1 PROCEDURE

Before final intersect could be made with the shaft approach, certain precautions had to be taken to assure that the integrity of the ventilation would not be compromised by this connection with an additional airshaft. The additional airshaft, if left uncontrolled, would imbalance the entire mine ventilation system due to a redistribution of quantities and pressures.

As the shaft was being constructed, ventilation was accomplished by a positive ventilation system, i.e., blowing-type fans. These fans, situated near the shaft coping, directed fresh air to the shaft bottom through flexible vent tubing.

Before the scheduled intersect occurred, the top of the shaft was capped and sealed with the exception of an open area for the exit of return air induced by the fans. As boring progressed to intersect, communications were established with personnel stationed on the surface at Shriver shaft. At the first indication of "break through" the surface crew was alerted to stop the shaft fans and quickly seal the opening left in the cap.

This action had the effect of including the shaft in the tunnel ventilation scheme. Enough intentional intake leakage was provided at the cap to eliminate dead areas in the shaft. The intersect was scheduled for a weekend when normal mine activity was in an idle state. Measurements taken before and after the intersect indicated a change of less than 10,000 cfm had occurred as a result of leakage at the shaft cap.

The following week was used to build regulators, etc., to ready the mine ventilation system for removal of the cap and the addition of another main air intake. Due to the presence of the degasification pipeline in the upper compartment, the quantity of air flowing in that compartment had to be wasted to the return. The wells in the lower compartment were shut-in as required by MSHA because of the intake air passing over them to the mine.

The weekend following was used to make the major ventilation change by removing the cap. Ventilation teams were deployed throughout the mine to make measurements and readjustments to the system as a whole. The presence of the machine in the approach, as well as the misalignment with the approach, functioned to regulate the quantity of air intaking at the shaft to the point that very little adjustment in the mine system was necessary.

The misalignment with the shaft approach made it necessary to excavate, by hand, a larger opening which otherwise would have been bored out by the machine. Boring forward to accomplish

this would have taken out the approach substructure as well as the left side of the approach proper. To further compensate for stopping short, an air-powered tugger was mounted in the opposite approach to pull components into hoisting position during demobilization.

Once adequate area was provided around the boring wheel, disassembly was started. At this point, the work force had been reduced to two 12-man crews working two shifts per day. Each 12-man crew was subdivided into two 6-man crews, with one crew working on each end of the machine.

The primary unit was disassembled first, followed by the secondary unit. Concurrently with this activity a crew was working on the disassembly of the California switch outby. The disassembled California switch components were taken out of the mine by rail haulage. To minimize delays to the normal mine operation, components that fell into the "major move" category, i.e., larger than a mine car, were hoisted out the shaft. This allowed removal of larger items than would have otherwise been possible and hastened the demobilization process.

Following the removal of the secondary unit was the skid unit and conveyors. Once the conveyors were removed, the temporary divider panels were removed and permanent flexicore panels were installed. Finally, the track was advanced to the shaft and the remaining shotcrete work was finished. Upon completion of the shotcreting, the final components of the system were removed. These included the air compressor, bridge crane, shotcrete machine and the transformer and switch gear.

5.2 PROBLEMS

Of all the problems encountered in the demobilization activity, the misalignment with the shaft approach was of the greatest magnitude. Misalignment occurred in both the vertical and horizontal directions. In addition to the access and hoisting complications caused by the horizontal error, the vertical error created pumping problems in the tunnel. A permanent pump situated in the shaft sump was frequently activated to dispose of water collected by the shaft water rings. After the tunnel was connected with the shaft, the shaft water problem became a tunnel water problem. The vertical error resulted in the tunnel's lower invert being some 7 feet below the shaft bottom, making the shaft sump totally ineffective for water collection and pumping.

The horizontal misalignment made access to bolts in the outer band of the wheel very difficult. Had direct access been possible, these bolts could have been removed with wrenches and the band sections hoisted from position with the shaft hoist. However, due to space limitations around the wheel, these bolts had to be

cut with a flame cutting torch and driven free. After each component was detached from the machine, it had to be pulled into hoisting position in the shaft by an air-operated tugger. Essentially, the misalignment with the shaft created a double handling situation for the heavier machine components.

The hoisting arrangement at the shaft was a double installation of stiff-leg derricks coupled with 3 drum hoists with a capacity of approximately 8 tons each. These hoists were used to sink the shaft and were left in position for demobilization. The only problem encountered in the hoisting effort was the capacity to lift one major component. The main tube and stator were taken underground by rail in two pieces where the final fit was achieved by the use of dowel pins. The two pieces were then field-welded together making it necessary to remove in one piece rather than two. The combined weight of the single piece approximated the total capacity of both hoists.

The decision on the final disposition of the removed TBM system was late in coming. The lateness of this decision led to uncertainty of storage time and location. Shipping arrangements could have been made that would have optimized handling, loading and storage times.

A minor problem emerged in loading and shipping the system components. The weights of the components were estimated as closely as possible to arrive at the maximum load per truck, however, this resulted in a hit-and-miss situation. Many trucks left the site underloaded while others had to return due to overweight.

5.3 IMPROVEMENTS

The most important concern, by far, is perfect alignment with the disassembly point. This situation could be enhanced by more precise surveys between the two points. In addition, a longer pilot hole could be drilled to further assure perfect alignment. A large degree of confidence could be obtained with a 300 foot pilot hole which would also allow considerable distance to make last minute corrections if deemed necessary.

The hoisting capacity was adequate for all but one component. A better system might be employed for field-joining the shaft tube with the stator that would lend itself to disassembly and ease of hoisting.

Portable truck scales at the loading site would take the guesswork out of truck weights and would allow shipping to be optimized.

An early decision on system disposition would allow better storage and shipping plans to be formulated.

TABLE I
RQD Log
TBM Project
Federal No. 2 Mine

<u>Hole Number</u>	<u>Overall Average</u>	<u>Boring Section</u>	<u>Roof Section</u>	<u>Floor Section</u>
1	62.7	30.2	70.1	93.7
2	45.8	18.9	56.6	50.7
3	36.3	29.3	41.4	28.8
4	47.4	39.5	48.5	56.9
5	48.4	32.8	50.1	80.2
6	34.7	19.9	38.2	61.9

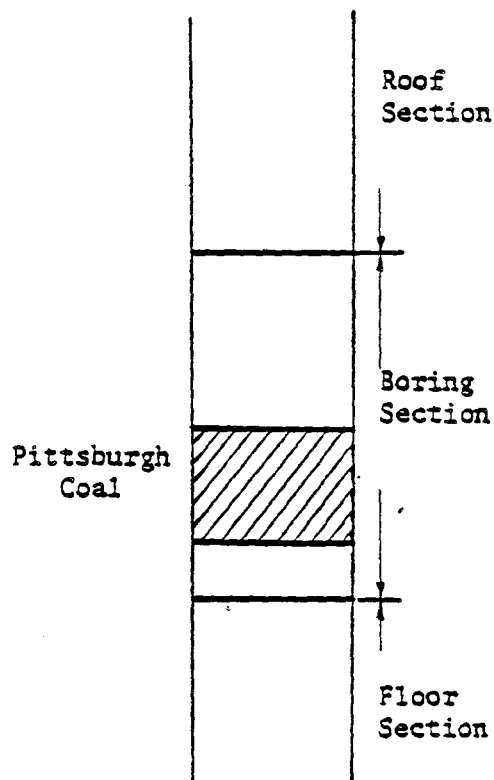


TABLE II
Rock Type Distribution (Ft.)
TBM Project
Federal No. 2 Mine

<u>Rock Type/ Hole No.</u>	<u>Coal</u>	<u>Clay</u>	<u>Limestone</u>		<u>Shale</u>							<u>Sand- stone</u>	<u>Total Length*</u>
			<u>Massive</u>	<u>Shaley</u>	<u>Coaly</u>	<u>Coaly- Sandy</u>	<u>Limey</u>	<u>Pyritic</u>	<u>Sandy</u>	<u>Sandy- Limey</u>	<u>Shale</u>		
1	8.3	2.1	1.9	12.7	6.8	4.3	25.0	1.9	4.4	1.4	3.9	-	64.4 (1016-1091)
2	8.5	0.8	1.8	15.4	4.9	-	12.6	3.8	3.6	0.4	9.5	-	61.3 (1072-1138)
3	9.5	0.3	2.0	11.2	7.8	-	14.1	4.8	-	-	9.6	-	59.3 (925-988)
4	11.3	-	9.6	6.0	1.8	1.9	14.7	7.7	5.0	0.8	3.7	2.8	65.3 (729-800)
5	1.5	1.0	8.8	23.5	4.6	-	7.6	4.0	-	-	9.3	-	60.3 (925-992)
6	7.0	2.0	1.0	6.5	1.1	1.3	9.0	1.7	-	-	10.6	0.7	40.9 (710-757)

* Numbers in parentheses are depth ranges.

TABLE III
Support Measures for Selected
Support Category*

Support Category	9	13	14	18	23	31	32	35
Method	sb (utg)	sb (utg)	B(tg) 1.5-2m	B(tg) 1-1.5m + c1m	B(tg) 1-1.5m	B(tg) 1m	B(tg) 1m	B(tg) 1m
		B(utg) 1.5-2m	+ C1m	B(utg) 1-1.5m + c1m	+ S(mr) 10-15 cm	ts(mr)5-12.5cm	+ S(mr) 40-60cm	+ S(mr) 30-100cm
of	B(utg)	B(utg) 1.5-2m	B(Sg) 1.5+2m	B(tg) 1-1.5m + S 2-3 cm	B(utg) 1-1.5m	S(mr)7.5-25cm	B(tg) 1m	CCA(sr) 60-200cm
	2.5-3m	+ S 2-3cm	+ S(mr)5-10cm	B(utg) 1-1.5m + S 2-3 cm	+ S(mr)5-10 cm	CCA 20-40cm	+ S(mr) 20-40cm	+ B(tg) 1m
Supports						+ B9tg) 1m	CCA(sr) 40-120cm	B(tg) 1m
						CCA(sr)30-50cm	+ B(tg) 1m	+ S(mr) 20-75cm
						+ B(tg) 1m		CCA(sr) 40-150cm
								+ B(tg) 1m

* Reproduced from Table IV, Page 27-30, of the report by Barton, et al 1974.

sb = spot bolting
B = systematic bolting
(utg) = untensioned, grouting
(tg) = tensioned
S = shotcrete
(mr) = mesh reinforced
c1m = chain link mesh
CCA = cast concrete arch
(sr) = steel reinforced

TABLE IV

Compressive Strength & Hardness Test
Hole No. 1
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Compressive Strength (psi)</u>	<u>Young Modulus X10⁶ psi</u>	<u>Poisson's Ratio</u>	<u>Schmidt * Hardness</u>
Shaley Limestone	SL ₁ C	26,566.80	3.82		
	SL ₂ C	16,838.52	2.96		
	SL ₃ C	25,645.60	4.14		
	SL ₄ C	16,459.00	3.29		
	SL ₅ C	19,318.40	2.29		
	SL ₆ C		overload		
	SL ₇ C	16,164.90	1.49		
	SL ₈ C	11,076.36	1.56		
	SL ₉ C	8,030.76	1.21		
	SL ₁₀ C	22,914.18	3.88		
	SL ₁₁ C	<u>23,456.35</u>	<u>3.55</u>		
	AVERAGE	18,647.08	2.82		

TABLE IV

Compressive Strength & Hardness Test (Continued)

Hole No. 1

Federal No. 2 Mine

TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Compressive Strength (psi)</u>	<u>Young Modulus X10⁶ psi</u>	<u>Poisson's Ratio</u>	<u>Schmidt * Hardness</u>
Limey Shale	LS ₁ C	6,427.91	1.04		
	LS ₂ C	14,323.08	2.44		
	LS ₃ C	14,865.16	3.70		
	LS ₄ C	4,395.14	1.04		
	LS ₅ C	3,836.90	1.71		
	LS ₇ C	16,160.60	2.86		
	LS ₈ C	15,169.60	1.36		
	LS ₉ C	<u>8,888.50</u>	<u>1.49</u>		
	AVERAGE	10,508.36	1.96		
Shale	S ₁ C	7,920.60	2.29		
	S ₂ C	<u>14,627.68</u>	<u>4.02</u>		
	AVERAGE	11,274.18	3.15		

TABLE IV

Compressive Strength & Hardness Test (Continued)
Hole No. 1
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Compressive Strength (psi)</u>	<u>Young Modulus X10⁶ psi</u>	<u>Poisson's Ratio</u>	<u>Schmidt * Hardness</u>
Sandy Shale	SS ₁ C	12,502.17	4.16		
	SS ₂ C	12,635.13	2.33		
	SS ₃ C	<u>11,681.50</u>	<u>7.47</u>		
	AVERAGE	12,272.93	4.65		
Conglomeratic Limey Shale	CLS ₁ C	4,012.20	.886		
Coaly Sandy Shale	CSS ₂ C	13,083.50	3.27		
Coal	C ₁ C	1,832.60	.224		
Dark Coaly Shale	DCS ₁ C	10,038.02	.804		

* Average of 3 readings. One on top and the other two on lateral surface.

TABLE V

Brazilian & Hardness Test
Hole No. 1
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Indirect Tensile Strength (psi)</u>	<u>Schmidt * Hardness</u>
Shaley Limestone	SL ₁ T	876.70	
	SL ₂ T	2,002.55	
	SL ₃ T	694.69	30.00
	SL ₄ T	574.70	13.00
	SL ₅ T	919.95	16.33
	SL ₆ T	492.74	16.00
	SL ₇ T	1,043.01	
	SL ₈ T	852.41	18.33
	SL ₉ T	1,189.42	
	SL ₁₀ T	923.70	
	SL ₁₁ T	1,162.40	20.00
	SL ₁₂ T	795.63	
	SL ₁₃ T	490.85	10.00
	SL ₁₄ T	<u>375.55</u>	<u> </u>
	AVERAGE	885.30	17.66
Sandy Shale	SS ₁ T	430.83	10.00
	SS ₂ T	1,486.69	15.66
	SS ₃ T	1,885.79	22.00

TABLE V

Brazilian & Hardness Test (Continued)
 Hole No. 1
 Federal No. 2 Mine
 TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Indirect Tensile Strength (psi)</u>	<u>Schmidt * Hardness</u>
Sandy Shale (Cont'd.)	SS ₄ T	877.19	13.00
	SS ₅ T	892.26	13.00
	SS ₆ T	631.60	13.00
	SS ₈ T	<u>1,064.70</u>	<u>15.66</u>
	AVERAGE	1,038.40	14.61
Clay	CL ₁ T	737.68	10.00
	CL ₂ T	<u>678.45</u>	<u>14.00</u>
	AVERAGE	708.06	12.00
Conglomeratic Limey Shale	CLS ₁ T	538.55	
Shale	S ₄ T	1,008.75	16.66

* Average of 3 readings. One on top and the other two on lateral surface.

TABLE VI
Compressive Strength & Hardness Test
Hole No. 2
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Compressive Strength (psi)</u>	<u>Young Modulus X106 psi</u>	<u>Poisson's Ratio</u>	<u>Schmidt Hardness</u>
Shaley Limestone	SL ₃ C	14,833.47	1.95		29
	SL ₄ C	6,793.40	1.94	0.14	32
	SL ₅ C	<u>20,041.93</u>	<u>4.35</u>	—	<u>31</u>
	AVERAGE	13,889.60	2.75	0.14	30.7
Liney Shale	L ₁ C	13,904.81	3.47	0.16	23
	L ₂ C	<u>9,991.57</u>	<u>3.80</u>	—	<u>25</u>
	AVERAGE	11,948.19	3.64	0.16	24
Dolomitic Shaley Limestone	DSL ₂ C	25,653.63	3.01		24
	DSL ₃ C	<u>10,192.90</u>	<u>1.51</u>	—	<u>22</u>
	AVERAGE	17,923.27	2.26		23
Coaly Shale	CS ₁ C	6,062.66	0.61		
	CS ₂ C	<u>4,472.58</u>	<u>0.75</u>		
	AVERAGE	5,267.62	0.68		
Shale	S ₄ C	12,792.26	1.89		27
Conglomeratic Liney Shale	COLS ₁ C	10,143.26	1.94	0.14	27
Coal	C ₁ C	1,269.33	0.29		
	C ₂ C	<u>2,131.09</u>	<u>0.55</u>		
	AVERAGE	1,715.21	0.42		

TABLE VII
Brazilian & Hardness Test
Hole No. 2
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Indirect Tensile Strength (psi)</u>	<u>Schmidt * Hardness</u>
Shaley Limestone	SL ₁ T	785.83	18
	SL ₂ T	865.27	16
	SL ₃ T	1,407.85	26
	SL ₄ T	1,428.03	24
	SL ₅ T	511.39	<10
	SL ₆ T	<u>846.15</u>	<u>22</u>
	AVERAGE	974.07	19
Limestone	L ₁ T	790.21	16
	L ₂ T	593.49	15
	L ₃ T	691.63	18
	L ₄ T	<u>793.38</u>	<u>12</u>
	AVERAGE	717.18	15
Dolomitic Shaley Limestone	DSL ₁ T	1,211.39	25
	DSL ₂ T	1,346.26	28
	DSL ₃ T	919.35	19
	DSL ₄ T	<u>789.97</u>	<u>22</u>
	AVERAGE	1,066.74	23.5
Conglomeratic Limey Shale	COLS ₁ T	833.77	17
	COLS ₂ T	<u>159.31</u>	—
	AVERAGE	496.54	17
Sandy Shale	SS ₁ T	788.16	<10
	SS ₅ T	196.07	10
	SS ₆ T	<u>297.80</u>	<u><10</u>
	AVERAGE	427.30	<10
Green Shale	GS ₁ T	591.67	10
Pyritic Shale	PS ₁ T	725.74	10

* Average of 3 readings. One on top and the other two on lateral surface.

TABLE VIII
Compressive Strength & Hardness Test
Hole No. 3
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Compressive Strength (psi)</u>	<u>Young Modulus X10⁶ psi</u>	<u>Poisson's Ratio</u>	<u>Schmidt * Hardness</u>
Shaley Limestone	SL ₁ C	5,370.87	3.22		18
	SL ₂ C	9,952.51	1.94	0.10	28
	SL ₃ C	8,145.25	3.89		16
	SL ₅ C	13,200.00	4.33		32
	SL ₆ C	<u>9,903.23</u>	<u>2.91</u>	—	<u>24</u>
	AVERAGE	9,314.37	3.26	0.10	23.6
Limy Shale	LS ₁ C	9,708.44	2.45	0.20	17
	LS ₂ C	17,651.71	3.06		30
	LS ₃ C	17,932.09	2.17		22
	LS ₄ C	<u>6,424.53</u>	<u>1.89</u>	—	<u>20</u>
	AVERAGE	12,929.19	2.39	0.20	22.3
Limestone	L ₁ C	10,123.00	3.03	0.22	22
	L ₂ C	<u>21,869.57</u>	<u>2.30</u>	<u>0.12</u>	<u>30</u>
	AVERAGE	15,996.29	2.67	0.17	26
Coal	C ₁ C	1,305.58	0.31	0.15	
	C ₂ C	2,097.95	0.24	0.35	
	C ₃ C	2,262.57	0.32	0.33	
	C ₆ C	<u>2,359.78</u>	<u>0.47</u>	<u>0.35</u>	
	AVERAGE	2,056.47	0.34	0.30	

* Average of 3 readings. One on top and the other two on lateral surface.

TABLE IX

Brazilian & Hardness Test
Hole No. 3
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Indirect Tensile Strength (psi)</u>	<u>Schmidt * Hardness</u>
Shaley Limestone	SL ₁ T	428.64	11
	SL ₂ T	552.62	18
	SL ₃ T	1,180.97	24
	SL ₇ T	534.99	15
	SL ₈ T	756.20	21
	SL ₉ T	<u>751.48</u>	<u>25</u>
	AVERAGE	700.82	19
Limy Shale	LS ₁ T	620.56	15
	LS ₂ T	604.24	11
	LS ₃ T	930.35	11
	LS ₄ T	1,131.75	21
	LS ₅ T	525.70	17
	LS ₆ T	529.54	18
	LS ₇ T	751.21	12
	LS ₈ T	622.18	10
	LS ₉ T	<u>570.88</u>	<u>22</u>
	AVERAGE	698.49	15.2
Limestone	L ₁ T	?	20
	L ₃ T	975.23	16
	L ₄ T	<u>1,047.14</u>	<u>20</u>
	AVERAGE	1,011.19	18.7
Pyritic Green Shale	PGS ₁ T	766.06	17
	PGS ₂ T	294.41	10
	PGS ₃ T	<u>133.06</u>	<u>12</u>
	AVERAGE	397.84	13
Coal	C ₁ T	37.13	
	C ₅ T	65.62	
	C ₇ T	<u>21.63</u>	
	AVERAGE	41.46	

TABLE X

Compressive Strength Test
Hole No. 4
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Compressive Strength (psi)</u>	<u>Young Modulus X10⁶ psi</u>	<u>Poisson's Ratio</u>
Coal	C ₃	2,424.90	0.15	
Limestone	L ₁	22,137.60	9.90	
	L ₂	<u>18,446.90</u>	<u>3.20</u>	
	AVERAGE	20,292.25	6.55	
Shale	S ₂	7,200.00	0.76	
Shaley Limestone	SL ₅	5,991.60	5.18	
	SL ₆	<u>4,097.90</u>	<u>4.64</u>	
	AVERAGE	5,044.75	4.91	
Sandstone	SS ₁	8,939.90	1.38	
	SS ₂	7,246.60	2.02	
	SS ₃	6,323.90	1.43	
	SS ₄	<u>5,578.40</u>	<u>2.11</u>	
	AVERAGE	7,022.00		

TABLE XI

Brazilian Test
Hole No. 4
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Indirect Tensile Strength (psi)</u>
Coal	C ₇	93.40
	C ₉	<u>202.90</u>
	AVERAGE	148.15
Limestone	L ₆	354.70
	L ₇	220.40
	L ₈	477.40
	L ₉	<u>284.50</u>
	AVERAGE	334.25
Limey Shale	LS ₇	645.80
	LS ₈	513.60
	LS ₉	<u>1,157.60</u>
	AVERAGE	579.25
Sandstone	SS ₈	734.46
	SS ₉	<u>284.20</u>
	AVERAGE	509.33

TABLE XII

Compressive Strength & Hardness Test
Hole No. 5
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Compressive Strength (psi)</u>	<u>Young Modulus X10⁶ psi</u>	<u>Poisson's Ratio</u>	<u>Schmidt * Hardness</u>
Shaley Limestone	SL ₁ C	11,930.00	4.72	-	-
	SL ₂ C	13,643.00	3.86	-	-
	SL ₃ C	11,048.50	2.35	-	28.66
	SL ₄ C	7,273.00	1.28	-	
	SL ₅ C	15,457.00	1.87	-	
	SL ₆ C	11,282.00	-	-	
	SL ₇ C	13,640.00	1.59	-	
	SL ₈ C	11,534.00	3.57	-	
	SL ₉ C	7,951.00	4.10	-	
	SL ₁₁ C	<u>16,636.00</u>	<u> </u>	<u> </u>	<u> </u>
	AVERAGE	12,040.00	2.91		28.66

TABLE XII

Compressive Strength & Hardness Test (Continued)
Hole No. 5
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Compressive Strength (psi)</u>	<u>Young Modulus X10⁶ psi</u>	<u>Poisson's Ratio</u>	<u>Schmidt * Hardness</u>
Limey Shale	LS ₂ C	7,665.40	1.85	-	-
	LS ₃ C	15,095.60	2.59	-	-
	LS ₄ C	5,247.50	1.40	-	-
	LS ₅ C	<u>11,893.70</u>	<u>1.73</u>	-	-
	AVERAGE	9,975.50	1.89		
Limestone	L ₁ C	31,640.00	6.24		34.60
	L ₂ C	19,737.50	7.92		
	L ₃ C	27,415.00	4.24		30.00
	L ₄ C	23,379.60	3.78		
	L ₆ C	<u>9,235.60</u>	<u>1.19</u>		
	AVERAGE	22,281.54	4.67		32.30

TABLE XII

Compressive Strength & Hardness Test (Continued)
Hole No. 5
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Compressive Strength (psi)</u>	<u>Young Modulus X10⁶ psi</u>	<u>Poisson's Ratio</u>	<u>Schmidt * Hardness</u>
Coaly Shale	CS ₁ C	2,194.50	.66		
	CS ₂ C	<u>3,265.40</u>	<u>.278</u>		
	AVERAGE	2,729.95	.469		
Pyritic Limey Shale	PLS ₁ C	9,855.00	1.59		
Pyritic Shaley Limestone	PSL ₁ C	6,875.00	2.65		
Dolomitic Shaley Limestone	DSL ₁ C	26,066.00	3.12		
	DSL ₂ C	<u>16,937.00</u>			
	AVERAGE	21,501.50	3.12		

* Average of 3 readings. One on top and the other two on lateral surface.

TABLE XIII

Brazilian & Hardness Test
Hole No. 5
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Indirect Tensile Strength (psi)</u>	<u>Schmidt * Hardness</u>
Limey Shale	LS ₁ T	1,432.42	
	LS ₂ T	764.10	
	LS ₃ T	1,062.40	
	LS ₄ T	873.13	
	LS ₅ T	1,234.67	
	LS ₆ T	1,478.47	22.00
	LS ₇ T	1,212.24	
	LS ₈ T	1,170.94	15.33
	LS ₉ T	906.46	12.00
	LS ₁₀ T	922.20	
	LS ₁₁ T	995.34	11.00
	LS ₁₂ T	<u>1,074.15</u>	<u>14.33</u>
	AVERAGE	1,093.87	14.93
Dark Coaly Shale	DCS ₁ T	1,018.57	10.00
	DCS ₄ T	<u>929.49</u>	<u>10.00</u>
	AVERAGE	974.03	10.00

TABLE XIII

Brazilian & Hardness Test (Continued)
Hole No. 5
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Indirect Tensile Strength (psi)</u>	<u>Schmidt * Hardness</u>
Coaly Sandy Shale	CSS ₁ T	1,089.08	12.00
	CSS ₂ T	930.90	9.00
	CSS ₃ T	<u>1,521.37</u>	<u>12.50</u>
	AVERAGE	1,180.45	11.16
Sandy Limey Shale	SLS ₁ T	1,478.68	10.00
	SLS ₂ T	<u>1,538.11</u>	<u>26.00</u>
	AVERAGE	1,508.39	18.00
Shaley Limestone	SL ₁ T	732.39	14.00
	SL ₂ T	1,043.82	11.00
	SL ₃ T	1,008.38	
	SL ₄ T	984.88	
	SL ₆ T	1,733.46	
	SL ₇ T	1,096.16	
	SL ₈ T	787.20	
	SL ₉ T	587.68	
	SL ₁₀ T	<u>656.21</u>	
	AVERAGE	958.90	12.50

TABLE XIII
 Brazilian & Hardness Test (Continued)
 Hole No. 5
 Federal No. 2 Mine
 TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Indirect Tensile Strength (psi)</u>	<u>Schmidt * Hardness</u>
Limey Shale	LS ₁ T	872.19	
	LS ₂ T	421.30	
	LS ₃ T	<u>1,127.65</u>	
	AVERAGE	807.04	
Limestone	L ₁ T	1,359.02	12.00
	L ₂ T	1,000.64	16.00
	L ₃ T	755.57	21.66
	L ₄ T	1,024.70	
	L ₅ T	726.00	
	L ₆ T	<u>708.30</u>	
	AVERAGE	929.00	16.55
Coaly Shale	CS ₁ T	448.11	-
	CS ₃ T	<u>638.73</u>	-
	AVERAGE	543.42	
Pyritic Shaley Limestone	PSL ₁ T	769.40	-
	PSL ₂ T	853.80	-
	PSL ₃ T	<u>419.00</u>	-
	AVERAGE	680.73	

TABLE XIII

Brazilian & Hardness Test (Continued)
Hole No. 5
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Indirect Tensile Strength (psi)</u>	<u>Schmidt * Hardness</u>
Dark Shale	DS ₁ T	839.90	
	DS ₃ T	<u>1,295.63</u>	
	AVERAGE	1,067.70	
Coal	C ₁ T	36.61	

TABLE XIV
Compressive Strength Test
Hole No. 6
Federal No. 2 Mine
TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Compressive Strength (psi)</u>	<u>Young Modulus X10⁶ psi</u>	<u>Poisson's Ratio</u>
Coal	C ₁	2,866.30	0.45	
Limey Shale	LS ₁	8,147.10	3.05	
	LS ₂	<u>6,245.60</u>	<u>2.15</u>	
	AVERAGE	7,196.35	2.60	
Sandstone	SS ₁	15,610.30	3.71	

TABLE XV
 Brazilian Test
 Hole No. 6
 Federal No. 2 Mine
 TBM Project

<u>Rock Type</u>	<u>Specimen Number</u>	<u>Indirect Tensile Strength (psi)</u>
Coal	C ₂	54.70
	C ₅	42.70
	C ₆	<u>40.20</u>
	AVERAGE	45.87
Sandstone	SS ₅	537.20
	SS ₆	<u>929.30</u>
	AVERAGE	733.25
Shaley Limestone	SL ₃	652.70
	SL ₄	<u>1,293.90</u>
	AVERAGE	973.30

TABLE XVI
Chemical Composition Limits of
Cutter Blade Material
Alloy Steel

<u>Element</u>	<u>4340</u>	<u>8630</u>	<u>Firmex Orange</u>
C	0.38/0.43	0.28/0.33	0.24/0.29
Mn	0.65/0.85	0.70/0.90	1.80/2.40
P	0.040	0.040	0.035
S	0.040	0.040	0.035
Si	0.020/0.35	0.20/0.35	0.15/0.30
Ni	1.65/2.00	0.40/0.70	0.70/1.00
Cr	0.70/0.90	0.40/0.60	
Mo	0.20/0.30	0.15/0.25	0.15/0.30
Cu			0.15/0.50
B			0.005
V			0.01/0.05
Al			0.05/0.10

TABLE XVII
 Typical Mechanical Properties of
 Cutter Blade Material
 Alloy Steel

<u>Alloy</u>	<u>4340</u>	<u>8630</u>	<u>Firmex Orange</u>
Tensile Strength (psi)	108,000	81,750	165,000
Yield Strength (psi)	68,500	54,000	135,000
Elongation (%)	22.0	29.0	18.0

TABLE XVIII
Equal Rating of Cutter Performance

<u>Position</u>	<u>Cutter No.</u>	<u>Tunnel Footage</u>	<u>Tracking Distance (Ft.)</u>
No. 1	Center	5,925	1,488,952
No. 2	2K	5,925	1,996,725
No. 3	3K	5,925	2,607,000
No. 4	4K	5,537	4,712,786
No. 4	172M	388	330,244
No. 5	5K	5,925	3,350,588
No. 6	6K	5,537	5,890,983
No. 6	188M	388	412,805
No. 7	7K	3,153	3,354,573
No. 7	95RI	2,772	2,014,149
No. 8	8K	5,537	7,069,165
No. 8	174M	388	495,365
No. 9	9K	953	778,410
No. 9	146R3	4,975	4,063,580
No. 10			
No. 10			
No. 11	11K	5,126	8,180,568
No. 11	178M	388	619,208
No. 11	128M	411	655,914
No. 12	12K	5,300	9,948,702
No. 12	181M	625	1,173,196
No. 13	13K	2,766	2,954,365
No. 13	116M	116	123,900
No. 13	122M	3,043	3,250,228
No. 14	14K	5,300	11,192,269
No. 14	158M	625	1,319,843
No. 15	15K	5,822	6,437,493
No. 15	118M	103	113,889
No. 16	16K	5,925	7,445,355
No. 17	17K	5,514	10,800,405
No. 17	114M	116	227,212

TABLE XVIII
Equal Rating of Cutter Performance (Continued)

<u>Position</u>	<u>Cutter No.</u>	<u>Tunnel Footage</u>	<u>Tracking Distance (Ft.)</u>
No. 17	123M	295	515,384
No. 18	18K	5,925	8,190,128
No. 19	19K	5,331	9,039,349
No. 19	113M	116	248,851
No. 19	119M	73	90,176
No. 19	126M	405	686,726
No. 20	20K	2,487	4,149,072
No. 20	190M	237	395,388
No. 20	126R1	1,860	3,052,931
No. 20	134M	1,180	1,840,318
No. 20	157M	161	291,338
No. 21	21K	2,494	4,113,445
No. 21	121M	659	900,066
No. 21	131R2	2,772	4,571,960
No. 22	22K	4,842	5,755,160
No. 22	36KRM	776	922,347
No. 22	102M	307	334,240
No. 23	23K	5,730	9,720,945
No. 23	164M	195	330,818
No. 24	24K	2,131	4,088,157
No. 24	35KRM	776	993,296
No. 24	156M	2,503	4,801,810
No. 24	101M	405	834,960
No. 24	106M	110	235,393
No. 25	25K	5,167	7,832,518
No. 25	116R1	80	226,308
No. 25	146R1	86	123,540
No. 25	143R2	298	397,952
No. 25	119R1	294	401,232
No. 26	26K	2,511	4,121,457
No. 26	13KRK	64	105,047

TABLE XVIII
Equal Rating of Cutter Performance (Continued)

<u>Position</u>	<u>Cutter No.</u>	<u>Tunnel Footage</u>	<u>Tracking Distance (Ft.)</u>
No. 26	99K	405	664,751
No. 26	115M	116	324,587
No. 26	124M	103	179,825
No. 26	133M	502	1,081,210
No. 26	159M	1,560	2,560,523
No. 26	141R1	39	74,644
No. 26	142R1	304	412,805
No. 26	143R1	321	418,460
No. 27	27K	725	1,411,765
No. 27	136R1	40	262,951
No. 27	161M	5,160	10,050,648
No. 28	28K	1,968	2,864,634
No. 28	6KPK	52	110,986
No. 28	14KPK	12	17,467
No. 28	14KRM	2,091	3,043,674
No. 28	34KRM	363	558,550
No. 28	182M	172	533,216
No. 28	102R1	20	28,953
No. 28	132M	502	1,153,291
No. 28	105R1	21	30,568
No. 28	131R1	19	15,683
No. 28	138R1	136	307,625
No. 28	117M	116	346,228
No. 28	187M	453	1,425,930
No. 29	29K	5,384	9,447,088
No. 29	98M	191	403,644
No. 29	106R1	350	488,750
No. 30	30K	2,728	7,734,488
No. 30	148M	26	42,298
No. 30	149M	10	17,417
No. 30	150M	19	16,173

TABLE XVIII
Equal Rating of Cutter Performance (Continued)

<u>Position</u>	<u>Cutter No.</u>	<u>Tunnel Footage</u>	<u>Tracking Distance (Ft.)</u>
No. 30	208M	237	543,659
No. 30	101R1	871	2,469,479
No. 30	139R1	103	197,807
No. 30	147R1	33	119,431
No. 30	104M	116	357,048
No. 30	111M	502	1,189,330
No. 30	155M	1,280	3,629,085
No. 31	31K	5,300	2,106,321
No. 31	184M	625	248,387
No. 32	32K	5,150	7,206,082
No. 32	105M	116	374,630
No. 32	117R1	556	777,977
No. 32	135R1	103	207,546
No. 33	33K	3,587	5,772,334
No. 33	171M	388	1,455,138
No. 33	112M	618	1,651,814
No. 33	144R1	1,332	2,143,504
No. 34	34K	66	148,249
No. 34	109M	1,066	2,394,449
No. 34	94K	867	1,947,455
No. 34	115R1	1,288	2,893,106
No. 34	124R1	960	215,635
No. 34	135R2	1,090	2,448,358
No. 34	145R2	588	1,320,766
No. 35	35K	66	149,285
No. 35	21R1	1,896	4,288,562
No. 35	173M	867	1,961,067
No. 35	125R1	1,060	2,397,614
No. 35	132R1	914	2,067,377
No. 35	117R2	1,122	2,537,852
No. 36	36K	66	Not Recorded

TABLE XVIII
Equal Rating of Cutter Performance (Continued)

<u>Position</u>	<u>Cutter No.</u>	<u>Tunnel Footage</u>	<u>Tracking Distance (Ft.)</u>
No. 36	33KRM	1,781	3,249,060
No. 36	162M	36	65,674
No. 36	168M	79	171,003
No. 36	140R1	42	99,073
No. 36	108M	618	1,686,960
No. 36	114R1	557	758,657
No. 37	105R2M	1,881	4,254,634
No. 37	163M	61	233,639
No. 37	177M	388	1,486,099
No. 37	192M	237	593,082
No. 37	160M	79	171,003
No. 37	105R2	1,881	7,204,516
No. 37	212M	242	547,380
No. 37	109R1	1,131	2,558,209
No. 37	137R1	574	1,298,331
No. 37	145R1	1,332	3,012,851

Key For Cutter Number Suffixes

K = Kennametal
 KRK = Kennametal Rebuilt by Kennametal
 KRM = Kennametal Rebuilt by McGuire
 M = McGuire
 R1 = First Rebuild
 R2 = Second Rebuild
 R3 = Third Rebuild

TABLE XIX
Hour Meter Readings

<u>1976</u>						
<u>Date</u>	<u>Station</u>	<u>H/M Reading</u>	<u>Dist.</u>	<u>Time</u>	<u>Total Dist.</u>	<u>Elapsed Time</u>
6/15	5 + 00	0.0	0			
3/28	10 + 05	89.6	505	89.6	505	89.6
6/24	14 + 96	166.8	491	77.2	996	166.8
9/8	20 + 02	269.9	506	103.1	1502	269.2
11/15	24 + 93	248.3	491	78.4	1993	348.3
<u>1977</u>						
<u>Date</u>	<u>Station</u>	<u>H/M Reading</u>	<u>Dist.</u>	<u>Time</u>	<u>Total Dist.</u>	<u>Elapsed Time</u>
5/24	30 + 00	410.0	507	61.7	2500	410.0
9/8	35 + 16	469.9	516	59.9	3016	469.9
11/22	39 + 94	536.8	478	66.9	3494	536.8
<u>1978</u>						
<u>Date</u>	<u>Station</u>	<u>H/M Reading</u>	<u>Dist.</u>	<u>Time</u>	<u>Total Dist.</u>	<u>Elapsed Time</u>
1/9	45 + 05	608.0	511	71.2	4005	608.0
2/26	49 + 94	722.3	489	114.3	4494	722.3
5/2	55 + 03	880.0	509	157.7	5003	880.0
6/27	58 + 02	978.5	299	98.5	5302	978.5

TABLE XX

<u>Cutter No.</u>	<u>Radius on Machine</u>	<u>Tracking Dist. Per Machine Revolution (Ft.)</u>	<u>Tracking Dist. @ 4 RPM of Machine (Ft.)</u>	<u>Cutter RPM @ 4 RPM of Machine</u>
Ctr. Cutter	12"	6.28	25.13	6.2
1	15"	7.85	31.42	7.7
2	18"	9.42	37.70	9.3
3	21"	11.0	44.00	10.8
4	24"	12.56	50.26	12.3
5	27"	14.14	56.55	13.9
6	30"	15.71	62.83	15.4
7	33"	17.28	69.11	17.0
8	36"	18.85	75.40	18.6
9	39"	20.42	81.68	20.1
10	42"	22.00	88.00	21.6
11	45"	23.59	94.25	23.2
12	48"	25.13	100.53	24.7
13	51"	26.70	106.81	26.3
14	54"	28.27	113.09	27.8
15	57"	29.84	119.38	29.4
16	60"	31.42	125.66	30.9
17	63"	32.98	131.94	32.5
18	66"	34.55	138.23	34.0
19	69"	36.13	144.51	35.6
20	72"	37.70	150.79	37.1
21	75"	39.27	157.08	38.7
22	78"	40.84	163.36	40.2
23	81"	42.41	169.65	41.8
24	84"	43.98	175.93	43.3
25	87"	45.55	182.21	44.9
26	90"	47.12	188.49	46.4
27	93"	48.69	194.78	48.0
28	96"	50.26	201.06	49.5

TABLE XX

<u>Cutter No.</u>	<u>Radius on Machine</u>	<u>Tracking Dist. Per Machine Revolution (Ft.)</u>	<u>Tracking Dist. @ 4 RPM of Machine(ft.)</u>	<u>Cutter RPM @ 4 RPM of Machine</u>
29	96-3/4"	50.65	202.63	49.9
30	99"	51.83	207.34	51.1
31	101-5/8"	53.21	212.84	52.4
32	103-7/8"	54.38	217.55	53.6
33	105-3/4"	55.37	221.48	54.5
34	107-1/4"	56.15	224.62	55.3
35	108"	56.55	226.19	55.7
36	108"	56.55	226.19	55.7

TABLE XXI
Monthly Production

<u>1976</u>				
	<u>Feet</u>	<u>Cars</u>	<u>Tons</u>	<u>Station</u>
January	56	78	806	00 + 56
February	46	65	663	01 + 02
March	219	307	3154	03 + 21
April	71	100	1023	03 + 92
May	35	49	504	04 + 27
June	75	105	1080	05 + 02
July	69	97	994	05 + 71
August	101	142	1455	06 + 72
September	55	77	792	07 + 27
October	0	0	0	07 + 27
November	0	0	0	07 + 27
December	0	0	0	07 + 27
<u>1977</u>				
	<u>Feet</u>	<u>Cars</u>	<u>Tons</u>	<u>Station</u>
January	42	59	605	07 + 69
February	76	107	1094	08 + 45
March	203	285	4104	10 + 48
April	21	30	302	10 + 69
May	220	308	3168	12 + 89
June	206	289	2966	14 + 95
July	234	328	3370	17 + 29
August	227	318	3269	19 + 56

TABLE XXI
Monthly Production
a

	<u>1977</u>			
	<u>Feet</u>	<u>Cars</u>	<u>Tons</u>	<u>Station</u>
September	167	234	2405	21 + 23
October	168	236	2419	22 + 91
November	378	530	5443	26 + 69
December	6	9	86	26 + 75

	<u>1978</u>			
	<u>Feet</u>	<u>Cars</u>	<u>Tons</u>	<u>Station</u>
January	0	0	0	26 + 75
February	0	0	0	26 + 75
March	54	76	778	27 + 29
April	142	199	2045	28 + 71
May	165	231	2376	30 + 36
June	163	229	2347	31 + 99
July	41	58	590	32 + 40
August	216	303	3110	34 + 56
September	87	122	1253	35 + 43
October	317	444	4565	38 + 60
November	224	314	3226	40 + 84
December	355	497	5112	44 + 39

	<u>1979</u>			
	<u>Feet</u>	<u>Cars</u>	<u>Tons</u>	<u>Station</u>
January	353	495	5083	47 + 92

TABLE XXI
Monthly Production

	<u>1979</u>			
	<u>Feet</u>	<u>Cars</u>	<u>Tons</u>	<u>Station</u>
February	258	362	3715	50 + 50
March	185	259	2664	52 + 35
April	229	321	3298	54 + 64
May	334	468	4810	57 + 98
June	9	13	130	58 + 07
July	0	0	0	58 + 07
August	118	167	1714	59 + 25

TABLE XXII
Summary of Tunnel Activities for 1977

	<u>May</u>	<u>June</u>	<u>July</u>	<u>Aug.</u>	<u>Sept.</u>	<u>Oct.</u>	<u>Nov.</u>	<u>Dec.</u>	<u>Weighted Avg. (1)</u>	<u>Total</u>
Shifts	36	51	43	27	42	40	40	8		287
Production Hrs. Avg. Per Shift									6.61	
Percent of Production Hrs.	6.3	6.4	6.6	6.7	7.1	6.8	6.5	6.3		
Boring Time	9.4	15.1	21.8	37.1	17.1	17.4	26.8	3.0	19.37	
Roof Bolting	1.0	2.3	0	.2	.1	0	1.4	0	0.76	
Hand Mucking	20.6	9.5	0	0	0	0	14.7	20.0	6.88	
Track & Pipe	.4	2.0	8.4	4.6	1.6	0	0	0	2.33	
Beam Installation	3.4	2.7	20.3	3.0	.3	1.3	4.2	0	5.04	
Shotcrete	7.0	3.0	.7	.7	0	0	2.6	0	1.94	
Flexicore	1.3	3.1	4.4	5.1	.4	0	2.3	0	2.23	
Haulage	12.2	1.3	3.9	7.3	3.1	4.2	6.4	0	4.96	
Conveyor	3.9	15.9	2.5	9.9	5.3	4.0	3.0	0	6.37	
Outside Supply	0	0	0	0	0	.2	0	0	0.03	
Move Switch	19.7	9.0	12.0	3.7	14.4	8.8	5.6	0	10.33	
Degas Drilling	0	0	0	0	0	0	0	0	0	
Mole	2.8	5.4	.2	4.0	30.2	55.1	25.6	0	17.38	
Ventilation	3.1	21.0	0	0	17.1	1.3	3.6	0	7.31	
Other	15.1	9.8	25.8	24.6	10.2	7.6	3.6	77	15.02	
Shift Travel in Hr. Per Shift	1.7	1.6	1.4	1.3	.9	1.2	1.5	1.7	1.39	
Penetration in Ft. Per Clock Hr.	8.4	5.7	5.3	4.6	4.9	3.6	8.2	1.0		
Advance (Ft. Per Month)	220	206	234	227	167	172	378	6		1610

Data was not collected prior to this time.

TABLE XXIII

Summary of Tunnel Activities for 1978

	<u>Mar.</u>	<u>Apr.</u>	<u>May</u>	<u>June</u>	<u>July</u>	<u>Aug.</u>	<u>Sept.</u>	<u>Oct.</u>	<u>Nov.</u>	<u>Dec.</u>	<u>Weighted Avg. (1)</u>	<u>Total</u>
Shifts		39	44	34	16	17	19	22	20	38		249
Production Hrs. Avg. Per Shift Percent of Production Hrs.		6.9	6.8	6.6	6.8	6.5	6.5	6.5	6.2	6.6	6.64	
Boring Time		10.1	12.8	12.4	3.5	40.9	11.3	36.2	19.7	29.5	18.70	
Roof Bolting		6.4	.3	0	2.1	2.5	1.7	2.1	0	.4	1.74	
Hand Mucking		44.7	4.9	0	.3	.6	0	0	0	0	7.93	
Track & Pipe		0	.4	.5	0	.3	0	0	0	2.8	0.59	
Beam Installation		1.2	2.7	1.8	0	1.3	.7	.7	0	.2	1.15	
Shotcrete		.7	1.6	2.2	0	0	9.9	4.2	3.1	4.7	2.79	
Flexicore		0	1.2	.6	.5	5.4	0	2.9	3.6	1.1	1.41	
Haulage		15.8	1.6	6.4	0	6.8	55.7	2.6	2.1	2.0	9.05	
Conveyor		8.3	4.2	17.5	1.8	6.5	2.5	1.9	3.5	13.9	7.75	
Outside Supply		0	0	.5	0	0	0	0	0	0	0.07	
Move Switch		0	8.5	17.3	0	1.8	0	14.0	12.8	12.6	8.13	
Degas Drilling		0	0	0	0	0	0	0	7.2	2.2	0.91	
Mile		9.6	22.4	8.5	0	12.8	16.5	2.7	6.0	2.5	9.86	
Ventilation		0	1.1	24.3	5.2	4.0	0	.4	0	3.7	4.72	
Other		3.1	38.0	17.8	86.6	16.5	1.4	32.1	41.8	24.0	25.80	
Shift Travel in Hr. Per Shift		1.1	1.2	1.3	1.2	1.5	1.5	1.5	1.7	1.3	1.32	
Penetration in Ft. Per Clock Hr.		9.3	9.3	9.2	9.1	8.5	6.7	9.1	9.5	7.5	9.55	
Advance (Ft. Per Month)	54	142	165	164	42	215	87	316	219	355	220	1759

No Records Recorded

TABLE XXIV

Summary of Tunnel Activities for 1979

	<u>Jan.</u>	<u>Feb.</u>	<u>Mar.</u>	<u>Apr.</u>	<u>May</u>	<u>June</u>	<u>July</u>	<u>Aug.</u>	<u>Sept.</u>	<u>Oct.</u>	<u>Nov.</u>	<u>Dec.</u>	<u>Weighted Avg. (1)</u>	<u>Total</u>
Shifts	44	39	37	39.43	44	22								
Production Hrs. Avg. Per Shift Percent of Production Hrs.	6.4	6.3	6.3	6.2	6.5	7.1								
Boring Time	33.9	35.5	22.0	37.1	48.2	2.7								
Roof Bolting	1.4	2.7	.2	0	.1	0								
Hand Mucking	0	.6	0	.03	.5	0								
Track & Pipe	1.5	0	0	0	0	0								
Beam Installation	.4	.2	0	.07	0	0								
Shotcrete	0	0	0	0	0	0								
Flexicore	0	1.8	.2	.04	0	0								
Haulage	1.5	.9	6.2	7.07	8.1	1.1								
Conveyor	1.7	9.4	15.2	6.3	5.6	0								
Outside Supply	0	0	0	0	0	0								
Move Switch	14.6	21.0	16.2	18.9	6.6	0								
Degas Drilling	0	3.0	0	0	0	0								
Mole	13.5	5.5	33.6	13.2	11.3	0								
Ventilation	3.7	3.9	5.2	3.5	1.3	1.9								
Other	27.8	15.5	1.2	13.1	18.3	146.76								
Shift Travel in Hr. Per Shift	1.6	1.7	1.7	1.8	1.4	.9								
Penetration in Ft. Per Clock Hr.	4.9	3.6	4.8	3.1	3.2	5.6								
Advance (Ft. Per Month)	353	256	185	229	334	9								

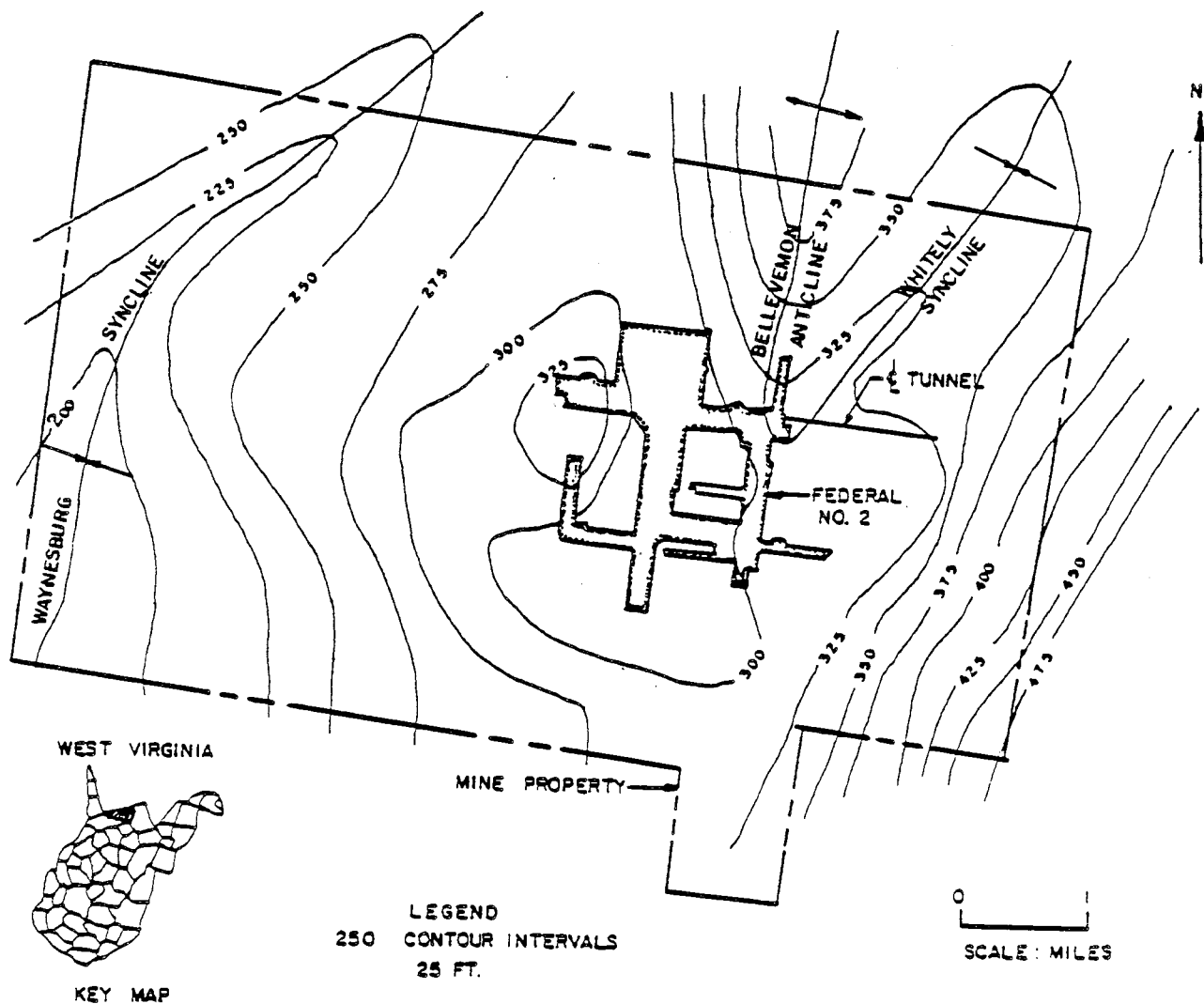
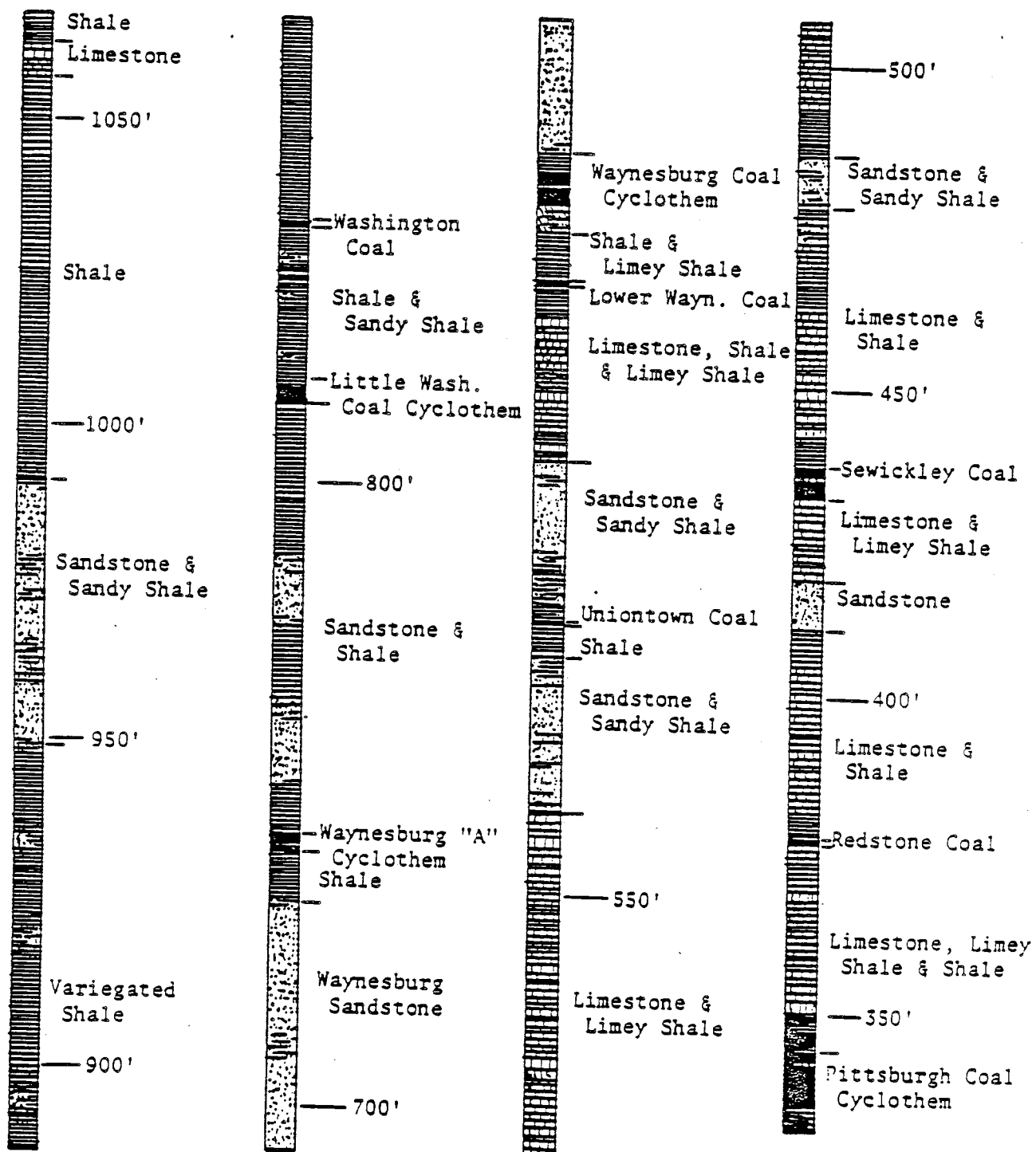


FIGURE 1

Structure Map Drawn on the Base of the Pittsburgh Coalbed
Federal No. 2 Mine



Scale: 1" = 25'

FIGURE 2

Generalized Columnar Section in Tunnel Area

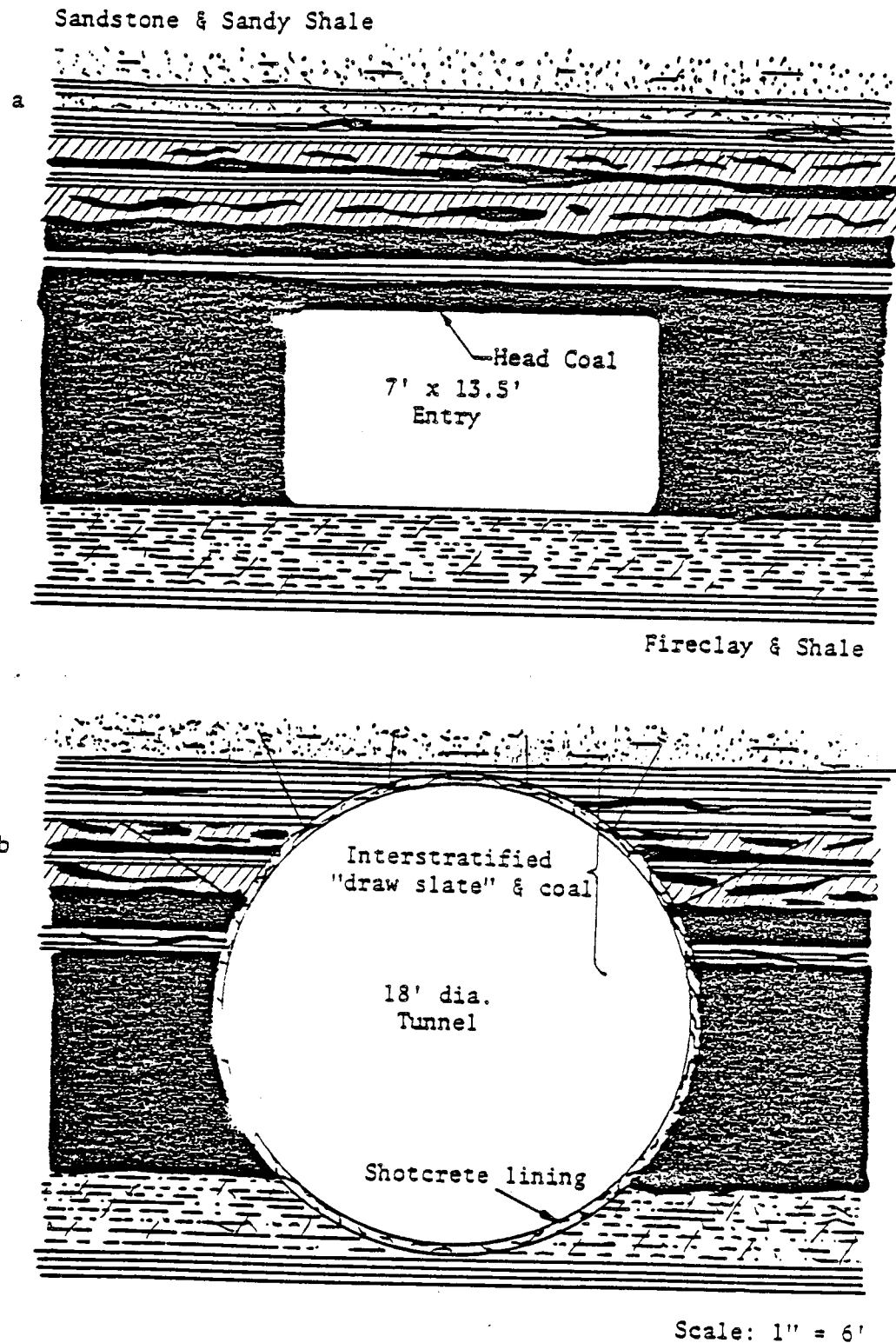


FIGURE 3

Entry and Tunnel Locations in Strata

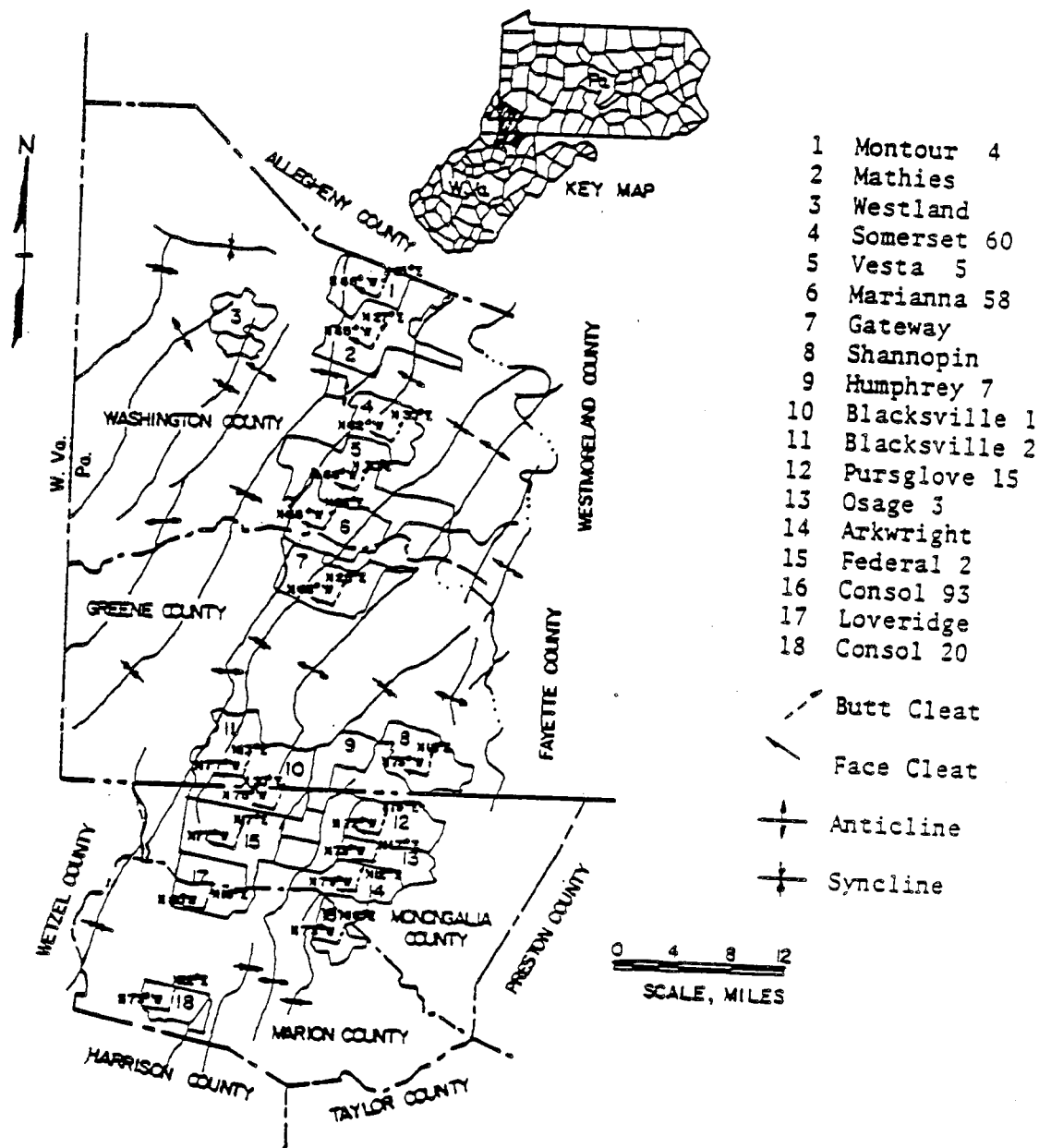


FIGURE 4
Coal Cleat Orientations in Pittsburgh Seam

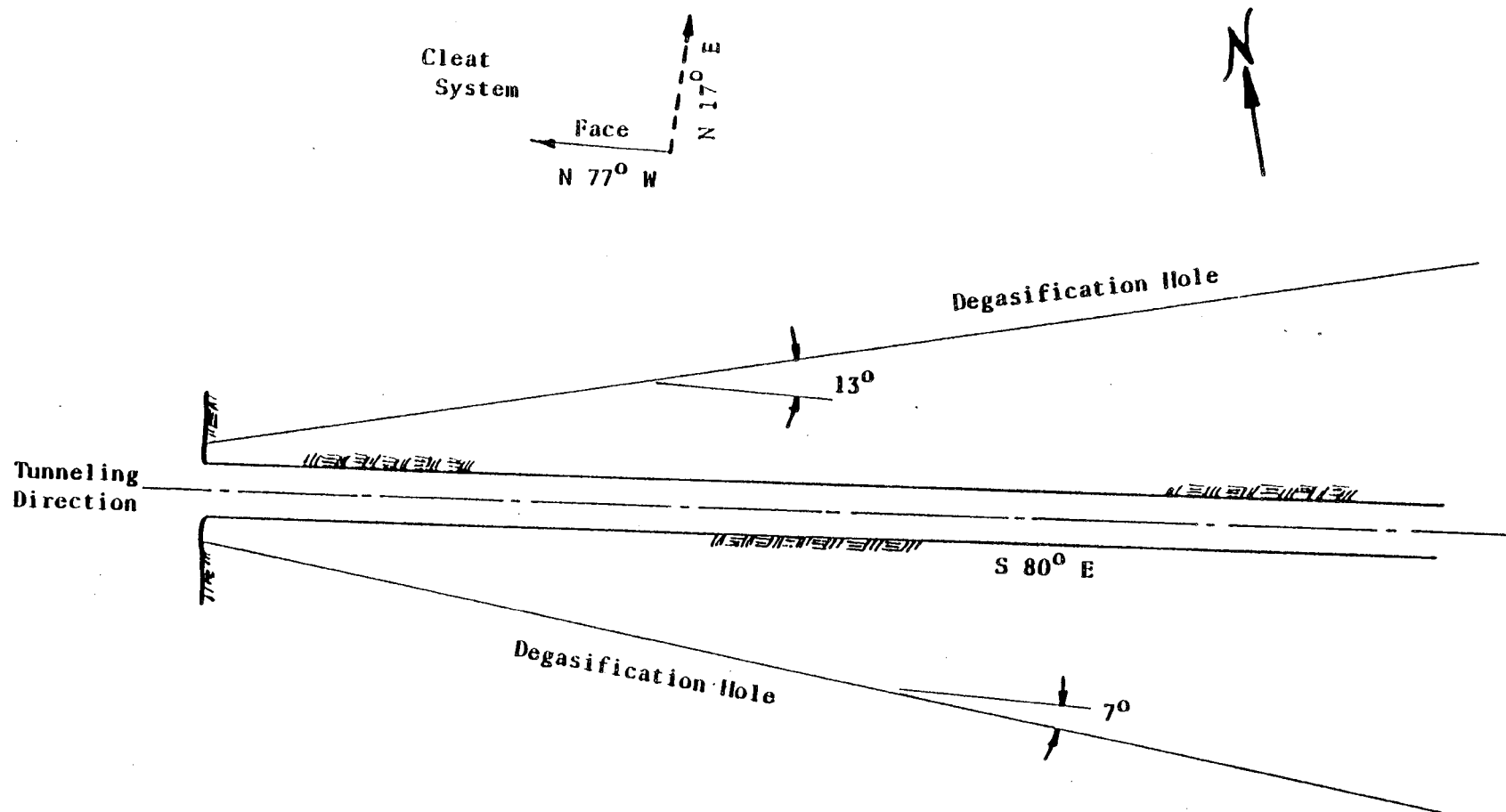


FIGURE 5
Degasification Hole Orientations With Respect to the Cleat System

TUNNEL BORING MACHINE

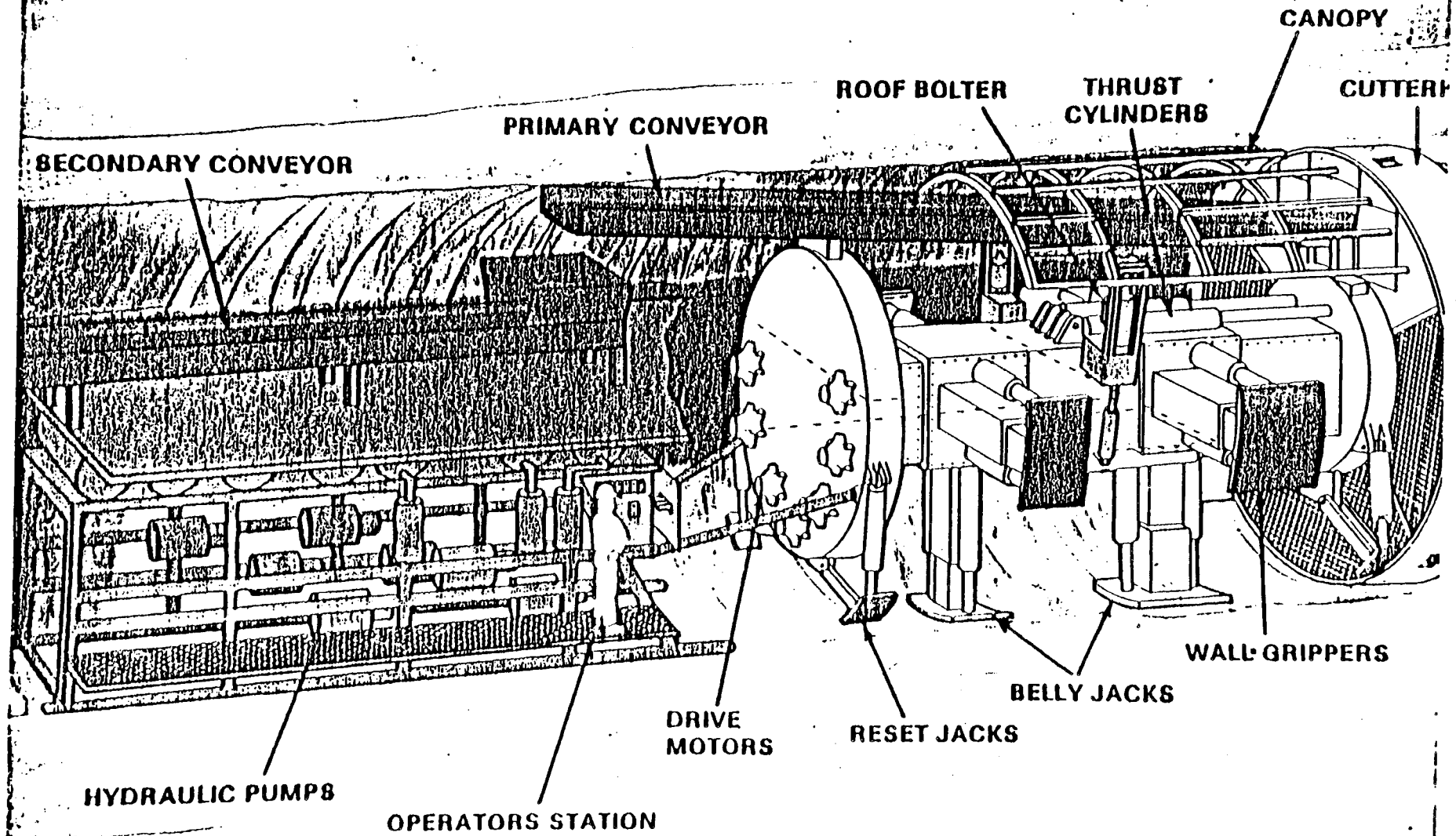


FIGURE 6A

Basic Machine

MATERIALS HANDLING SYSTEM

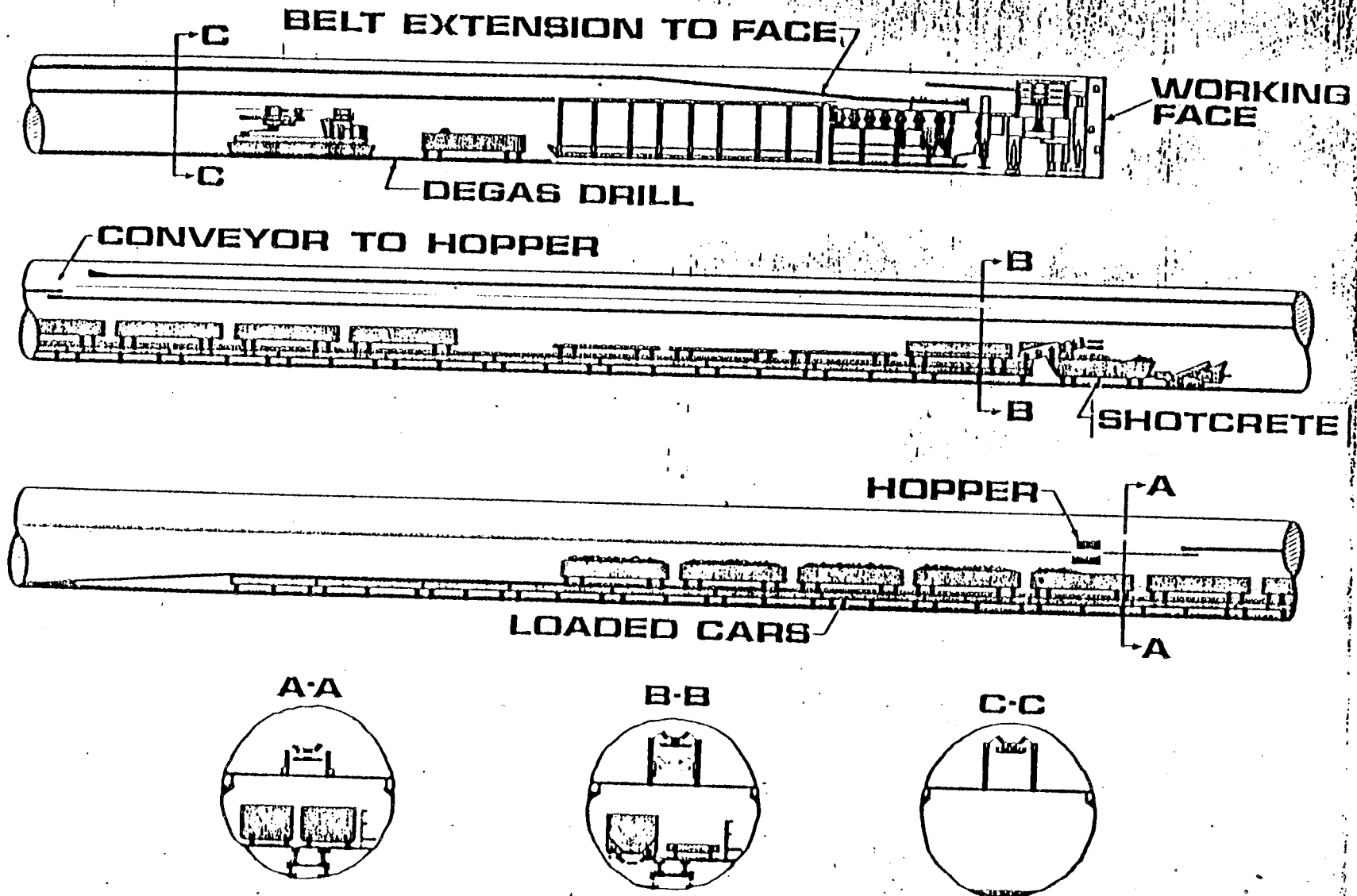


FIGURE 6B

Basic Machine

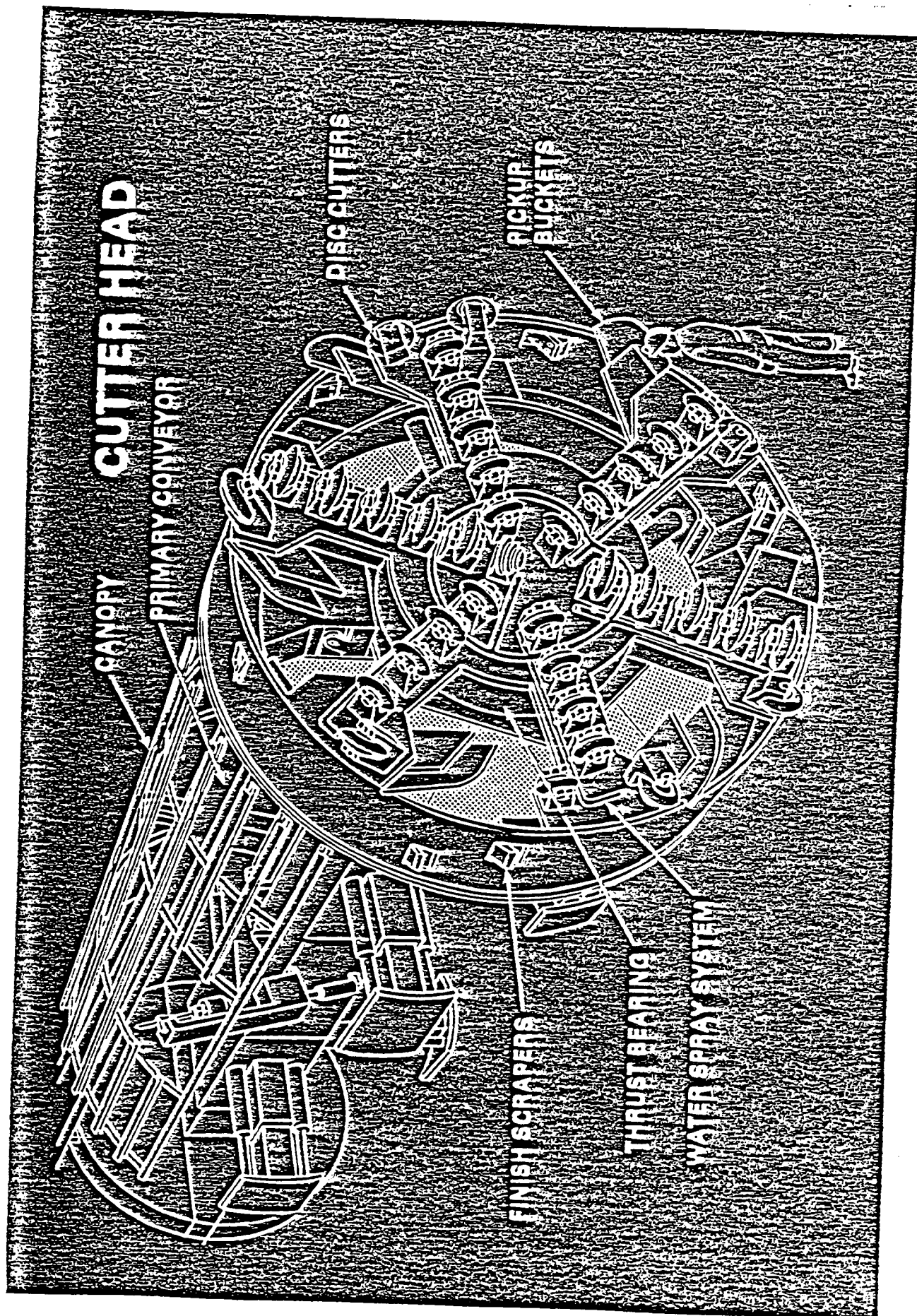


FIGURE 7

Machine Cutting Head

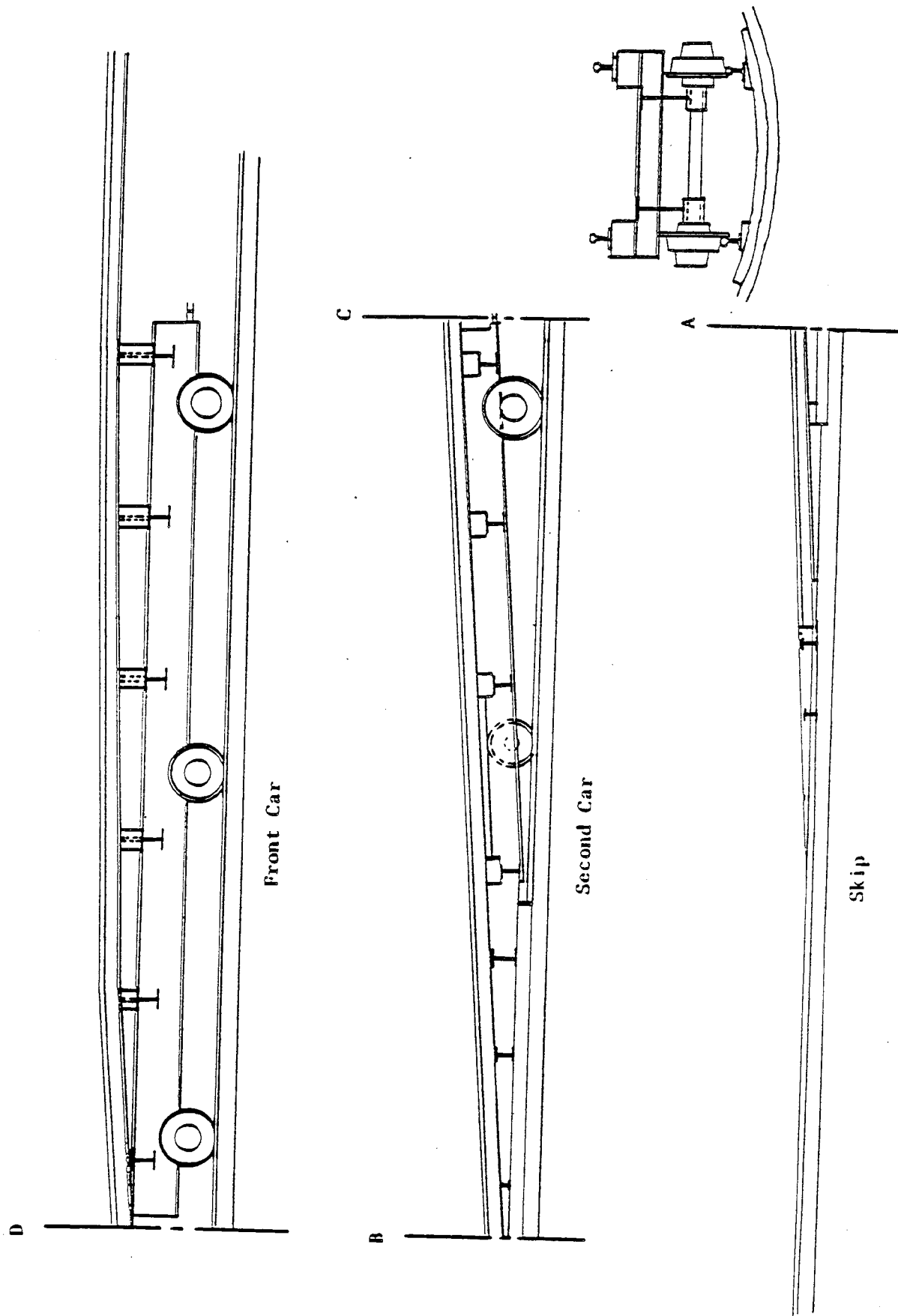


FIGURE 8A
Ramp For California Traveler

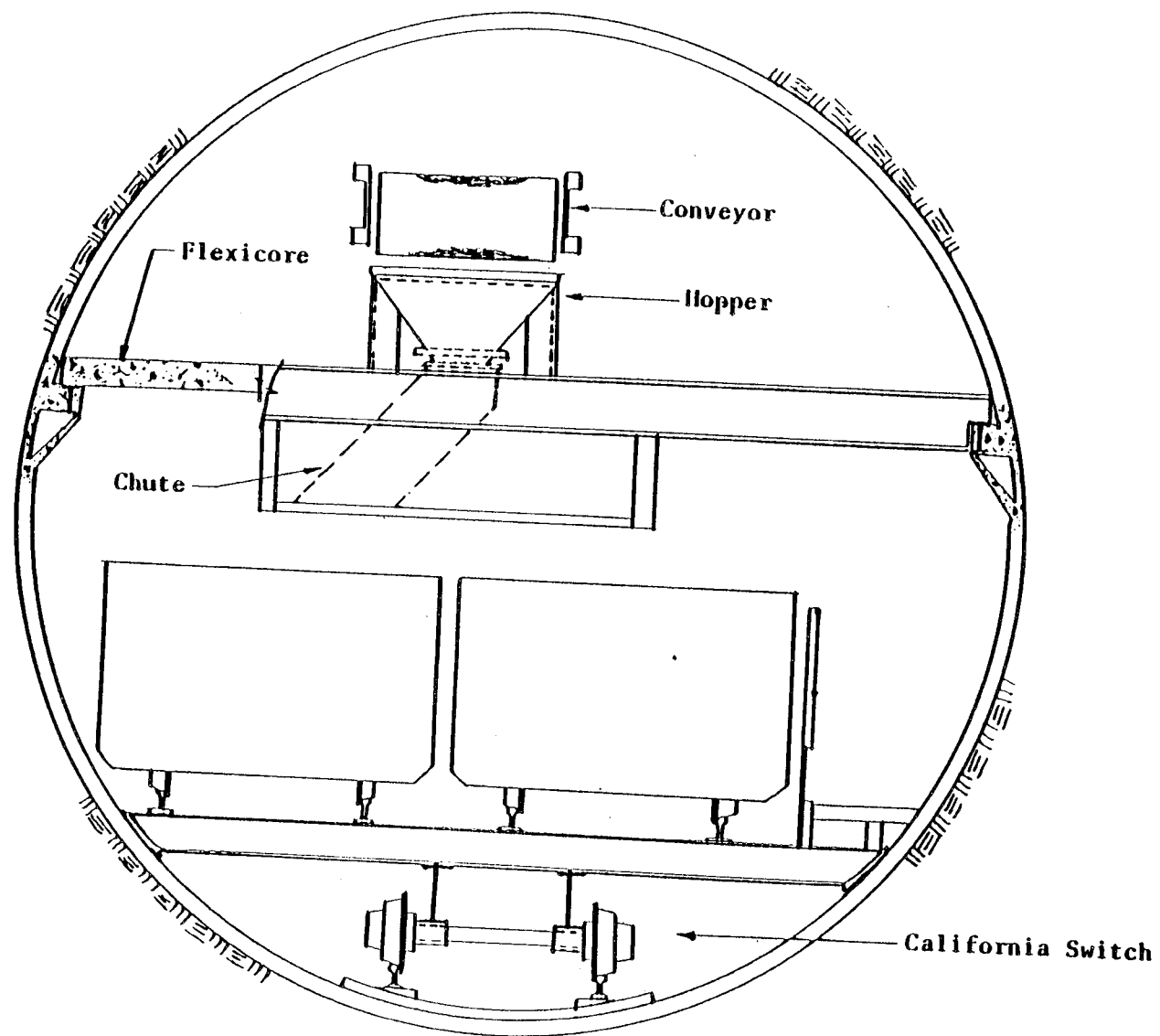


FIGURE 8B

Arrangement Hopper and California Switch

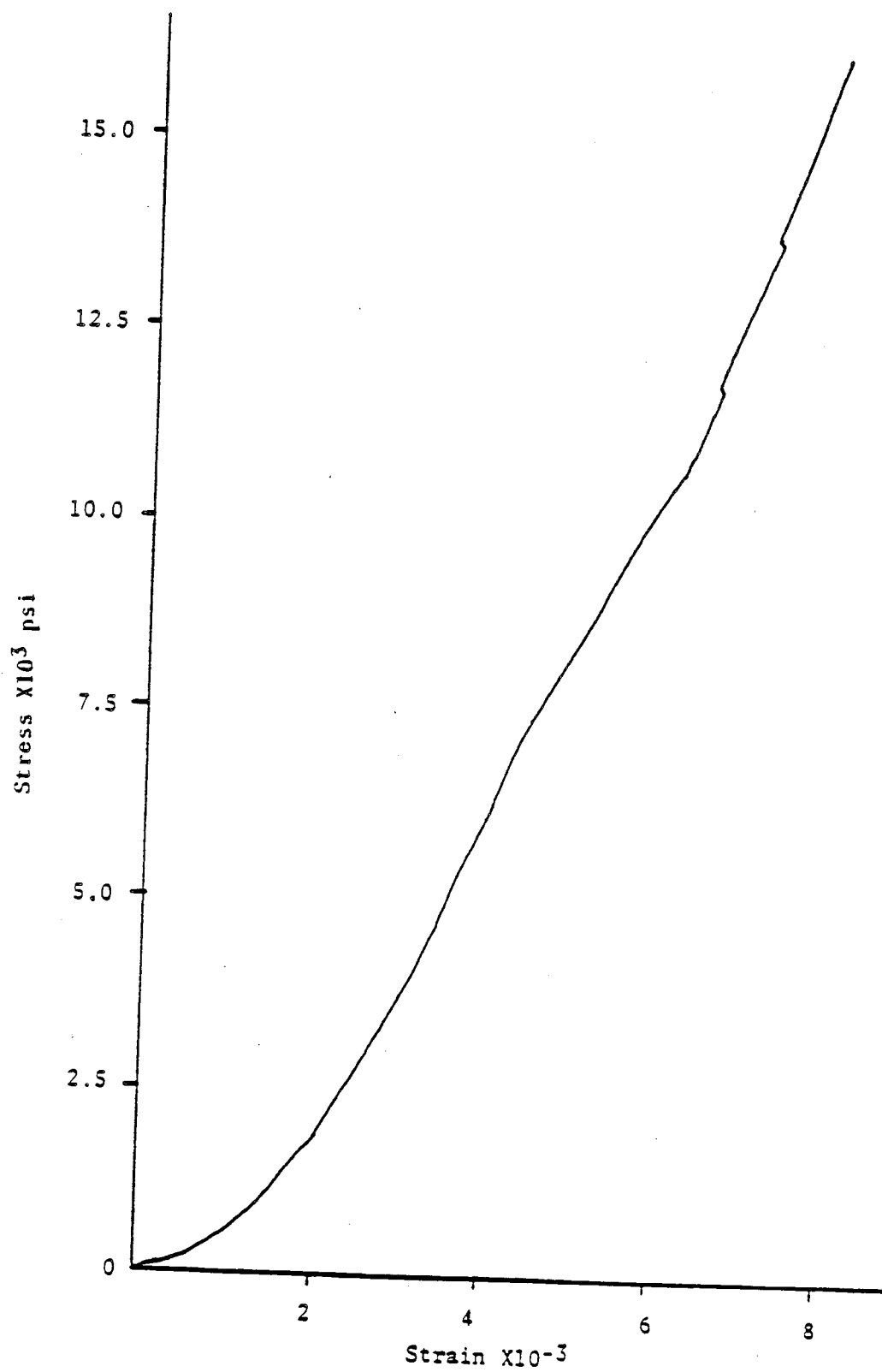


FIGURE 9

Typical Stress-Strain Curve For Shaley Limestone

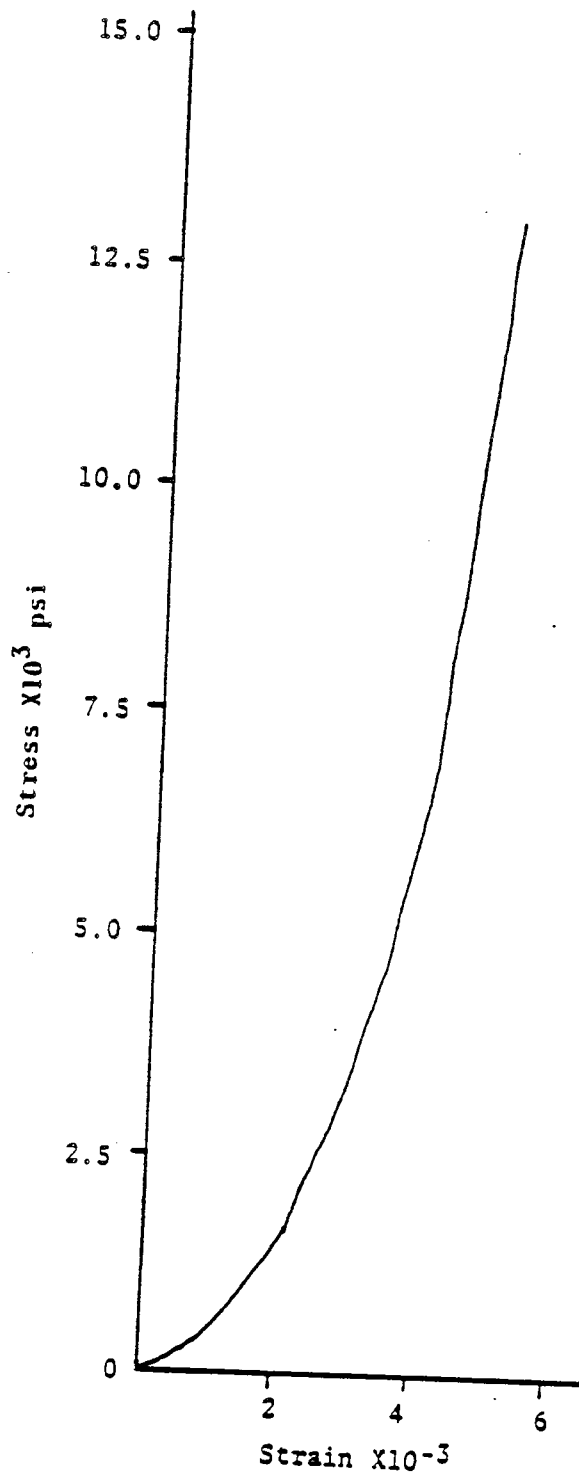


FIGURE 10
Typical Stress-Strain Curve For Coaly Sandy Shale

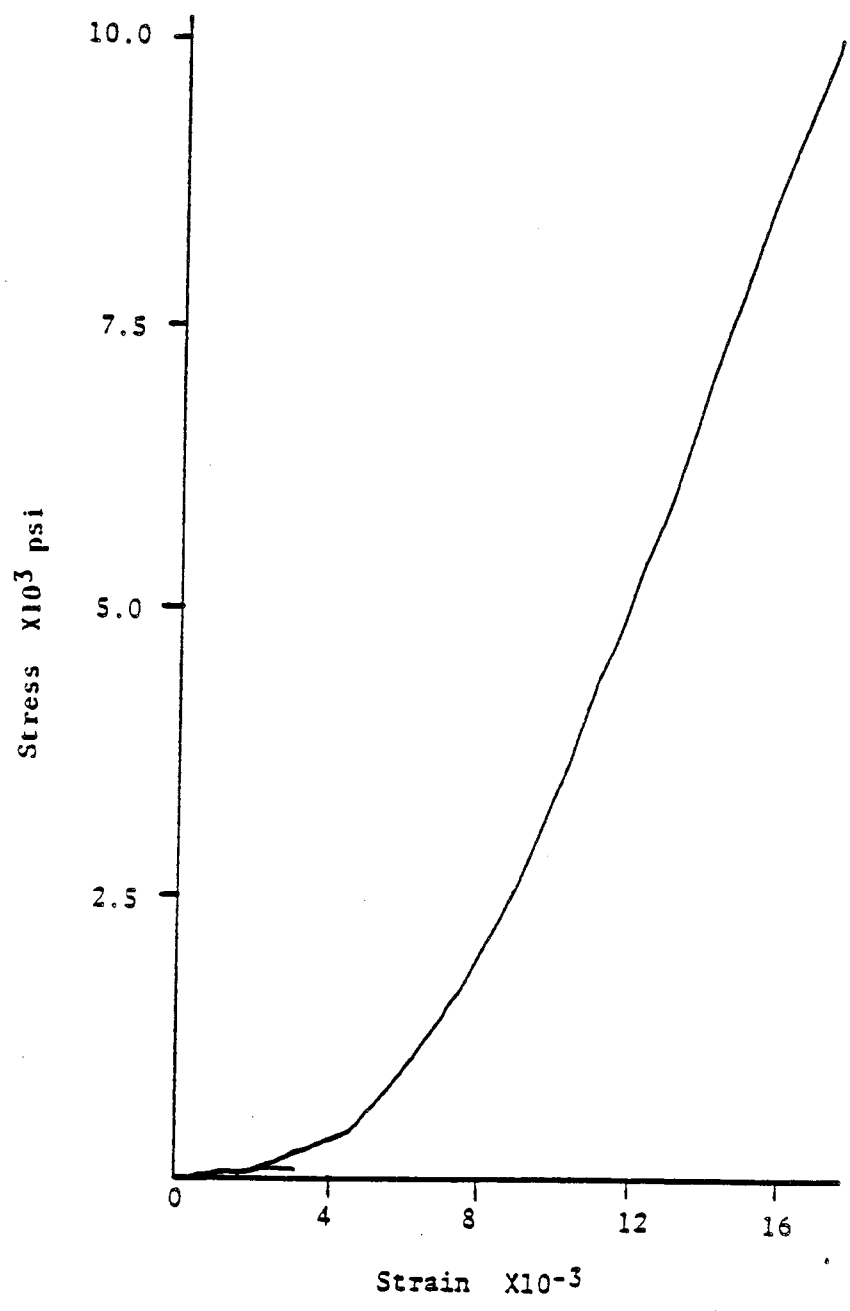


FIGURE 11
Typical Stress-Strain Curve For Dolomitic Conglomeratic Shale

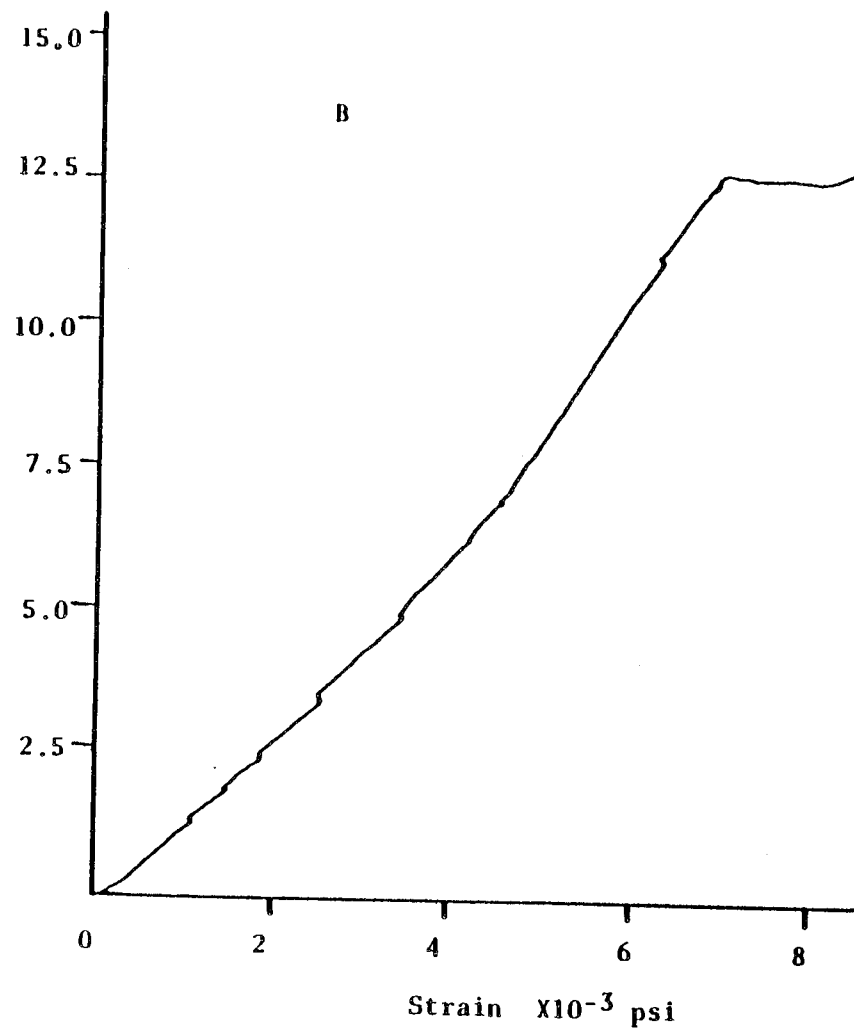
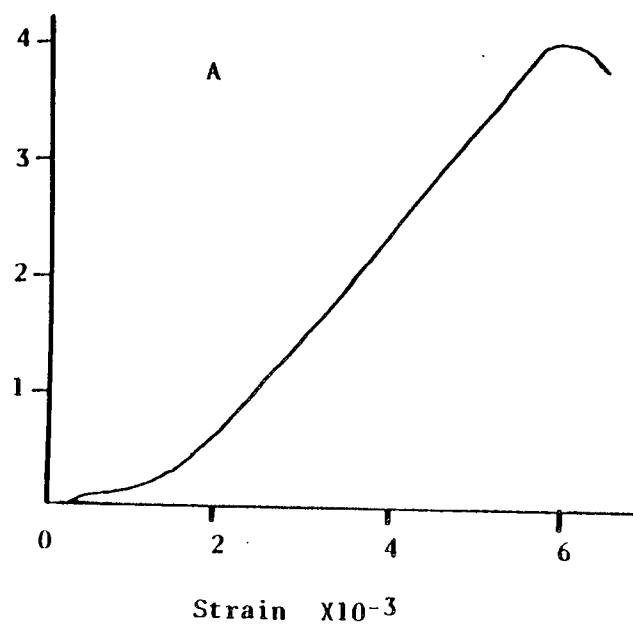


FIGURE 12
Typical Stress-Strain Curve For (A) Coaly Limey Shale and (B) Limey Shale

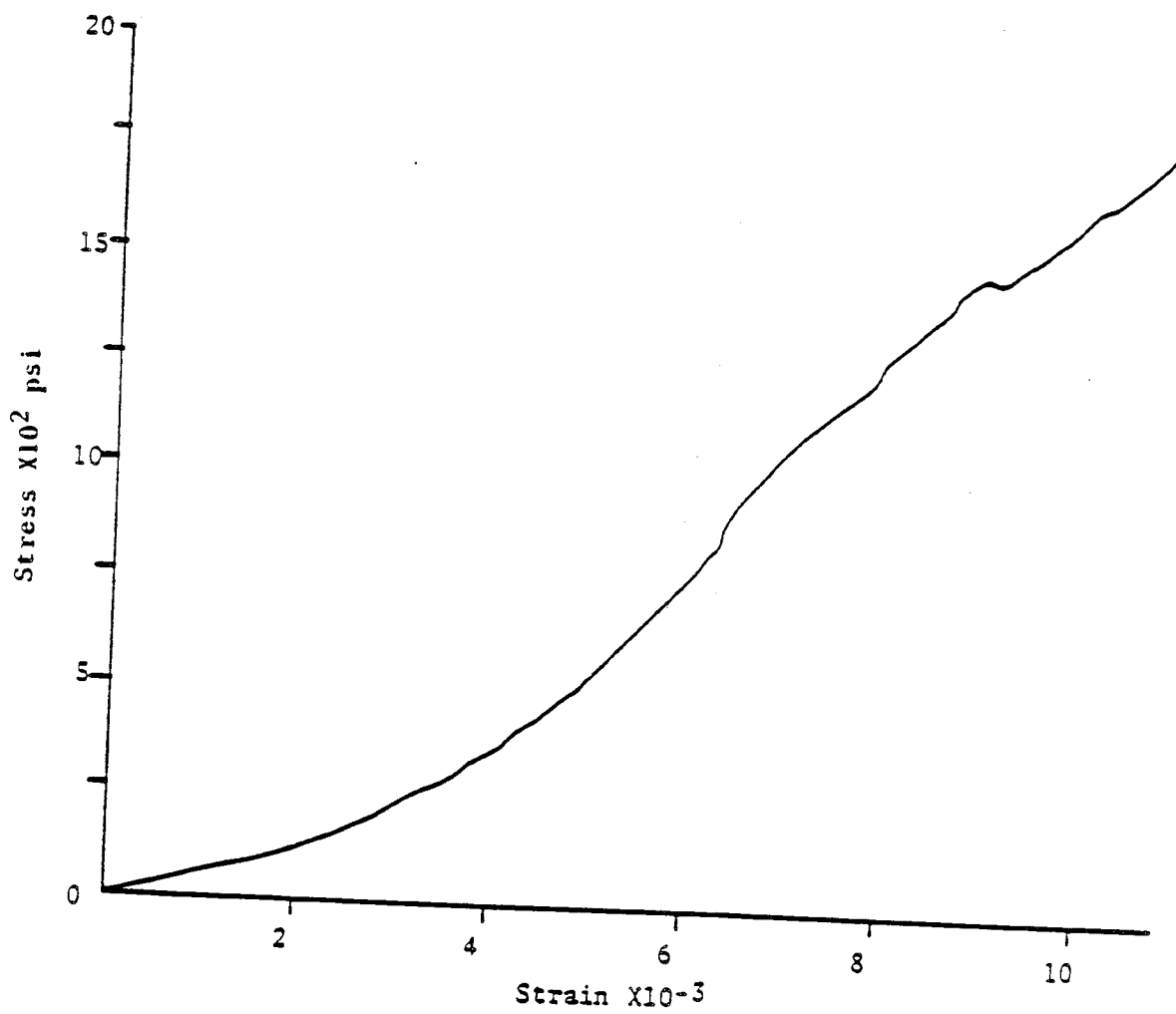


FIGURE 13

Typical Stress-Strain Curve for Coal

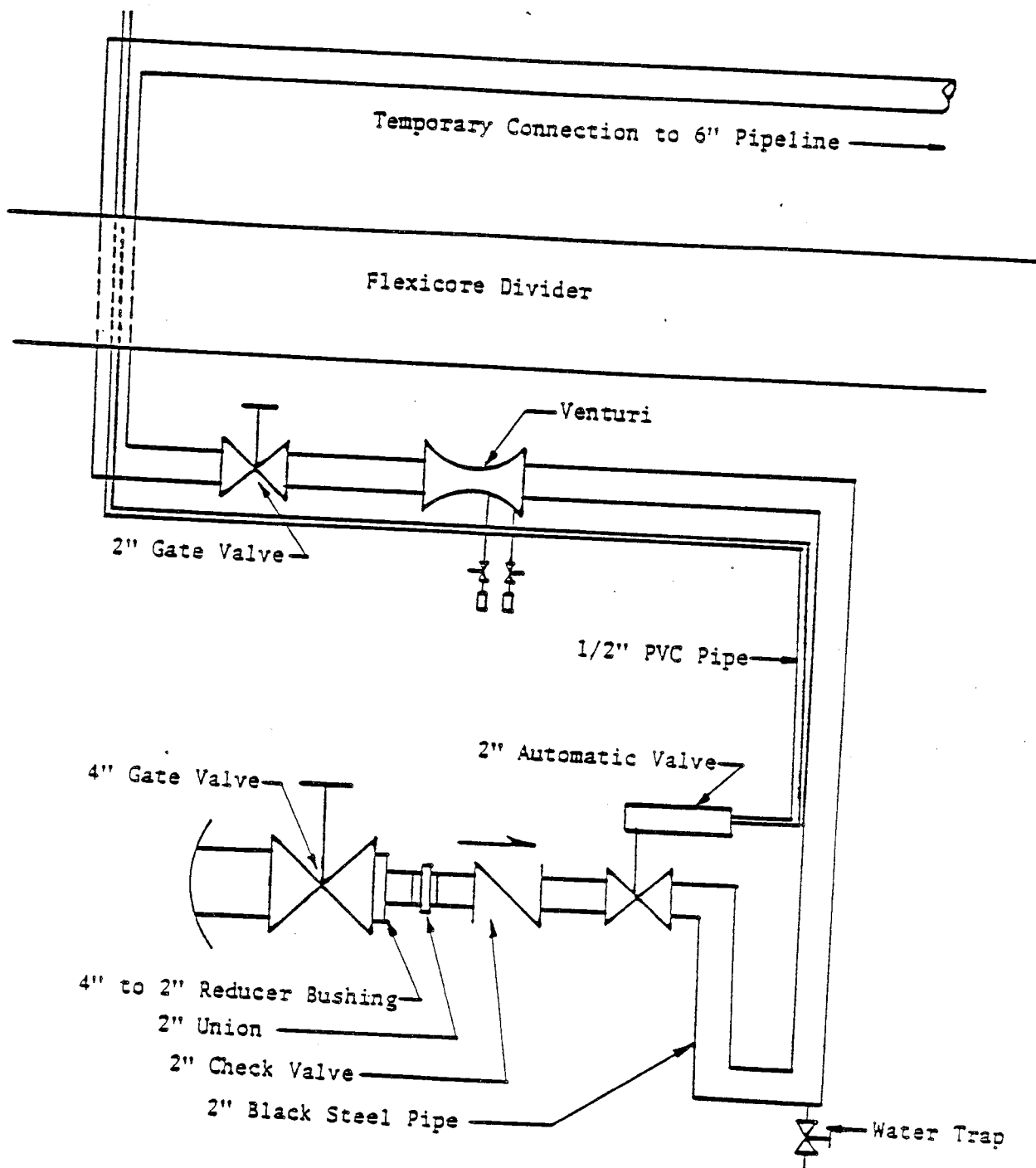


FIGURE 14
Standardized Schematic of Wellhead Plumbing

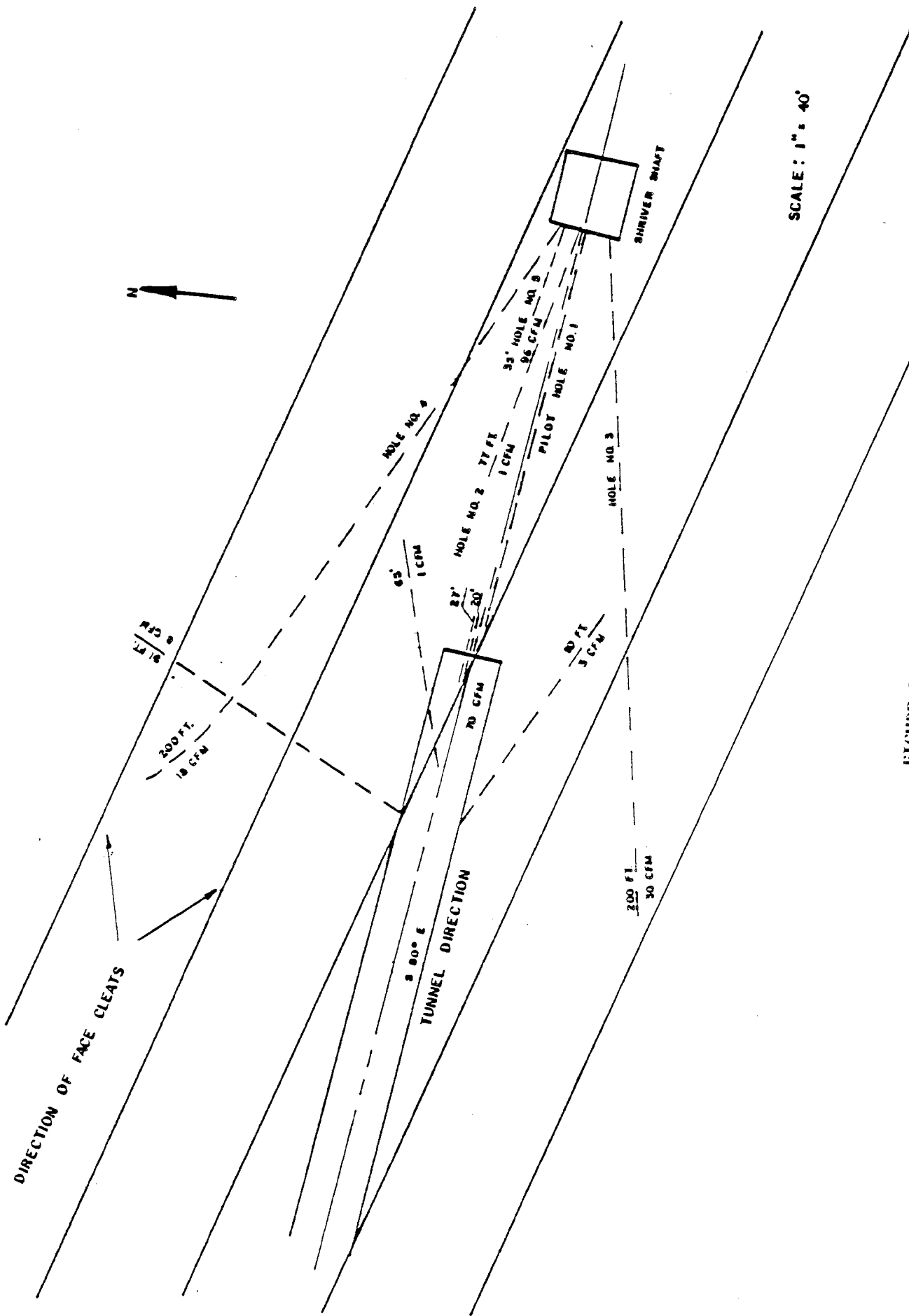
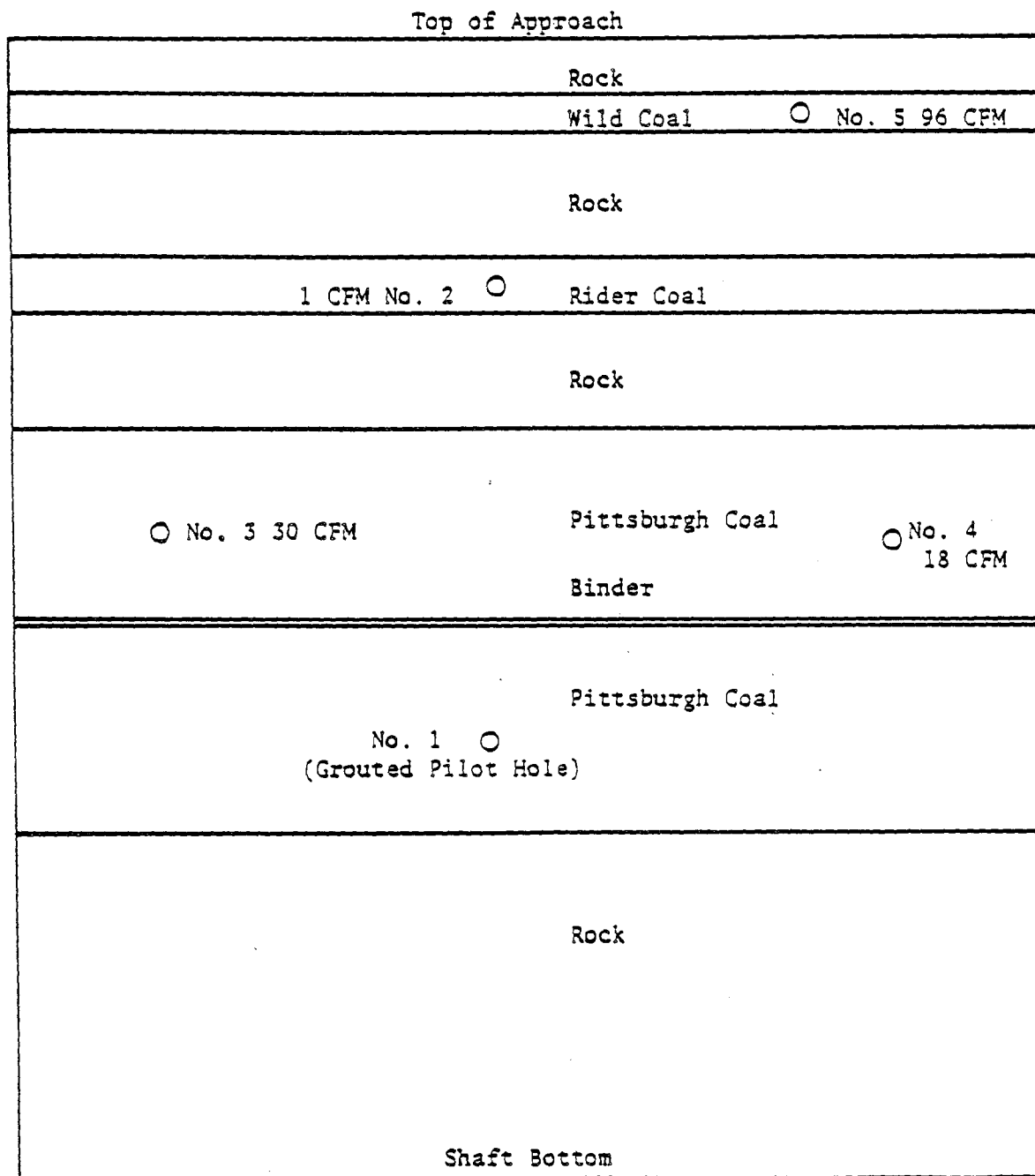


FIGURE 15

Location of Degas Holes in Shaft and Tunnel



Scale: 1" = 3'

FIGURE 16
Location of Holes in Shaft

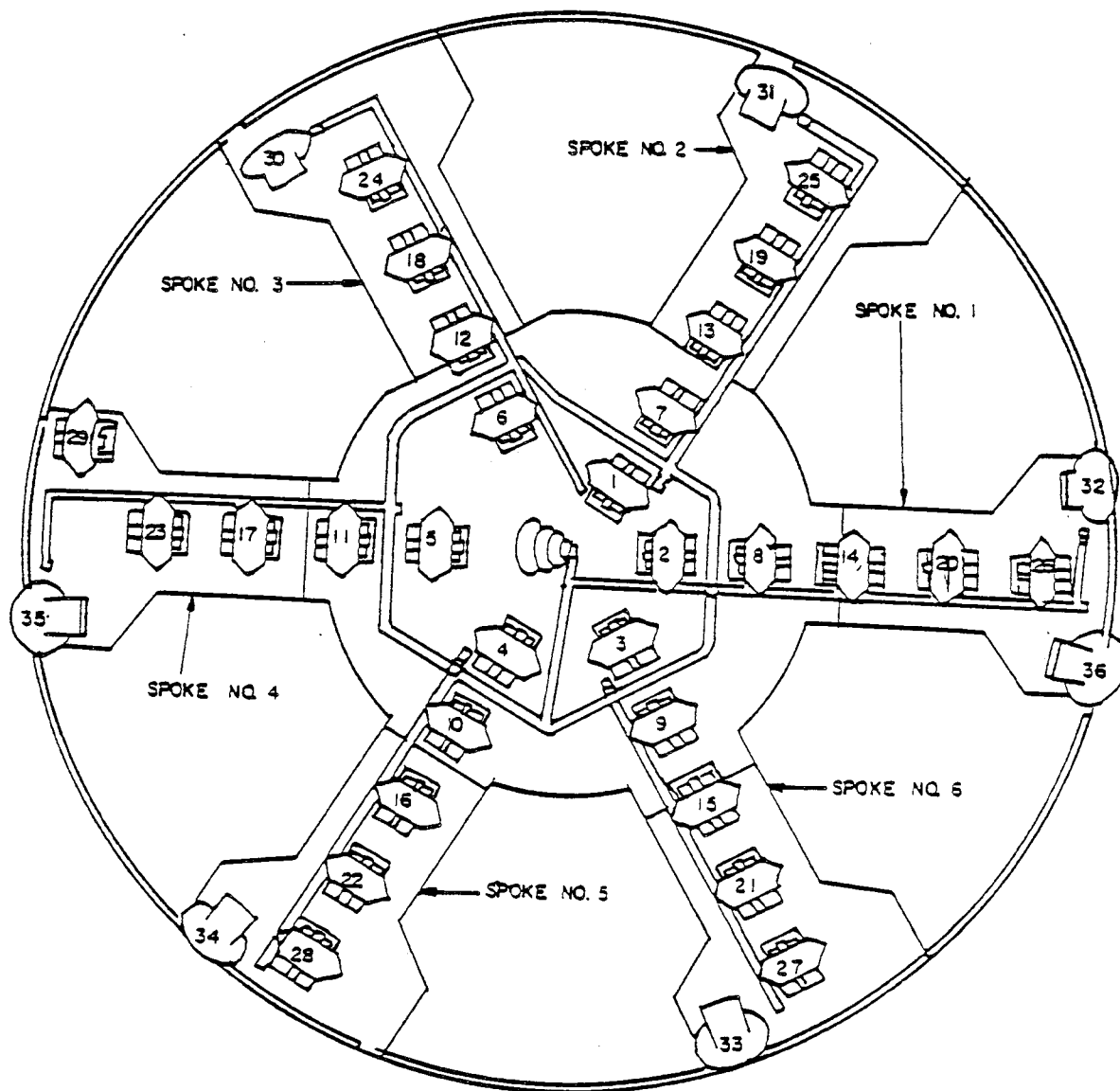
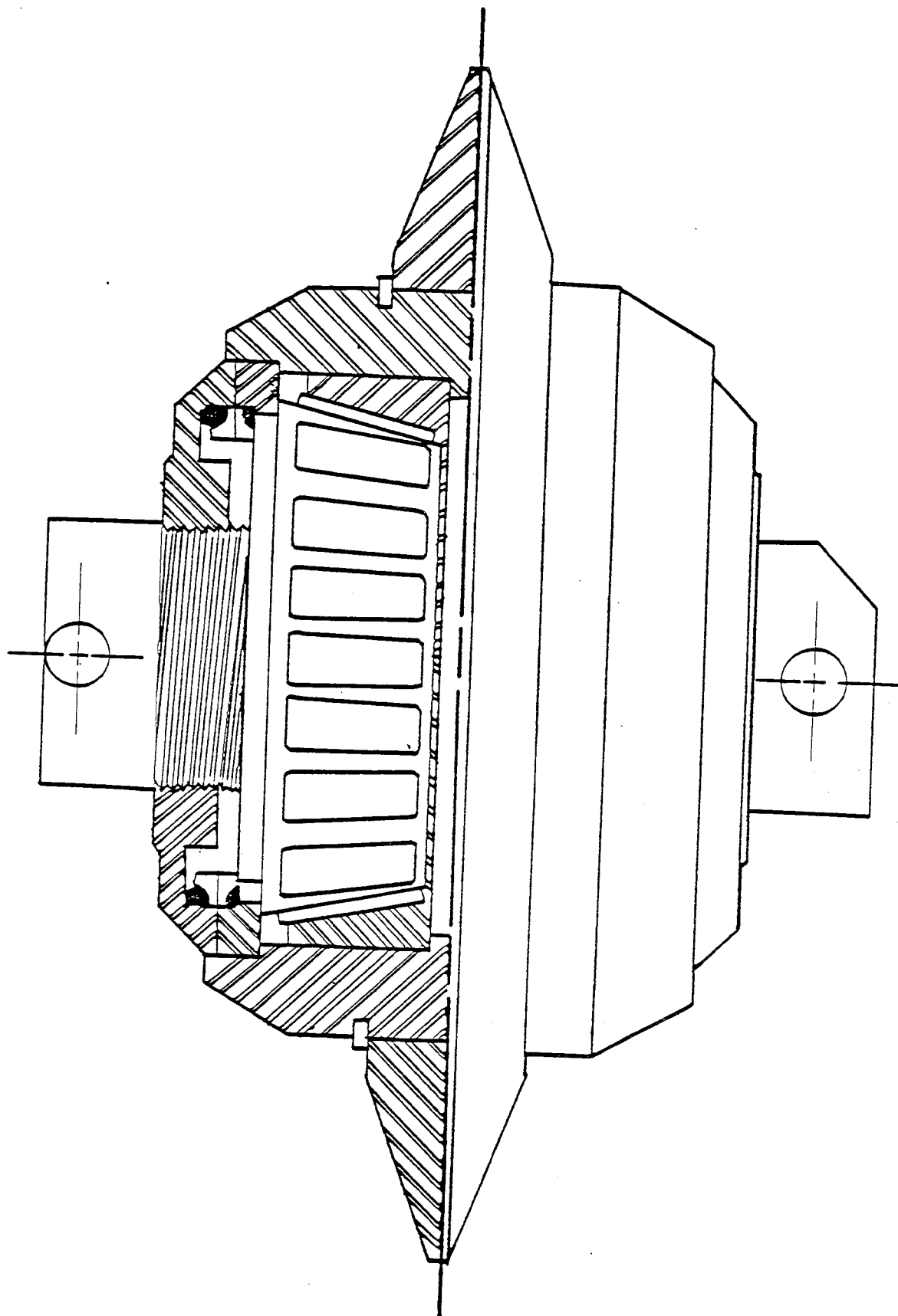


FIGURE 17
Sketch of Cutter Head Showing Cutter Position



15-1/2 Inch Rolling Disc Cutter

FIGURE 18

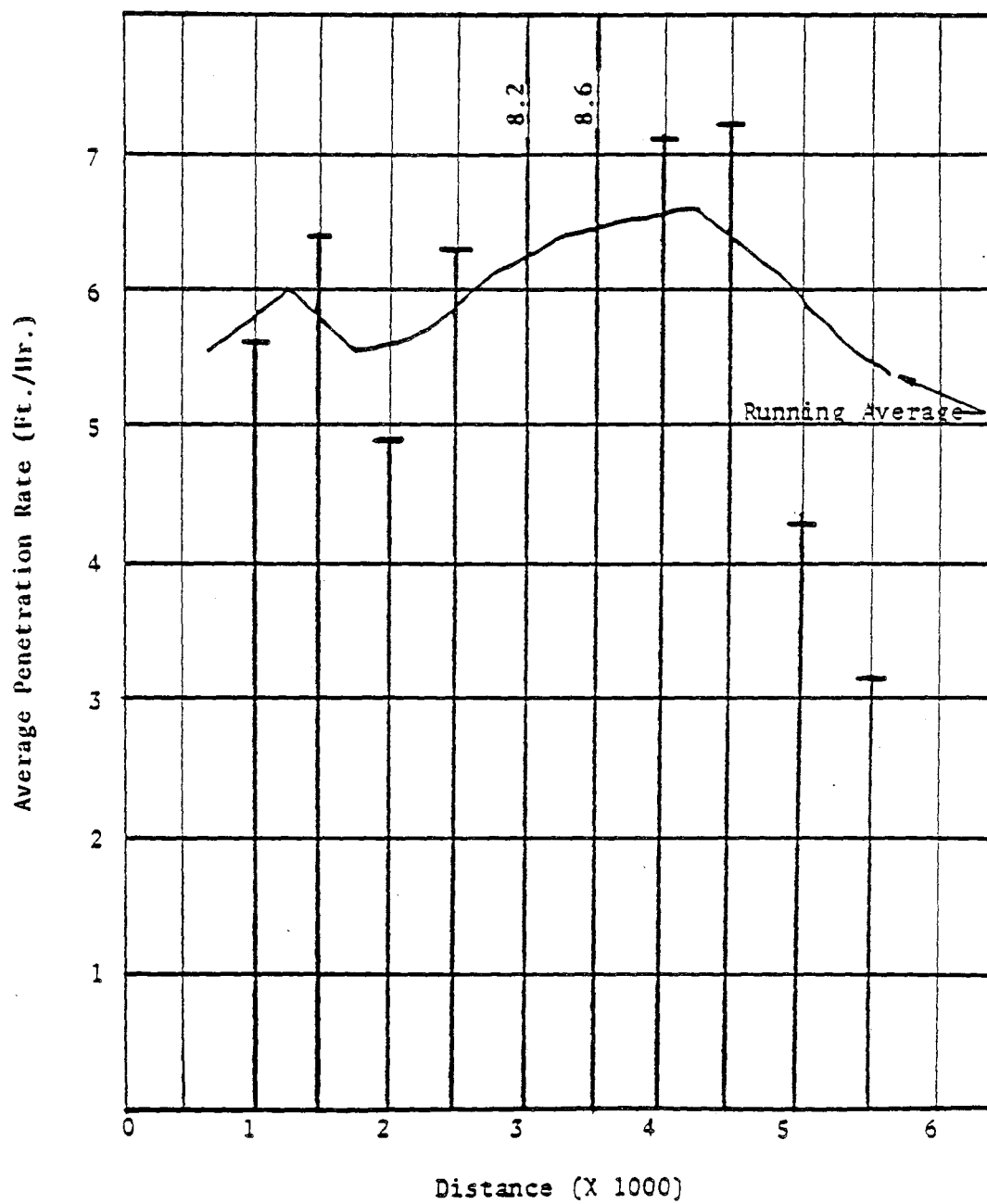


FIGURE 19

Graph of Penetration Rate

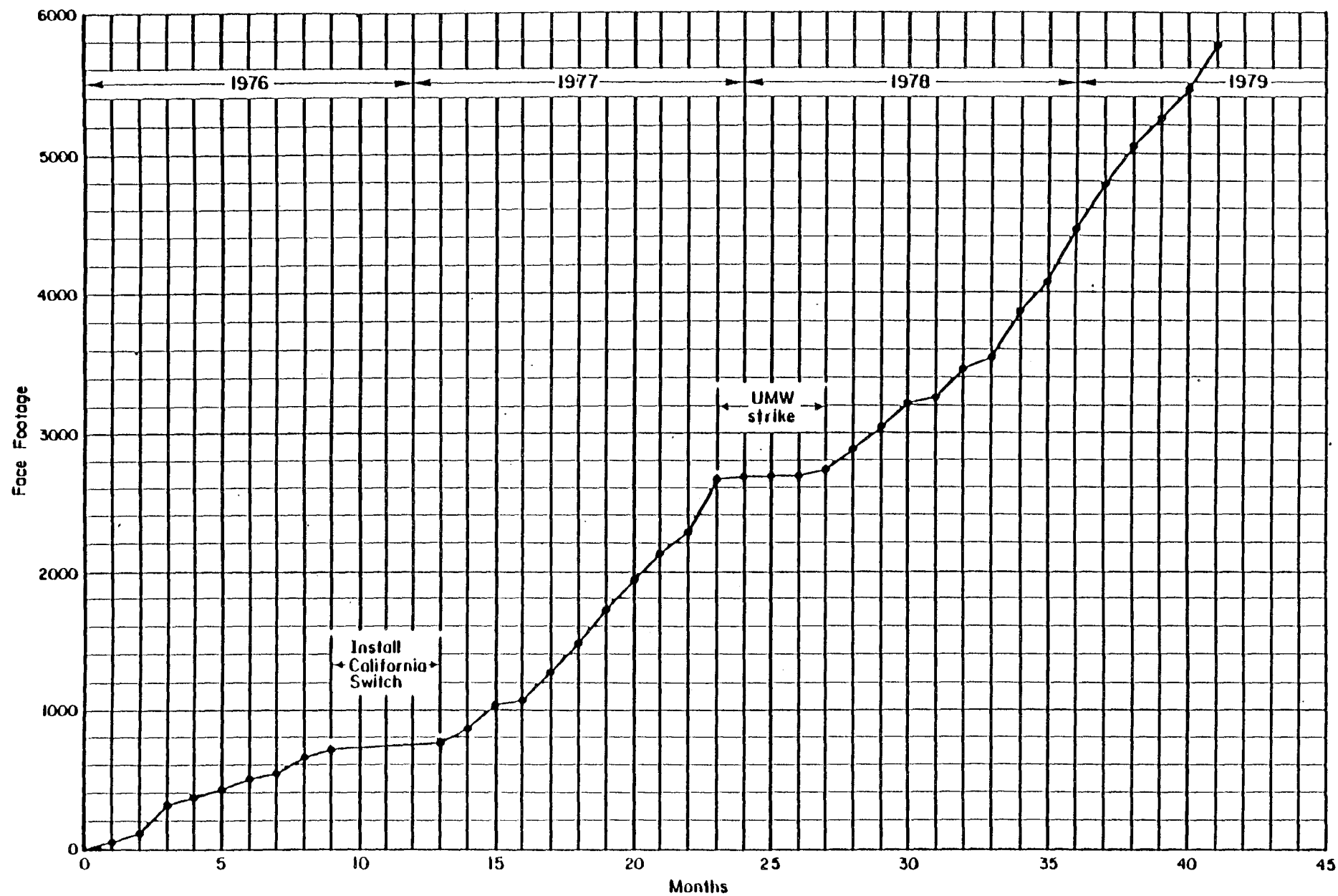


FIGURE 20

Graph of Monthly Progress

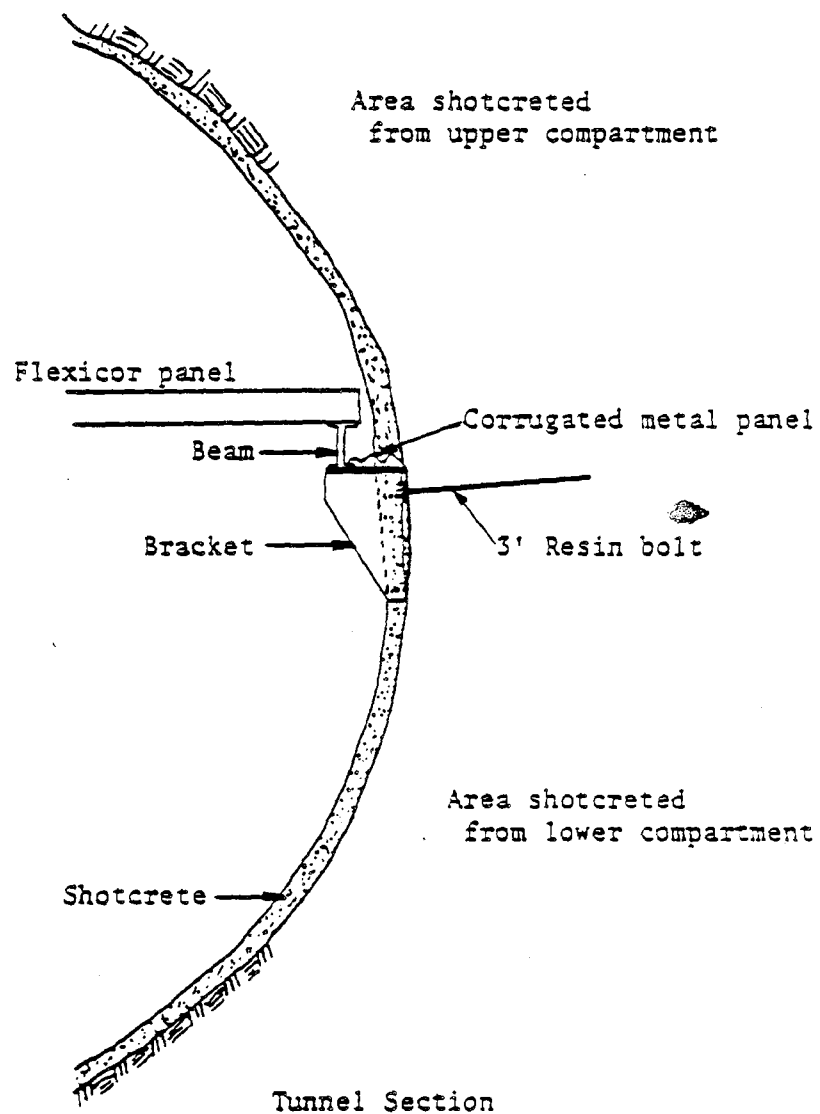


FIGURE 21

Placement of Shotcrete Liner

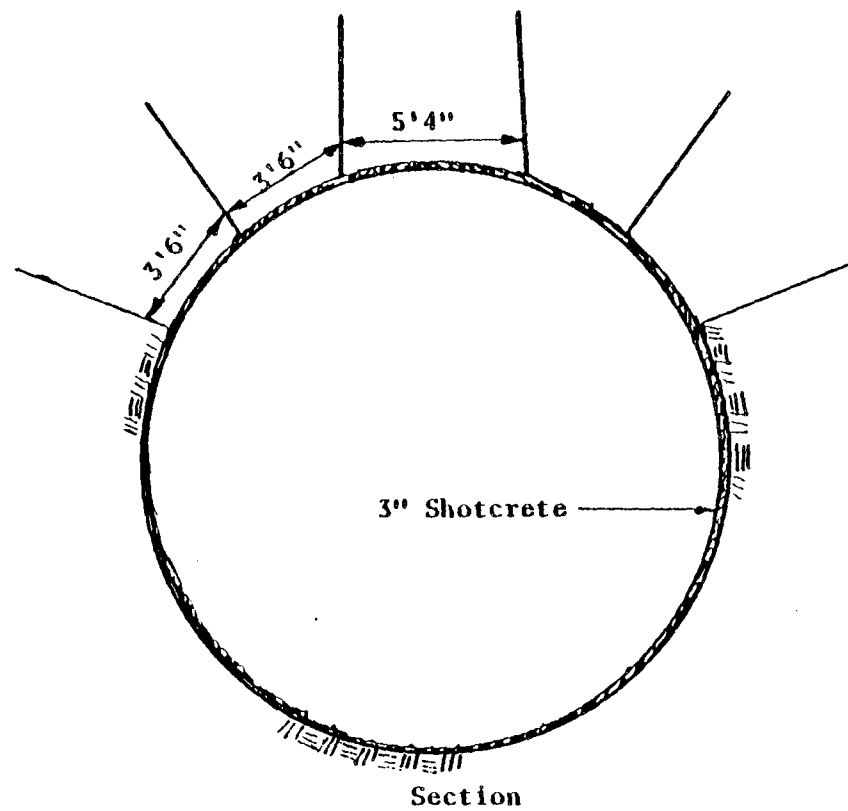
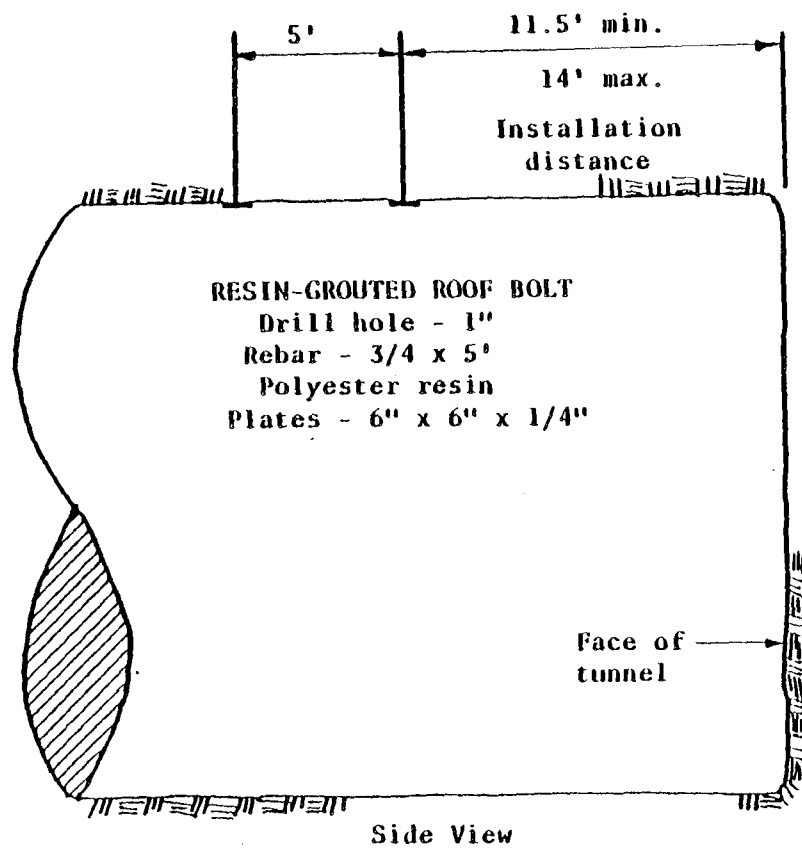


FIGURE 22

Roof Control Plan for TBM Tunnel

APPENDIX A

CORE LOGGING

Each core was handled with extreme care so that the "natural condition as delivered" was preserved as closely as possible. This condition included fractures and moisture. Each core was logged for rock types and simultaneously inspected for fractures (including fracture orientation and size). The overall rock type distribution for each bore hole is shown in Table I.

METHODS USED IN Q QUALITY DETERMINATION AND SUPPORT REQUIREMENTS

A Q Quality log was performed on roof core extending up to 20 feet above the crown of the projected tunnel. It was believed that the roof of an underground opening would become self-supporting after having assumed an arched shape, the height of which was approximately equal to the width of the opening.

Since each bore hole was 1000 feet apart, Q Quality was assessed for each bore hole in order to evaluate its variation along the entire (6000 feet) length of the projected tunnel.

Among the six parameters used in Q Quality determination, the following two were assumed to be constant throughout; J_w (joint water reduction factor) = 1.0 for dry excavation (5A Category) and SRF (stress reduction factor) = 2.0 for high stress, very tight structure (6K Category), because $\sigma_c/\sigma_1 = 5.5 - 8.9$ and $\sigma_2/\sigma_1 = 0.35 - 0.73$ assuming the major (vertical) principal stress due to an overburden height of 750 - 1100 feet averaged 1000 psi. The selection of 6 (b) competent rock, rock stress problems was debatable, because coal mine entries in the Pittsburgh Seam were known to have encountered clay veins which would place SRF under 6 (a) Category. However, due to the lack of any field data in the projected tunnel site, the selection of SRF = 2 seemed to be justified as reasoned.

The other four parameters were determined by careful inspection of the cores.

Table I is the RQD log showing first the overall average and then three separate sections; boring, roof and floor. For Q Quality determination only the roof section is of interest.

The strike and dip of each joint (only those considered as naturally-occurred joints were logged; these joints were characterized by aged surfaces) were determined together with J_r and J_d . The strike of each joint was not true azimuth. It was measured against a fixed point (as the north) determined arbitrarily at the starting point of each bore hole log. The

METHODS USED IN Q QUALITY DETERMINATION AND SUPPORT REQUIREMENTS (Cont'd.)

joints for each bore hole were then plotted on a stereo net and each preferred orientation was defined as a joint set. However, with the exception of Bore Hole No. 1, certain sections of the core run were highly crushed or broken. In this case, J_n were directly determined using the guideline set by Barton et al. This is especially true for Bore Hole No. 3. It should be noted that most of the joints logged were more or less subhorizontal. The drilling of vertical bore holes eliminated the inclusion of vertical or subvertical joints.

Since J_n , J_a and J_r for each joint were not necessarily similar in each bore hole, the most frequently occurring case was defined as the average condition and recommended as the basis for support design. However, there is an uncertainty associated with this design method. This uncertainty will probably be reduced somewhat by knowing the extreme cases, i.e., the best and worst conditions. Judging from the fact that the higher the Q number, the less support required, the best condition will be smaller J_n and J_a with larger J_r . The opposite is true for the worst condition.

Q QUALITY DETERMINATION

$$Q \text{ Quality: } Q = RQD/J_n \times J_r/J_a \times J_w/SRF$$

For Hole No. 1

- a. Average Condition $J_n = 3.0$, $J_r = 1.5$, $J_a = 4.0$
 $Q = 70.1/3.0 \times 1.5/4.0 \times 1.0/2.0 = 4.38$ (fair)
- b. Best Condition $J_n = 3.0$, $J_r = 4.0$, $J_a = 0.75$
 $Q = 70.1/3.0 \times 4.0/0.75 \times 1.0/2.0 = 62.31$ (very good)
- c. Worst Condition $J_n = 3.0$, $J_r = 1.0$, $J_a = 8.0$
 $Q = 70.1/3.0 \times 1.0/8.0 \times 1.0/2.0 = 1.46$ (poor)

For Hole No. 2

- a. Average Condition $J_n = 6.0$, $J_r = 1.0$, $J_a = 3.2$
 $Q = 56.6/6.0 \times 1.0/3.2 \times 1.0/2.0 = 1.47$ (poor)
- b. Best Condition $J_n = 6.0$, $J_r = 3.0$, $J_a = 0.75$
 $Q = 56.6/6.0 \times 3.0/0.75 \times 1.0/2.0 = 18.89$ (good)
- c. Worst Condition $J_n = 20.0$, $J_r = 1.0$, $J_a = 13.0$
 $Q = 56.6/20.0 \times 1.0/13.0 \times 1.0/2.0 = 0.11$ (very poor)

For Hole No. 3

- a. Average Condition $J_n = 15.0$
 $Q = 41.4/15.0 \times 0.5/4.0 \times 1.0/2.0 = 0.17$ (very poor)

Q QUALITY DETERMINATION (Cont'd.)

For Hole No. 3 - Cont'd.

- b. Best Condition $J_n = 20.0, J_r = 0.5, J_a = 12.0$
 $Q = 41.4/6.0 \times 3.0/3.0 \times 1.0/2.0 = 3.45$ (poor)
- c. Worst Condition $J_n = 6.0, J_r = 3.0, J_a = 3.0$
 $Q = 41.4/20 \times 0.5/12.0 \times 1.0/2.0 = 0.04$ (extremely poor)

For Hole No. 4

- a. Average Condition $J_n = 2.0, J_r = 1.0, J_a = 4.0$
 $Q = 48.5/3.0 \times 3.0/3.0 \times 1.0/2.0 = 8.08$ (fair)
- b. Best Condition $J_n = 20.0, J_r = 0.5, J_a = 6.0$
 $Q = 48.5/3.0 \times 3.0/2.0 \times 1.0/2.0 = 12.13$ (good)
- c. Worst Condition $J_n = 20.0, J_r = 3.0, J_a = 2.0$
 $Q = 48.5/20.0 \times 1.0/12.0 \times 1.0/2.0 = 0.10$ (extremely very poor)

For Hole No. 5

- a. Average Condition $J_n = 2.0, J_r = 1.0, J_a = 4.0$
 $Q = 50.1/2.0 \times 1.0/4.0 \times 1.0/2.0 = 3.13$ (poor)
- b. Best Condition $J_n = 20.0, J_r = 0.5, J_a = 6.0$
 $Q = 50.1/2.0 \times 3.0/2.0 \times 1.0/2.0 = 18.79$ (good)
- c. Worst Condition $J_n = 2.0, J_r = 3.0, J_a = 2.0$
 $Q = 50.1/20.0 \times 0.5/6.0 \times 1.0/2.0 = 0.11$ (very poor)

For Hole No. 6

- a. Average Condition $J_n = 20.0, J_r = 3.0, J_a = 1.0$
 $Q = 38.2/20.0 \times 3.0/1.0 \times 1.0/2.0 = 2.87$ (poor)
- b. Best Condition $J_n = 20.0, J_r = 0.5, J_a = 12.0$
 $Q = 38.2/20.0 \times 3.0/1.0 \times 1.0/2.0 = 14.33$ (good)
- c. Worst Condition $J_n = 2.0, J_r = 3.0, J_a = 1.0$
 $Q = 38.2/2.0 \times 0.5/12.0 \times 1.0/2.0 = 0.04$ (extremely poor)

SUPPORT REQUIREMENT

$$\text{Equivalent dimension of the tunnel} = \frac{\text{Diameter of Tunnel}}{\text{ERS}}$$

SUPPORT REQUIREMENT (Cont'd.)

1. For Hole No. 1 area (refer to Table III for support measures of each support category)
 - a. Average Condition - Support Category No. 18
 - b. Best Condition - Support Category No. 9 and no support
 - c. Worst Condition - Support Category No. 23
2. For Hole No. 2 (Table III)
 - a. Average Condition - Support Category No. 23
 - b. Best Condition - Support Category No. 13
 - c. Worst Condition - Support Category No. 32
3. For Hole No. 3 (Table III)
 - a. Average Condition - Support Category No. 31
 - b. Best Condition - Support Category No. 31
 - c. Worst Condition - Support Category No. 35
4. For Hole No. 4 (Table III)
 - a. Average Condition - Support Category No. 18
 - b. Best Condition - Support Category No. 14
 - c. Worst Condition - Support Category No. 32/35
5. For Hole No. 5 (Table III)
 - a. Average Condition - Support Category No. 23
 - b. Best Condition - Support Category No. 13
 - c. Worst Condition - Support Category No. 32
6. For Hole No. 6 (Table III)
 - a. Average Condition - Support Category No. 23
 - b. Best Condition - Support Category No. 14
 - c. Worst Condition - Support Category No. 35

The support requirements determined above are intended for roof support. The wall support requirements could be different. Since wall materials consist of mostly coal, the wall support requirement will probably lean toward the worst condition for each bore hole location.

PHYSICAL PROPERTY DETERMINATION

Specimen Preparation

Specimens were received in the form of NX-sized cores, the length of which varied with bore hole numbers. The last letter in each specimen number was either "C" or "T" which meant for "Compressive Test" or "Indirect Tensile Test", respectively. The selection of specimen location

PHYSICAL PROPERTY DETERMINATION (Cont'd.)

was a compromise between total length of individual rock type and availability of core length (larger than 4.5 inches for Compression Test, for instance) for specific test.

Three types of tests were performed: uniaxial compressive test for uniaxial compressive strength, Brazilian test for indirect tensile strength and Schmidt test for Schmidt hardness index. Uniaxial compressive tests were conducted as defined in ASTM Standard D-2938-71a.

To prepare the specimens, the cores were first cut on both ends to a predetermined length by a standard diamond blade saw used by geologists. Extreme care was exercised during cutting to make sure that both end surfaces are visually parallel, because this will facilitate the subsequent lapping operation in achieving the required parallelism.

The next step was lapping. The specimen was hand-held straight and laid endwise on a lapping wheel rotating at 300 rpm. The wheel was first covered with 120 grit silicone carbide paper at the beginning of the lapping. It was later changed to 240 grit to provide smoother end finish. It generally requires 1-2 hours to achieve an end parallelism of 0.001 inch. The lapping machine is a standard one used by geologists to prepare thin sections or polished sections.

During lapping, the parallelism of end surface of the specimens was frequently checked by a dial gage mounted on a horizontal bar whose support rested on a smooth, flat glass. The specimen was laid on the glass right under the dial gage such that the pin of the dial gage barely contacts the end surface. By moving the specimen diametrically in any direction and checking the movement of the needle, the degree of parallelism along certain directions can be determined in terms of difference in dial gage readings. In real operation, lapping and checking of parallelism go hand-in-hand. The ultimate parallelism of end specimens ranged from 0.001 to 0.020 inch.

Testing Technique

A universal testing machine built by Satec Systems Inc., with a maximum capacity of 120,000 lbs. was used. The testing started by placing the specimens at the center of the lower platen with the top side up and then bringing the lower platen gradually up until the top end surface of the specimen barely contacted the upper platen. A deflectometer with a resolution of 0.0005 inch was set up in such a way that the movement of the lower platen against the upper one could be monitored. The loading rate was regulated by turning the hydraulic valves to a predetermined setting which was calibrated by the manufacturer. But for double-checking the calibration, the testing time of each specimen was monitored by a precision stop watch. The instantaneous load and corresponding deformation were simultaneously recorded in an x-y recorder built into the instrument panel of the machine.

PHYSICAL PROPERTY DETERMINATION (Cont'd.)

Two men were generally required: one was to operate the machine and record the testing time, and the other recorded the characteristics of behaviors, e.g. loading levels for occurrence of first popping noise and chipping, where cracking first initiated, etc., specimen failure was determined by signs of extensive and violent cracking noise with considerable loosening or flying out of fragments from the surfaces.

In the Brazilian tests, only ultimate loads were recorded. The loading rate was approximately similar to uniaxial compressive tests.

Schmidt hardness tests were done for specimens from roof and floor rock immediately before testings. The single reading shown in each specimen was the average of three readings: one from top end surface and two from cylindrical surface.

Calculation

A. Loading rate = $\frac{P/A}{T}$

where

P = total load at specimen failure

A = load bearing area of the specimen end surface

T = total loading time

The loading rates ranged from 160 psi/min to 175 psi/min with an average of 172 psi/min which corresponded to a strain rate of approximately 1.5×10^{-3} /sec. The maximum difference of 6 psi/min in loading rate was insignificant to affect any testing result.

B. Stress = load/load bearing area of the specimen end surface.

C. Strain = deformation/specimen length.

D. Strength

1. Uniaxial compressive strength, $\sigma_c = \frac{P_{max}}{A}$

2. Indirect tensile strength, $\sigma_t = \frac{2 P_{max}}{Dt}$

where

P_{max} = maximum applied load at which specimen fails

D = diameter of the specimen

t = thickness of the specimen

PHYSICAL PROPERTY DETERMINATION (Cont'd.)

E. Young Modulus and Poisson's Ratio.

The ratio of stress to strain along the linear portion of the stress-strain curve of each specimen was defined as the Young Modulus of the specimen. If the stress-strain curve does not have a perfect linear portion, a best fit straight line was plotted along the more or less linear portion of the curve to determine the Young Modulus. The corresponding strain in lateral direction was used for Poisson's Ratio determination

Results

Tables IV to XV show the results of uniaxial compressive, Brazilian and Schmidt tests. Figures 9 to 13 are stress-strain curves for typical rock types tested in uniaxial compressive strength testing.

Considerable variations were encountered for each rock type, especially uniaxial compressive strength. This was attributed partly to specimen variation in texture and/or existence of cracks, and partly to the difficulty of classifying each rock type accurately without resorting to precise chemical analysis. In other words, some rock defined in one hole as limey shale could be very close to shaley limestone in another hole. In some cases, it occurred within the same hole.²

² Peng, S. "Rock Quality and Property Determination, Federal No. 2 Mine", TBM Project, Unpublished Report, January, 1976.

APPENDIX B

FINANCIAL REPORT¹

Government Charges by Subcontractors	1974						1975			
	July	August	September	October	November	December	January	February	March	April
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	9,932	-	-	16,963	-	1,012	-	3,729	8,977	11,357
III. Construction of Tunnel	56,650	-	-	14,988	8,645	17,284	-	12,689	16,990	11,279
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	126,845	99,739	99,127	64,837	87,407	69,756	74,109	117,448	138,170	75,818
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	-	-	-	-	-	-	-	-	-	-
e. Fringes & Supervision	-	-	3,287	7,016	5,834	7,510	17,200	13,950	18,160	19,732
f. G & A Overhead	20,031	14,374	53,260	17,556	9,432	13,477	16,941	22,121	47,639	22,769
VI. Total Before Profit	213,458	114,113	156,374	121,360	111,318	109,039	108,250	169,937	229,936	140,955
VII. Profit Allowance	21,346	11,411	15,637	12,136	11,132	10,904	10,826	16,993	22,994	14,038
VIII. Grand Total	<u>\$234,804</u>	<u>\$125,524</u>	<u>\$172,011</u>	<u>\$133,496</u>	<u>\$122,450</u>	<u>\$119,943</u>	<u>\$119,076</u>	<u>\$186,930</u>	<u>\$252,930</u>	<u>\$154,993</u>

	1975								1976	
	May	June	July	August	September	October	November	December	January	February
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	33,045	11,798	7,594	10,766	8,435	11,208	8,568	9,827	11,597	6,100
III. Construction of Tunnel	1,481	11,101	56,961	18,867	(2,064)	15,292	41,658	31,517	40,770	42,956
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	74,900	94,277	46,648	105,200	76,901	67,409	34,915	54,926	39,578	51,205
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	-	-	-	-	-	-	-	-	-	-
e. Fringes & Supervision	30,048	19,051	55,168	14,353	19,516	23,919	33,044	12,791	20,215	574,550
f. G & A Overhead	19,917	26,968	28,102	23,229	22,734	21,242	17,462	20,724	52,221	19,858
VI. Total Before Profit	159,391	163,195	194,473	172,415	125,522	139,070	135,647	129,785	164,381	717,765
VII. Profit Allowance	15,821	16,319	19,447	16,928	12,794	13,907	13,565	12,972	16,438	13,489
VIII. Grand Total	<u>\$175,212</u>	<u>\$179,514</u>	<u>\$213,920</u>	<u>\$189,343</u>	<u>\$138,316</u>	<u>\$152,977</u>	<u>\$149,212</u>	<u>\$142,764</u>	<u>\$180,819</u>	<u>\$731,254</u>

¹ NOTE: Financial report is preliminary pending final disposition of tunnel machinery.

APPENDIX B

FINANCIAL REPORT

	1976									
	March	April	May	June	July	August	September	October	November	December
Government Charges by Subcontractors										
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	3,268	9,582	4,417	890	1,644	159	223	1,066	2,621	4,151
III. Construction of Tunnel	42,786	60,612	32,774	42,754	19,678	12,708	24,101	25,137	34,177	35,554
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	23,019	30,882	112,972	36,119	34,630	16,906	78,430	68,100	73,332	73,515
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	-	-	-	-	-	-	-	-	-	-
e. Fringes & Supervision	24,934	69,216	37,100	39,621	57,243	24,026	30,499	43,965	41,335	35,294
f. G & A Overhead	13,021	19,185	36,943	8,980	19,371	10,319	15,718	77,626	29,834	32,706
VI. Total Before Profit	<u>107,028</u>	<u>189,477</u>	<u>224,206</u>	<u>128,364</u>	<u>132,566</u>	<u>64,118</u>	<u>148,971</u>	<u>215,894</u>	<u>182,636</u>	<u>181,220</u>
VII. Profit Allowance	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000
VIII. Grand Total	<u>\$132,028</u>	<u>\$214,477</u>	<u>\$249,206</u>	<u>\$153,364</u>	<u>\$157,566</u>	<u>\$ 89,118</u>	<u>\$173,971</u>	<u>\$240,894</u>	<u>\$207,636</u>	<u>\$206,220</u>

	1977									
	January	February	March	April	May	June	July	August	September	October
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	943	1,564	336	777	177	3,846	2,184	1,011	3,447	1,029
III. Construction of Tunnel	40,376	53,694	64,536	44,314	56,671	74,568	70,173	55,132	93,316	57,504
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	34,408	37,689	59,546	63,770	38,057	21,998	38,025	57,069	23,981	47,229
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	613	1,101	2,012	798	1,230	1,297	1,024	20,148	532	6,376
e. Fringes & Supervision	59,453	42,417	39,497	86,148	37,190	85,704	80,877	43,251	57,585	153,554
f. G & A Overhead	29,477	29,126	22,771	23,825	39,823	25,138	30,367	25,288	25,923	28,675
VI. Total Before Profit	<u>165,270</u>	<u>165,591</u>	<u>188,698</u>	<u>219,632</u>	<u>173,148</u>	<u>212,551</u>	<u>222,650</u>	<u>201,899</u>	<u>204,784</u>	<u>294,367</u>
VII. Profit Allowance	25,000	25,000	8,490	-	-	-	-	-	-	-
VIII. Grand Total	<u>\$190,270</u>	<u>\$190,591</u>	<u>\$197,188</u>	<u>\$219,632</u>	<u>\$173,148</u>	<u>\$212,551</u>	<u>\$222,650</u>	<u>\$201,899</u>	<u>\$204,784</u>	<u>\$294,367</u>

APPENDIX B

FINANCIAL REPORT

Government Charges by Subcontractors

	1977 November	December	1978 January	February	March	April	May	June	July	August
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	4,608	485	-	-	-	1,106	1,721	4,130	-	-
III. Construction of Tunnel	87,435	21,817	4,307	260	1,246	50,314	103,153	60,959	47,197	56,752
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	36,405	6,843	-	-	762	15,171	24,230	11,960	24,798	14,738
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	1,428	7,049	1,443	26,069	-	282	26,985	1,913	2,120	-
e. Fringes & Supervision	76,988	46,532	39,861	24,610	44,006	36,923	80,472	99,796	125,497	46,049
f. G & A Overhead	33,527	21,171	53,435	10,358	7,442	19,514	33,760	30,126	148,690	28,685
VI. Total Before Profit	240,391	103,897	99,046	61,297	53,456	123,310	260,321	208,884	348,302	146,224
VII. Profit Allowance	-	-	-	-	-	-	-	-	-	-
VIII. Grand Total	\$240,391	\$103,897	\$99,046	\$61,297	\$53,456	\$123,310	\$260,321	\$208,884	\$348,302	\$146,224

	1978 September	October	November	December	1979 January	February	March	April	May	June
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	-	146	-	-	-	-	-	-	-	-
III. Construction of Tunnel	67,395	75,218	85,572	91,893	80,534	115,323	80,718	81,368	113,468	37,604
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	15,582	22,164	22,371	18,197	8,753	14,249	24,342	10,156	4,581	3,196
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	10,884	384	-	2,625	-	3,162	-	719	873	-
e. Fringes & Supervision	51,672	100,178	61,009	69,970	121,764	67,270	66,352	159,606	84,772	124,511
f. G & A Overhead	11,799	31,435	47,006	28,607	27,294	33,750	24,297	31,207	35,023	30,680
VI. Total Before Profit	157,332	229,525	215,958	211,292	238,345	233,754	195,709	283,056	238,717	195,991
VII. Profit Allowance	164,610	18,290	18,290	18,290	18,290	18,290	18,290	18,290	18,290	18,305
VIII. Grand Total	\$321,942	\$247,815	\$234,248	\$229,582	\$256,635	\$252,044	\$213,999	\$301,346	\$257,007	\$214,296

APPENDIX B

FINANCIAL REPORT

	1979 <u>July</u>	<u>August</u>	<u>September</u>	<u>October</u>	<u>November</u>	<u>Total To Date</u>
Government Charges by Subcontractors						
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	-	-	-	-	-	226,439
III. Construction of Tunnel	7,750	37,158	21,765	4,051	-	2,671,686
IV. Degasification of Tunnel	-	-	-	-	-	-
V. Costs Related to Entire Project						
a. Sub-Contract Indirect	660	344	208	40	-	2,848,642
b. Air Shaft	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-
d. Other	-	-	21,493	6,377	-	724,824
e. Fringes & Supervision	63,054	36,489	84,674	42,451	-	3,184,067
f. G & A Overhead	38,498	18,391	19,143	9,348	-	1,811,297
VI. Total Before Profit	<u>109,962</u>	<u>92,382</u>	<u>147,281</u>	<u>63,067</u>	-	<u>11,466,955</u>
VII. Profit Allowance	-	-	-	-	-	936,829
VIII. Grand Total	<u>\$109,962</u>	<u>\$ 92,382</u>	<u>\$147,281</u>	<u>\$63,067</u>	<u>\$ -</u>	<u>\$12,403,784</u>

APPENDIX B

FINANCIAL REPORT

Government Charges for Additional Work

	1974						1975			
	July	August	September	October	November	December	January	February	March	April
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	-	-	-	-	-	-	-	12,000	-	-
III. Construction of Tunnel	-	-	-	-	-	-	-	-	-	-
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	57,279
V. Costs Related to Entire Project	-	-	-	-	-	-	-	-	-	-
a. Sub-Contract Indirect	-	-	-	-	-	-	-	2,734	330	245
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	-	-	-	-	-	-	-	-	-	-
e. Fringes & Supervision	-	-	3,218	-	-	-	-	-	-	-
f. G & A Overhead	-	-	-	-	-	-	-	-	-	-
VI. Total Before Profit	-	-	3,218	-	-	-	-	14,734	330	57,524
VII. Profit Allowance	-	-	-	-	-	-	-	-	-	-
VIII. Grand Total	\$ -	\$ -	\$ 3,218	\$ -	\$ -	\$ -	\$ -	\$ 14,734	\$ 330	\$ 57,524

	1975								1976	
	May	June	July	August	September	October	November	December	January	February
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	-	-	-	72,523	-	-	-	-	11,961	-
III. Construction of Tunnel	37,048	-	-	-	-	-	-	-	-	-
IV. Degasification of Tunnel	-	-	-	-	-	-	1,586	36,945	69	3,018
V. Costs Related to Entire Project	-	-	-	-	-	-	-	-	-	-
a. Sub-Contract Indirect	435	470	272	381	248	339	309	157	309	211
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	-	12,307	11,996	1,000	11,768	7,836	6,539	-	-	-
e. Fringes & Supervision	-	-	-	13,909	-	-	-	-	2,644	1,610
f. G & A Overhead	-	-	-	-	-	-	-	-	-	-
VI. Total Before Profit	37,483	12,777	12,268	87,813	12,016	8,175	8,434	37,102	14,983	4,847
VII. Profit Allowance	-	-	-	-	-	-	-	-	-	-
VIII. Grand Total	\$ 37,483	\$ 12,777	\$ 12,268	\$ 87,813	\$ 12,016	\$ 8,175	\$ 8,434	\$ 37,102	\$ 14,983	\$ 4,847

APPENDIX B

FINANCIAL REPORT

	1976									
	March	April	May	June	July	August	September	October	November	December
Government Charges for Additional Work										
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	-	-	-	-	-	-	-	-	-	403
III. Construction of Tunnel	3,465	3,180	3,909	3,370	800	1,651	3,171	3,514	3,264	2,857
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	328	239	253	239	157	318	232	157	157	398
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	-	-	539	450	-	-	-	-	-	-
e. Fringes & Supervision	1,856	6,234	2,095	1,806	3,540	885	1,700	1,884	1,750	1,531
f. G & A Overhead	-	-	-	-	-	-	-	-	-	-
VI. Total Before Profit	5,649	9,653	6,796	5,865	4,497	2,854	5,103	5,555	5,171	5,269
VII. Profit Allowance	-	-	-	-	-	-	-	-	-	-
VIII. Grand Total	\$ 5,649	\$ 9,653	\$ 6,796	\$ 5,865	\$ 4,497	\$ 2,854	\$ 5,103	\$ 5,555	\$ 5,171	\$ 5,269

	1977									
	January	February	March	April	May	June	July	August	September	October
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	2,230	-	-	-	-	-	-	-	-	-
III. Construction of Tunnel	3,101	2,987	3,416	2,140	5,303	4,970	4,616	5,321	6,645	5,132
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	230	-	475	157	318	242	270	240	157	240
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	-	-	-	-	-	-	-	-	-	-
e. Fringes & Supervision	2,397	1,601	1,831	1,147	2,842	2,664	2,474	2,645	3,562	2,751
f. G & A Overhead	-	-	-	-	-	-	-	-	-	-
VI. Total Before Profit	7,966	4,588	5,722	3,444	8,463	7,876	7,360	8,206	10,364	8,123
VII. Profit Allowance	-	-	-	-	-	-	-	-	-	-
VIII. Grand Total	\$ 7,966	\$ 4,588	\$ 5,722	\$ 3,444	\$ 8,463	\$ 7,876	\$ 7,360	\$ 8,206	\$ 10,364	\$ 8,123

APPENDIX B

FINANCIAL REPORT

	1977		1978							
	November	December	January	February	March	April	May	June	July	August
Government Charges for Additional Work										
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation										
III. Construction of Tunnel	4,452	1,769	1,860	1,508	90	3,685	3,791	3,894	2,692	400
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	240	288	290	319	244	243	243	243	239	244
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	-	-	-	-	-	-	-	-	-	-
e. Fringes & Supervision	2,386	1,103	1,152	808	48	1,975	2,032	2,087	1,443	214
f. G & A Overhead	-	-	-	-	-	-	-	-	-	-
VI. Total Before Profit	7,078	3,160	3,302	2,635	382	5,903	6,066	6,224	4,374	858
VII. Profit Allowance	-	-	-	-	-	-	-	-	-	-
VIII. Grand Total	\$ 7,078	\$ 3,160	\$ 3,302	\$ 2,635	\$ 382	\$ 5,903	\$ 6,066	\$ 6,224	\$ 4,374	\$ 858

	1978		1979							
	September	October	November	December	January	February	March	April	May	June
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation										
III. Construction of Tunnel	200	470	219	28,482	6,139	5,048	6,122	6,405	6,714	9,483
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	247	244	335	254	344	249	246	157	571	196
b. Air Shaft	-	-	-	-	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-	-	-	-	-
d. Other	-	-	-	-	-	-	-	-	-	-
e. Fringes & Supervision	107	142	107	15,253	3,291	2,706	3,281	3,433	3,597	5,082
f. G & A Overhead	-	-	-	-	-	-	-	-	-	-
VI. Total Before Profit	554	856	661	43,989	9,774	8,003	9,649	9,995	10,882	14,761
VII. Profit Allowance	-	-	-	-	-	-	-	-	-	-
VIII. Grand Total	\$ 554	\$ 856	\$ 661	\$ 43,989	\$ 9,774	\$ 8,003	\$ 9,649	\$ 9,995	\$ 10,882	\$ 14,761

APPENDIX B

FINANCIAL REPORT

	1979 <u>July</u>	<u>August</u>	<u>September</u>	<u>October</u>	<u>November</u>	<u>Total To Date</u>
Government Charges for Additional Work						
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	-	-	-	-	-	99,197
III. Construction of Tunnel	-	-	-	-	-	302,180
IV. Degasification of Tunnel	-	-	-	-	-	-
V. Costs Related to Entire Project						
a. Sub-Contract Indirect	193	-	-	-	-	16,884
b. Air Shaft	-	-	-	-	-	-
c. Mine Support	-	-	-	-	-	-
d. Other	-	-	-	-	-	-
e. Fringes & Supervision	-	-	-	-	-	52,435
f. G & A Overhead	-	-	-	-	-	118,831
VI. Total Before Profit	<u>193</u>	<u>-</u>	<u>-</u>	<u>-</u>	<u>-</u>	<u>-</u>
VII. Profit Allowance	-	-	-	-	-	589,527
VIII. Grand Total	<u>\$ 193</u>	<u>\$ -</u>	<u>\$ -</u>	<u>\$ -</u>	<u>\$ -</u>	<u>\$589,527</u>

APPENDIX B

FINANCIAL REPORT

Eastern Associated Coal Corp. Direct Contribution

	1974						1975		
	July	August	September	October	November	December	January	February	March
I. Site Preparation	\$ -	\$ 4,538	\$ 26,288	\$ 31,635	\$ 10,463	\$ 12,107	\$ 36,393	\$ 24,393	\$ 28,262
II. Tunnel Machine Preparation	-	-	-	1,008	10,725	12,013	31,535	19,694	6,644
III. Construction of Tunnel	-	-	-	-	-	-	-	-	-
IV. Degasification of Tunnel	-	-	-	-	-	-	-	-	-
V. Costs Related to Entire Project	-	-	-	-	-	-	-	-	-
a. Sub-Contract Indirect	-	-	-	-	-	-	-	-	-
b. Air Shaft	-	-	-	-	-	-	-	-	-
c. Mine Support	-	116	3,705	4,223	1,323	1,822	8,542	5,183	9,113
d. Other	-	-	-	-	-	-	-	-	-
e. Fringes & Supervision	-	2,412	14,023	16,410	6,759	11,500	17,294	19,390	16,727
f. G & A Overhead	-	240	1,769	2,130	3,171	1,825	5,122	686	3,530
VI. Total Before Profit	-	7,306	45,785	57,406	32,441	39,267	100,886	69,346	71,852
VII. Profit Allowance	-	-	-	-	-	-	-	-	-
VIII. Grand Total	\$ -	\$ 7,306	\$ 45,785	\$ 57,406	\$ 32,441	\$ 39,267	\$ 100,886	\$ 69,346	\$ 71,852

	1975								
	April	May	June	July	August	September	October	November	December
I. Site Preparation	\$ 41,696	\$ 67,216	\$ 49,320	\$ 37,806	\$ 33,891	\$ 40,409	\$ 33,858	\$ 31,094	\$ 4,552
II. Tunnel Machine Preparation	573	454	1,840	-	(56,292)	-	-	-	457
III. Construction of Tunnel	-	1,948	2,118	549	41,112	15,671	39,504	544	1,399
IV. Degasification of Tunnel	-	8,490	660	-	8,792	1,929	7,040	744	58,289
V. Costs Related to Entire Project	-	-	-	-	-	-	-	-	-
a. Sub-Contract Indirect	-	-	-	-	-	-	-	-	-
b. Air Shaft	4,300	-	19,659	-	61,436	-	1,325	45,662	98,166
c. Mine Support	22,841	18,965	4,945	4,105	30,323	40,598	6,360	37,198	12,237
d. Other	-	-	-	-	-	-	-	-	-
e. Fringes & Supervision	15,530	25,450	21,663	18,645	480	22,800	24,755	16,935	10,451
f. G & A Overhead	20,583	14,728	5,315	27,692	8,776	2,922	8,210	4,081	5,401
VI. Total Before Profit	105,523	137,251	105,528	88,797	130,518	124,329	121,052	136,258	190,552
VII. Profit Allowance	-	-	-	-	-	-	-	-	-
VIII. Grand Total	\$ 105,523	\$ 137,251	\$ 105,528	\$ 88,797	\$ 130,518	\$ 124,329	\$ 121,052	\$ 136,258	\$ 190,552

APPENDIX B

FINANCIAL REPORT

Eastern Associated Coal Corp. Direct Contribution

	1976								
	January	February	March	April	May	June	July	August	September
I. Site Preparation	\$ 14,147	\$ (10,763)	\$ 606	\$ 331	\$ 9	\$ 227	\$ 131	\$ 129	\$ 96
II. Tunnel Machine Preparation	3,451	-	9	1,724	-	1,107	-	-	-
III. Construction of Tunnel	3,083	19,140	3,237	20,866	4,914	6,075	7,550	24,466	1,573
IV. Degasification of Tunnel	78	2,998	1,133	1,664	7,113	15,506	3,884	4,315	7,101
V. Costs Related to Entire Project									
a. Sub-Contract Indirect	-	-	-	-	-	-	-	-	-
b. Air Shaft	74,754	16,407	42,547	32,679	65,416	50,816	52,535	47,188	71,009
c. Mine Support	10,900	3,743	6,025	3,896	3,661	4,869	3,378	4,584	3,389
d. Other	-	-	-	-	-	-	-	-	-
e. Fringes & Supervision	10,383	6,537	7,276	7,310	10,646	10,370	5,924	8,711	7,770
f. G & A Overhead	3,580	7,366	4,729	3,745	4,236	4,866	3,431	3,821	3,617
VI. Total Before Profit	120,376	45,428	65,562	72,215	95,995	93,836	76,833	93,214	94,555
VII. Profit Allowance	-	-	-	-	-	-	-	-	-
VIII. Grand Total	\$120,376	\$ 45,428	\$ 65,562	\$ 72,215	\$ 95,995	\$ 93,836	\$ 76,833	\$ 93,214	\$ 94,555

	1976				1977					
	October	November	December	January	February	March	April	May	June	
I. Site Preparation	\$ 65	\$ 129	\$ 65	\$ -	\$ 129	\$ 1,600	\$ -	\$ 2	\$ 260	
II. Tunnel Machine Preparation	693	1,105	633	800	1,482	1,210	499	490	4,273	
III. Construction of Tunnel	1,464	1,531	1,668	1,464	1,468	1,732	1,722	1,976	1,706	
IV. Degasification of Tunnel	6,269	10,179	783	5,687	5,884	6,505	5,103	11,571	7,864	
V. Costs Related to Entire Project										
a. Sub-Contract Indirect	-	-	-	-	-	-	-	-	-	
b. Air Shaft	52,432	76,543	70,816	34,229	62,301	61,207	44,302	20,701	25,429	
c. Mine Support	3,831	3,069	3,205	4,538	5,372	6,692	3,390	7,310	7,576	
d. Other	-	-	-	-	-	-	-	-	-	
e. Fringes & Supervision	8,813	7,225	6,214	10,044	9,774	8,993	9,054	12,304	14,060	
f. G & A Overhead	3,637	4,469	3,500	2,001	5,471	4,051	3,227	3,603	3,050	
VI. Total Before Profit	77,204	104,250	86,884	58,763	91,881	91,990	67,297	57,957	64,218	
VII. Profit Allowance	-	-	-	-	-	-	-	-	-	
VIII. Grand Total	\$ 77,204	\$104,250	\$ 86,884	\$ 58,763	\$ 91,881	\$ 91,990	\$ 67,297	\$ 57,957	\$ 64,218	

APPENDIX B

FINANCIAL REPORT

	1977						1978	
	July	August	September	October	November	December	January	February
Eastern Associated Coal Corp. Direct Contribution								
I. Site Preparation	\$ 65	\$ 138	\$ 324	\$ 260	\$ -	\$ -	\$ -	\$ -
II. Tunnel Machine Preparation	2,324	-	-	-	-	-	-	-
III. Construction of Tunnel	1,955	2,085	1,663	1,548	158,108	28,369	1,448	1,264
IV. Degasification of Tunnel	6,382	7,684	5,906	11,382	6,403	2,046	-	-
V. Costs Related to Entire Project								
a. Sub-Contract Indirect	-	-	-	-	-	-	-	-
b. Air Shaft	16,690	45,284	26,066	32,009	43,324	26,501	-	-
c. Mine Support	6,702	7,098	6,861	5,652	9,639	2,457	2,194	1,949
d. Other	-	-	-	-	-	-	-	-
e. Fringes & Supervision	11,579	8,897	10,581	11,347	10,911	5,491	5,293	4,673
f. G & A Overhead	3,199	3,529	3,085	3,602	9,053	5,790	4,043	2,236
VI. Total Before Profit	48,896	74,715	55,486	65,800	237,438	70,654	12,978	10,122
VII. Profit Allowance	-	-	-	-	-	-	-	-
VIII. Grand Total	<u>\$ 48,896</u>	<u>\$ 74,715</u>	<u>\$ 55,486</u>	<u>\$ 65,800</u>	<u>\$ 237,438</u>	<u>\$ 70,654</u>	<u>\$ 12,978</u>	<u>\$ 10,122</u>

	1978							
	March	April	May	June	July	August	September	October
I. Site Preparation	\$ -	\$ -	\$ 194	\$ 649	\$ 260	\$ 194	\$ 194	\$ 248
II. Tunnel Machine Preparation	-	-	-	-	-	-	-	-
III. Construction of Tunnel	1,681	3,516	1,499	1,700	1,630	1,607	3,219	2,442
IV. Degasification of Tunnel	2,269	13,228	12,973	9,289	6,117	2,442	10,164	7,670
V. Costs Related to Entire Project								
a. Sub-Contract Indirect	-	-	-	-	-	-	-	-
b. Air Shaft	-	-	-	-	-	-	-	-
c. Mine Support	3,581	6,097	10,145	3,859	2,527	6,800	5,673	10,906
d. Other	-	-	-	-	-	-	-	-
e. Fringes & Supervision	7,121	14,637	13,711	10,513	5,535	7,329	9,687	11,867
f. G & A Overhead	3,238	3,464	4,244	6,051	7,575	10,715	54,448	21,340
VI. Total Before Profit	17,890	40,942	42,766	32,061	23,644	29,087	83,385	54,473
VII. Profit Allowance	-	-	-	-	-	-	-	-
VIII. Grand Total	<u>\$ 17,890</u>	<u>\$ 40,942</u>	<u>\$ 42,766</u>	<u>\$ 32,061</u>	<u>\$ 23,644</u>	<u>\$ 29,087</u>	<u>\$ 83,385</u>	<u>\$ 54,473</u>

APPENDIX B

FINANCIAL REPORT

Eastern Associated Coal Corp. Direct Contribution

	1978 November	December	1979 January	February	March	April	May	June
I. Site Preparation	\$ 324	\$ 324	\$ 917	\$ 553	\$ 825	\$ 399	\$ 334	\$ 134
II. Tunnel Machine Preparation	-	-	-	-	-	-	-	-
III. Construction of Tunnel	10,024	2,026	1,913	2,101	15,370	1,970	14,406	8,946
IV. Degasification of Tunnel	12,845	13,838	11,488	10,859	12,360	8,914	15,558	9,262
V. Costs Related to Entire Project								
a. Sub-Contract Indirect	-	-	-	-	-	-	-	-
b. Air Shaft	-	-	-	-	-	-	-	-
c. Mine Support	10,187	10,721	12,929	8,538	10,707	10,295	13,817	3,548
d. Other	-	-	-	-	-	-	-	-
e. Fringes & Supervision	13,539	15,323	14,768	12,751	15,783	12,674	14,023	11,509
f. G & A Overhead	10,392	25,936	13,920	22,538	20,658	17,393	17,068	12,081
VI. Total Before Profit	57,311	68,168	55,935	57,340	75,703	51,645	75,206	45,480
VII. Profit Allowance	-	-	-	677	2,955	10,747	6,173	213
VIII. Grand Total	\$ 57,311	\$ 68,168	\$ 55,935	\$ 58,017	\$ 78,658	\$ 62,392	\$ 81,379	\$ 45,693

	1979 July	August	September	October	November	Total To Date
I. Site Preparation	\$ -	\$ -	\$ -	\$ -	\$ -	\$ 529,420
II. Tunnel Machine Preparation	-	-	5,130	6,172	387	62,140
III. Construction of Tunnel	6,638	8,617	6,026	1,726	-	501,977
IV. Degasification of Tunnel	9,672	10,138	252	-	-	408,722
V. Costs Related to Entire Project						
a. Sub-Contract Indirect	-	-	-	-	-	-
b. Air Shaft	-	-	-	-	-	-
c. Mine Support	207	8,325	2,954	291	-	1,333,846
d. Other	-	-	-	-	-	484,149
e. Fringes & Supervision	5,474	19,288	12,698	5,397	256	725,287
f. G & A Overhead	13,122	8,477	11,695	6,053	2,678	516,204
VI. Total Before Profit	35,133	54,845	38,755	19,639	3,321	4,561,745
VII. Profit Allowance (Paid to subcontractor by EACC)	-	-	-	-	-	20,765
VIII. Grand Total	\$ 35,133	\$ 54,845	\$ 38,755	\$ 19,639	\$ 3,321	\$ 4,582,510

NOTE: Final costs are not complete pending certain governmental decisions.

APPENDIX B

FINANCIAL REPORT

Explanation of Eastern Associated Coal Corp. direct contribution.

- I. Provide labor, materials, and equipment necessary to furnish 50,000 CFM of fresh air to the project and track facilities to deliver and store supplies for the project and remove muck from the project.
- II. Provide labor, materials, and equipment to construct a stall for the assembly of the boring machine, deliver the machine from the surface to the job site, and fire bossing during the excavation of the stall and machine assembly period.
- III. Provide labor, materials, and equipment to remove muck during the boring period from the job site to the rotary dump including track materials necessary to install track haulage in the tunnel during boring.
- IV. Provide labor, materials, and equipment to degasify the tunnel area as boring progressed including a gas collection system.
- V. Provide labor, materials, and equipment as necessary for mine support services including, but not limited to, engineering, power, bathhouse facilities, research and development, supply crews, project manager, pumps-pressure and dewater, drainage pipe and fittings, water pipe and fittings, air pipe and fittings, and electric and telephone cable.
- VI. Self-explanatory.
- VII. Profit allowance cost incurred in subcontract not reimbursable in the contract.
- VIII. Self-explanatory.

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